Part X

Technical Report

An Independent Competent Persons' Report on the Material Properties of Global Enterprises Corporate Limited

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Executive Summary

Introduction

SRK Consulting (South Africa) (Proprietary) Limited ("SRK") has been commissioned by the directors of Global Enterprises Corporate Limited ("GEC") to prepare an independent competent persons' report ("CPR") on the material assets and liabilities associated with the following Cu-Co projects located near Kolwezi in the Katanga Province of the Democratic Republic of Congo (the "Material Properties"):

- KOV, a mining project;
- Kananga, an exploration project; and
- Tilwezembe, an exploration project.

GEC has a 75% interest in each of the Material Properties, with the remaining 25% held by Gécamines (*la Générale des Carrières et des Mines*). The exploitation rights to the Material Properties are held in a joint venture vehicle DCP SARL (DRC Copper and Cobalt Project, "DCP").

These have been valued by SRK using valuation techniques appropriate to the stage of development of each project.

SRK was approached by GEC to prepare a CPR on the Material Properties in accordance with the admission requirements of the Alternative Investment Market ("AIM") of the London Stock Exchange in anticipation of a listing of the Company on AIM during 2006. This CPR has been prepared in accordance with the Guidance Note for Mining, Oil and Gas Companies issued by AIM on 16 March 2006.

Details of data/information used to prepare this CPR are as follows:

- Gécamines Western Proposal—pre-feasibility study, Report No M3037, Bateman Metals (as lead consultant) 15 June 2005;
- DRC Copper/Cobalt Project SARL—Definitive Feasibility Study ("DFS"), compiled by GEC, May 2006 (with input from a consortium of independent consultants);
- Financial Evaluation Model for the KOV Project, as part of the Definitive Feasibility Study for the KOV Project, compiled by Global Enterprises Corporate, May 2006;

The effective date (the "Effective Date") of this report is deemed to be 1 July 2006, and is co-incident with the Valuation Date and cashflow projections as incorporated herein.

The achievability of LoM Plans, budgets and forecasts are neither warranted nor guaranteed by SRK. The forecasts for the KOV project as presented and discussed herein have been developed by SRK based on information supplied to it by GEC, and cannot be assured; they are necessarily based on economic assumptions, many of which are beyond the control of GEC. Future cash flows and profits derived from such forecasts are inherently uncertain and actual results may be significantly more or less favourable.

GEC, DCP and the Material Properties

Global Enterprises Corporate Limited ("GEC"), a company registered in the British Virgin Islands, is 100% owned by Nikanor plc. Following execution of a Memorandum of Understanding on 5 May 2004, GEC and Gécamines entered into a Joint Venture Agreement ("JVA") on 9 September 2004 in relation to the rehabilitation and exploitation of three brownfield Cu-Co mines within Gécamines' "Group West" operational zone located near Kolwezi in the Katanga Province of the Democratic Republic of Congo ("DRC") (see Figure 2.1). The JVA comprises the mines of KOV, Kananga and Tilwezembe, the processing facilities of the Kolwezi Concentrator and an existing electro-refinery at Luilu (the "Existing Electro-Refinery"), and certain rights in relation to the Shituru refinery in Likasi.

DCP SARL (DRC Copper and Cobalt Project, "DCP") is a joint venture vehicle established by Gécamines and GEC under the laws of the DRC for the exploitation of the KOV, Kananga and Tilwezembe deposits. DCP, a company registered in the DRC, is held 75% by GEC and its nominees and 25% by Gécamines.

Three exploitation permits for the KOV, Kananga and Tilwezembe open pit mines were transferred from Gécamines' existing mining title Exploitation Permit No 525 (*Permis d'Eploitation*, "PE"). The three exploitation permits provide DCP with the exclusive right to conduct exploration, development and exploitation activities with respect to copper and cobalt (as well as related mineral substances) within the relevant mining zones.

A key commercial term of the JVA involves the lease or granting of certain rights to DCP with respect to following processing facilities (together the "Processing Facilities"):

- The Kolwezi Concentrator ("KZC");
- The Existing Electro-Refinery (together with waste sites and certain infrastructure together "Luilu facilities")
- Certain buildings and workshops (such as those at SKM) ("KOV facilities");
- Treatment of concentrates at the Shituru hydro-metallurgical treatment plant in Gécamines' Central Group operational zone around the town of Likasi.

In return for the use of the KZC, the Luilu facilities and the KOV facilities, a percentage of net sales receipts (2% for the first four years and 1.5% thereafter) will be payable to Gécamines.

The JVA also gives DCP a right of first refusal on surplus production of acid and/or lime from any of Gécamines' neighbouring plants.

As the Processing Facilities are not required for the planned operations at KOV, the lease/ granting of rights has been treated as a material contract, although it will not form part of the valuation of GEC.

