TECHNICAL REPORT ON THE MINERAL RESOURCES FOR THE KONKOLA NORTH COPPER PROJECT, CHILILABOMBWE, ZAMBIA

Report Prepared for

TEAL EXPLORATION & MINING INCORPORATED

Report No 384408

March 2008

SRK Consulting

Engineers and Scientists

TECHNICAL REPORT ON THE MINERAL RESOURCES FOR THE KONKOLA NORTH COPPER PROJECT, CHILILABOMBWE, ZAMBIA

TEAL EXPLORATION & MINING INCORPORATED

SRK Project Number 384408

SRK Consulting
265 Oxford Road
Illovo
2196
South Africa

P O Box 55291
Northlands
2116
South Africa

Tel: (011) 441-1111

Fax: (011) 880-8086

Victor Simposya vsimposya@srk.co.za

March 2008

Authors:
V M Simposya (Principal Geologist and Partner-SRK)
P C L Hart (Process Engineer DRA Mineral Projects)

Reviewed by:
HG Waldeck
Partner

Executive Summary

TEAL Exploration & Mining Incorporated (“TEAL”) approached SRK Consulting South Africa for an independent technical review of the Mineral Resources and for the compilation of an independent Technical Report which conforms to the requirements of the Canadian Security Administrators National Instrument 43-101 (“NI 43-101”) for the Konkola North Copper Project (“KNP”).

The KNP is located within the greater Konkola area of the Zambian Copperbelt, close to the village of Konkola, near the town of Chililabombwe, Copperbelt Province, Republic of Zambia. The KNP consists of a large scale mining licence (LML 20) covering an area of approximately 44 square kilometres.

Copper mineralisation at KNP was discovered in 1924, and between 1936-38 a total of 26 boreholes for 6,640 meters were drilled by Bancroft Mines Limited the first registered owner of the project. In 1953 Bancroft Mines Limited sank the No. 2 shaft to a depth of 423 meters and completed two ventilation shafts and three development levels into the South Limb deposit. In 1959 the mine was put into care and maintenance due to depressed copper prices. Between 1967 and 1978 a further 10,900 meters were drilled by Anglo American Corporation and Zambia Consolidated Copper Mines. Following the nationalization of the Zambian copper mines in 1972, the Government of Zambia redrafted legislation and invited interested parties to participate in the privatization of its mines and resources in 1995. As part of the government’s privatization efforts, TEAL acquired the rights to the Konkola North Project in 1997.

Konnoco (Zambia) Limited, an indirectly held, wholly owned subsidiary company of TEAL, holds the large scale mining licence for the Konkola North Copper Project. TEAL currently has 100% ownership of the KNP with ZCCM Investment Holdings plc, a company controlled by the Zambian government, retaining the option of buying a stake of up to 20%, of which 5% is a ‘free-carry’.

The copper mineralisation at the KNP is hosted in the OS1 Member (“Ore Shale”), a siltstone/shale unit. The zones of copper mineralisation at the KNP have been subdivided into three (previously five) different zones, namely the East Limb, the South Limb and Area ‘A’ (previously also Area ‘A’ Extension and Area ‘B’). The copper mineralisation is characterised by essentially two different types, namely chalcocite-rich (East Limb) and chalcopyrite-bornite rich (South Limb and Area A), and variable amounts of malachite and other copper minerals occurs throughout.

A total of 65,230m of diamond drilling has been undertaken on the KNP, which is inclusive of the 6,640 meters drilled by Bancroft Mines Limited, 10,900 meters drilled by Anglo American Corporation and Zambia Consolidated Copper Mines (“ZCCM”) and the 47,685.55m by TEAL undertaken during 1997 to 1998.

TEAL is presently in the process of concluding a Feasibility Study focussed on the exploitation and processing of copper ores from the exiting shaft infrastructure by mining the South and East Limb mineralised zones.

A geologist from SRK, the Independent Qualified Person, visited the site during the period 3rd – 5th October 2007 and was shown exposures of mineralised zones in open stopes and underground haulage developments as well as collar positions of selected holes drilled from the underground mining haulages.

The QP also examined the drill cores from selected holes drilled under the TEAL programme and visited the core shed, the repository of all the KNP drillholes and examined the sample preparation facilities that were used in the preparation of the pulp samples prior to shipment to the laboratories.

The sample preparation process and sample dispatch system in place during the drilling programme was also explained.

SRK has reviewed the technical studies undertaken on the KNP leading to the estimation of Mineral Resources. SRK has found the work undertaken to be of acceptable quality to allow for the classification of Mineral Resources according to the guidelines and definitions of The Samrec Code.

SRK has examined the distributions of the composite data against the block model estimates and found that generally there is good correspondence between the estimates and the composite data in areas where there is adequate drillhole coverage, but is less so is sparsely drilled areas.

SRK has reviewed the data and grade distribution, the spatial continuity, bulk density and the QA/QC results with respect to the categorisation of the Mineral Resources.

SRK has not re-estimated the Mineral Resource grades, but reviewed the process leading up to the estimates and consider the work done to be satisfactory. SRK has slightly modified the previous classification of the Mineral Resources by African Rainbow Minerals Limited, taking into account the foregoing. Table 1 presents SRK's audited Mineral Resources.

Table 1: Audited Mineral Resources for KNP, as of February 2008

Classification		**Measured**			**Indicated**			**Inferred**		
Resource Area	**Sub-area**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**
South Limb	**Block 1**	10.0	2.23	0.60	-	-	-	-	-	-
	Block 3	-	-	-	22.2	2.13	0.26	-	-	-
	Block 5	-	-	-	-	-	-	16.2	2.22	0.25
Sub-Total		**10.0**	**2.23**	**0.60**	**22.2**	**2.13**	**0.26**	**16.2**	**2.22**	**0.25**
East Limb	**Block 2**	7.1	2.34	1.07	-	-	-	-	-	-
	Block 4	-	-	-	11.7	2.87	0.39	-	-	-
	Block 7	-	-	-	-	-	-	10.7	2.83	0.41
Sub-Total		**7.1**	**2.34**	**1.07**	**11.7**	**2.87**	**0.39**	**10.7**	**2.83**	**0.41**
Area A	**Block 6**	-	-	-	-	-	-	219.5	2.64	1.09
Sub-Total		**-**	**-**	**-**	**0.0**	**0.00**	**0.00**	**219.5**	**2.64**	**1.09**
Total by Category		**17.1**	**2.28**	**0.80**	**33.8**	**2.38**	**0.30**	**246.4**	**2.62**	**1.01**
Total Measured and Indicated		**51.0**	**2.35**	**0.47**						

SRK's has noted that dense drilling highlights the local grade variability inherent in the project. SRK therefore recommends that the level of drilling be increased to provide coverage to relatively the same detail as shown in the densely drilled areas of the South and East Limbs.

Table of Contents

List of Tables

List of Figures

Peter Clive Lyon Hart

SRK Consulting
Engineers and Scientists

SRK House
265 Oxford Road, Illovo
Johannesburg 2196

P O Box 55291
Northlands 2116
South Africa

e-Mail: johannesburg@srk.co.za
URL: http://www.srk.co.za

Tel: +27 (0) 11 441 1111
Fax:+27 (0) 11 880 8086

5th March 2008

384408

TECHNICAL REPORT ON THE MINERAL RESOURCES FOR THE KONKOLA NORTH COPPER PPROJECT, CHILILABOMBWE, ZAMBIA

1 Introduction and Terms of Reference

TEAL Exploration & Mining Incorporated ("TEAL") approached SRK Consulting South Africa (Pty) Limited ("SRK") for an independent technical review of the Mineral Resources and for the compilation of an independent Technical Report which conforms to the requirements of the Canadian Security Administrators National Instrument 43-101 ("NI 43-101") for the Konkola North Copper Project ("KNP"). The KNP is located near the town of Chililabombwe in Zambia.

TEAL is registered on the Toronto Stock Exchange and the KNP was part of listing document, entitled "Independent Technical Report on the Central African Mining Properties of TEAL Exploration & Mining Incorporated" generated by RSG Global dated August 2005 and filed on www.sedar.com. Since the listing, TEAL has also undertaken a secondary listing on the JSE Limited.

TEAL is in the process of undertaking a technical assessment of the KNP to the level and detail of a feasibility study and have generated Mineral Resources for the project which is the subject of this review.

1.1 Sources of information and data

The information contained in this report with respect to the Mineral Resources estimates is based on information provided to SRK by TEAL and is inclusive of drillhole data, resource models and third party technical reports prepared by independent contractors or consultant, who are listed in the Reference section of this report.

1.2 Personal Inspection

A site visit to the KNP site was undertaken by the Qualified Person ("QP"), a geologist from SRK, during the period 3rd – 5th October 2007. The QP was taken on an underground tour of the No.2 Shaft and observed exposures of mineralised zones in open stopes and underground haulage developments. The QP was also shown the collar positions of selected holes drilled to intersect the mineralised zones at various locations within the development haulages.

The QP also examined the drill cores from selected holes drilled under the TEAL programme and visited the core shed, the repository of all the KNP drillholes and examined the sample preparation facilities that were used in the preparation of the pulp samples prior to shipment to the laboratories.

The sample preparation process and sample dispatch system in place during the drilling programme was also explained.

1.3 Qualifications and Independence

The SRK Group comprises approximately 600 staff, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership largely rests with its staff. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues.

The SRK Group has a demonstrated track record in undertaking independent mineral property valuations, resources and reserves estimations, project evaluations and audits, Competent Person's Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining operations and projects providing mining industry consultancy service inputs and has specific experience in transactions of this nature.

Victor Simposya has more than 28 years experience in the mining and exploration industry throughout Africa and other international locations, with particular specialisation in Mineral Resource estimation. He is a Principal Geologist and Partner of SRK, a Member of the SAIMM and a registered Professional Earth Scientist (Pr.Sci.Nat. Reg. No. 400352/03) with the statutory body South African Council for Natural Scientific Professions. Mr Simposya qualifies as a QP as defined in NI 43-101.

Neither the author nor SRK have any material interest, either direct, indirect or contingent in TEAL, nor in the mineral properties included in this report, nor in any other TEAL asset, nor has any interest existed in the past. Fees for the preparation of this report are being charged at commercial rates. Payment of these fees is in no way contingent upon the conclusions reached in the report.

2 Disclaimer

The observations and comments presented in the report represent SRK's opinion as of March 2008 and are based on personal inspection, discussion with TEAL and its consultants and the review of the data and reports provided to SRK.

Whilst exercising all reasonable diligence in checking, confirming and testing this information SRK, in formulating its opinion, have accepted that the data presented by TEAL is reasonable. Also SRK has endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the third party technical data upon which this report is based. However, SRK does not warrant the authenticity or completeness of any such third party information.

Further, the SRK scope did not allow for a due diligence on the legal aspects of the property and the metallurgical and mineral processing aspects of the project, studies of which are still in progress. The ownership, agreements, environmental and permitting and all the information presented in this report have been provided by TEAL. SRK has not verified or confirmed any of this information.

Assessments of metallurgical and mineral processing aspects of the project (included in Section 13) have been signed off by Peter C L Hart (Process Director DRA Process Engineers) DRA Mineral Projects (Pty) Ltd of South Africa, who has acted as a consulting metallurgist to KNP.

3 Property description and Location

The KNP is situated in the northern Copperbelt Province, Republic of Zambia, adjacent to the border with Democratic Republic of Congo ("DRC") (Figure 3.1). The property lies 300 km north of Lusaka, and immediately north of the town of Chililabombwe. Konnoco (Zambia) Limited ("Konnoco") acquired a Large Scale Mining Licence (LML 20) on 30 May 1997, which confers exclusive mining rights to Konnoco for a period of 23 years for the elements copper, cobalt , gold, silver , selenium , tellurium and sulphur. LML20 covers an area of 44 square kilometres. The surface rights of the area were acquired by the Konnoco.

A vertical shaft, the No.2 shaft, is situated on the licence LML 20. The shaft is to a depth of 423 meters, and associated infrastructure includes two ventilation shafts and three development haulages into the ore body with numerous sub-haulages on the various levels.

TEAL is presently in the process of concluding a Feasibility Study focussed on the exploitation and processing of ores from the exiting shaft infrastructure by mining the South and East Limb mineralised zones.

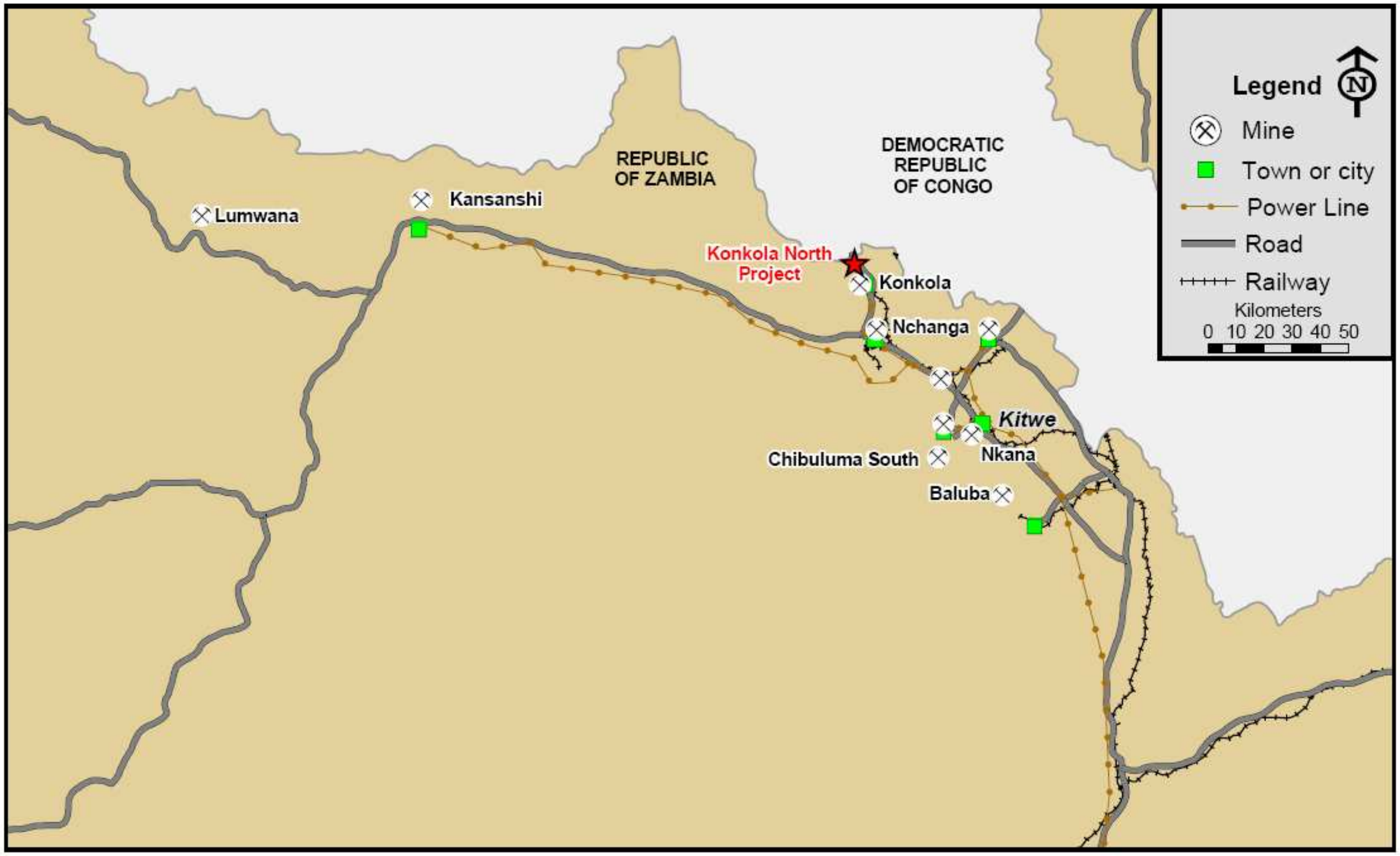

Figure 3.1: Map of the Zambian Copperbelt showing the location of the KNP

The KNP area is separated from the adjacent LML 3 held by Konkola Division of Konkola Copper Mines Plc ("KCM") along the local KCM 40 000 m grid line. The following beacons (UTM 27°) define the 40 000 m boundary line:

KN1 (W beacon):	+583 515.417 mE;	+8 636 030.510 mN
KN2 (E beacon):	+592 844.061 mE;	+8 636 004.500 mN

The remaining LML 20 area and beacon co-ordinates are as previously defined for Konkola Division Bancroft ML7.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the KNP is via paved road which connects the village of Konkola through Chililabombwe in Zambia to Kasumbalesa in the DRC. The Chililabombwe paved road connects the KNP to the towns within the Zambian Copperbelt, to Lusaka, the capital city of Zambia and beyond that to the ports in South Africa, Namibia and Tanzania. The Zambian Copperbelt towns are also on the Zambian railway network which provides rail link to the ports in South Africa and Dar es Salaam in Tanzania.

The village of Konkola is serviced with electricity connections and a 11 kilovolt substation is located next to the No.2 Shaft. Potable water is pumped from the water reservoir at Chililabombwe to Konkola. In addition, a water treatment plant is situated close to the project. The Konkola North Copper Project is situated some 12 kilometers north of the operating Konkola Copper Mine adjacent to the town of Chililabombwe which possess all necessary utilities to support a major mining operation.

The KNP is located in the northern portion of the country where there are generally two seasons, a wet season from November to March and a dry season from April to October. The wet season is characterised by heavy downpours of rain sometimes over short or over prolonged periods of time. The earlier part of the dry season, is characterised by lower temperatures from May through July (2^0 C to 27^0 C) becoming warmer during August to October.

Typical elevations within the project area are of the order of 1350m amsl.

5 History

Copper mineralisation was first discovered in the present Konkola North area by the prospector James Williams in 1924. Williams traced malachite-stained outcrops in the vicinity of the

Konkola Dome. No significance was attached to the discovery for another four years, until, under the systematic prospecting campaign initiated by Dr J. Austen Bancroft, the mineralisation was followed up and correlated with that part of the geological sequence known from other parts of the Copperbelt to contain the main copper-bearing formations. This initial investigation of the deposit seemed disappointing, however, and work was abandoned during the 1930s' depression. At the same time, Belgian geologists, following up on the mineralisation across the border, in the current DRC, outlined a substantial orebody at the town of Musoshi. Work resumed on the Zambian side in 1936, and, between 1936 and 1938, the mineralised ore shale was outlined in an initial drilling programme, amounting to 26 drill holes totalling 6,640 m. The No. 2 Shaft was sunk 16 years later in 1954, and the first copper was produced in July 1957. Owing to the prevailing low copper price at the time, production was halted in March 1958 and the mine put on care and maintenance until 1962 when it was allowed to flood. A total of 749,000 tons at a block grade of 2.29 % copper is recorded to have been depleted by mining during that period.

A subsequent drilling programme of 15 drill holes totalling 10,900 m was carried out between 1967 and 1978. This was aimed at investigating the mineralisation at depth, mainly in the central part of the Konkola North area. Several drill holes intersected mineralisation in this area, with this resource later becoming known as the Saddle Lode. The drillholes were assayed for total copper ("TCu") and acid soluble copper ("AsCu"). Results from the historical drilling are indicated in Table 5.1.

Table 5.1: Summary of intersections from the historical drilling

	Depth, m		Thickness, m`				
BHID	**From**	**To**	**Drilled**	**True [1]**	**%TCu**	**%AsCu**	**Comments**
KK1	97.84	101.80	3.96	3.67	1.51	0.77	
KK4	271.58	282.55	10.97	8.03	3.20	0.23	
KK15	-	-	-	-	-	-	Total core loss in ore zone
KK20	-	-	-	-	-	-	Hole abandoned
KK25	213.97	215.19	1.22	1.11	1.28	0.01	
KK26	-	-	-	-	-	-	<1%TCu
KK28	266.70	271.85	5.15	4.81	2.14	0.31	
KK29	-	-	-	-	-	-	<1%TCu
KLB82	749.81	760.17	10.36	9.74	3.82	2.57	
KLB85	815.96	833.03	17.07	12.28	2.37	0.68	
KLB90	-	-	-	-	-	-	<1%TCu
KLB115	-	-	-	-	-	-	Abandoned
KLB115R	-	-	-	-	-	-	Abandoned

1 true thickness calculated from the core angle of the strata relative to the inclination of the hole

The historical database was acquired by Konnoco and includes borehole core and logs from both phases of drilling, as well as geological reports and mine plans. Where borehole core was available this was re-logged, and integrated into the database for geological and grade modelling purposes. The core was also re-sampled where possible in order to attain a confidence level in the historical assay results.

Initial estimates of Mineral Resources for the KNP were first reported by African Rainbow Minerals Limited ("ARM") in June 2001 and updated in August 2001 (Table 5.2). These Mineral Resources did not include discounts for previous mining in the No. 2 Shaft area, due to the non-availability of production reconciliations records.

Table 5.2: Historical KNP Mineral Resources, ARM, 2001

	Amended Pre-feasibility, August 2001			KNP, June 2001		
	Mt	**%TCu**	**%AsCu** [1]	**Mt**	**%TCu**	**%AsCu**
Measured - East Limb	13.7	2.61		14.0	2.45	0.7
- South Limb	19.0	2.25		18.7	2.10	0.5
Total Measured	**32.7**	**2.40**		**32.8**	**2.25**	**0.6**
Indicated - East Limb	14.0	3.01		15.4	2.67	0.4
- South Limb	31.0	2.23		30.6	2.13	0.3
Total Indicated	**45.0**	**2.47**		**46.1**	**2.31**	**0.3**
Total Measured and Indicated	**77.7**	**2.44**		**78.8**	**2.29**	**0.4**
Inferred - Area A	107	2.30				
- Area A Ext.	63	3.88				
- Area B	41	- [2]				
Total	**211**	**-** [3]				

1 %ASCu grade not estimated, August 2001

2 no %TCu grades provided for Area B

3 average %TCu grade for Total Inferred not computed due to unavailability of %TCu grade for Area B

The classifications of the historical Mineral Resource estimates were based on the application of confidence intervals of the estimated mean, which were derived from the Sichel-t test. Initial independent reviews by SRK in 2004 and RSG Global in 2005 indicated uncertainties in the geological interpretations and the three dimensional wireframe modelling and the Measured Mineral Resources estimates were downgraded to the Indicated category.

There have been several re-evaluations of Mineral Resources on the project: RSG Global, July 2005, GijimaAST in December 2006, ARM in January 2007, July 2007, January 2008 and the current estimate, February 2008.

6 Geological Setting

The Konkola North copper deposit is one of approximately 30 major copper +/- cobalt deposits occurring within the Central African Copperbelt. It is located at the north-western extremity of the Zambian portion of the Copperbelt and represents an autochthonous “Zambian-type” deposit within the late-Proterozoic Lufilian Arc fold and thrust belt. The deposit is hosted within sediments which accumulated in an intracratonic rift, which was subsequently closed during the Lufilian Orogeny.

The Pan-African Neo-Proterozoic Lufilian fold and thrust belt in Central Africa spans 750 km in length and 550 km in width and comprises rocks of the Katanga Sequence and pre-Katanga basement inliers (Figure 6.1). Sediments comprising the Katanga Sequence were deposited in an intracontinental rift, which was subsequently closed during a protracted period of folding and thrusting known as the Lufilian Orogeny.

Depositional ages for the Katanga Sequence are poorly constrained; maximum ages are denoted by the age of the Nchanga granite at 880 Ma and by the age of bimodal basalts and rhyolites at the base of the sequence near Kafue (879 +/- 19) Ma., Microcline-bearing veins in the Lower Roan Formation yield a Rb-Sr age of 870 +/- 42 Ma. A minimum age would be that of the late orogenic Hook Granite at 560-530 Ma. Thus, deposition of the sequence can broadly be constrained between 880 Ma and 530 Ma.

The Lufilian Orogeny is dated at 656 to 503 Ma .U-Pb and K-Ar ages of approximately 500 Ma are common throughout the Lufilian Orogeny and probably represent cooling ages. The Lufilian fold belt is flanked by Kibaran-age (1300-1100 Ma) rocks to the north-west, the Archaean Kasai Shield to the west, the Bangweulu Block to the north-west and the Irumide fold belt (Kibaran age) to the south-east.

The Lufilian fold belt is separated from the divergent Zambezi belt to the south by the Mwembeshi dislocation zone, a major north-east trending sinistral transcurrent shear zone. Structures in the Zambezi Belt verge to the south and the Mwembeshi dislocation zone is considered the suture zone between the Kalahari and Congo cratons. The granite plutons characteristic of magmatic arcs related to subduction zones are represented by the Hook granite (560-530 Ma) and satellite syenite intrusions, and numerous other post-tectonic granites around 500 Ma are exposed over large areas of eastern Zambia.

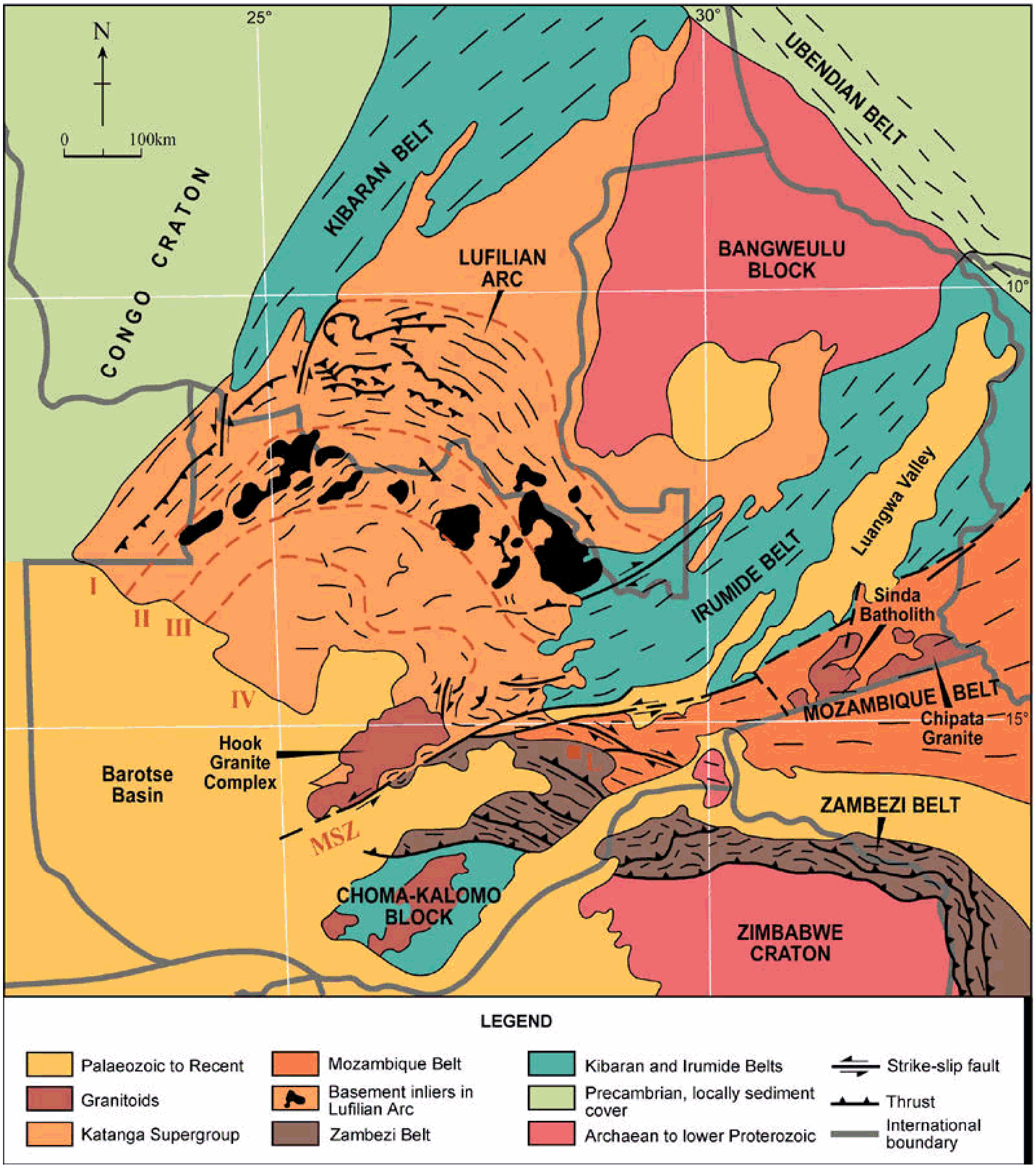

Figure 6.1: Regional Geology (source: zambia-mining.com)

The Lufilian fold belt is thrust over relatively undeformed sediments of the extensive Kundelungu graben to the north-east. This graben is interpreted as an aulacogen, the failed northeast-trending arm of a triple rift junction. Major conglomerate units and basaltic flows follow the fault-bounded margins of the graben. Trachyandesites, tuff horizons and rhyolites are also associated with the Lufilian fold belt, and this bimodal signature, together with thickened conglomerate wedges along fault scarps, indicates that the Katangan Sequence formed in an intracontinental rift. Late tectonic granites and syenites dominated by the Hook granite intrude the Katanga Sequence.

The predominant direction of thrusting was to the northeast with thrust-bounded sequences striking NW-SE. An inner unit with deep-seated thrust faults is recognised, followed by a central unit of thick-skinned thrust sheets, presumably detached, incorporating basement inliers and an outer unit of tightly folded thin-skinned thrust sheets.

The Zambian Copperbelt deposits are located within the central unit and the Konkola area is located on the outer margins of this unit. Although the host rocks to mineralisation are autochthonous in the Konkola area and are preserved in structural lows, there is much evidence for the transition to a thin-skinned thrust dominated terrain higher in the sequence. Interpretation of aeromagnetic survey data indicates prominent discontinuities and truncation of units in the middle and upper parts of the sequence.

6.1 Local Geology

The Konkola North deposit forms a continuation of a NW-SE trending line of deposits which extend from Bwana Mkubwa, southeast of Ndola to Musoshi in the DRC. Within the northern portion of this belt, mineralisation is continuous over a strike of approximately 52 km, with major operating mines at the towns of Chililabombwe, Chingola, Kitwe and Luanshya. The copper mineralisation found in the greater Konkola area is continuous and can be traced from the No. 3 Shaft of the Konkola Copper Mines ("KCM") via the KNP to Musoshi in the DRC.

6.2 Stratigraphy

Numerous stratigraphic classifications exist for the Zambian Copperbelt; with each mine adopting its own subdivisions, which are largely based on convenient mining terms rather than subdivisions done on a geological basis. For this project, Konnoco redefined the stratigraphy based on sedimentological processes observed during bore hole core logging. The stratigraphy was subdivided into sequences bounded by unconformities to better define the development of the sedimentary basin through time. The new stratigraphic subdivision is illustrated in Figure 6.3.

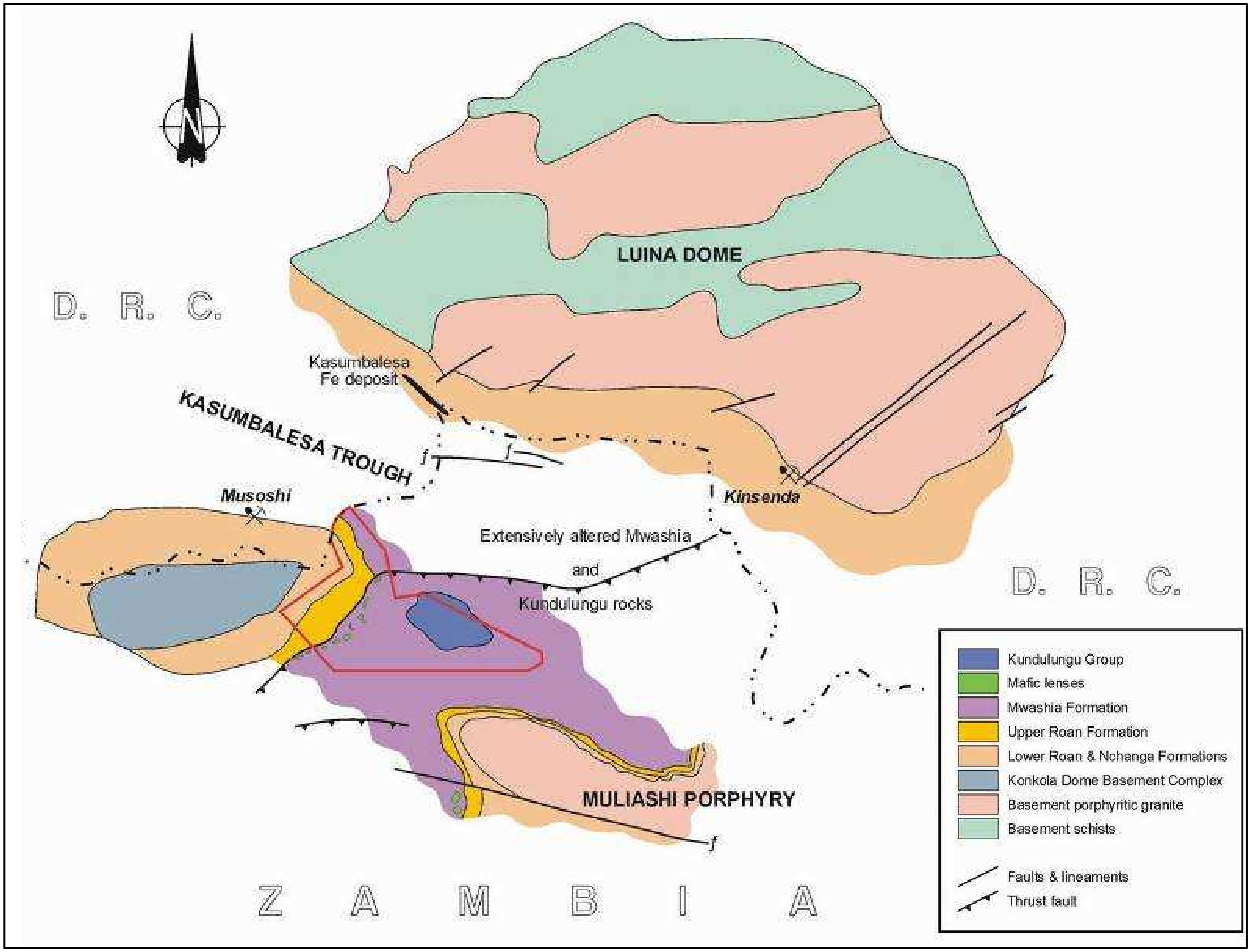

Figure 6.2 Regional Geology of the Konkola North Area(KNP property outlined in red)

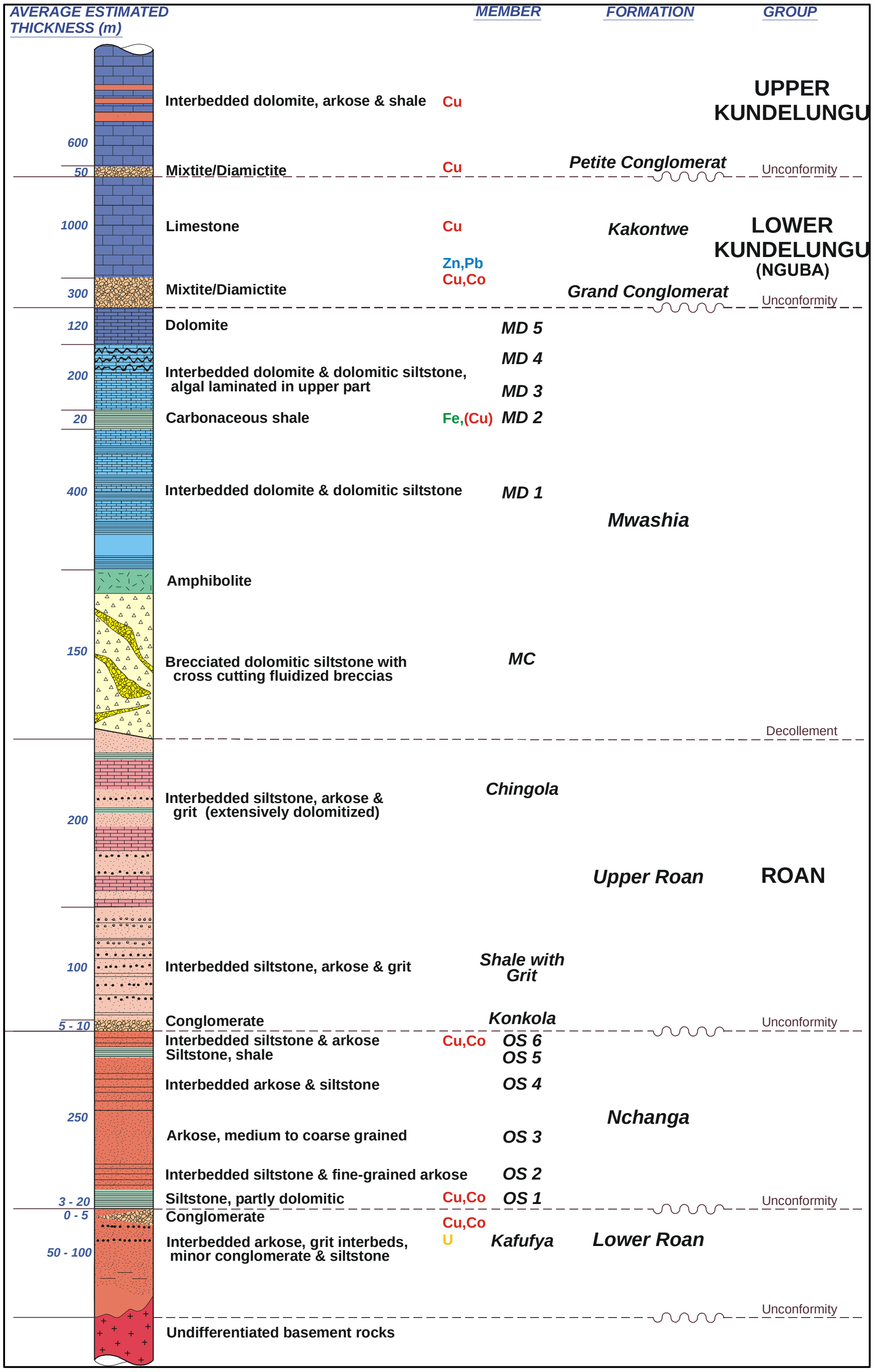

Figure 6.3: Stratigraphy of the KNP

The stratigraphic units are described below:

Basement Complex: Within the Konkola North area the Katangan sedimentary sequence uncomformably overlies coarse porphyritic granite. The granite outcrops as the Muliashi Porphyry and the Luina Dome to the southeast and northeast respectively. This granite contains large orbicular microcline phenocrysts and is dated at 1882 +23/-19 Ma. The Konkola Dome to the northwest comprises Kibaran-age intrusives and metasediments.

The Basement Complex in this area forms part of an extensive magmatic arc terrane, formed during subduction of the Congo Craton beneath the Tanzanian and Zimbabwe cratons. This belt trends NE-SW and is dominated by the Bangweulu Block in northern Zambia. It represents a major crustal forming event at around 2.0 Ga, and is correlated with similar rocks in Namibia and the Northwestern Cape of South Africa.

Lower Roan Formation: In the Konkola North area, the Lower Roan Formation is 1000–1500 m thick and comprises a thick fining upwards sequence of polymictic boulder conglomerate and arkose deposited in a proximal alluvial fan setting (Chimfunshi Member). The thickness of the boulder conglomerate sequence between Konkola and Nchanga indicates proximity to basin margin faults and rapid uplift along these faults. The Chimfunshi Member is overlain by the Kafufya Member (50-100 m thick), represented by a sequence of immature interbedded arkoses and conglomerates with subordinate argillaceous rocks and characterised by large-scale cross bedding with heavy mineral lags indicative of a braided stream mid-fan depositional environment. Dissolution casts after possible evaporites have been recorded.

Most Konkola North boreholes were terminated about 30 m below the OS1 Member, and the typical arkose from this zone is grey, porous, massive in structure, and contains abundant dissolution cavities, possibly after evaporates. On the East Limb, the Lower Roan formation is capped by a 10-15 m thick conglomerate which pinches out on the South Limb and in the Central part. The conglomerate is clast supported with an average pebble size of 12 mm, with K-feldspar and arkose clasts predominating.

For the most part the Lower Roan formation have a feldspar content of 10-15% (mostly secondary plagioclase) and may be classified as subarkoses. The entire formation is oxidised and constitutes a typical “red bed” sequence. Evidence of leaching and removal of specular hematite and iron oxide coatings of detrital grains is widespread, resulting in a bleached appearance.

Nchanga Formation: The OS1 Member abruptly overlies conglomerate or arkose of the Kafufya Member and is the main host to mineralisation in the area. It varies from 3m to 12m in true thickness and is defined strictly as a fine-grained siltstone. Occasional thin sandy lenses are observed in the OS1 and, where they occur, are preferentially oxidised and leached. This unit has been partially dolomitised, resulting in thin white interbeds; however, the extent of dolomitisation decreases to the north. The typical composition of the OS1 is microcline (40%), K-feldspar (30%), quartz (30%), dolomite (2%) and subordinate rutile, tourmaline, zircon and white mica.

The OS1 is extremely fine grained, with an average grain size of 65 %< 75 µm. The grain size appears to be slightly finer on the South Limb.

The OS1 is gradationally overlain by a 5-30m thick sequence of interbedded siltstone and fine-grained arkose of the OS2 Member. The arkose grain size and proportion increase upwards. The OS1-OS2 contact is taken at the base of the first obvious fine-grained arkose bed. The interbedding is on a centimetre to decimetre scale and load; flame and dewatering structures are commonly observed together with ripple cross lamination and starved ripples.

The transition into the OS3 Member is gradational, with the top of the OS2 Member defined at the last observed siltstone unit. The OS3 Member is a medium- to coarse-grained arkose, gritty in parts and typically extensively replaced by K feldspar. This replacement is partial to pervasive, with remnant darker grey argillaceous material commonly observed. This unit varies considerably in thickness (3-213 m drilled thickness). It is overlain by a sequence of interbedded arkose and subordinate siltstone (OS4 Member), varying in drilled thickness from 83 m to 310 m.

The overlying OS5 Member comprises a 0-47m thick siltstone-shale-dolomite unit similar in appearance to the OS1. Throughout the area, the top and bottom contacts of the OS5 are characterised by soft sediment folding and crumpling thought to originate from a regional tectonic event such as an earthquake or alternatively mobile slurries that flowed over water-saturated strata. In places, this unit contains scapolite and nodular dolomite pseudomorphs after possible anhydrite/sulphate evaporites. As in the OS1 Member, where sedimentary layering wraps around such nodules, it is clear that the precursor minerals grew *in situ*. Although not present in all boreholes, the OS5 constitutes a marker bed in the Konkola North area and is frequently mineralised by fine-grained chalcopyrite, though this mineralisation does not attain economic proportions. In certain boreholes, the OS5 Member grades into an interbedded siltstone and arkose, defined as the OS6 Member, which varies in drilled thickness from 3m to 22m.

Upper Roan Formation: A well-developed unconformity separates the Upper Roan Formation from the Nchanga Formation. The base of the Upper Roan Formation is defined by a 20m thick pink to red clast-supported conglomerate (Konkola Member), with an erosional lower contact. The conglomerate has dominantly K-feldspar pebbles averaging 7-8 mm in size to a maximum of 40 mm, set in a gritty matrix. Heavy mineral-delineated cross stratification is common. The conglomerate grades upwards into a 60m to 288m thick sequence of thinly interbedded arkose, siltstone and grit cycles, commonly known as Shale with Grit Member ("SWG"). The interbedding in the SWG comprises thin fining-upward cycles of grit, ripple cross laminated arkose and siltstone, the cycles ranging from 5cm thick at the base of the SWG and increasing in thickness upwards to about 50cm thickness at the top. These cycles are interpreted to have been deposited as thin, but a really extensive sheet outwash flows in a playa lake setting. Dewatering structures are common as are grit infilled mudcracks at the top of individual cycles. The middle and upper parts of the SWG are extensively dolomitised and are referred to as the Chingola Member. This member varies in drilled thickness from 112m to 603m, largely as a function of the extent of dolomitisation. The dolomite replacement is characteristically hematite stained and pink to red in colour. The dolomites are abruptly overlain by a 20m-thick hematite-rich arkose unit.

Mwashia Formation: A well-developed *decollement* separates the Mwashia Formation (“MW”) from the Upper Roan Formation. The MW1 Member comprises brecciated laminated dolomitic siltstone and arkose, intruded extensively by fluidised dolomitic breccias. The fluidised breccias, although stratabound, are locally discordant and are thought to be genetically related to the giant megabreccias observed in the Katanga Province, DRC. They are red to grey-green in colour, matrix supported, and lack internal deformation. The matrix is predominantly dolomite, albite, and chlorite, and the clasts mainly siltstone and dolomite. Extensive albitisation has resulted in near complete replacement of the original matrix mineralogy. The clasts are subangular to well rounded and, together with the laminated structures in the breccias, have often been misinterpreted as sedimentary in origin. It is thought that these tectonic breccias were sourced from, and controlled by, the existence of evaporitic horizons.

Highly altered mafic intrusives occur within the MW1 and may be traced at this position from immediately south of Konkola Division though the project area. The intrusives are not laterally continuous but occur in lenses which have a characteristic magnetic signature. These mafic bodies are spatially related to the breccias described above, both here and at other localities on the Zambian Copperbelt. The intrusives are coarse grained and comprise mainly biotite, plagioclase and epidote.

Drillhole observations indicate that the MW1 breccia unit varies greatly in thickness from 50m to 573m and extends upwards into intact laminated dolomitic siltstone, carbonaceous shale and limestone (MW2-6 Members). The MW2 varies from 355m to 778m in thickness and is a well-laminated greenish-grey dolomitic siltstone, evaporitic in parts. Biotite porphyroblasts are common within certain beds, reaching 3 mm in size and overgrowing the lamination. Within two distinct horizons, the biotite porphyroblasts contain cores of magnetite. These two units are marker horizons and are mappable on the aeromagnetic plans. Biotite also occurs as a selvedge to ubiquitous (metamorphic secretion) calcite veins present in the MW2. Dolomite is present throughout, but varies in abundance. Locally, white dolomite replacement is massive and obliterates sedimentary structures. Subhedral pyrrhotite and minor pyrite also overgrow the sedimentary structures.

A 15m-30m thick pyritic carbonaceous dolomitic shale (MW3 Member), with minor calcite secretion veins overlies the MW2 Member. The MW4 Member comprises laminated dolomitic siltstone, with 20-30% anhydrite developed along bedding, possibly after evaporites. Disseminated fine-grained biotite overgrows the sedimentary lamination. Interbedded dolomite and dolomitic siltstone constitute the MW5 Member, which is strongly folded and sheared. Talc-infilled bedding parallel shears are common. Abundant disseminated and vein-hosted anhydrite is observed. An algal dolomite unit is developed towards the top of this unit. Wavy algal lamination with possible low amplitude stromatolites are observed, together with zones of reworked algal material and coarse clastic carbonate debris. This 30m thick algal unit is directly underlain by a 40 m thick microbrecciated dolomitic siltstone characterised by a strong fabric. Medium-grey fragments (1-5 mm) are set in a dark-grey matrix and are elongate and flattened in the plane of foliation.

The MW5 Member is capped by a 120m-thick fine-grained dolomite unit (MW6), which ranges in colour from pink at the base through white to grey. Dissolution cavities and stylolites occur throughout. Minor remnant bedding is observed.

In the DRC, the equivalent of the Mwashia Formation (the Mwashya Group or R-4), comprises arenaceous redbeds deposited in alluvial fan complexes extending into braided intertidal channel deposits and probable playa lake facies. These are overlain by lagoonal dolomites deposited over evaporitic fine-grained oxidised intertidal sediments in a transgressive marine environment. In this context, the reduced dolomitic siltstones observed in the Konkola area are interpreted as distal lagoonal/shallow marine time equivalents. In the DRC, an igneous event is identified within the Lower Mwashya Group (R-4.1), in the form of pyroclastic flows in the central Katanga arc (Likasi / Kambove area).