The KOV mining project comprises the four large orebody fragments of Kamoto East, Oliveira and Virgule (hence the name KOV) and FNSR. The open pit operations of KOV and Kamoto East are situated at Latitude 10°42'S and Longitude 25°25'E, some 5km west of the town of Kolwezi in the Katanga Province in the south-eastern part of the DRC (Figure 2.2).

From the aerial photo of the KOV and Kamoto East pits (Figure 2.3), it can be seen that the existing opencast mine workings are filled with water. A programme to dewater the pits and the surrounding slopes is planned to start in the fourth quarter of 2006.

A significant asset within the existing infrastructure at KOV is a KRUPP waste conveyor system, complete with in-pit crushers and a large spreader, capable of handling 64Mt of waste per year. This conveyor system operated for only 6 months in 1997. Thyssen-Krupp engineers inspected the system in March 2006 and reported that the conveyor system was in very good condition.

The Kananga project is situated at Latitude 10°40'S and Longitude 25°28'E some 5km north of the town of Kolwezi. The deposit has not been adequately drilled but is believed to continue largely uninterrupted over a strike length of about 6km.

The ore is mainly oxide in nature with very little sulphide material in the mineralogy. Predominant copper minerals are malachite and pseudo-malachite associated with the cobalt mineral, heterogenite. The host rock is both dolomitic and siliceous. Copper and cobalt head grades are reasonably well defined using both current and historical records from the Gècamines geology database and from head grades of ore processed in the KZC concentrator. The resulting oxide concentrate was leached and refined at the 'Old' Luilu and Shituru refineries, located at Kolwezi and Likasi respectively, producing a "B" grade copper cathode and cobalt metal.

The Kananga pit is located close to the Dilala River and wetland and is also within 20m of the Lubumbashi-Lobito railway line.

The Tilwezembe project is located at Latitude 10°47'S and Longitude 25°42'E some 27km south east of Kolwezi, close to the Lualaba River on the road to Likasi. Mining has taken place intermittently since 1999.

A rail siding and contractors yard was established close to the site. Due to a shortage of rolling stock and locomotives, a stockpile of ore had been created close to the siding.

The deposit has not been adequately drilled but is believed to continue largely uninterrupted over a strike length of about 6km.

The ore is mainly oxide in nature with very little sulphide material in the mineralogy. Predominant copper minerals are malachite and pseudo-malachite associated with the cobalt mineral, heterogenite. The host rock is both dolomitic and siliceous. Copper and cobalt head grades are reasonably well defined using both current and historical records from the Gècamines geology database and from head grades of ore processed in the KZC concentrator. The resulting oxide concentrate was leached and refined at the 'Old' Luilu and Shituru refineries, located at Kolwezi and Likasi respectively, producing a "B" grade copper cathode and cobalt metal. The Tilwezembe deposit is very rich in cobalt, but also contains a high proportion of manganese that requires an adjustment to the processing method for this ore.

GEC and Gécamines entered into a JVA in relation to the rehabilitation and exploitation of the KOV, Kananga and Tilwezembe mines near Kolwezi. The JVA was structured for consistency with the provisions of the Code and included the following key elements:

- Obligation to pay customs duties, taxes (30% tax on profits, 10% turnover tax on local sales of goods made and 5% for services rendered locally) and royalties (due to the Public Treasury at 2% of sales proceeds for non-ferrous metals, net of transport, insurance and sales costs);
- Obligation to pay surface area fees (US$5/ha/yr to cover costs of services rendered, plus a surface area tax of US$0.08/ha/yr, for the area covered by the PE);
- Obligations with respect to the protection of the environment; and
- Leasing by Gécamines (as Lessor) of the Processing Installations to the JVA (as Lessee), including obligations of the Lessee to maintain asset condition, meet taxes, observe all laws and make investments to ensure development.

In terms of the JVA, GEC is required to provide/procure financing necessary for the rehabilitation of the mines and related processing installations and infrastructure in order to bring them up to anticipated production levels and implement the necessary construction and upgrade works.

The three exploitation permits (PE4960, PE4961 and PE4963) grant DCP the exclusive right to carry out, within the areas over which they have been granted (see Figure 2.2), exploration, development, construction and exploitation works in connection with the mineral substances for which the permits were granted (i.e. copper, cobalt and associated mineral substances).

Formation of DCP has been authorised by the President of the DRC and the three PE's for KOV, Kananga and Tilwezembe have all been registered with the Mining Registry. The local law opinion provided by Djunga & Risasi provided on or about the date hereof indicates that DCP has legitimate and secure tenure to the mineral rights of the three projects and is authorised to exploit the KOV, Kananga and Tilwezembe orebodies.

Geology, Mineral Resources and Reserves

The KOV, Kananga and Tilwezembe orebodies are located at the western end of the Katangan Copperbelt, one of the great metallogenic provinces of the world, and which contains some of the world's richest copper, cobalt and uranium deposits, (Figure 3.1).