Kundelungu Group: The Grand Conglomerat Member of the Kundelungu Group was intersected in one borehole and described as a diamictite with rounded carbonate clasts of less than 10cm set in a mud matrix.

In the Konkola area, the Mwashia Formation and Kundelungu Group are thrust over the middle and lower parts of the Roan Group along a prominent detachment zone at the base of and within the brecciated MW1 Member and are considered allochthonous. Numerous crosscutting fluidised dolomitic breccias are observed, which acted as zones of lubrication during thrusting similar to the albite-dolomite breccias and the Sole Dolomite, in particular, as described from the Damara Sequence in Namibia. The lack of a prominent fabric in these breccias may be due to later post-kinematic recrystallisation from a syn-tectonic fluidised mass. The detachment plane truncates progressively lower stratigraphic units south of Konkola. In the western part of the Chambishi basin, similar breccias and associated gabbros are logged in the Upper Roan Formation in association with talc carbonate schists (deformed dolomite) while, further to the south in the Chibuluma area, breccia and gabbro overlie Lower Roan Formation arenites. In the latter case, the ore shale may have been tectonically removed rather than not having been deposited.

Mineralised sections hosted within megabreccias in the DRC may have originated from depositories in the inner and central parts of the fold and thrust belt. Where gabbro occurs within dolomite, no chilled margins or skarnification is seen and this, together with the typical discontinuous lens-type geometry of these bodies in the northern Copperbelt, may imply that they were intruded during thrusting. In the Konkola area, the mafic intrusives are surrounded by pervasive alteration, possibly due in part to assimilation of sedimentary country rock, representing allochthonous blocks within a breccia melange.

Throughout the Zambian Copperbelt the dominant structural grain is northwest trending. These structures represent the largest and most laterally continuous structures observed. This is very apparent from structural interpretation of Landsat TM imagery, aerial photography and airborne magnetic coverage. From this data, three populations of structural trends are observed: The prominent NW trend, N30°-70°E trending structures and N-S trending structures.

6.2.1 Local Structural Setting

The Katanga Sequence rocks onlap onto Basement Complex rocks of the Konkola and Kirila Bombwe Domes to the northwest and southeast respectively. The axis of the Kirila Bombwe antiform trends N50°-60°W and plunges 5°-15°NW. An antiformal axis trends southeast-wards at depth beneath the Katanga sediments away from the Konkola Dome. Three sub-basins, partly juxtaposed by major structures, are interpreted from the aeromagnetic data.

In the Konkola area, sedimentation was initiated during intracontinental rifting by deposition of proximal alluvial fan boulder conglomerates. The sequence is upward fining, with boulder conglomerates overlain by trough cross-bedded mid-fan grits and arkoses. Paleocurrent directions are mainly from the southeast, indicating the Muliashi Porphyry as the dominant provenance. The mineralogical composition of the Lower Roan Formation sediments corresponds very closely to that of the Muliashi Porphyry.

A sudden transgression resulted in the deposition of the OS1 Member. Further uplift along proximal fault scarps saw deposition of interbedded grits, arkoses and siltstone (OS2-6 Members). This deposition was overlain by playa lake sheet wash flows of partially dolomitised interbedded siltstone, arkose and grit occurring as very thin upward fining cycles. The initial phase of rifting was followed by basin subsidence and a marine transgression with deposition of platform carbonate and pelitic sequences.

Mafic volcanism in the Luina Dome area is related to block faulting in the basement, mainly along N50°-70°E structures. Major magnetic lineaments in the Luina Dome trend N57°-68°E. The major trend of rifting in the Kundelungu aulacogen is N30°-40°E, which corresponds to the trend of conglomerate-bounded syn-sedimentary fault scarps observed from the Musoshi-Kinsenda area.

Two fold trends are recognised. Earlier N20°E trending folds plunge to the south with an axial plane dipping 60°SE. The north-south folds are characterised by zones of brecciation, contortion and a highly variable mineral content. These folds are crosscut by a later dominant fold set which trends N70°W in the regional structural grain of the fold and thrust belt, with fold axes plunging to the NW and axial planes dipping 60°-80°NE. In both sets, fold hinge zones are sites of potential dilatancy and jointing, which act as permeability channelways to groundwater movement. Fold intersections are commonly sites of oxidation, leaching and brecciation of ore.

Tensional faults are observed in the hinge zones of folds. Where present, northwest-trending faults are normally high angle reverse faults and are parallel to fold axes. Displacements are minor and do not exceed 15m.

7 Deposit Types

The Zambian Copperbelt is developed within a major geological structure, the Lufilian Arc, stretching for more than 500 km from Kolwezi in southern DRC to Luanshya in Zambia. This arc is host to extensive high-grade copper-cobalt mineralisation in very large stratiform deposits, with many large copper mining operations working in this area. The KNP area is one such type of deposit occurring in the ore shale (“OS”).

7.1 Mineralisation

The KNP area mineralisation is defined as the ore shale type of mineralisation. It is developed on a southwest-trending dome/anticline controlled by basement topography a short distance to the north-northwest of Konkola. Copper mineralisation is largely hosted within the OS1 Member, the first major reducing horizon above a 1000m thick sequence of oxidised arkoses and conglomerates. The OS1 Member siltstone unconformably overlies the Kafufya Formation and represents a sudden transgressive event. Although copper mineralisation occurs mostly within the OS1 Member, it is crosscutting on a gross scale and pinches out towards the east within the overlying OS2 Member.

Regionally, the mineralised portion of the OS1 Member is usually between 15m and 20m thick and at individual mines may have a strike length of over 10km. A complex paragenesis of copper oxides and sulphides is present. Copper also occurs in its native form and within the lattice of metamorphic micas, i.e. the Chingola refractory ores.

The OS1 Member is the lowest well-defined and continuous shale/siltstone/schist unit of the Katanga Sequence and has been interpreted as a reductant, impervious trap site and depositional site for mineralised fluids.

The true thickness of the OS1 Member varies from 3m to 14m. Typically, the lowermost 1.5m of the OS1 is massive in structure and contains very little copper. The transition to >1% TCu is abrupt and takes place over centimetres, above a thin red iron oxide-rich marker layer, which probably acted as a reducing-oxidation (“redox”) boundary. The upper contact of the >1%ºTCu zone (assay hangingwall) is also well defined in the assay profile, but is not as sharp as the assay footwall contact.

Mineralisation occurs as finely disseminated sulphides assuming the grain size of the host rock, as coarser grains along bedding planes and cleavage, in thin veinlets, and in lenticles and stringers. These minerals comprise chalcocite, chalcopyrite, bornite, digenite, covellite, pyrite and carrollite in approximate order of abundance. A large proportion of the non-sulphide copper minerals occur along fractures and veins. The following are the main non-sulphide minerals which are observed, in order of abundance: malachite, pseudomalachite, chrysocolla, cuprite, azurite and native copper.

Vertical zonation of the copper minerals is observed; with an increase in copper content and a decrease in iron and sulphur content with depth. Redistribution of copper has taken place. An idealised orebody profile has chalcocite enrichment in the lower part followed by bornite and then chalcopyrite and pyrite. This is confirmed by the typical grade pattern, which is asymptotic, with the highest grades at the base. The acid-soluble minerals are generally concentrated towards the base, in the zone of most intense leaching. A large spread of copper values is observed in the shallow boreholes, testifying to the mobility of copper in the weathering environment.

Chalcocite replacement of chalcopyrite and bornite took place from the base of the orebody, diminishing upwards, with subsequent leaching and oxidation being associated largely with structure. Such leaching and oxidation may occur at any position in the orebody, resulting in low grade patches. It is, however, concentrated at the base of the OS1 where the contact between the siltstone and conglomerate/arkose represents a permeability channel way. Iron oxide after weathering of chalcopyrite and bornite is common in leached zones. An analysis of the iron oxide aggregates indicates mixtures of hematite, iron oxide-hydroxides and iron oxide-hydroxide-hydrates. Hematite forms ordered intergrowths of hematite lamellae and is mostly related to the later K-feldspar-hematite-quartz alteration event. The limonitic aggregates occur as concentrically zoned material, containing traces of secondary copper minerals and weathered copper sulphides. The copper-phosphate pseudomalachite normally occurs as spheroidal aggregates, often surrounded by concentrically zoned shells of limonite.

7.1.1 South Limb

Primary mineralisation on the South Limb comprises mainly disseminated extremely fine-grained chalcopyrite. The grain size is in the order of 70% < 75 µm, with 5-10% of the grains < 20 µm. The typical ore mineralogy below 300 m depth comprises chalcopyrite (65%), chalcocite (17%), malachite (15%), bornite (1%), digenite (1%) and covellite (1%). At above 300 m, the orebody comprises chalcocite and lesser malachite as the main ore minerals with an average ratio of acid-soluble to total copper of 33%. The average true thickness of the orebody is 4.7 m, with an average apparent dip of 60°.

7.1.2 East Limb

On the East Limb, mineralisation transgresses at a low angle and occurs partially within the OS2 Member downdip.

The average true thickness and apparent dip of the East Limb orebody is 5.3m and 45° respectively. Mineralisation is slightly coarser grained with 62% of the sulphide grains passing 75 µm. The typical ore mineralogy below a depth of 200m comprises chalcocite (63%), malachite (27%), bornite (4%), chalcopyrite (4%), digenite (1%) and covellite (1%). Mineralogical logging of ore intersections indicates that primary mineralisation was vertically zoned from bornite at the base to chalcopyrite at the top. Above 200 m, the orebody comprises chalcocite and malachite as the major ore minerals. Above a depth of 30 m, the orebody is usually completely leached.

7.1.3 Block "A" – Block "A" Extension

Mineralisation is continuous between the South and East Limb areas and the known mineralisation at KCM's Konkola Division. Over this large area, the orebody occurs at between 760m and 1600m depth, with a variable dip of between 0° and 40°. The thickness of the orebody increases towards the south, where it averages 9 m true thickness and straddles the OS2 and Kafufya Members.

7.2 Distribution of Mineralisation

The results of the drilling programme indicate continuous mineralisation over an area of 28.2 km^2, contiguous with the Musoshi orebody in the north and the KCM's Konkola Division North orebody in the south.

The distribution of copper minerals varies across the area, largely owing to the effects of structurally controlled leaching and supergene processes operating to significant depths. The vertical distribution is also variable being controlled by joint, fold, fault and fracture patterns. The supergene enrichment results in the secondary pattern of mineral distribution largely obscuring the primary ore mineral pattern. However, remnant signatures of hypogene mineralisation have been observed. Lateral zonation from bornite-chalcopyrite on the East Limb through chalcopyrite-dominated mineralisation on the South Limb to pyrite near the western boundary is observed. There are indications that the primary sulphide mineralisation was originally more extensive in a lateral and vertical sense, and that the tenor of mineralisation has been diminished by later oxidation and leaching.

The distribution of the %TCu, interpreted from the grid contouring of the drillhole intersections, indicate a NW-SE trend parallel to the dominant regional structural grain, as evidenced by the trend of structural features mapped from aeromagnetics, gravimetrics, satellite imagery and aerial photography

Two of these zones define the East and South Limb deposits. The higher grade nature of the East Limb deposit is clearly evident. The intervening low grade zone is defined by 12 drill holes.

On the East Limb, two higher grade zones are distinguished: a dominant N45°W trending zone defining the main body of mineralisation and a weaker N70°W trending zone. At depth, the orebody is abruptly cut off to the east by a zone of intense fracturing, brecciation and K-Fe alteration related to the intersection of NW-SE trending faults and a major E-W discontinuity. This event was superimposed on existing mineralisation, with the K-Fe alteration event serving to replace and obliterate the pre-existing mineralisation. The same alteration event is responsible for cut-off of mineralisation in the extreme northern borehole, KN8.

The East Limb deposit probably links up with the Saddle Lode deposit, but this has not been confirmed due to the failure of several boreholes to reach their target.

The South Limb deposit trends N70°W. In this area, late north-south fractures have facilitated remobilisation of mineralisation, resulting in two north-south trending overprints. One of these has clearly controlled the distribution of higher grade supergene mineralisation along three closely spaced parallel fractures.

In the central Saddle Lode area, two subparallel N40°W trending higher grade zones are evident, separated by a lower grade block. This trend is disrupted by a N50°E trending zone of elevated grade defined by four boreholes over three kilometres. The latter trend is also structurally controlled as it coincides with regional lineaments of similar orientation.

The above observations indicate that the distribution of mineralisation is closely controlled by regional and local structures and that these structures post-date mineralisation and were responsible for localising supergene and leaching processes.

The following observations can be made regarding the distribution of acid-soluble copper:

- AsCu mineralisation represents 39% of TCu in the near-surface mineralisation of the East Limb orebody and 42% within the South Limb orebody. This elevated AsCu content is related to near-surface weathering processes operating to depths of 300-400 m. Below this depth, the AsCu content is reduced.

- Below this depth, throughout the area, AsCu content and its distribution are related to structure.

- Within the Saddle Lode area (Block "A" Extension), AsCu mineralisation accounts for some 59% of total copper mineralisation.

- Two sub-parallel zones of high AsCu content, about 500 m wide, trend N40°W in the Saddle Lode area. These zones are directly related to structural highs, being the crests of two sub-parallel antiforms which join towards the NW. A direct correlation exists between a high proportion of AsCu and the flatter lying (0°-17° apparent dip) part of the orebody in the Saddle Lode area.

The pattern of AsCu content appears to be locally influenced by N-S trends. It is critical to map in detail the distribution and proportion of AsCu as this strongly influences metallurgical process options and metal recoveries.

7.2.1 Cobalt Distribution

Cobalt mineralisation is restricted to the southern boundary and a small area on the western extension of the South Limb. The average grade of mineralised intersections in the remainder of the area is 0.01-0.02%TCo. Within the defined cobalt resource, cobalt occurs as carrollite in close association with chalcopyrite, in the form of disseminations and as blebs and stringers along bedding. In zones of deep oxidation and weathering, complete transition to heterogenite, wad and asbolite (soft, black earthy mineral, often classed as wad, that contains up to 32%TCo) occurs over distances of <600m.

The cobalt occurrence on the South Limb is defined by three borehole intersections with an average grade of 0.17%TCo. The cobalt occurs as carrollite severely inter-grown with chalcocite, covellite and digenite and collectively rimming chalcopyrite.

7.2.2 Silver Distribution

The distribution of silver appears to be contained within chalcocite and as such is more abundant towards the base of the orebody. The silver grade is highly variable, with an average grade of 0.42 g/t in the orebody over the Konkola North area and a maximum single sample result of 14.62 g/t in drill hole KNS32 on the East Limb.

7.2.3 Orebody Geometry and Attitude

In the No.2 Shaft area, the orebody is thinnest in the low grade block between the East and South Limbs. This correlates with the structural highs mapped from the geophysics data. Throughout the entire area, there is a correlation between grade and thickness. The dip of the orebody is steepest along an E-W trend on the South Limb. The orebody on the East Limb appears to lie within a trough.

A broad flat dipping structural high covers most of the Saddle Lode area. The dip of the orebody steepens abruptly to the NE with the edge of the structural high coinciding with a major magnetic lineament.

7.2.4 Alteration

Alteration processes have locally coarsened, concentrated and depleted the orebody. Several stages of alteration in the orebody are observed. An early stage of chalcocite replacement of primary chalcopyrite and bornite was followed by oxidation and leaching and the formation of secondary minerals such as copper carbonate-hydroxide minerals, copper phosphates and silicates and iron oxides. With the earlier stage, small relicts of some of the larger chalcopyrite grains are observed, in places rimmed by covellite. This earlier alteration does not appear to be structurally controlled, whereas the later oxidation through supergene solutions is localised by folding, faulting, joints and fractures. Alteration and replacement fronts are commonly very sharp, resulting in zoned mineralogical changes over centimetres and even millimetres. Dark-grey siltstone with secondary chalcocite is in knife-edge contact with reddish oxidised siltstone, with an accompanying disappearance of chalcocite over millimetres and its replacement by limonite, malachite and chrysocolla.

The lowermost part of the OS1 Member is very low grade to barren, and is separated by >1% TCu with a thin red iron oxide line, which marks a redox boundary between barren oxidised rock and reduced metalliferous rock.

K-feldspar-quartz-hematite veins and associated replacement fronts in the OS1 Member pertain to the same alteration event observed in the overlying units. Surrounding these veins, the dark-grey siltstone becomes pervasively reddened by K-feldspar, with accompanying disseminated hematite, and associated obliteration of pre-existing mineralisation. Alteration becomes more pronounced towards the east, and results in a complete hematite overprint of the OS1 Member.

8 Exploration

Geophysical exploration work undertaken on behalf of TEAL included:

- **Aerial Photography**- undertaken by Photosurveys (Pty) Ltd from 21 to 22 June 1997 mainly to obtain baseline environmental data at the outset of the programme. Black and white aerial photography was flown at 1:10 000 scale and colour infrared at 1:15 000 scale. This data was digitally mapped and contoured at a 1 m contour interval to produce orthophotos and a digital elevation model. A high resolution "quickbird" satellite image of the mine licence area was acquired by Konnoco in 2006;

- **Airborne and Ground-based Surveys-** this was a high resolution helicopter-borne aeromagnetic survey undertaken by Geodass (Pty) Ltd in September 1997. The aim of these surveys was to obtain detailed information on the geological structure of the area to assist in planning of the drilling programme. Data from this programme was acquired at low level (+/- 40 m ground clearance) with a flight line spacing of 50 m. A semi-detailed grid-based gravimetric survey was undertaken in December 1997 on grid lines 500 m apart with a 200 m sample spacing TEAL had already established aeromagnetics as an excellent mapping tool for its Copperbelt Joint Venture licence areas;

- **Semi-detailed Gravity**- As a result of the low contrasts in density between the various rock types, the gravimetric data does not show much detail. The data shows a bouger low anomaly coincident with the low density Lower Roan, Nchanga and Upper Roan Formation arkoses, grits and siltstones. Dolomitisation of the Upper Roan Formation is secondary and only partial and therefore does not result in an increased density that would reflect in the gravimetrics. The bouger low pertaining to these rocks extends into the major ESE trending thrust zone as these rocks are caught up in the detachment zone;

- **Reflection Seismic Test Survey -** A conventional 2D seismic reflection survey was carried out in April 1997 by Geoplan Manx Limited, over four lines totalling 15 line kilometres within the Konkola North area. The results were inconclusive because of a lack of seismic penetration;

- **High Resolution Aeromagnetics**: Lithomagnetic units have been mapped and correlated with the known geology. Prominent lithomagnetic units in the Kundelungu Group are not discussed, as these lie mostly outside the area of interest. Undifferentiated basement lithologies of the Konkola Dome appear as magnetic highs and are overlain by a unit with a stippled signature, probably the basal boulder conglomerate comprising abundant clasts of basement material. This unit is correlated immediately south of Chililabombwe. Most of the Lower Roan, Nchanga and Upper Roan Formations are magnetically quiet and transparent, but yield a broad higher magnetic response around the axis of the Konkola Dome. This elevated signature coincides well with the extent of mineralisation (>1%TCu) on the East and South Limbs of the Konkola Dome and may reflect an alteration event associated with the mineralisation. Alternatively, magnetic profiling across the axis of this trend suggests a deeply buried basement high (extension of the Konkola Dome) plunging to the southeast.

The broad structure of the “saddle” structural high between the Karila Bombwe and Konkola Domes is clearly denoted by the aeromagnetics. The trend of the structural axis through the Karila Bombwe “antiform” is NW-SE, whereas the axis of the Konkola Dome trends E-SE.

No indication of faulting or displacement along the supposed Lubengele fault exists; however, a magnetic lineament or trend is apparent.

9 Drilling

A total of 47,689.55 m of diamond drilling was undertaken by TEAL during the course of the exploration programme between 1997 to 1998. This is in addition to the 6,640 meters drilled by Bancroft Mines Limited and the 10,900 meters drilled previously by Anglo American Corporation and Zambia Consolidated Copper Mines (“ZCCM”). The TEAL drilling was undertaken by three drilling contractors, Boart Longyear, Stanley Mining Services and RedrilSA.

The drilling was monitored by TEAL’s field geologists. Logging of the holes was undertaken on site and logged according to standard TEAL core logging procedures.

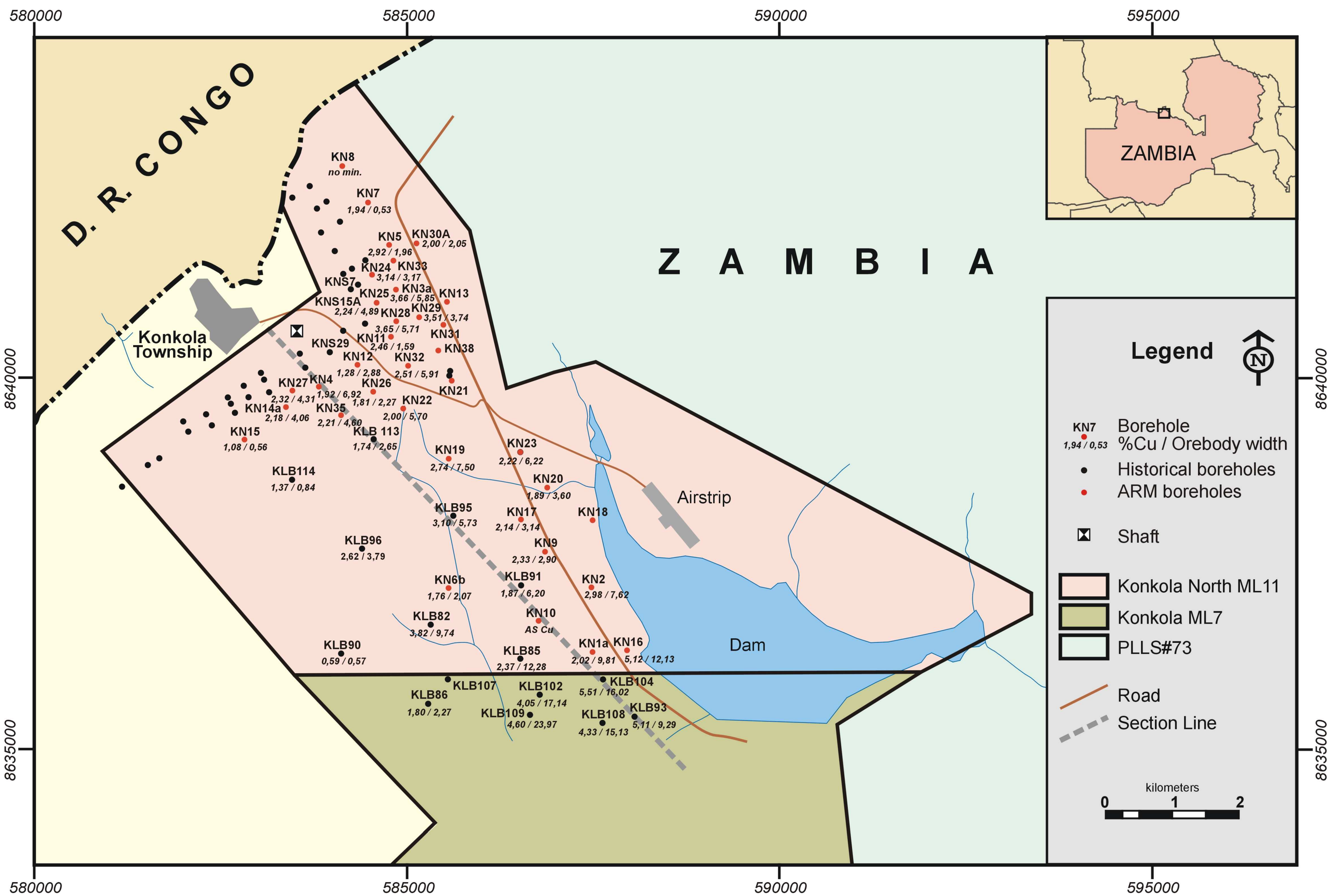

Figure 9.1: Plan view of the KNP showing location of drillholes

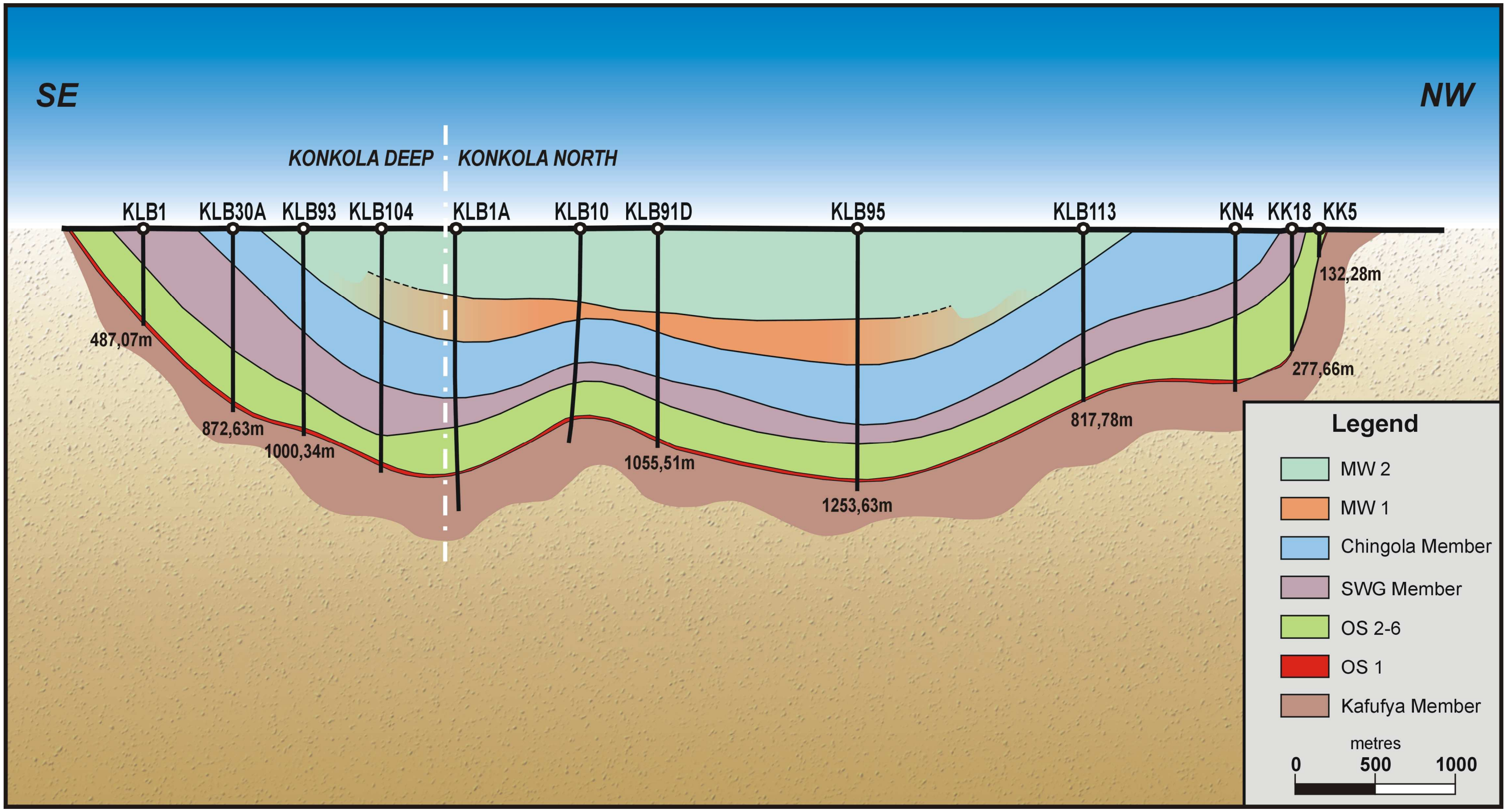

Figure 9.2: Geological section through the KNP

10 Sampling method and Approach

Prior to the commencement of the drilling, a drilling and sampling protocol for the project was drawn up by TEAL. The protocol outlined procedures and the methodologies adopted and executed by TEAL personnel during the drilling, sample collection, core sampling, sample dispatch, security, and Quality Assurance and Quality Control ("QA/QC").

The sampling work was undertaken in 1999, prior to the introduction of international standards for reporting of Mineral Resources and the requirement for independent observations on the sampling protocols and review of the sample dispatch, security and QA/QC programmes.

All the work was undertaken by TEAL personnel.

10.1 Selection of laboratories

Prior to commencement of the drilling programme, TEAL undertook a complete review and audit on the ZCCM analytical laboratory at Kalulushi and the Anglovaal Research Laboratory ("AVRL") in Johannesburg. TEAL decided to use the Kalulushi facility for the Konkola North programme for the following reasons:

- The laboratory had already been extensively used for analysis of soil and core samples from the Anglovaal (Zambia) Limited Copperbelt exploration programme;
- The review and audit deemed the laboratory to be of sufficiently good standard. (Both the ZCCM and AVRL facilities are ISO 9002 accredited);
- The proximity of the laboratory to the field base would offer definite logistical advantages.

The AVRL laboratory was used as an umpire facility. Analytical techniques for the analyses of the samples were discussed with both laboratories to ensure consistency in the analyses, this was particularly important in the case of acid soluble copper analyses. A description of the methodology was agreed upon and adopted for both laboratories.

10.2 Sample collection and recovery procedures

The following is an extract of the TEAL sampling protocol:

- The site geologist inspects the core and defines the positions of the mineralised intercepts. The core is photographed for geotechnical and reference purposes;
- The core is metre-marked and recoveries calculated for individual runs. Thereafter, the core is marking for splitting by diamond saw, making sure that the same side is sampled throughout the mineralized section. Broken core is pieced together and secured with masking tape;

- After splitting, both halves are placed in the core tray with split surfaces facing upwards and the core logged in detail. The geologist then marks the core for sampling using a china marker, taking cognizance of variations in lithology, mineralization and alteration and utilizing natural breaks as far as possible;

- Sample widths approximate 0.5m. The corresponding sample intervals and numbers are then marked on the remaining half core using enamel paint;

- Sampling details are captured on a standardized spreadsheet (sample no., from and to depths, dip, drilled thickness, true thickness). Sample numbers are prefixed with the borehole number and increase sequentially down the hole;

- The geologist then bags each sample, inserting 2 pre-marked sample tickets, one of which is used for splitting at the next stage;

- The samples are then crushed to –2mm using a jaw crusher. At all stages of the sampling process, strict measures are taken to avoid contamination. The jaw crusher is thoroughly cleaned between sample batches with compressed air and clean quartz, and between individual samples with compressed air;

- The crushed sample is then split twice, one half from each split returned to the original sample bag as the assay sample, and the other half from each split to a new bag, which is labelled, purged with nitrogen, sealed, and stored as a metallurgical sample;

- The assay sample is then pulverized in a sieb mill, inserting 100g at a time to avoid coarse fragments. The mill pots and rings are cleaned with compressed air and a brush between samples, and with crushed clean quartz after every tenth sample;

- The pulverized sample is then split, retaining half as a back-up sample and the other half representing the final assay sample;

- Quality control samples are inserted and carefully recorded on a standardized spreadsheet. The samples are assigned coded sample numbers, to conceal the identity of the borehole and of the control samples. A minimum of 10 samples in each batch are further split as duplicate samples and no less than 10 standard reference samples inserted at intervals throughout the batch.

Figure 9.3 is a schematic showing the protocol adopted by TEAL for the sample handling and preparation.

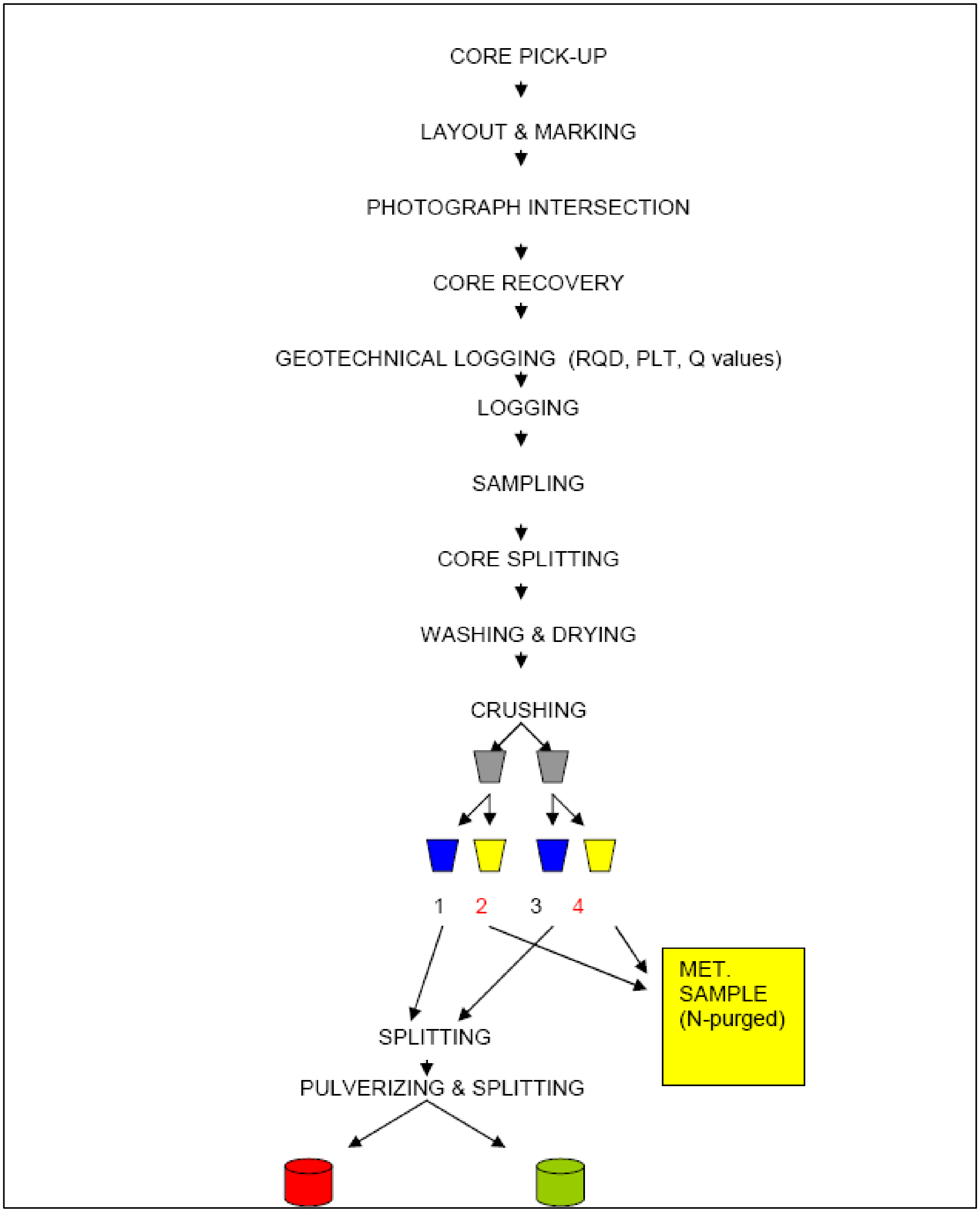

Figure 9.3: Sample handling and preparation protocols

10.3 Sample Analyses

Prior to commencement of the drilling programme, TEAL undertook a complete review and audit on the ZCCM analytical laboratory at Kalulushi and the Anglovaal Research Laboratory (“AVRL”) in Johannesburg. TEAL decided to use the Kalulushi facility for the Konkola North programme for the following reasons:

- The laboratory had already been extensively used for analysis of soil and core samples from the Anglovaal (Zambia) Limited Copperbelt exploration programme;
- The review and audit deemed the laboratory to be of sufficiently good standard. (Both the ZCCM and AVRL facilities are ISO 9002 accredited);
- The proximity of the laboratory to the field base would offer definite logistical advantages.

The AVRL laboratory was used as an umpire facility. Analytical techniques for the analyses of the samples were discussed with both laboratories to ensure consistency in the analyses, this was particularly important in the case of acid soluble copper analyses. Brief descriptions of the methodologies, agreed to and adopted by both laboratories, are indicated below.

10.3.1 Procedure for determination of %TCu and %TCo

- Weigh 2g of the sample into a PTFE beaker and add 10ml of HCl and 5ml HF. Heat to dryness;
- Add 10ml of 1:4 perchloric/HNO_3 mixture and heat to dryness;
- Dissolve the residue in 15ml of HCl and about 10ml of H_2O with heating;
- Filter and dilute the filtrate to 100ml and determine the elements by Flame AAS using suitable calibration standards.

10.3.2 Procedure for determination of %AsCu and %AsCo

- A 1g sample is weighed in a 250ml plastic bottle with screw cap;
- The sample is leached with 25ml of 5% solution of H_2SO_4 saturated with SO_2 for 1 hour on a mechanical shaker (the addition of SO_2 suppresses sulphide mineral dissolution);
- The solution is filtered and diluted to a known volume;
- The amounts of copper and cobalt are determined by AAS.

The presence of native copper is recognised as it tends to smear during sample preparation. Samples containing native copper are analysed using a screen assay method. In this process, the sample is crushed and sieved, the coarse material dissolved in total and analyzed; the fine material is pulverized, then homogenized and split for analysis. The final result is a proportional combination of the two assays.

10.4 Quality Control and Quality Assurance

A 60kg run-of-mine sample was obtained from No. 1 and No. 3 Shafts of KCM's Konkola Division, and processed as the standard reference sample to be used for the duration of the programme. This sample was crushed and pulverized to the specification of at least 80% < 70µm, thoroughly blended and split into 36 sub-samples. At this stage, a homogeneity test was carried out on 18 November 1997 at both the ZCCM Kalulushi and AVRL laboratories.

These sub samples were further split to produce a total of 1152 individual standard samples stored in plastic bottles. The samples were individually labelled according to the splitting routine. This ensured determination of the total homogeneity of the sample as it was possible to select the (30) aliqots spread through the complete standard sample. The results from the homogeneity test were used for setting the acceptance/re-analysis criteria for the Kalulushi and AVRL laboratories, and both laboratories were held to this.

Strict quality control procedures were implemented and adhered to throughout the programme in order to ensure that all analytical data was subjected to accept/reject criteria to ensure compatibility of data, to monitor variations within batches and between batches and to monitor the benchmark repeatability of the laboratory over time.

The following quality control procedures were adopted:

- Within each batch, 10 per cent of samples (minimum 10 samples) within a batch were repeated as duplicate samples to check repeatability (average batch size = 42 samples);
- In between batch control - a minimum of 10 standard samples per batch were included to monitor between batch variation;
- 10% of samples from each batch were re-analyzed as "quality control" batches at an external laboratory.

10.5 Sample analyses

A total of 3076 borehole core samples were analyzed in 83 batches, with some batches being re-analysed at AVRL. After the first 16 batches were analyzed by the Kalulushi laboratory, the laboratory gave notice that it would be suspending operations due to the privatization of ZCCM.

It was decided to continue the analyses of all KNP core samples at the AVRL laboratory and to use the Anglo American Research Laboratories ("AARL") as an umpire laboratory facility.

Prior to sending routine core samples to the AVRL laboratory, a batch of 30 previously analyzed standard samples were submitted to ensure compatibility. This batch returned higher than expected values for %TCu and %AsCu and was assumed to indicate a problem at the ZCCM laboratory. The values obtained for %TCu compared well with the results of the AVRL homogeneity test of the standard sample. External check analyses from these initial batches had already been submitted to AVRL and AARL as Control Batches 1 and 2 respectively.

Batches 17-29 were analyzed at AVRL. The %TCu values on the standard sample showed a wide variance, and were consistently higher than ZCCM and AARL. AVRL %AsCu results were also consistently higher than ZCCM results and showed a laboratory drift over time. It was established that AVRL was not using the standardized procedure for acid soluble analysis. AVRL reported short-term instrumental drift on their ICP machine. Batches 30-33 were analyzed at AARL until such time that the quality control problems at AVRL were identified and corrected.

In the investigation on quality control problems at AVRL, it was established that the ICP was not providing the accurate reproducible results required. This was confirmed on a number of the samples re-read on another ICP. The best reproducible results were obtained from AAS and the following remedial measures were undertaken:

- All samples for Batches 17-29 originally reporting TCu >1% were reanalyzed on the AAS;
- Adherence to the standard %AsCu determination method. Although this had been adopted, residue material was not being removed after the standard leaching time, resulting in higher than expected values. All samples for Batches 17-29 originally reporting >0.4% ASCu were reanalyzed on the AAS;
- Amended assay certificates were issued for the above batches, and the original assay certificates were destroyed;
- In accordance with QC procedures for an NLA accredited facility, all observations and changes made to the procedures were well documented and filed within the AVRL quality control system;
- Batches 30-33 analyzed by AARL were reanalyzed by AVRL;
- Samples from Batches 1-16 analyzed by ZCCM Kalulushi containing >0.1% TCu were reanalyzed by AVRL. These results are incorporated in Batches 26 and 56.

Only AVRL results for all boreholes were accepted as the official results for the entire programme while having full internal QC procedures in place and assigning AARL as the external umpire/check laboratory.

Results were received electronically and subjected to quality control analysis prior to approval, acceptance and payment.

10.5.1 Historical core resampling programme

In order to check the reliability of the historical assay data, a re-sampling programme of the available core was undertaken, and the samples submitted to the ZCCM laboratory in Kalulushi. Historical assay results exist for the two periods of drilling, i.e. the initial drilling in the 1930's and the drilling programme carried out in the 1970's.

In general, the assay results of the re-samples showed good correlation for %TCu and %TCo with the historical results with the historical. The %AsCu was on average 15% lower than the resample assay results. The variation in the %AsCu is thought to be the effect of exposing the samples to surface conditions over a considerable period of time and being oxidised.

Based on the favourable results of the statistical comparison, the historical data was accepted into the database.

10.6 Reliability of assay results

10.6.1 Duplicates

Half Absolute Relative Deviation (HARD) plots for %TCu and %AsCu have been prepared to compare the original and duplicate assays (Figure 10.1). This comparison strictly examines the analytical reproducibility between the two laboratories and does not examine any variance attributable to sample splitting, other than that variance that arises from the extraction of a sample aliquot from the original pulp sample.

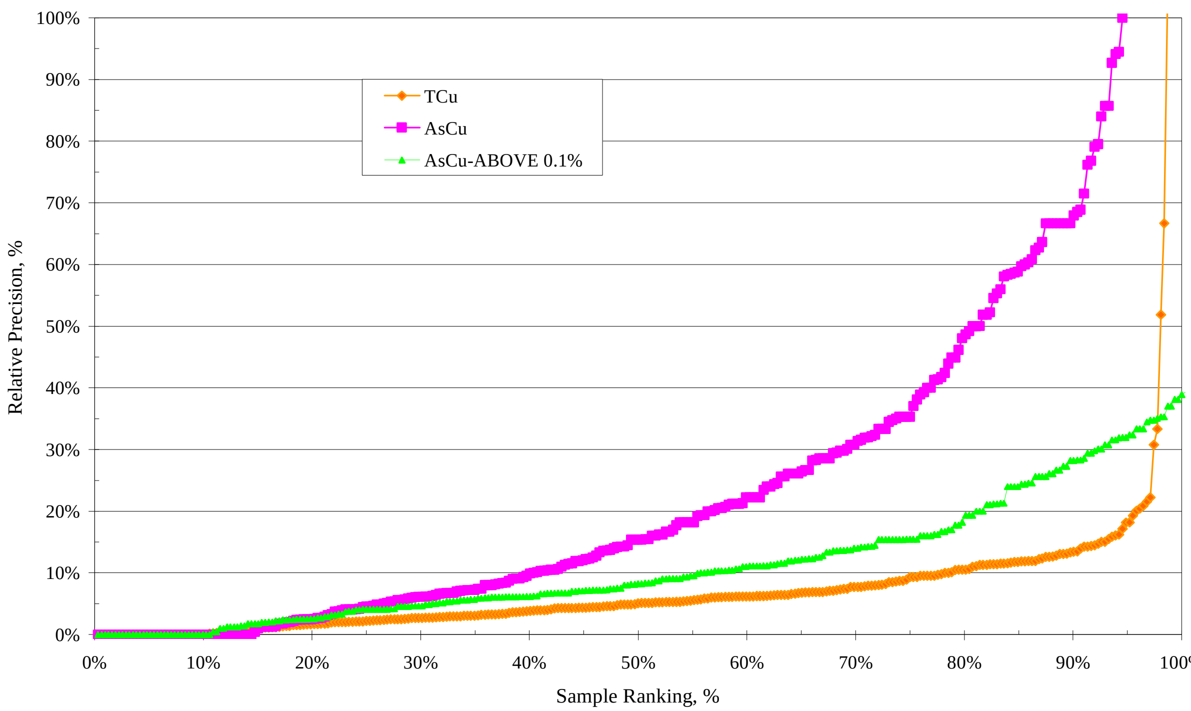

Figure 10.1: Relative repeatability of analyses for %TCu and %AsCu

The HARD plot for %TCu shows good repeatability of the analyses between the laboratories with 80% of the data being within 10% variation. For the %AsCu in total, is generally poor with 40% of the results being within 10% variation. However, when the assays above 1.0% are considered, there is a general improvement to just below 60% within 10% variation.

10.6.2 Reference material

Figures 10.2 and 10.3 show the performance of the laboratories measured against the known values of the standard reference material for %TCu and %AsCu respectively. The values returned are considered acceptable within 10% of the expected value. The average %Cu value from the analyses is 3.10% and the analyses vary within a narrow band of 2.94% and 3.26%, well within the 10% margin.

For the %AsCu, the average value is 0.48 and varies within a band of 0.40% to 0.55%.