Stratigraphically, the rich copper and cobalt deposits found in Zambia and the DRC occur localised in the Roan Supergroup ("Roan"). The generalised stratigraphy of the Katangan System is shown in Figure 3.2.

Primary mineralisation, in the form of sulphides, within the Lower Roan is associated with the D Strat and RSF for the Lower Orebody and the SDB and SDS for the Upper Orebody and is thought to be syn-sedimentary in origin. Typical primary copper sulphide minerals are bornite, chalcopyrite, chalcosite and occasional native copper while cobalt is in the form of carrolite. The mineralisation occurs as disseminations or in association with hydrothermal carbonate alteration and silicification.

Supergene mineralisation is generally associated with the levels of oxidation in the sub-surface sometimes deeper than 100m below surface. The most common secondary supergene minerals for copper and cobalt are malachite and heterogenite. Malachite is the main mineral mined within the confines of the current KOV pit.

There are three main individual “fragments” hosting mineralised Lower Roan lithologies within the KOV pit area. These are Kamoto East, Oliveira and Virgule, from which the name KOV is derived. A fourth and smaller fragment, the FNSR, is a remnant of the Musonoi West fragment mined to the east of KOV pit. The FNSR lies below and is sub-parallel to the Virgule orebody.

The fragments that make up the KOV orebody occur in an east-west striking synclinal structure consisting of a steeply dipping southern limb and a shallow dipping northern limb named the Kamoto East and Virgule orebody respectively, while the Oliveira fragment occurs as a shallower dipping orebody in faulted contact with and below the Virgule orebody.

Based on the interpretation by Gécamines, the Kananga orebody forms the northern limb of the Kananga-Dilala syncline. The deposit plunges to the south. The Kananga orebody outcrops and forms a ridge with a NNE strike. The ridge falls quite rapidly towards the south and has been cut to form part of the embankment for the Lobito railway line which runs parallel to the ridge between 10m to 20m away for most of the strike length of the orebody. The deposit is divided by the Musonoi River into two hills called Kananga East and Kananga West. Mining activities have taken place on the western edge of the Kanaga East hill.

Based on the exploration work undertaken, the interpretation by Gécamines indicates that the Tilwezembe deposit is located in an NE-SW anticlinal structural lineament, which extends further to the east hosting four other known copper and cobalt deposits (Kisanfu, Myunga, Kalumbwe and Deziwa).

Mineralisation generally occurs as infilling of fissures and open fractures associated with the brecciation. The typical mineralisation consists mainly of copper minerals (chalcopyrite, malachite and pseudomalachite) and cobalt minerals (heterogenite, carrolite and spherocobaltite) and manganese minerals (psilomelane and manganite).

GEC’s budget to conduct exploration-related activities at the Kananga and Tilwezembe orebodies for the three years 2006 to 2008 amount to US$8.6million.

The KOV Project

The information used in the generation of the KOV orebody models consisted of:

- **Surface Topography Plans**—on four 1:1000 scale sheets consisting of contours and spot heights.
- **Geological Cross Sections**—seventeen cross-sections on a 1:2000 scale and at a 100m spacing from X-600 to X-2200 across the deposit. The sections contained projections of drillholes annotated with lithological units, Gécamines geological interpretation and the latest pit surface topography as at December 1989 for the Kamoto Pit and December 1991 for the KOV pit. Also on the sections were tabulations of composite orebody intersections in each drillhole.
- **Drillhole Logs**—copies of drillhole logs for 176 holes pre-fixed KOV and KTO within the KOV pit area. This did not include KTO drillhole logs specific to the Kamoto East orebody which were provided by Gécamines in September 2005. For each drillhole there were several sections in the log, each section was of multiple pages. The main sections were collar page, laboratory report, sample logs and summary of intersections. Down the hole surveys were contained in the collar report. Excluding the geological logs, all the sections were captured in Microsoft Excel worksheets.
- **Bench Plans**—bench plans at 10m intervals from 1330m to 960m elevation and on a 1:1000 scale were also collected for the Kamoto East and KOV pits. There were 47 copies on A0 size, on average 2 plans per bench.

GEC commenced a drilling programme in December 2005 primarily to collect samples for metallurgical testwork and also to confirm the general orebody intersections and grade.

By end May 2006, a total of 7 holes had been completed. The orebody intersections in these holes are comparable to the Gécamines holes.

There are a total of 215 drillholes that have orebody intersections in the database of which 64 are KTO holes and 151 are pre-fixed KOV. The distribution of the holes is shown in Figure 4.1. Some of the holes drilled did not intersect the Roan stratigraphy and have therefore been excluded from the drillhole database for resource modelling.