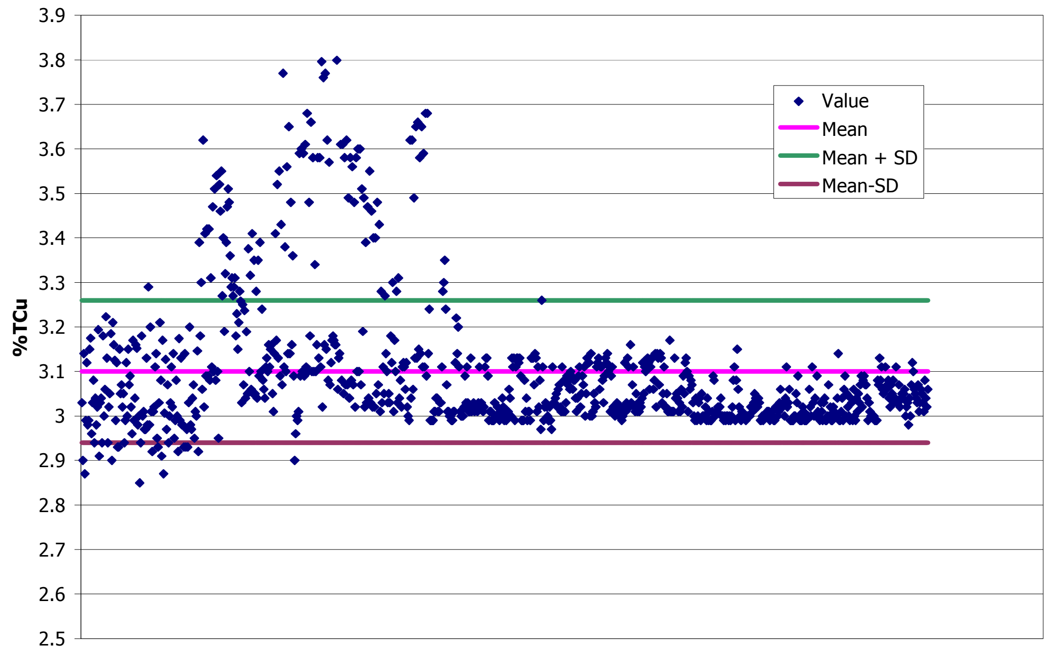

Figure 10.2: Repeatability in the analyses of reference material, %TCu

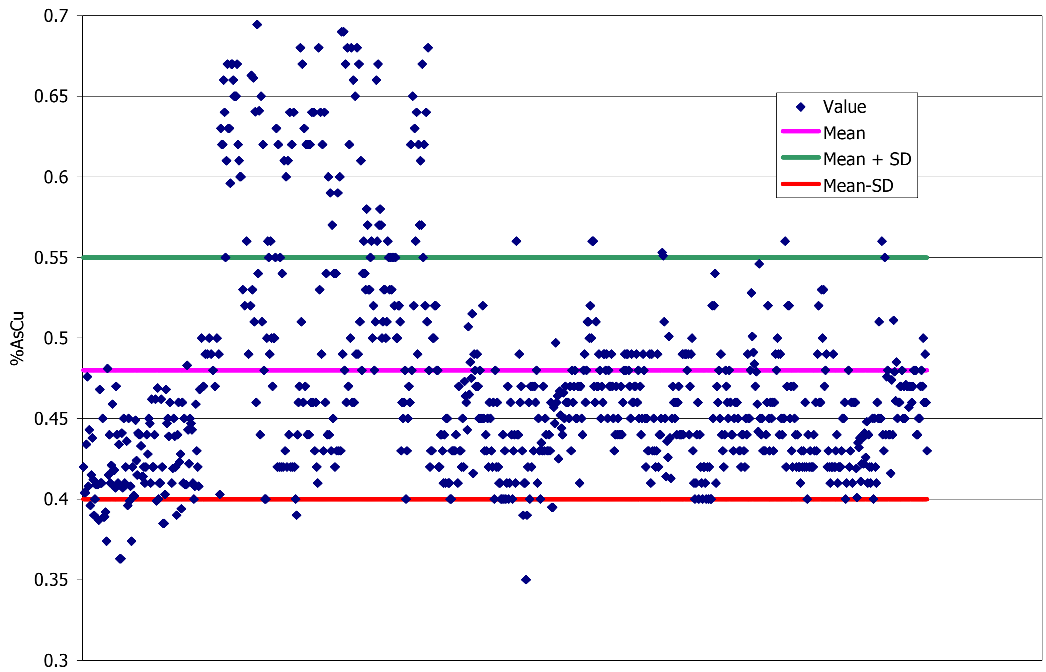

Figure 10.3: Repeatability in the analyses of reference material, %AsCu

Figures 10.2 and 10.3 show the variance in the analyses of the Standard reference material for %TCu and %AsCu respectively. Generally, the bulk of the analyses did show good reproducibility being within the bands of acceptability. The %AsCu shows a bit more scatter, but this can be associated with the problems of analyses at such low values.

SRK have reviewed the quality control procedures and the results are satisfied with the work undertaken and consider the results as representative to the material analysed and as such suitable for use in the Mineral Resource estimation process.

10.7 Bulk Density Determinations

A total of 879 half-core samples were dispatched to Alfred H Knight Laboratories in Kitwe, Zambia for bulk density ("BD") determinations. The half-cores were from various drillholes and from various intersections within the OS1 member.

The BD determinations were undertaken using Archimedes principle: where the half-core is weighed in air first (W1), then the sample is waxed and weighed again in air (W2) and then submerged in water and then weighted again (W3).

The BD is calculated using the following formula:

$$BD = \left\{ \frac{W1}{[W2-W3]-[W2-W1]/0.877} \right\}$$

where

W1 = Weight of dry sample in air.
W2= Weight of dry sample coated with wax in air.
W3 = Weight of sample coated with wax in water.
0.877= Specific gravity of wax.

Statistics from the database are indicated in Table 10.1.

Table 10.1: Statistics from the Relative Density database

	Relative Density, g/cm3	
	ALL	TCu>1.0%
Minimum	1.84	2.12
Maximum	3.14	2.92
Average	**2.57**	**2.59**
Standard deviation	0.16	0.14
Coefficient. of variation	0.06	0.05
Number of Records	879	69

The BD data is very consistent varying within a narrow band of 6% and 5% for the complete dataset and for the dataset with TCu above 1.0% respectively with average values of 2.57/m^3 and 2.59 t/m^3 respectively.The distribution of the BD against %TCu is presented in Figure10.4.

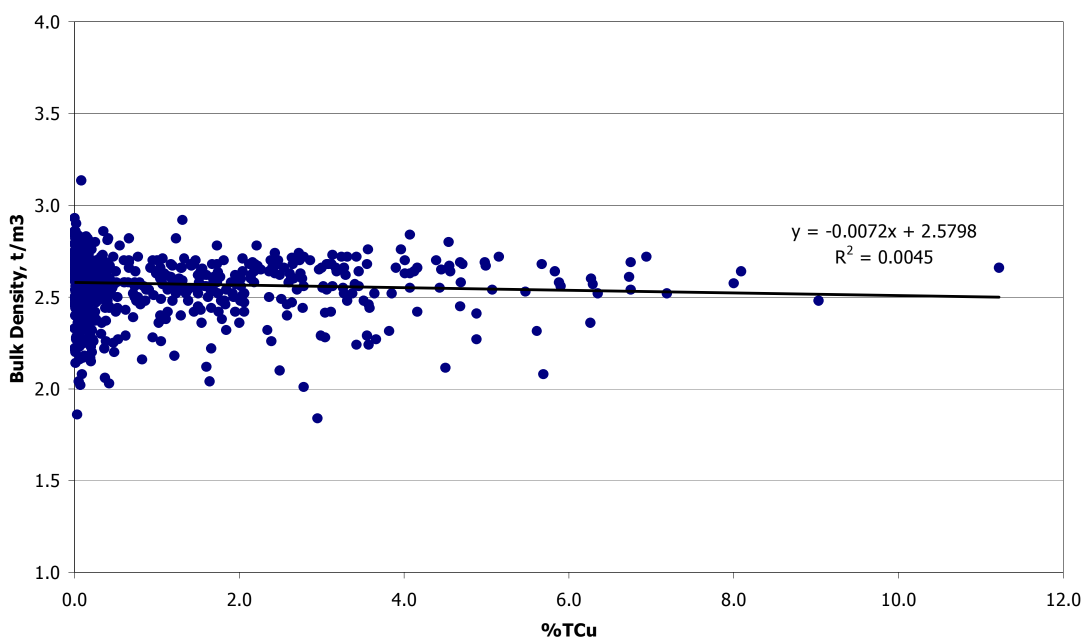

Figure 10.4: Variation of BD with %TCu grade

The BD appears relatively constant irrespective of the copper content (Figure 10.4). This observation supports the use of an average BD of 2.57 t/m^3 for the conversion of volumes to tonnages for the Mineral Resources. This is comparable to the BD of 2.58 t/m^3 in use at the operations of KCM adjacent to the KNP property.

11 Data verification

A validation exercise was undertaken to verify all data before its incorporation into the database for Datamine geological and grade modelling. The purpose of this exercise was to ensure a high degree of standardization and conformity between geologists for all aspects of data capture, as well as to ensure the accuracy of such data and of subsequent calculations and conclusions. As part of the procedures instituted by TEAL, the following parameters were rigorously checked for each borehole:

- Depth meter markings on the core;
- Core recoveries through mineralized intersections;
- Accuracy and conformity in geological, alteration and structural logging;
- Definition of stratigraphic units and correlation thereof;
- Down-the-hole borehole surveys (depth, inclination, direction);
- Spot checks on RQD measurements and joint (Q values) descriptions;
- Sequential sample numbering;
- Sample “from” and “to” depths;
- Sample capture sheets and encoding of sample numbers;
- Core bedding angle measurements in mineralized sections;
- Quality control (statistical analysis of duplicate and standard samples);
- Incorporation of assay data and grade calculations;
- Standardized capture of this information;
- Verification and approval of the final “Ore Zone Report” for each borehole.

The drillhole database was captured into a SABLE database and editing of the SABLE database was carried out by the Senior Geologist, who also approved and signed off the SABLE summary geological and assay logs, grade calculations, quality control reports and final reports on the intercepts of mineralised zones within each borehole.

The database was also the subject of independent verification by outside consultants, GijimaAST in August 2006 and Sphynx in June 2007. SRK has reviewed the database as part of this study and found it to be acceptable for Mineral Resource estimation.

12 Adjacent Properties

The deposits of the Lufilian Arc have produced significant quantities of base metals over the last century. Not withstanding the significant production from this area, there are still large mineral resources and reserves being reported. After a century of mining, the Central African Copperbelt is estimated to contain more than half of the total copper previously defined as mineral resources and reserves. Production in Zambia peaked at about 730 000 t of copper metal in 1969, and the Congo metal production averaged about 450 000 t over many years. Recent production in Zambia has been about 490 000 t.

There are numerous major copper-cobalt mines on the Zambian and Katangan Copperbelt. KCM operate the No. 1 and No. 3 Shafts immediately adjacent to the KNP licence area, extracting mineralised copper bearing rocks from the OS1 member, the same host to the copper mineralisation observed at KNP.

Recent additional resources and reserves of previously known deposits and the recent development of these deposits (i.e. Kansanshi, Lumwana and the Konkola Deep Project) have been reported. The additional investment in primary and secondary development on the Zambian mines is expected to increase substantially finished copper production in Zambia in the next few years.

13 Mineral Processing and Metallurgical Testing

AVRL conducted metallurgical test work between 1997 and 1999 on drill core composites generated during the exploration programme at Konkola North. TEAL restarted the test work in 2006 as part of the KNP feasibility study.

For the feasibility study only the South and East limbs were considered. It was reported that, in general, the mineralised zone from the East Limb had a higher ratio of chalcocite to chalcopyrite, and a rather higher proportion of oxide minerals than the South Limb. There was a general tendency for the %AsCu fraction to decrease with depth for the first 500 vertical metres of the mineralised zone, from as high as 80% to 90% in some places down to 10% to 15%.

The combination of oxide and sulphide copper in the mineralised zone made the metallurgical processing more challenging. Generally, the sulphide copper could be recovered by flotation into a smelter grade concentrate. The oxide copper, on the other hand, did not float as well and poor recoveries and grades were achieved. Since 20% to 25% of the copper in the Konkola North ore is present as oxide copper, the overall flotation recoveries were low. The oxide recovery was increased by the addition of reagents (NaSH or hydroxamate collector) in the oxide flotation stage, albeit at the cost of oxide concentrate grade.

The oxide recovery can also be increased by leaching in sulphuric acid. Various leaching options were considered during the feasibility study, including leaching of concentrates and flotation residues. Several leaching tests were performed to investigate this option.

The AVRL testwork focused mainly on the flotation and leaching of the Konkola North ore. TEAL continued with the work in 2006 and additional flotation, concentration, leaching and communition test work was done.

In addition to the historical test work, the following additional test work was carried out by TEAL:

- Release flotation tests were done to determine the maximum copper recovery from the ore (Alfred H Knight, Zambia, 2006);

- A series of tests was conducted on drillcore material to measure the communition characteristics of the ore (Mintek, RSA and JKTech, Australia, 2006);

- Channel samples collected from the 2nd mining level were used to evaluate a new hydroxamate collector for copper oxides (Mintek, RSA, 2007);

- Channel samples were subjected to heavy-liquid (HLS) and optical sorting tests to upgrade the ore prior to milling (Mintek, RSA, 2007);

- A bulk sample of 43 tons was used during a pilot milling campaign. The campaign was designed to verify the Semi Autogenous Grinding (SAG) / Ball milling design proposed in the DRA feasibility study (Mintek, RSA, 2007);

- Flotation test work was done on the milled ore from the pilot campaign (Mintek, RSA, 2007);

- Leaching test work was done on products from the flotation campaign above to investigate alternative flowsheets. These flowsheets include flotation and leaching processes and were designed to maximise copper oxide recovery (Mintek, RSA, 2007).

In general, the test work indicated that a grind to 80% -75µm is required for optimum flotation recovery. Communition test work ranked the Konkola North ore as a medium to moderately hard ore.

Flotation recovery of the acid-insoluble copper ranged from 70% to 93%. To produce a saleable copper concentrate, a concentrate re-grind, cleaning and re-cleaning were required. Oxide flotation recoveries were lower than the acid insoluble copper ("AICu") recoveries and ranged from 30% to 77% depending on the flotation regime employed.

The test work also indicated that the oxides minerals were readily leachable.

Concentration of the ore by means of heavy-liquid ("HLS") and optical sorting tests were performed at Mintek on –20 mm material. HLS was unsuccessful as upgrading beyond 3% Cu was not achieved. Optical sorting was somewhat more successful, but the upgrading and recoveries were still below requirements.

14 Mineral Resource and Mineral Reserve Estimates

14.1 Mineral Resource Estimation Methodology

The current mineral resources for KNP were estimated by ARM as of February 2008. The estimation methodology is a refinement of the work undertaken by GijimaAST in December 2006. The refinement has been the re-definitions of the mineralised intercepts using the rules to account for intervening lower grade zones where there is a split intersection of mineralised zones. The rules for the re-definitions are documented in this Section.

The 2006 GijimaAST estimate is the precursor to the current estimate. The estimation methodology adopted is summarised.

14.1.1 The 2006 estimation methodology

GijimaAST were contracted by TEAL to undertake a comprehensive geological modelling and resource estimation exercise on the existing data in the KNP. The GijimaAST estimates were based ordinary kriging using a 1.0%TCu cut-off and block models supplied by TEAL.

A two-dimensional ("2D") block modelling approach was adopted: where the block models consisted of fundamental blocks of size 500m x 500m in the X and Y directions, sub-celled to 250m x 250m on the wireframe edges. In the Z-plane, one block of a large dimension was used.

Consistent with the 2D modelling approach, GijimaAST computed the weighted average grade and thickness of intersections of mineralised zone within each drillhole from which the accumulation ("m%"), which is the average %TCu multiplied by the true thickness of the intersection of mineralised zone, was computed. The shape of the wireframe of the mineralised zone was used to estimate the true dip at the intersection of the drillhole and the wireframe. True thickness was then calculated by dividing the drilled thickness by the cosine of the true dip.

For the variography, GijimaAST examined the spatial distribution of the accumulation for TCu and AsCu and the true thickness in the unfolded space and generated omni-directional variograms for from the drillhole intersections within KNP. The omni-directional variograms are listed in Table 14.1.

Table 14.1: Omni-directional variogram parameters for thickness and m%

Variable	nugget, (c0)	sill1, (c1)	range1 (a1), m	sill2, (c2)	range2 (a2), m
True_Thick	0.1120129	0.18849	275	0.699497	456

mTCu (m%)	0.3653837	0.634614	600		
mAsCu (m%)	2.58	2.68	241.94		

For each block, estimates were made from the drillhole data for the variables, True_thick, m%TCu and m%AsCu using the respective variogram parameters listed above and samples sourced within a search neighbourhood of 500m in the first pass. In the second pass, blocks not estimated within the first pass were estimated using samples within a neighbourhood search of twice the first pass or 100m. Blocks that remained un-estimated after the second pass were estimated using samples within a neighbourhood search of four times the first pass or 2000m.

The number of minimum and maximum number of samples used to estimate a block were 3 and 100 respectively in the first pass, 2 and 5 for the second and third passes. The %TCu and %AsCu grades were then back-calculated from the estimates of accumulation and true thickness.

GijimaAST used a density of 2.75 t/m^3 to convert volume to tonnages and declared Mineral Resources for KNP. These are tabulated in Table 14.2 and compared with previous estimates by ARM.

Table 14.2: 2006 Mineral Resources for the KNP compared with ARM's 2005 estimate

	GijimaAST 2006			ARM 2005		
	Mt	%TCu	%AsCu	Mt	%TCu	%AsCu
East Limb	30.762	2.52	0.70	29.43	2.52	0.48
South Limb	41.864	1.80	0.58	49.4	1.92	0.41
Total	**72.626**	**2.11**	**0.63**	**78.83**	**2.14**	**0.44**

14.2 Data Validation

A total of 237 boreholes were referenced in the construction of the wireframes. These included parent holes and up to two deflections. Of these boreholes, 141 holes were drilled within the TEAL property and the remaining 96 holes are on the KCM property but were made available to TEAL to assist in the geological interpretations on the edges of the licence boundaries and also for the grade interpolation.

The sample logs and assay data were extensively validated as stated in Section 11 of this report.

14.3 Definition of Mineralised Zones

The top and bottom contacts of the mineralised zones are based on a 1% TCu cut-off which appears to be sharply defined. However within the mineralised zone are internal lower grade zones (less than 1.0% TCu) which result in a split orebody intersection consisting of a top mineralised zone, an intermediate lower grade zone and a bottom mineralised zone. The characterisation of the mineralised envelope from these three intersections is critical in the determinations of the Mineral Resources. Rules were developed to ascertain whether:

- to combine the three zones into one mineralised zone, or;

- to take either the top or bottom mineralised zone as the main mineralised zone.

Sphynx Consulting cc ("Sphynx") was contracted in 2007 to review the drillhole data and generate the geological model for the project. As part of this process, Sphynx re-examined all the drillhole intercepts of mineralised zones and reviewed the criterion adopted in the generation of the intercepts. The criterion for the split mineralised zone was to test whether the average weighted grade of the combined mineralised zones and lower grade zone was above the 1.0%TCu cut-off grade. This test was performed separately on the combination of the top mineralised zone and lower grade zone and also for the bottom mineralised zone and lower grade zone.

Where both combinations were above the cut-off grade, the mineralised zone is defined as being the top of the top mineralised zone and extends all the way to the bottom of the bottom mineralised zone.

Where one of the combinations is not above the cut-off grade, then a selection is made on whether the top mineralised one or the bottom mineralised zone is selected as the main mineralised zone.

Table 14.3 illustrates the concept, where the combination of top and waste zone and bottom and waste zone are both above 1.0% TCu cut-off. The mineralised zone is then 17.06m with average TCu grade of 2.54%.

Table 14.3: Illustration of split mineralised zones combined into one

BHID	LENGTH	%TCu	Zone		zonal averages	combined Avgs	Final Zone
KK04_D0	0.92	0.06					
KK04_D0	0.30	0.06					
KK04_D0	0.61	1.97	1				
KK04_D0	0.61	2.73	1				
KK04_D0	0.61	3.19	1		2.40% @ 3.05m		
KK04_D0	0.91	2.05	1				
KK04_D0	0.31	2.05	1			1.13% @ 7.23m	
KK04_D0	0.30	0.21					
KK04_D0	0.31	0.21					
KK04_D0	0.61	0.21			0.3% @ 3.05m		
KK04_D0	1.22	0.06					
KK04_D0	0.61	0.96					2.54% @ 17.06m
KK04_D0	0.61	1.95	1				
KK04_D0	1.22	3.21	1			2.57% @ 14.01m	
KK04_D0	1.21	3.27	1				
KK04_D0	1.21	2.76	1				
KK04_D0	1.22	2.87	1		6.41% @ 10.96m		
KK04_D0	1.22	3.63	1				
KK04_D0	1.22	3.87	1				
KK04_D0	1.22	3.04	1				
KK04_D0	1.22	3.74	1				
KK04_D0	0.61	2.88	1				
KK04_D0	0.61	0.02					
KK04_D0	0.61	0.04					

Table 14.4 illustrates the case where the grades of one of the mineralised zones cannot carry the lower grade zone above the cut-off.

Table 14.4: Illustration of split mineralised zones

BHID	LENGTH	%TCu	Zone	zonal averages	combined Avgs	Final Zone
KK16_D0	0.28	0.04				
KK16_D0	0.61	0.57				
KK16_D0	0.33	1.16	2			
KK16_D0	0.28	1.16	2	1.19% @ 1.22m		
KK16_D0	0.33	1.22	2			
KK16_D0	0.28	1.22	2			
KK16_D0	0.33	0.75			0.82% @ 2.43m	
KK16_D0	0.28	0.75				
KK16_D0	0.33	0.12		0.44% @ 1.21m		
KK16_D0	0.27	0.12				
KK16_D0	0.01	1.47	1			
KK16_D0	0.33	1.47	1			
KK16_D0	0.27	1.47	1			
KK16_D0	0.61	1.44	1			
KK16_D0	0.33	1.61	1			
KK16_D0	0.28	1.61	1		1.50% @ 6.4m	
KK16_D0	0.33	0.95	1			
KK16_D0	0.28	0.95	1	1.75% @ 5.19m		1.75% @ 5.19m
KK16_D0	0.33	2.39	1			
KK16_D0	0.28	2.39	1			
KK16_D0	0.61	1.49	1			
KK16_D0	0.33	2.39	1			
KK16_D0	0.28	2.39	1			
KK16_D0	0.33	1.95	1			
KK16_D0	0.28	1.95	1			
KK16_D0	0.31	2.4	1			
KK16_D0	0.02	0.44				
KK16_D0	0.28	0.44				

In Table 14.4, the combination of the top mineralised zone and the intervening lower grade zone average 0.82%TCu over a thickness of 2.43m, below the cut-off grade of 1,.0%TCu.

In this case, the mineralised zone for the drillhole KK16 is 1.75%TCu over a thickness of 5.19m.

The exercise of defining the intercepts of the ore zones was a manual process checking to ensure that the criteria were met. These re-defined drillhole intercepts were used to define the footwall and hangingwall envelopes of the mineralised zones within each of the three resource areas, East Limb, South Limb and Area A.

14.4 Summary of drillhole intersections

The data base contains at total of 125 drillholes of which 99 have intersected mineralised zones and 26 were either stopped short or did not intersect mineralised zones above the requisite cut-off grade within the OS1 member. Table 14.5 lists the drillholes with intersections of mineralised material at a 1.0% TCu cut-off.

Table 14.5: Drillhole intersections of mineralised zones on a 1.0% TCu cut-off

BHID	From	To	Drilled	True Thick [1]	Core Recvy, %	%TCu	%AsCu
KK1	97.84	98.76	0.92	**0.85**	100	**3.68**	-
KK3	129.86	137.16	7.30	**6.26**	82	**2.59**	**0.22**
KK2	75.29	83.21	7.92	**5.89**	85	**2.77**	**1.03**
KK4	271.58	282.55	10.97	**8.03**	96	**3.20**	**0.23**
KK5	99.06	113.08	14.02	**7.22**	58	**2.55**	**0.77**
KK6	149.96	151.79	1.83	**1.63**	81	**2.25**	**0.20**
KK7	292.00	293.22	1.22	**0.95**	70	**1.64**	**0.39**
KK8	254.51	259.99	5.48	**4.20**	91	**3.22**	**0.63**
KK9	137.46	145.08	7.62	**5.20**	75	**2.53**	**0.33**
KK10	195.68	203.00	7.32	**5.60**	100	**2.81**	**0.23**
KK11	81.08	88.10	7.02	**4.79**	78	**2.39**	**1.36**
KK12	270.05	276.76	6.71	**5.41**	98	**3.07**	**0.21**
KK13	140.82	143.87	3.05	**1.48**	100	**2.00**	**1.25**
KK14	171.91	178.00	6.09	**5.17**	92	**2.13**	**0.57**
KK16	267.00	272.19	5.19	**3.73**	81	**1.75**	**0.04**
KK17	498.96	502.01	3.05	**2.43**	87	**2.69**	**0.02**
KK18	570.59	575.46	4.87	**3.44**	77	**2.50**	**0.57**
KK19	510.24	515.72	5.48	**4.38**	85	**3.05**	**0.02**
KK21	452.63	478.84	26.21	**5.90**	82	**1.75**	**0.16**
KK22	292.61	296.27	3.66	**3.29**	90	**2.05**	**0.02**
KK24	240.18	243.54	3.36	**2.84**	74	**2.46**	**0.75**
KK25	213.97	215.19	1.22	**1.11**	67	**1.28**	**0.01**
KK27	268.96	276.28	7.32	**5.77**	98	**2.77**	**0.32**
KK28	270.66	271.01	0.35	**0.33**	100	**1.82**	**0.04**
KK30	161.85	165.24	3.39	**2.81**	97	**2.56**	**0.14**
KK31	127.04	134.72	7.68	**5.71**	93	**1.97**	**0.82**
KK32	235.66	239.38	3.72	**3.12**	94	**2.65**	**0.47**
KLB82	749.81	760.17	10.36	**9.74**	95	**3.82**	**2.57**
KLB85	815.96	833.03	17.07	**12.28**	98	**2.37**	**0.68**
KLB91	1039.06	1041.20	2.14	**1.84**	100	**1.69**	**1.49**
KLB95	1199.42	1205.21	5.79	**5.73**	97	**3.10**	**0.46**
KLB96	1033.98	1038.07	4.09	**3.79**	100	**2.62**	**1.31**
KLB113	814.48	817.20	2.72	**2.65**	85	**1.74**	**0.29**
KLB114	730.01	730.92	0.90	**0.84**	100	**1.37**	**1.01**
KN2	1131.05	1140.41	9.36	**8.87**	98	**3.12**	**0.28**
KN3A	733.40	745.10	11.70	**5.89**	100	**3.93**	**1.00**
KN4	735.60	743.53	7.93	**6.93**	99	**2.10**	**0.23**
KN5	750.53	752.53	2.00	**1.97**	100	**3.07**	**0.27**
KN6B	1070.82	1073.21	2.39	**2.07**	100	**1.77**	**0.20**
KN7	614.94	615.59	0.65	**0.53**	100	**1.99**	**0.29**
KN9	1137.00	1139.88	2.88	**2.88**	100	**2.38**	**1.22**
KN10	905.54	906.50	0.96	**0.96**	100	**1.07**	**0.96**
KN11	679.14	681.39	2.25	**1.59**	82	**2.35**	**0.16**
KN12	705.50	709.92	4.42	**2.88**	100	**1.28**	**0.03**
KN14A	525.85	530.50	4.65	**4.06**	100	**2.18**	**0.23**
KN15	414.30	414.88	0.58	**0.56**	100	**1.08**	**0.82**
KN16	1280.45	1293.00	12.55	**9.75**	97	**5.62**	**3.08**
KN17	1267.90	1270.48	2.58	**2.45**	98	**1.76**	**1.16**
KN18	1590.73	1595.12	4.39	**3.77**	97	**2.34**	**0.63**
KN19	1128.66	1137.42	8.76	**7.50**	96	**2.74**	**1.13**
KN1A	1163.09	1174.00	10.91	**9.81**	98	**2.10**	**0.09**
KN20	1461.95	1466.00	4.05	**3.60**	100	**1.89**	**0.79**
KN22	1114.95	1121.08	6.13	**5.70**	100	**2.00**	**0.50**
BHID	**From**	**To**	**Drilled**	**True**	**Core**	**%TCu**	**%AsCu**

				Thick [1]	Recvy, %		
KN23	1642.68	1651.52	8.84	**6.22**	100	**2.22**	**0.12**
KN24	499.73	505.70	5.97	**3.17**	97	**3.14**	**0.17**
KN25	389.70	395.26	5.56	**4.89**	100	**2.24**	**0.25**
KN26	1033.76	1036.29	2.53	**2.27**	96	**1.74**	**0.21**
KN27	572.37	576.80	4.43	**4.01**	100	**2.32**	**0.20**
KN28	587.58	595.56	7.98	**5.70**	99	**3.28**	**0.63**
KN29	1016.75	1022.95	6.20	**3.50**	100	**3.20**	**0.62**
KN30A	921.48	925.88	4.40	**2.05**	100	**1.97**	**0.20**
KN32	1271.84	1280.26	8.42	**5.91**	100	**2.51**	**0.34**
KN34	562.63	564.84	2.21	**2.09**	97	**2.37**	**0.42**
KN35	768.15	772.61	4.46	**3.76**	99	**2.60**	**0.40**
KN36	380.15	384.68	4.53	**4.32**	99	**2.94**	**0.61**
KN37	1049.07	1056.13	7.06	**3.15**	97	**3.08**	**0.14**
KNS1	187.74	197.84	10.10	**5.83**	100	**2.37**	**0.20**
KNS2	112.50	120.11	7.61	**5.42**	99	**4.20**	**0.58**
KNS4	122.39	127.13	4.74	**3.76**	100	**1.86**	**0.86**
KNS3	115.50	121.90	6.40	**4.62**	99	**3.54**	**1.32**
KNS5	135.66	144.15	8.49	**6.31**	100	**1.97**	**0.29**
KNS6	70.60	75.82	5.22	**4.82**	100	**3.47**	**0.96**
KNS7	90.73	98.15	7.42	**4.46**	100	**2.18**	**1.06**
KNS8	112.24	120.00	7.76	**6.36**	100	**2.77**	**0.71**
KNS9	82.68	90.69	8.01	**6.56**	100	**2.58**	**0.94**
KNS10	104.35	110.90	6.55	**5.52**	100	**2.22**	**0.40**
KNS11	60.45	68.07	7.62	**6.27**	100	**3.12**	**0.62**
KNS12	58.14	64.45	6.31	**5.72**	100	**2.48**	**1.64**
KNS13	81.27	87.53	6.26	**5.21**	99	**4.01**	**0.71**
KNS14	36.19	42.89	6.70	**3.97**	99	**1.71**	**1.51**
KNS16	66.12	68.58	2.46	**2.30**	100	**2.00**	**1.58**
KNS17	44.00	47.30	3.30	**2.18**	100	**2.09**	**2.00**
KNS19	69.43	76.30	6.87	**4.49**	96	**1.91**	**1.59**
KNS20	48.13	50.52	2.39	**1.54**	100	**1.27**	**0.89**
KNS23	63.00	64.57	1.57	**1.05**	100	**2.63**	**2.31**
KNS27	90.95	92.13	1.18	**0.88**	98	**2.33**	**1.31**
KNS28	96.97	99.45	2.48	**1.61**	98	**1.79**	**0.99**
KNS29	81.93	82.27	0.34	**0.24**	97	**2.49**	**2.38**
KNS30	128.04	131.67	3.63	**3.27**	100	**2.38**	**0.32**
KNS31	48.10	49.17	1.07	**0.90**	100	**1.86**	**1.77**
KNS32A	142.05	154.78	12.73	**7.38**	98	**3.02**	**1.01**
KNS33	80.15	87.61	7.46	**6.35**	100	**2.67**	**0.66**
KNS34	148.56	160.23	11.67	**5.71**	100	**2.86**	**0.50**
KNS35	104.42	114.00	9.58	**7.27**	99	**2.34**	**0.55**
KNS36	9.00	9.70	0.70	**0.45**	55	**1.16**	**1.09**
KNS37	92.79	95.93	3.14	**2.40**	98	**2.13**	**0.19**
KNS38	107.83	108.34	0.51	**0.33**	96	**1.35**	**1.10**
KNS39	163.80	164.65	0.85	**0.74**	100	**2.36**	**0.61**
KNS40	86.73	87.97	1.24	**0.88**	99	**1.99**	**0.63**

1 ***true thickness calculated from the core angle of the strata relative to the inclination of the hole***

14.5 Data distribution within the Resources Areas

The KNP has been sub-divided into three Resource areas, South Limb, East Limb and Area A. Figure 14.1 shows the drillhole data distribution the average intersected %TCu grade in each drillhole within the resources areas.

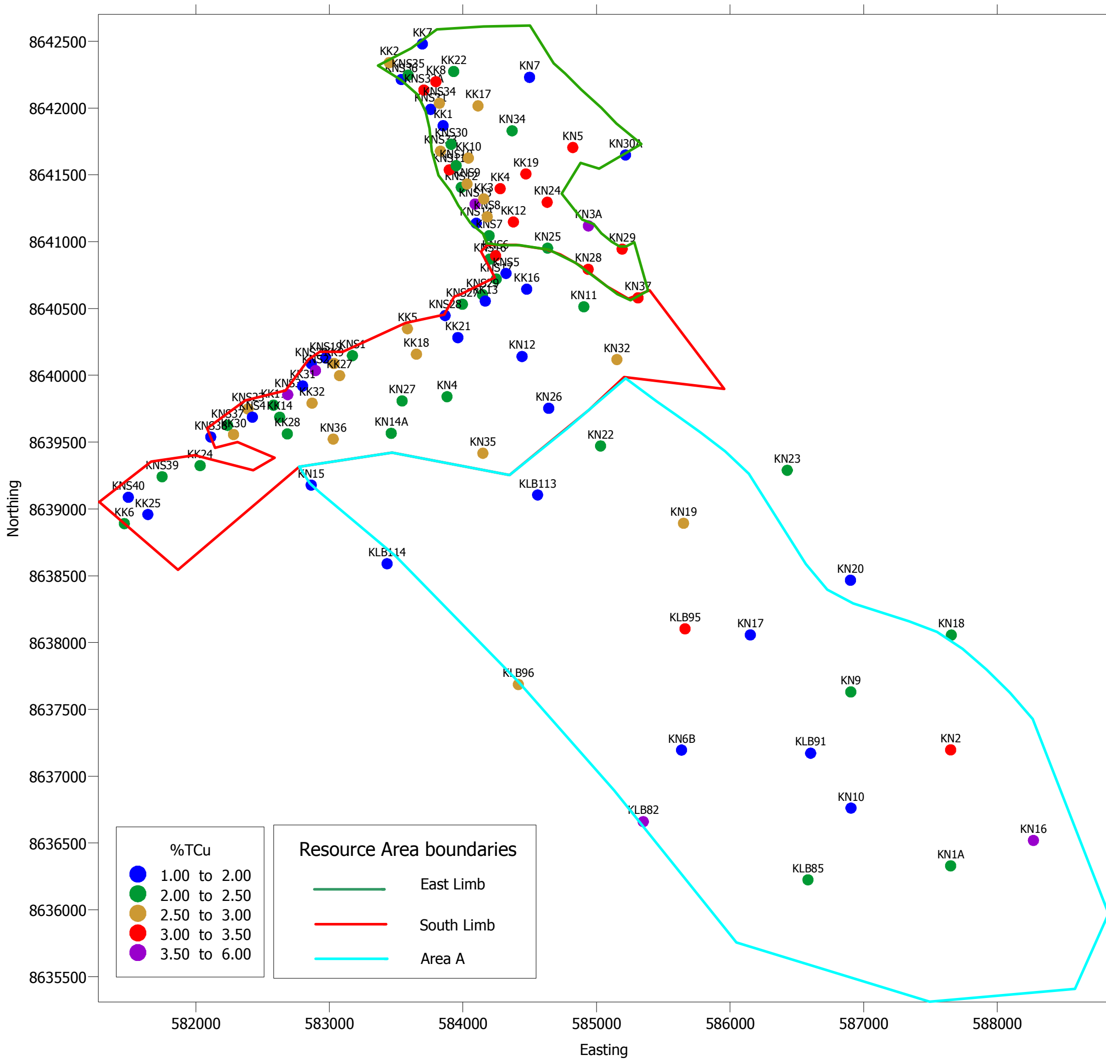

Figure 14.1: Distribution of drillhole intersections and defined resource areas

14.6 Modelling of the mineralised zones

ARM used Datamine software package for the geological and grade modelling.

Wireframes were generated within each of the resource areas defining the extent of the mineralised zones. A 2D block modelling approach was adopted where the block sizes of 100m x 100m in the X and Y directions were generated with one 3000m high block in the Z direction. This allows for the generation of one block in the Z direction.

14.7 Grade Interpolation

For the grade estimation, ARM adopted the 2D approach used by GijimaAST in 2006, where estimates for True thickness, m%TCu and m%AsCu were interpolated into each of the blocks within the wireframes in each of the three project areas using the 2006 omni-directional variograms generated from the unfolded space.

The %TCu and %AsCu grades were then back-calculated from the estimates of accumulation and true thickness.

14.8 Mineral Resources

Mineral Resources are reported according to the defined project areas into East Limb, South Limb and Area A.

Based on bulk density determinations, an average value of 2.57 t/m^3 has been used to convert volume to tonnages. All previous Mineral Resource declarations had been based on a tonnage factor of 2.75 t/m^3 and as such the current Mineral Resource statement is lower than previous estimates on this account.

For the classification of the Mineral Resources, ARM reviewed the GijimaAST omni-directional variograms and selected the range for the True Thickness variable of 275m and downgraded it to 250m. ARM used the 250m range as the criterion for the classification into the various Mineral Resource categories taking into account the spatial distribution of the drillhole data and generated circles around drillhole location points of radii. The following criteria was applied:

- Within two-thirds of the 250m range (165m) were considered as within the Measured Mineral Resources;
- Within 165 to 250m were considered as Indicated Mineral resources;
- Beyond 250m were classified into the Inferred Mineral resource.

Table 14.6 presents ARM’S estimates and classification of the Mineral resource as of February 2008 within the KNP while Figure 14.2 shows ARM’s categories of Mineral Resources.

Table 14.6: ARM's Classified Mineral Resources statement as of February 2008

Classification		Measured			Indicated			Inferred		
Resource Area	**Sub-area**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**
South Limb	Block1	10	2.34	0.52	-	-	-	-	-	-
	Block 3	-	-	-	1.1	1.86	0.66	-	-	-
	Block 4	-	-	-	12.49	2.31	0.47	-	-	-
	Block 5	-	-	-	3.47	2.11	0.38	-	-	-
	Block 7	-	-	-	-	-	-	3.06	2.16	0.67
	Block 9	-	-	-	-	-	-	18.25	2.20	0.37
Sub-Total		**10**	**2.34**	**0.52**	**17.06**	**2.24**	**0.46**	**21.31**	**2.19**	**0.41**
East Limb	Block 2	7.25	2.55	0.62	-	-	-	-	-	-
	Block 6	-	-	-	11.25	2.82	0.49	-	-	-
	Block 8	-	-	-	-	-	-	11.08	2.63	0.5
Sub-Total		**7.25**	**2.55**	**0.62**	**11.25**	**2.82**	**0.49**	**11.08**	**2.63**	**0.50**
Area A		-	-	-	-	-	-	220.78	2.63	1.09
		-	-	-	-	-	-	-	-	-
Sub-Total		-	-	-	-	-	-	**220.78**	**2.63**	**1.09**
Total by Category		**17.3**	**2.43**	**0.56**	**28.31**	**2.47**	**0.47**	**253.17**	**2.59**	**1.01**
Total Measured and Indicated		**45.6**	**2.45**	**0.51**	-	-	-	-	-	-

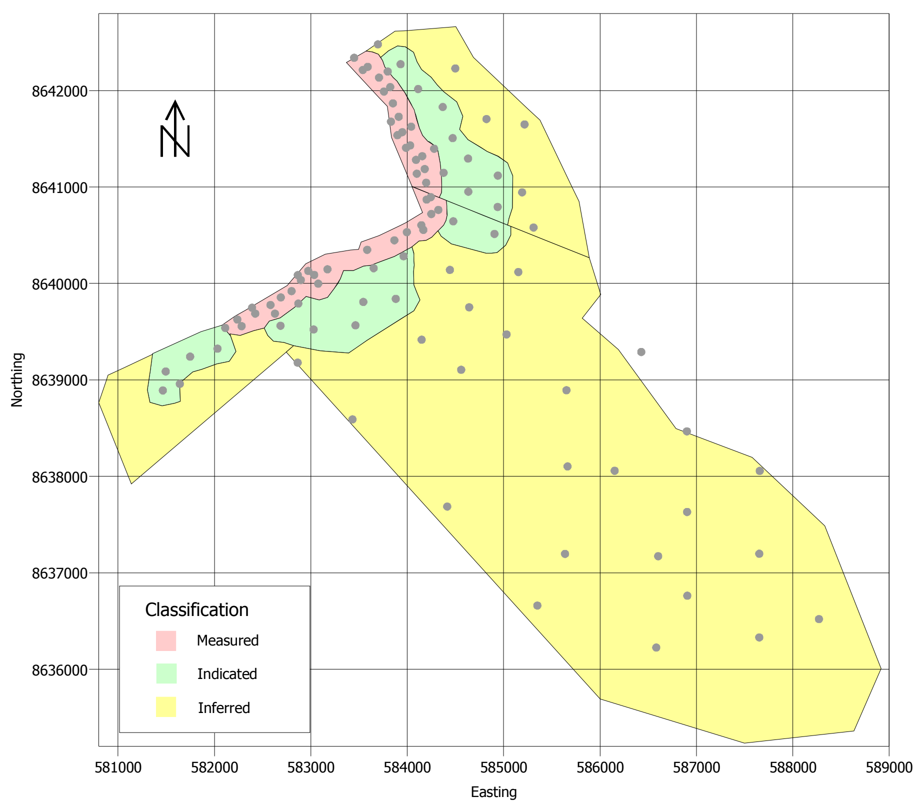

Figure 14.2: ARM's classification of Mineral Resources for the KNP

14.9 Mineral Reserves

Currently there are no Mineral Reserves estimates for TEAL concerning the KNP. The KNP has only Mineral Resources.

15 SRK's Review of the Mineral Resources

SRK has reviewed the input data provided and the model estimates generated by ARM. SRK has done random checks on the database and to provide a level of comfort in the estimates generated by ARM. The Mineral Resource estimates quoted in this report have been generated by ARM. However these have been verified by SRK who independently modelled the Mineral Resource models for KNP and geostatistically estimates grades into the model and these were found to be comparable to the estimates by ARM.

For each of the project areas, East Limb, South Limb and Area A, composite data and block models containing the ARM's estimates were provided. The composite files provided contained one intercept of the mineralised zone for each drillhole.

15.1 Data distribution

Figure 15.1 shows the distribution of the drillholes within each of the resource areas, South Limb, East Limb and Area A, and.

Figures 15.1, 15.2 and 15.3 provide more details in the drillhole and %TCu grade distributions for the South Limb, East Limb and Area A respectively.