Cores from the orebody intersections were sampled for chemical analyses. The lengths of core sampled varied and it is SRK's understanding that this was a consequence of the sample recovered within each run. In the Gécamines logging sheet is a column for percentage recovery where values ranging from 1% to 100% are entered to describe the amount of core recovered in the sample length. Core recoveries are only recorded for cores that were sampled.

SRK has reviewed the database and the following issues with data quality are highlighted:

- Poor core recovery within the orebody varying between 65-75%, the worst recovery being in the RSC lithology;
- The cutting of the total copper grade to 12%, if above 12%. However, not all the samples with total copper grades above 12% in the database have been cut;
- The RSC formation in contact with the SDB and the RSF was sampled, but near the middle of the formation, selectively sampling was on the basis of visible copper mineralisation;
- The orebody zones are sampled for total copper and total cobalt, but assays for acid soluble copper and cobalt and for calcium oxide are limited;
- Samples with zero core recovery, mostly from the earlier drilling in Kamoto East, are shown with high total copper values; presumably from the analyses of the sand collected in the absence of core.

The assay database is incomplete for %ASCu and %CaO as less than 50% of the samples have assays for %ASCu and less than 20% of the samples have assays for %CaO. Geostatistical modelling of these two variables would lead to misleading results. SRK have excluded these two variables in the grade estimation and resource reporting processes downstream.

The %ASCu and %CaO grade values have been used to provide for a feel of the distribution of oxide-sulphide and dolomitic-siliceous ore types. For the oxide-sulphide, the ore is considered oxide when the proportion of %ASCu to the %TCu is higher, and sulphide with low %ASCu content relative to the %TCu. As an exercise, SRK have used a ratio of 0.5 as the division between the two ore types: when the %ASCu to %TCu ratio is less than 0.5, the ore is considered sulphide and when the ratio is higher than 0.5 the ore is considered oxide.

For the conversion of volumes into tonnes, Gécamines assigned density values based on an empirical criterion obtained from standardised values generated from the exhaustive data available from all Gécamines operations within the Katangan Copperbelt. The criterion was based on the categorisation of the ore type into either dolomitic or siliceous based on the relationship between %TCu and %CaO described above.

Generally the oxide ores have been assigned values of 2.2t/m^3, the mixed ores 2.4t/m^3 and the sulphide ores 2.6t/m^3. In the models generated for KOV, SRK has used a density of 2.2t/m^3 based on the inference that the majority of the ore is oxide.

For modelling purposes, the lithological units within each of the fragments were modelled as separate entities and a bounding string was used to indicate the limit of the each fragment.

Each of the digitised sections were imported into Datamine and modelled by lithological units as 3- dimensional wireframe surfaces. For each lithological unit, a 3-dimensional block model was generated from the adjacent surfaces. Parent blocks of dimensions 100m x 100m x 10m sub-celled to 25m x 25m x10m in the X, Y and Z directions were filled above each of the generated surfaces.

A lithological block model was generated by superimposing the two stratigraphically successive lithologies models and selecting the blocks lying in between the surfaces for the particular lithology.

The surface topography was generated from a combination of the digitised contours and spot heights and the pit profile digitised from the section plots. Whereas every effort was made to reproduce the spot height elevations as accurately as possible, the quality of the copies of the spot height plans was very poor.

Grades were estimated into the block model using geostatistical techniques.

Omni directional variograms were computed and modelled for the package and within each orebody zone for %TCu and %TCo. The variograms modelled indicated that samples were correlatable over distances of 300m to 600m for %TCu and 200m to 300m for %TCo. The variograms were not well structured with the nugget effect constituting about 50% to 90% of the total variance.

Grades were estimated into each of the orebody zones using the respective composites data files and the omni-variogram models for each zone. The terms and definitions used to present the statement of mineral resources are those given in the SAMREC Code.

Table 4.2: Updated Indicated Mineral Resources for the KOV Project, dated April 2006

Orebody	Mt	%TCu	%TCo
Kamoto	34	5.39	0.33
Virgule	64	5.09	0.41
Kamoto + Virgule	**98**	**5.19**	**0.38**
Oliveira	57	4.73	0.68
FNSR	17	5.65	0.44
Total Indicated	**172**	**5.09**	**0.49**

Kananga

The orebody has been defined over a 300m strike length based on the seven drillholes with assay data spaced on average about 50m apart supplemented by surface trenching information. The orebody is sub-vertical and dips steeply towards the south.

Except for one, the drillholes intersecting the Kananga orebody were drilled southwards at various inclinations and based on the geological interpretations, the drillholes are sub-parallel to the dip of the strata and can be interpreted to be sampling the same unit down the hole.

The individual rock block models were added together to generate an ore body model for Kananga. Blocks of waste were added outside of the ore models and blocks above the surface topography were removed from the final geological model. The %TCu and %TCo grades were estimated into the