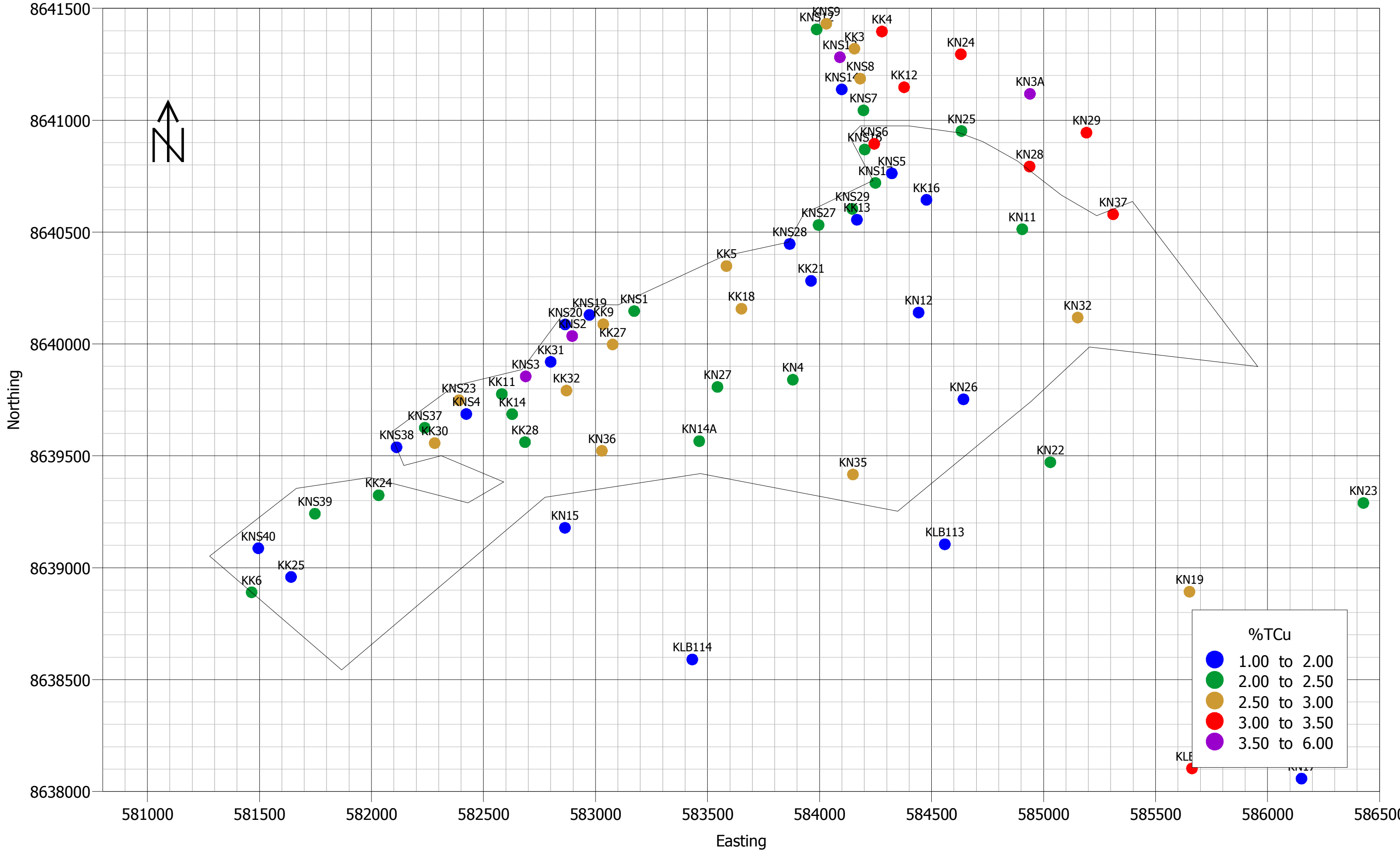

Figure 15.1: South Limb - Drillhole and %TCu grade distribution

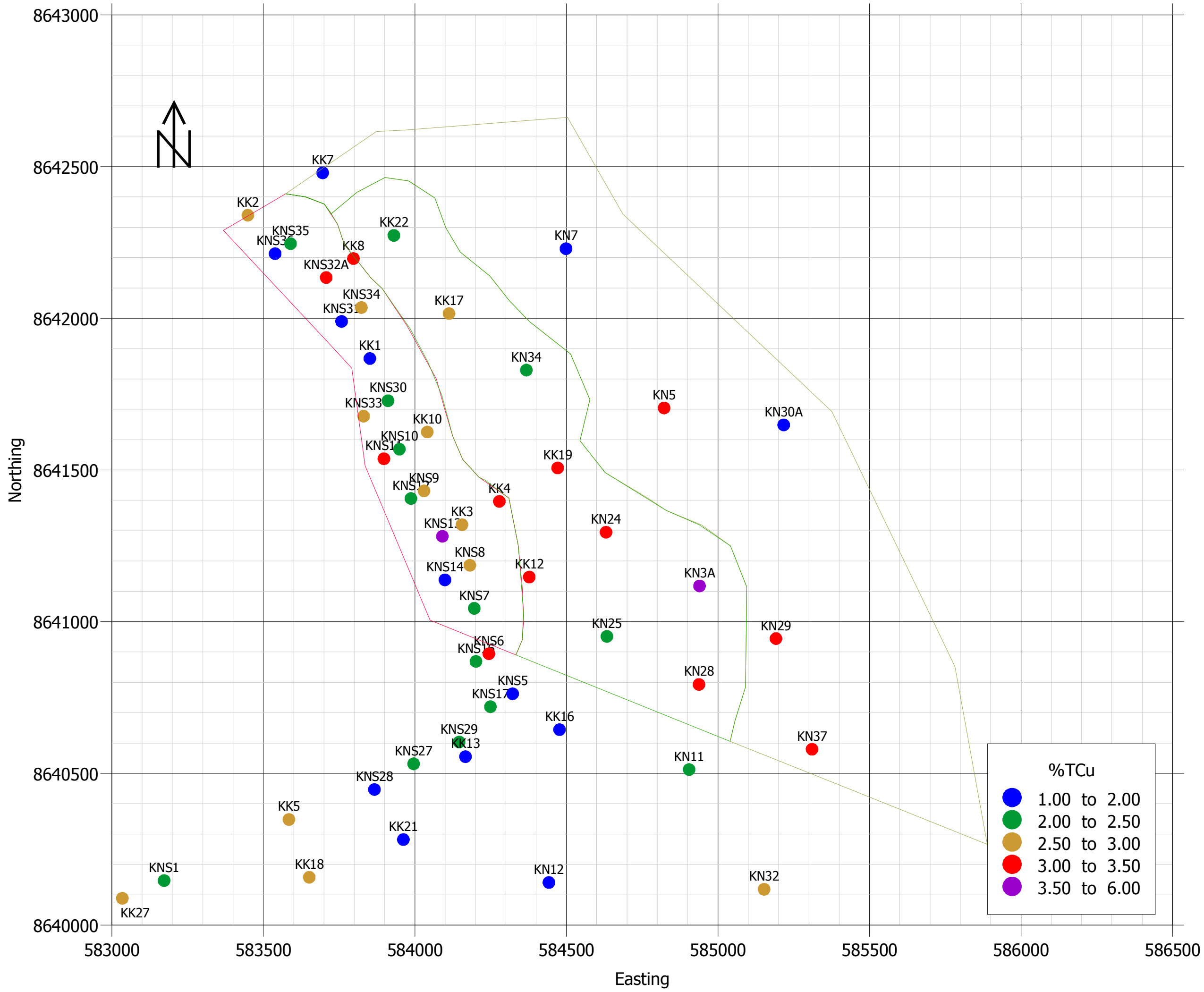

Figure 15.2: East Limb - Drillhole and %TCu grade distribution

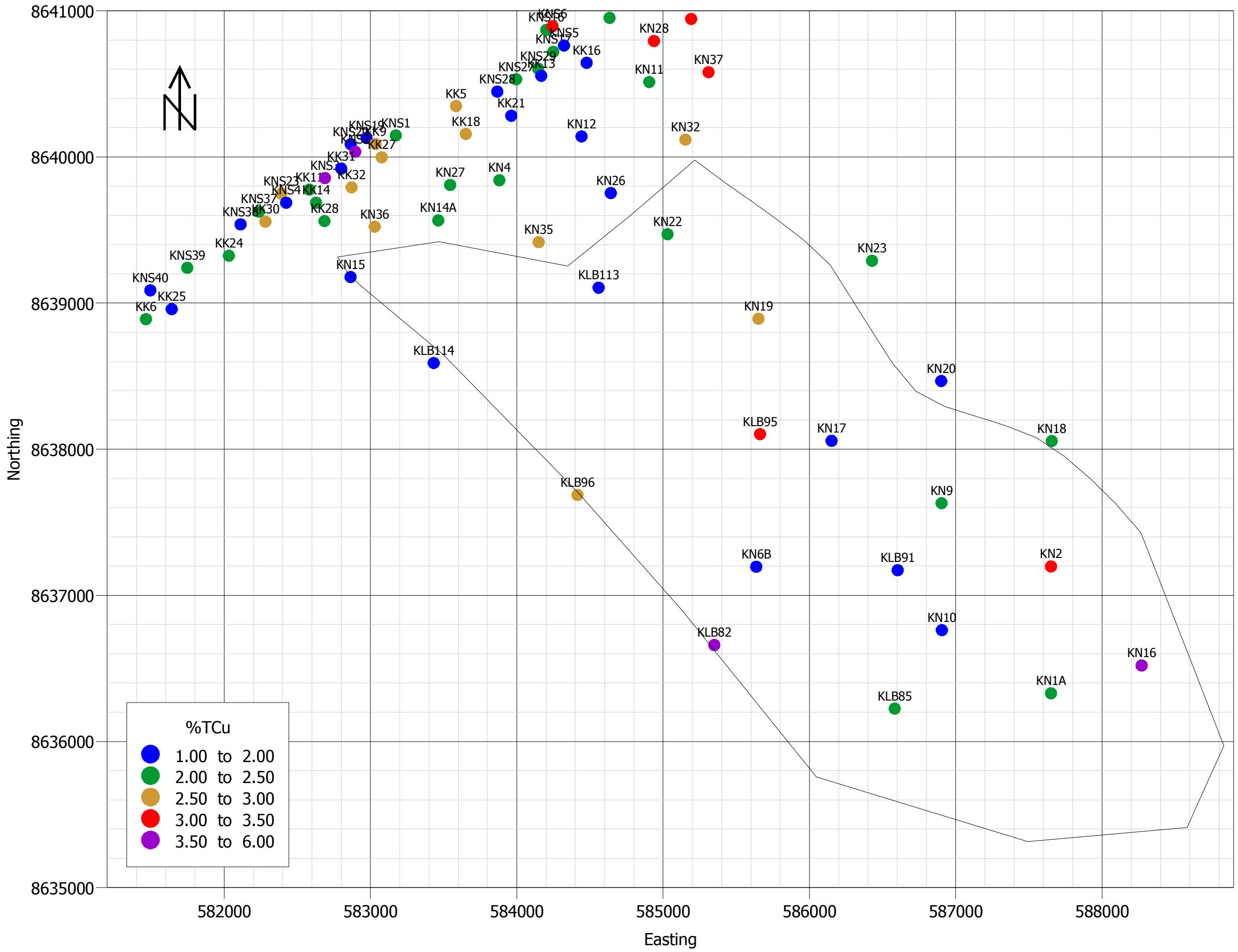

Figure 15.3: Area A - Drillhole and %TCu grade distribution

15.1.1 South Limb

The drillhole spacing in the South limb varies from about 100m to over 400m along strike and the bulk of the data along the up-dip section of the limb is covered by drillholes within 150m spacing. In most cases there are at least two drillholes from the same location, but drilled at different orientations to provide two intersections of the mineralised zone about 70 to 150m apart down-dip.

The grade distribution indicates relatively lower grades in the south-west, and extreme north-east surrounding relatively higher grades in the central potion of the limb (figure 15.1). Overall the grade varies between 2% to 3% TCu.

15.1.2 East Limb

In the East limb, the drillholes are spaced on average about 120m to 170m, averaging abut 150m along strike. As in the South Limb, there are two intersections from most drillhole positions in the up-dip portion of the limb. A secondary fence line runs about 300m to 500m along the strike and is sub-parallel to the up-dip line of holes. Drillholes along this fence line are 300m to 400m apart and generally only one drillhole was done. Further down-dip are scattered holes at much wider spacing.

Generally, the intersections in the East Limb average 2% to 3% TCu, but vary locally with relatively higher %TCu intersections further down dip (Figure 15.2).

15.1.3 Area A

Area A is much bigger in area than the South and East limbs and has been investigated by drillholes along 5 fence lines spaced between 500m to 800m apart. Along the fence lines the minimum spacing between drillholes is about 500m and the maximum over 1500m.

In terms of grade distribution, there block contains relatively lower grade intersections with sporadic higher grade intersections (Figure 15.3).

15.2 Comparisons of composite data and block estimates

The distributions of the block estimates are compared with those of the composite data.

15.2.1 Statistics

Histograms and statistical tables of the composites within each resource area are compared with the block estimate for the respective resource zone in Figures 15.4, 15.5 and 15.6.

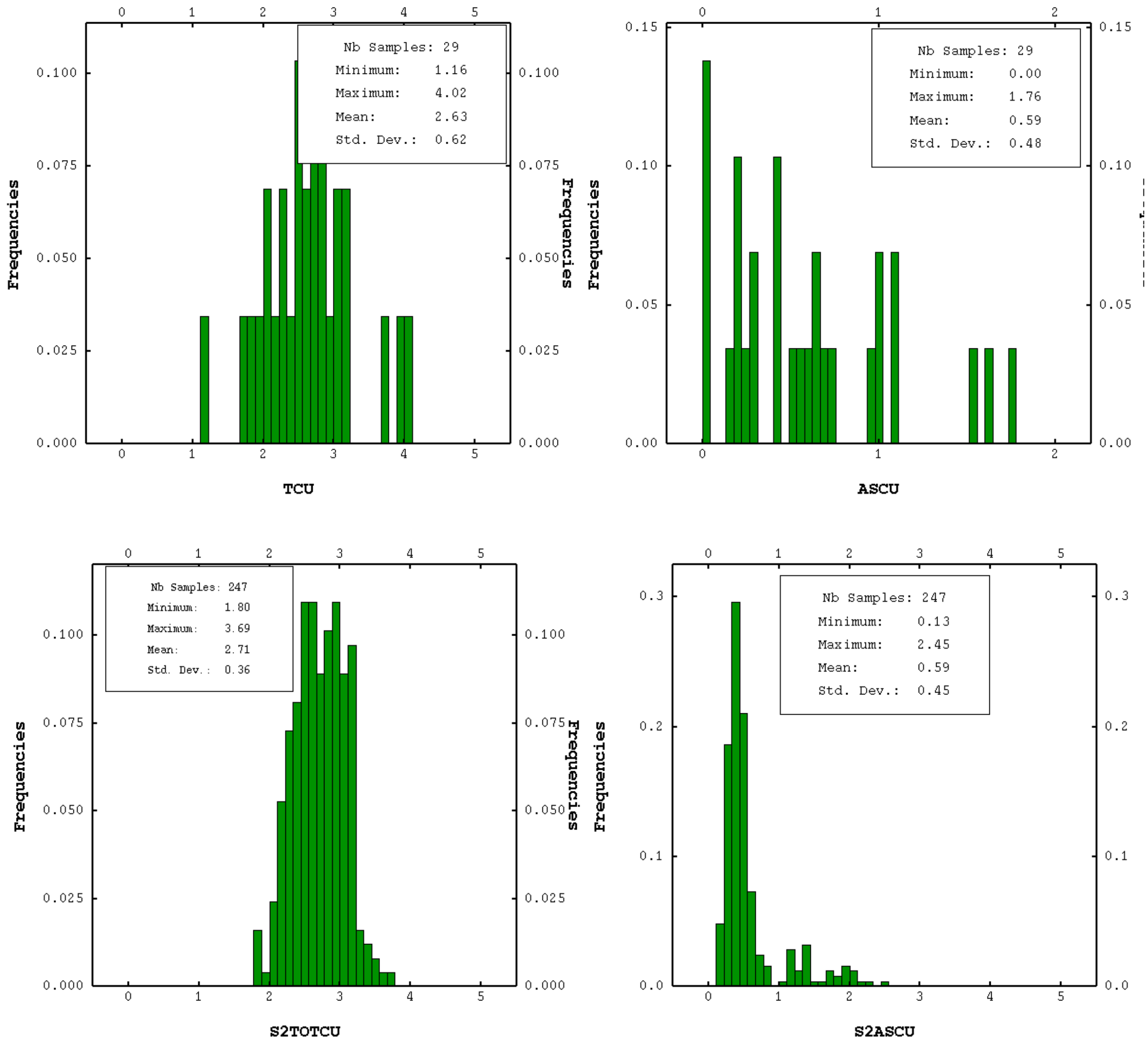

Figure 15.4: East limb histograms - Comparisons of composite data and block model

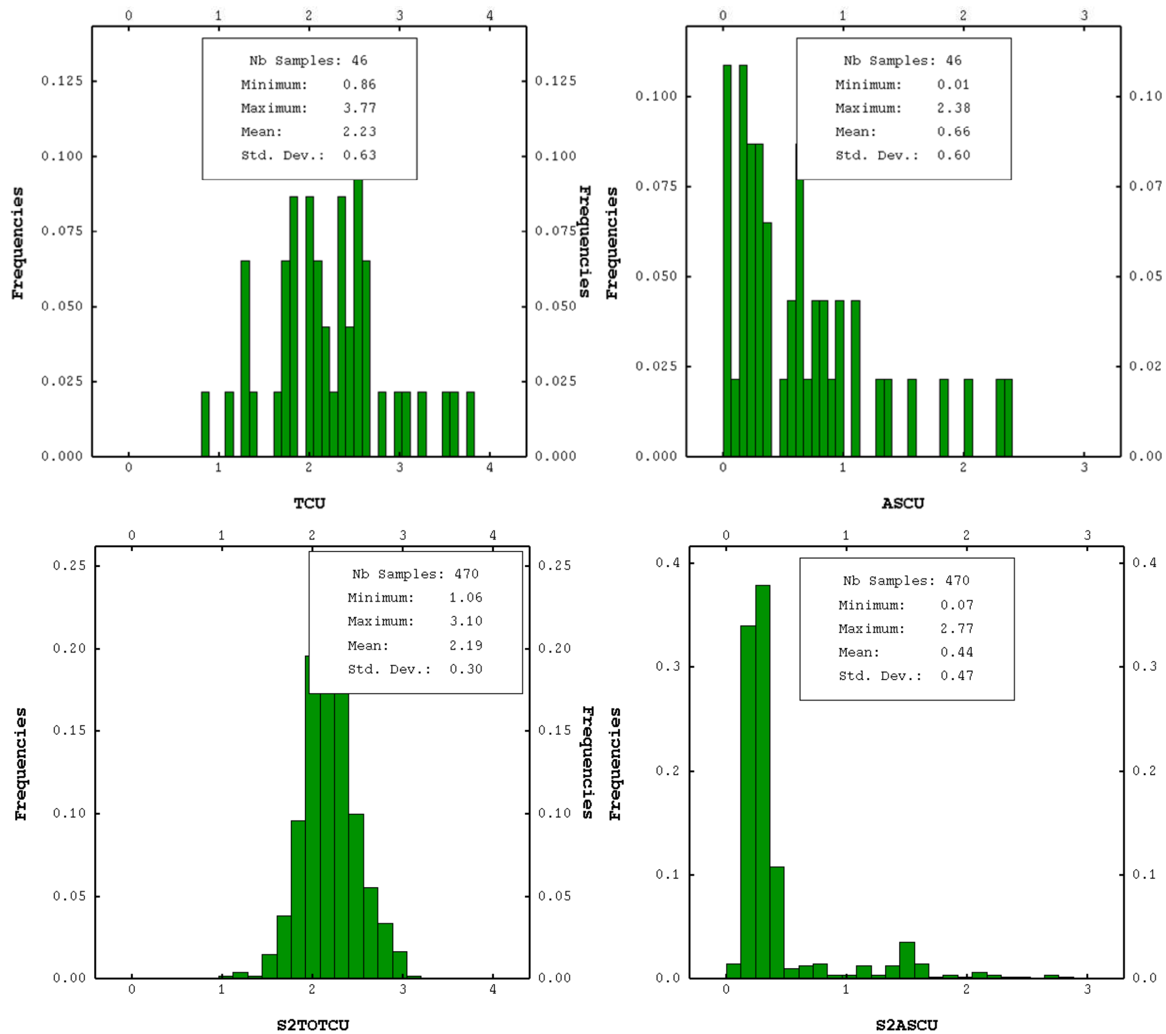

Figure 15.5: South Limb histograms - Comparisons of composite data and block model

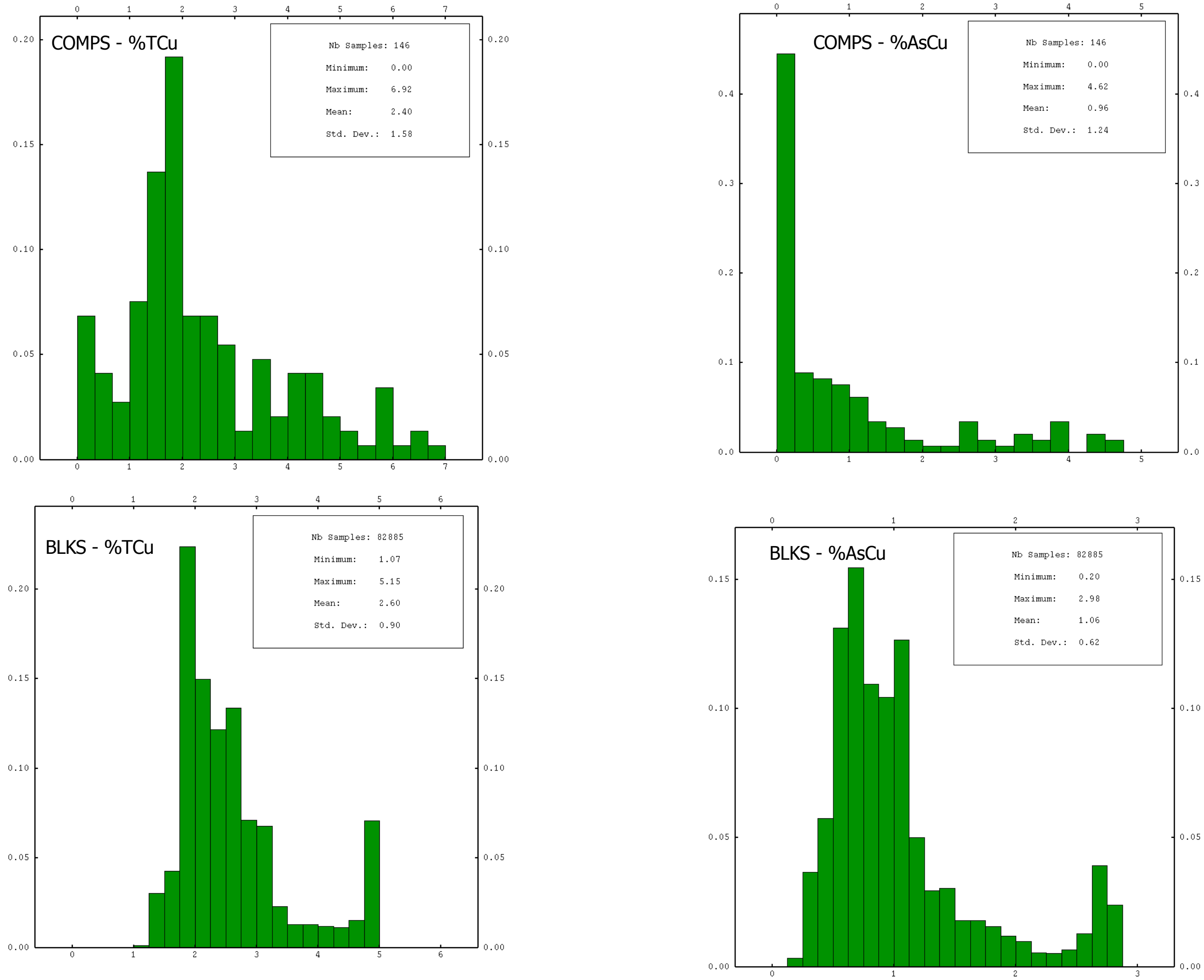

Figure 15.6: Area A histograms - Comparisons of composite data and block model

15.2.2 Grade distributions

The %TCu and %ASCu block histograms generally show that the distributions of the estimates are consistent with the distributions of the composite data. This is to be expected due to the inverse distance estimation applied in the current estimates.

SRK regularised the block models and compared the estimates against the composite drillhole intersected. The %TCu grade distribution in each of the orebody areas is plotted against the composite block model %TCu grade in Figures 15.7 to 15.9 for each of the three project areas.

The estimated block grade distribution in the East Limb is consistent with the composite grades in the densely drilled areas

Similar observations noted above for East Limb are applicable for South Limb. Also there are blocks in the north of the East Limb that remain an un-estimated.

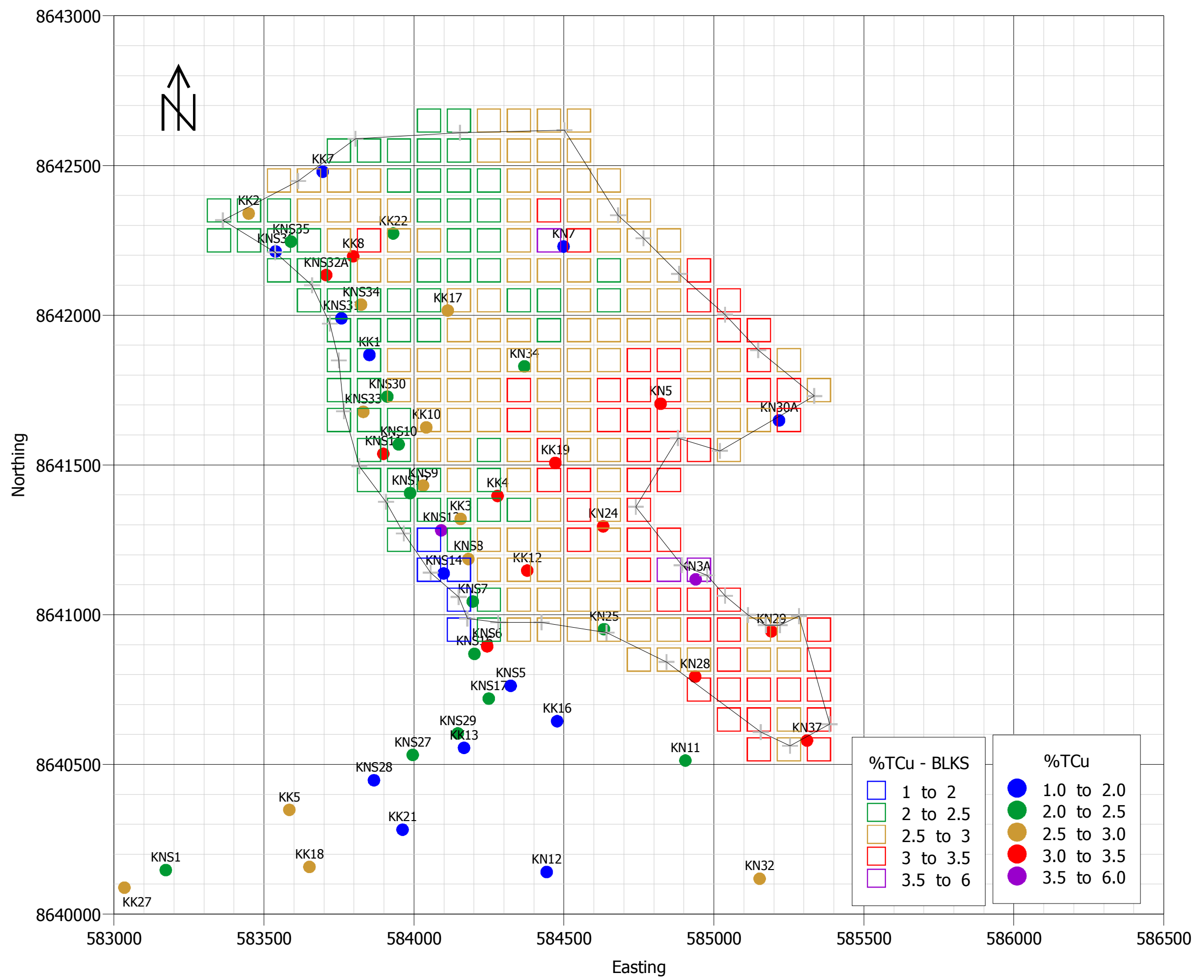

Figure 15.7: East Limb- Grade distributions of the %TCu composite data

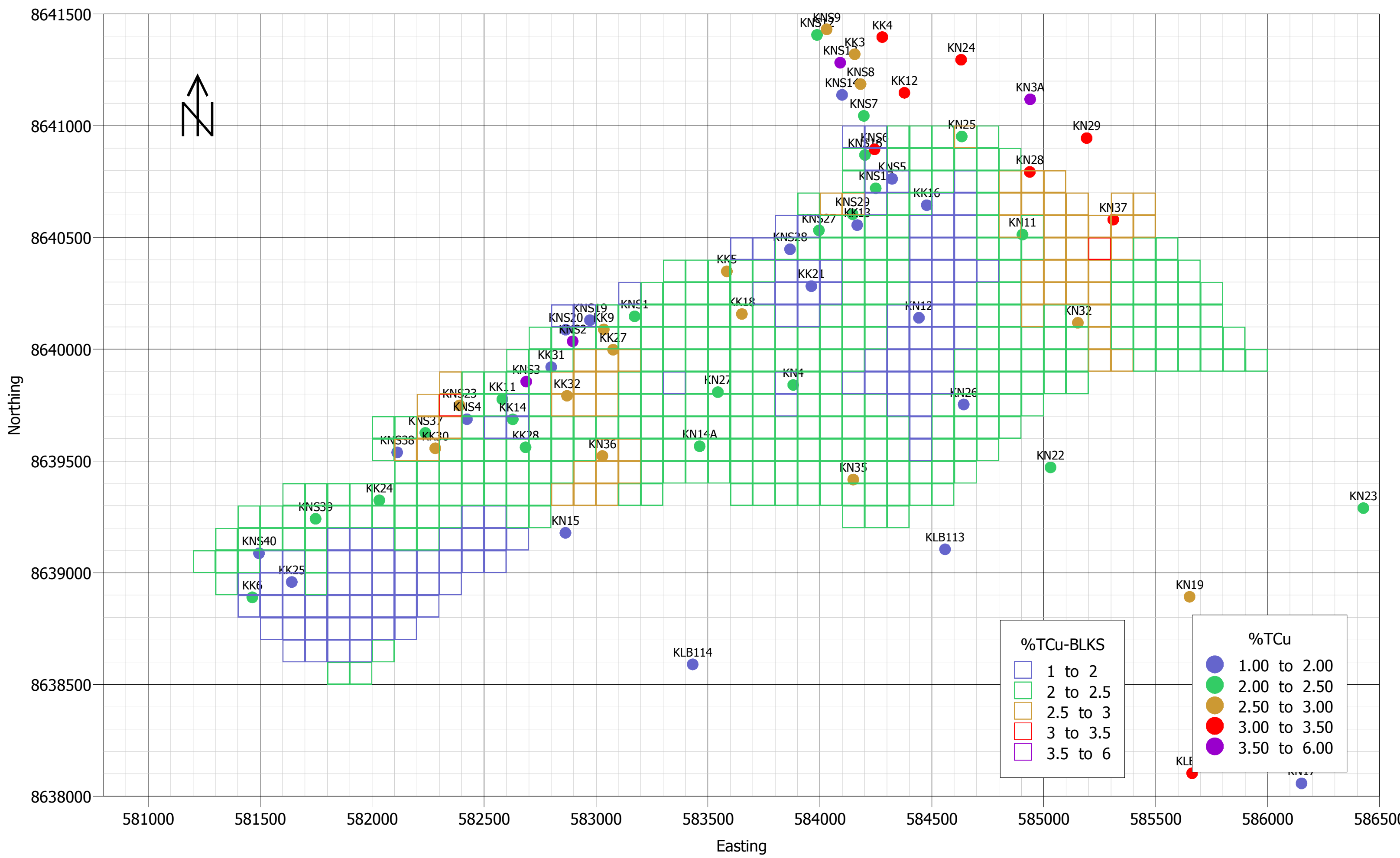

Figure 15.8: South Limb- Grade distributions of the %TCu composite data

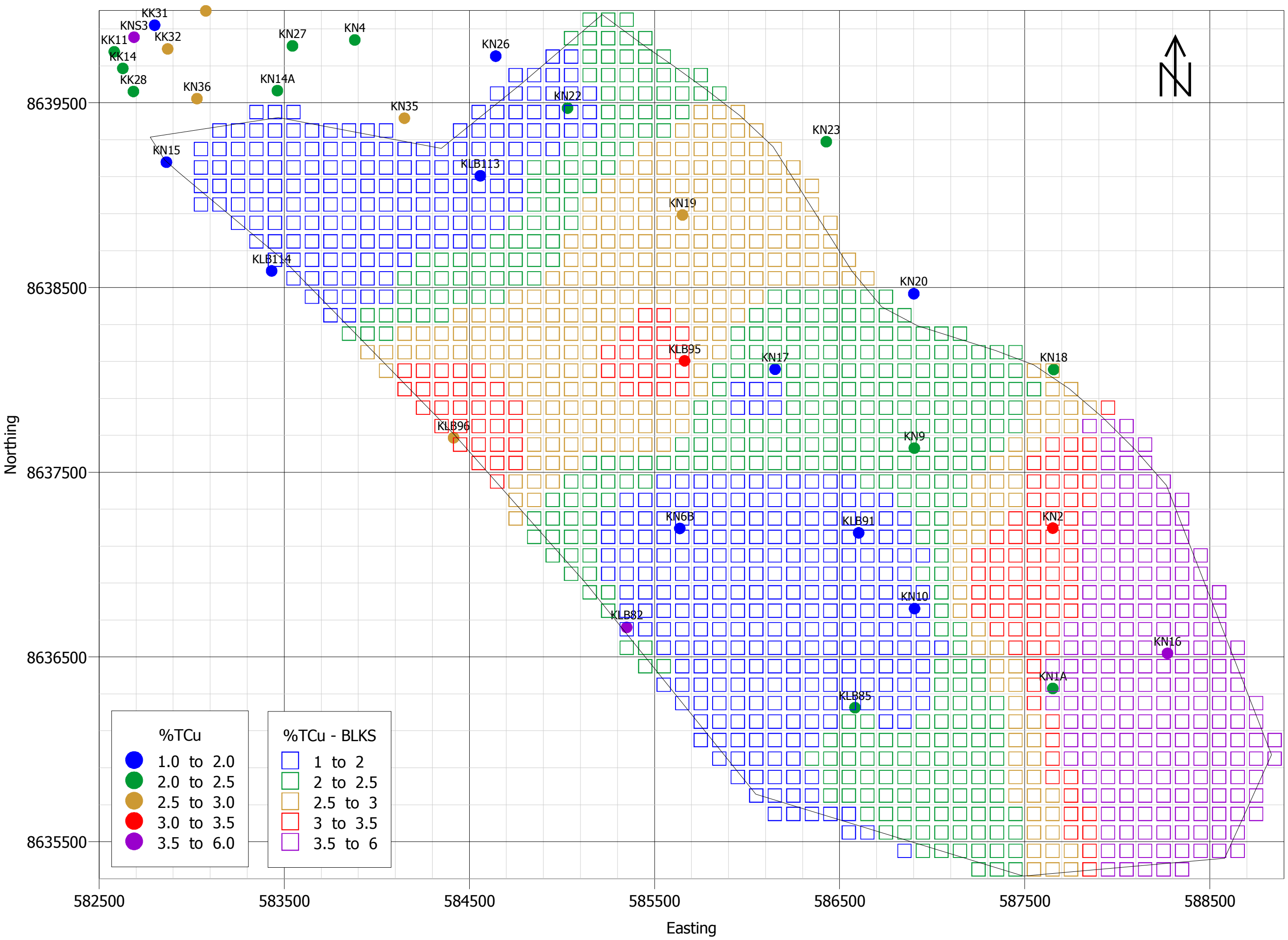

Figure 15.9: Area A- Grade distributions of the %TCu composite data

15.3 Variography

SRK split the data according the resource areas into generated omni-directional variograms for m%TCu, m%AsCu (accumulations of TCu and AsCu) and True Thickness. It should be stated here that SRK did not unfold the data prior to the variography, as this was not in the scope, and therefore the results are not directly comparable to the work undertaken by GijimaAST. SRK work was to provide a level of comfort in the spatial continuity of the data.

SRK's omni-directional variograms indicated spatial continuity for m%TCu, m%ASCu and True Thickness of 647m, for 682m and for 430m respectively. Only the m%TCu range is comparable to the GijimAST range of 660m. SRK's omni-directional variograms are shown in Figure 15.10.

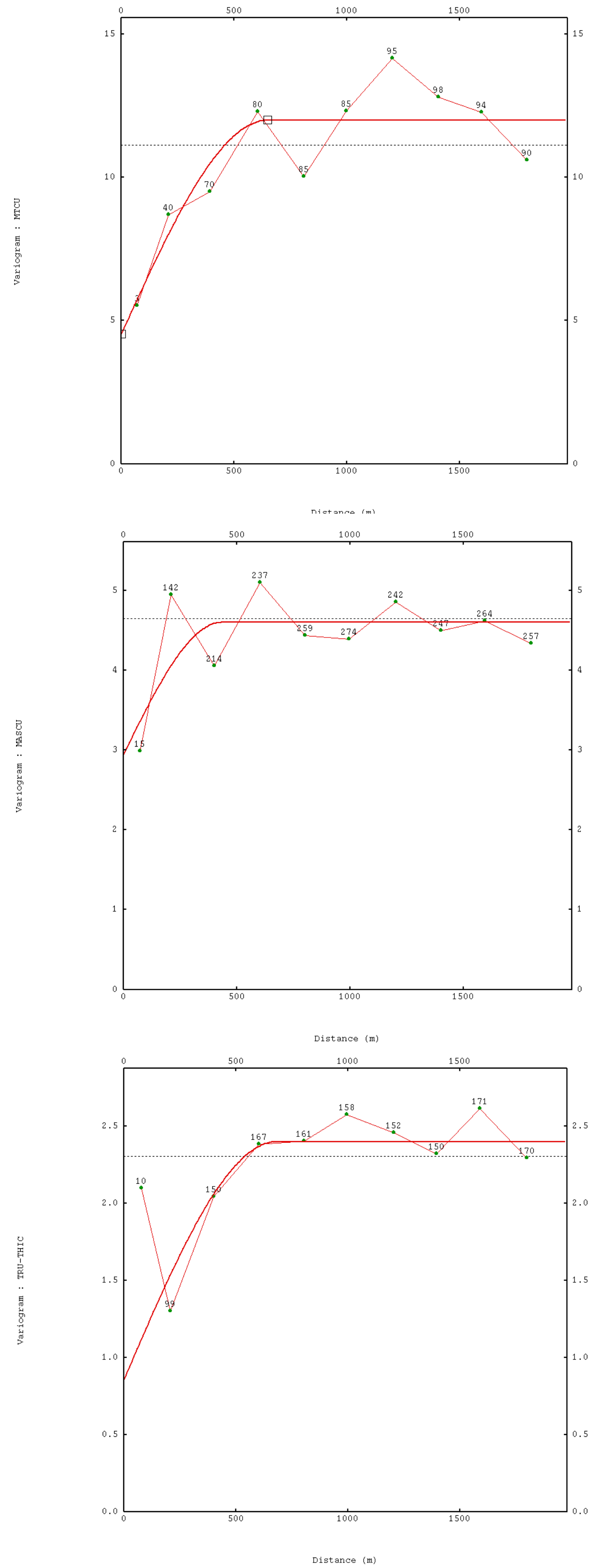

Figure 15.10: Omni-directional variograms for m%TCu, m%AsCu and True Thickness

15.4 Mineral Resources and Classification

SRK has reviewed of the data distribution, the models, the methodology adopted in the estimation process and the factor used for the conversion of volumes to tonnes. SRK's findings under these headings are:

- **Data distribution -** The drillhole spacing in the South Limb varies from about 100m to over 400m along strike and the bulk of the data along the up-dip section of the limb is covered by drillholes within 150m spacing. In the East Limb, the drillholes are spaced on average about 120m to 170m, averaging abut 150m along strike. As in the South Limb, there are two intersections from most drillhole positions in the up-dip portion of the limb. Area A is interested by widely spaced drillholes;

- **Grade distributions -** The grade distribution indicates relatively lower grades in the south-west, and extreme north-east surrounding relatively higher grades in the central potion of the limb (figure 15.2). Overall the grade varies between 2% to 3%TCu. Generally, the intersections in the East Limb average 2% to 3% TCu, but vary locally with relatively higher intersections further down dip. The review has indicated local grade variations with more closer spaced drilling;

- **Spatial continuity-** Omni-directional variograms generated by SRK on the accumulated data set indicate spatial continuity of over 600m for the m%TCu and m%AsCu and 400m for the True Thickness. this confirms the GijimaAST range for m%TCu of 660m;

- **QA/QC -** the performance of the laboratories measured against the known values of the standard reference material for %TCu and %AsCu returned within the acceptable 10% band of variation and SRK considered the results to be representative to the material analysed and as such suitable for use in the Mineral Resource estimation process;

- **Bulk density -** an average bulk density of 2.57 t/m^3 has been used to convert volume of mineralised material to tonnage. This is based on over 800 determinations which statistically vary within a narrow limit around the mean value and do not seem to be affected by the tenor of mineralisation in the rock. The average bulk density from the determinations is comparable to the 2.58 t/m^3 value being used at the adjacent operations of KCM, for the mineralised rock units of the OS1 member.

SRK has slightly modified ARM's classification of the Mineral Resources taking into account the foregoing. Table 15.1 presents SRK's audited Mineral Resources while Figure 15.11 shows the spatial location of SRK's revised classification boundaries.

Table 15.1: Audited Mineral Resources for KNP, as of February 2008

Classification		Measured			Indicated			Inferred		
Resource Area	**Sub-area**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**	**Mt**	**%TCu**	**%AsCu**
South Limb	**Block 1**	10.0	2.23	0.60	-	-	-	-	-	-
	Block 3	-	-	-	22.2	2.13	0.26	-	-	-
	Block 5	-	-	-	-	-	-	16.2	2.22	0.25
Sub-Total		**10.0**	**2.23**	**0.60**	**22.2**	**2.13**	**0.26**	**16.2**	**2.22**	**0.25**
East Limb	**Block 2**	7.1	2.34	1.07	-	-	-	-	-	-
	Block 4	-	-	-	11.7	2.87	0.39	-	-	-
	Block 7	-	-	-	-	-	-	10.7	2.83	0.41
Sub-Total		**7.1**	**2.34**	**1.07**	**11.7**	**2.87**	**0.39**	**10.7**	**2.83**	**0.41**
Area A	**Block 6**	-	-	-	-	-	-	219.5	2.64	1.09
Sub-Total		**-**	**-**	**-**	**0.0**	**0.00**	**0.00**	**219.5**	**2.64**	**1.09**
Total by Category		**17.1**	**2.28**	**0.80**	**33.8**	**2.38**	**0.30**	**246.4**	**2.62**	**1.01**
Total Measured and Indicated		**51.0**	**2.35**	**0.47**						

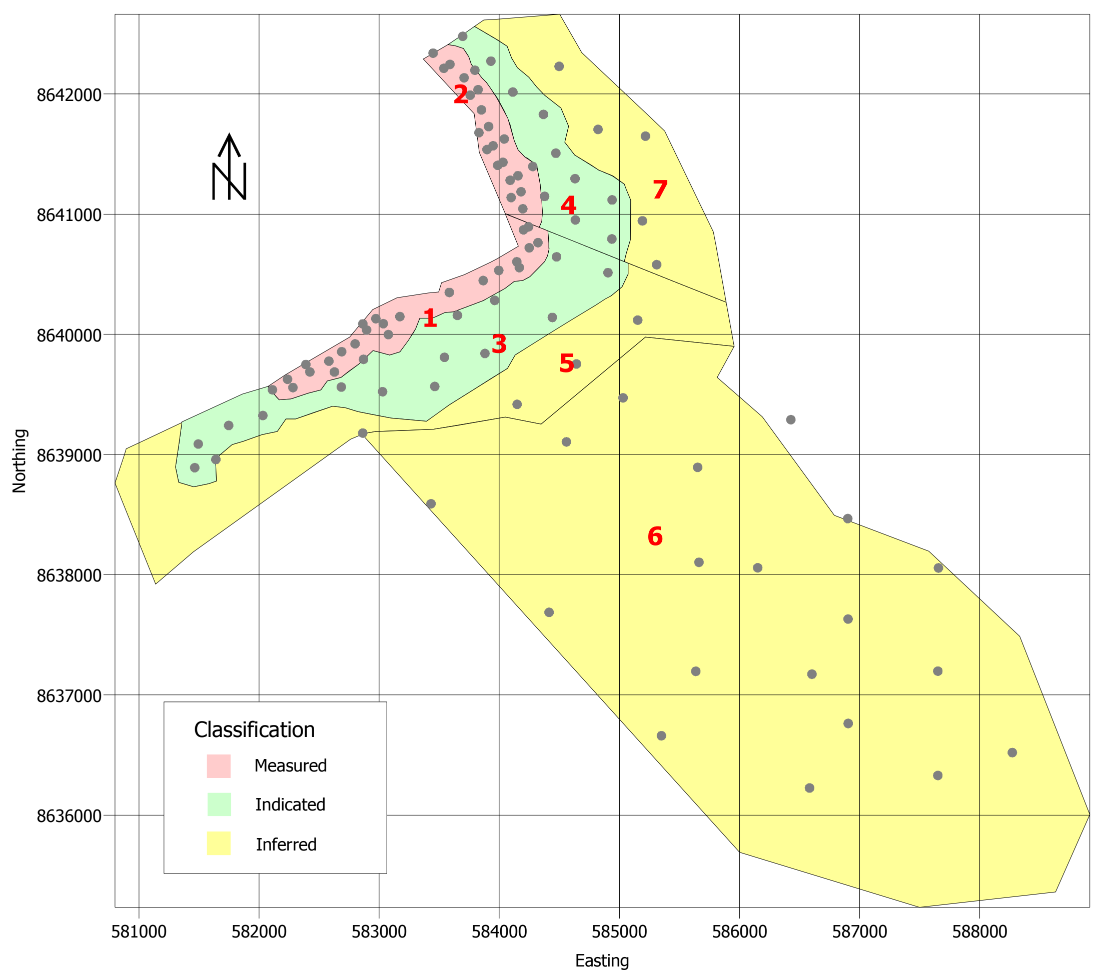

Figure 15.11: SRK's revised categories of Mineral Resources

16 Other Relevant Data and Information

There is currently no other relevant data or information.

17 Interpretation and Conclusions

SRK have reviewed the technical studies undertaken on the KNP leading to the estimation of Mineral Resources. SRK has found the work undertaken to be of acceptable quality to allow for the classification of Mineral Resources according to the guidelines and definitions of The Samrec Code.

SRK has examined the distributions of the composite data against the block model estimates and found that generally there is good correspondence between the estimates and the composite data in areas where there is adequate drillhole coverage, but is less so in sparsely drilled areas.

SRK has reviewed the data and grade distribution, the spatial continuity, bulk density and the QA/QC results with respect to the categorisation of the Mineral Resource:

Data distribution - The drillhole spacing in the South Limb varies from about 100m to over 400m along strike and the bulk of the data along the up-dip section of the limb is covered by drillholes within 150m spacing. In the East Limb, the drillholes are spaced on average about 120m to 170m, averaging abut 150m along strike. As in the South Limb, there are two intersections from most drillhole positions in the up-dip portion of the limb. Area A is interested by widely spaced drillholes;

- **Grade distributions -** The grade distribution indicates relatively lower grades in the south-west, and extreme north-east surrounding relatively higher grades in the central potion of the limb (figure 15.2). Overall the grade varies between 2% to 3%TCu. Generally, the intersections in the East Limb average 2% to 3% TCu, but vary locally with relatively higher intersections further down dip. The review has indicated local grade variations with more closer spaced drilling;

- **Spatial continuity-** Omni-directional variograms generated by SRK on the accumulated data set indicate spatial continuity of over 600m for the m%TCu and m%AsCu and 400m for the True Thickness. this confirms the GijimaAST range for m%TCu of 660m;

- **QA/QC -** the performance of the laboratories measured against the known values of the standard reference material for %TCu and %AsCu returned within the acceptable 10% band of variation and SRK considered the results to be representative to the material analysed and as such suitable for use in the Mineral Resource estimation process;

- **Bulk density -** an average bulk density of 2.57 t/m^3 has been used to convert volume of mineralised material to tonnage. This is based on over 800 determinations which statistically vary within a narrow limit around the mean value and do not seem to be affected by the tenor of mineralisation in the rock. The average bulk density from the determinations is comparable to the 2.58 t/m^3 value being used at the adjacent operations of KCM, for the mineralised rock units of the OS1 member.

On the basis of the foregoing, SRK are of the opinion that there is adequate drillhole coverage continuous grade distributions supported by the spatial continuity of the mineralisation for the Mineral Resources to be classified into Measured Indicated and Inferred categories based on the guidelines and definitions of The Samrec Code.

18 Recommendations

SRK's has noted that dense drilling highlights the local grade variability inherent in the project. SRK therefore recommends that the level of drilling be increased to provide coverage to allow for the understanding of the spatial continuity of mineralisation at closer spacing.   This will form part of the production planning prior to the extraction by mining.

VM Simposya Pr. Sci. Nat., MSAIMM

Principal Geologist

SRK Consulting

19 References

Anglovaal Mining, (1999a). KNP Prefeasibility Study – Detailed Technical Work Volume 1 - Report. Konnoco (Zambia) Limited, October 1999.

Anglovaal Mining, (1999b). KNP Prefeasibility Study – Volume 2 - Appendices. Konnoco (Zambia) Limited, October 1999.

Anglovaal Mining, (June 2001). KNP Ore Resource Evaluation Konnoco (Zambia) Limited, June 2001.

Anglovaal Mining, (2001). KNP Amended Prefeasibility Study – Volume 1. Konnoco (Zambia) Limited, August 2001.

ARM Ltd (2007) Konkola North Copper Mine: Mineral resource Estimation Methodology, July 2007.

ARM Ltd (2008) Konkola North Copper Mine: Mineral resource Estimation Methodology, February 2008.

GMSI Consulting, (2007): Teal Mining-KNP-Geological Modelling report, August 2007.

GMSI Consulting, (2006): Teal Mining-KNP-Data validation report, August 2006.

GijimaAst Consulting, (2006): Geostatistical report-Estimation and Evaluation of Resources Mining-KNP-Geological Modelling report. Teal Mining- Konkola North Geological Modelling, December 2006.

Sphynx Consulting, (2007): A report on the creating of a geological model for Konkola North. Geological model June 2007, June 2007.

20 CERTIFICATE OF THE QUALIFIED PERSON

I, Victor M. Simposya, Pr Sci. Nat., an employee of SRK Consulting, South Africa, 265 Oxford Road, Illovo, 2196 Johannesburg, South Africa, hereby certify that:

1. I am a graduate of Montana Tech, Butte Montana with a M.Sc. degree in Mining Engineering (1990) majoring in geostatistics and mine planning, and a graduate of the University of Zambia, Lusaka, Zambia, with a B.Sc. degree in geology (1979).

2. I have over 28 years experience in the mining industry; 17 years as a geologist on the Zambian Copperbelt, 2 years as a mine planning engineer for a diamond mining company and over 9 years as a geologist for SRK Consulting. I have been involved in feasibility studies, due diligences and project reviews for base metals, platinum group metals, chrome, and diamond projects in various locations within and outside of South Africa.

3. My most recent visit to the KNP was from 3rd October 2007 to 5th October 2007, to inspect the core from the drilling and review the interpretations of the geology and the mineralisation.

4. I am a member in good standing of the South African Institute of Mining and Metallurgy ("SAIMM"), member No. 702167 and the regulatory body, the South African Council for Natural Scientific Professions ("SACNASP") Registration No. 400052/03.

5. I personally prepared, all of the sections, with the exception of the section on Mineral Processing: and Metallurgical Testing", of this technical report entitled "Technical Report on the Mineral Resources for the Konkola North Copper Project, Chililabombwe, Zambia" dated March 2008.

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101º("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 experience I fulfill the requirements to be a qualified person for the purposes of NI43 101.

7. I have read the NI 43-101 and the technical report has been prepared in compliance therewith.

8. I am not aware of any material fact or material change with respect to the subject matter of the technical report which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

9. I have no interests of any form in the reporting issuer (TEAL) apart from the payment for the services rendered in the preparation of this report. I therefore, am considered independent of TEAL in respect of this report.

10. I consent to the filing of the Technical report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 4th day of March 2008 and signed,

Victor M Simposya, M.Sc., Pr. Sci. Nat.

CERTIFICATE OF QUALIFIED PRSON

I Peter Clive Lyon Hart do hereby certify that:

I am a Professional Engineer with Dowding Reynard & Associates, DRA House, No. 3 Inyanga Close, Sunninghill.

I am a graduate of:

- University of the Witwatersrand, BSc Metallurgical Engineering, 1975.
- University of South Africa, BCom.

I am a member of good standing with:

- The Association of Professional Engineers.
- South African Institute of Mining and Metallurgy.

I have practiced my profession continuously since graduation.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that, by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI43-101.

My relevant experience with respect to the project includes Metallurgical Process Development and Metallurgical Engineering.

I am responsible for:

- Section: Mineral Processing and Metallurgical Testing.
- Report: Technical Report on the Konkola North Copper Project.

I am not aware of any material fact or change with respect to the report for which I am responsible that is not reflected in the report.

I am independent to the issuer of the actual report.

As of the date of this Certificate, to my knowledge, the Report contains all technical information that is required to make the report not misleading.

I therefore consent to the filing of the Technical Report with the stock exchange and other regulatory authority and any publication by them, including electronic publications in the public company files on their website accessible by the public.

Dated 21 May 2008

Signature

SRK Report Distribution Record

Complete this form and include it as the final page for each copy of the report produced.

Report No. 384408

Copy No. 1

Name/Title	Company	Copy	Date	Authorised by
Mr. C Schlegel	TEAL	PDF	5 March 07	VM Simposya
VM Simposya	SRK	2	5 March 07	VM Simposya
SRK Library	SRK	3	5 March 07	VM Simposya

Approval Signature: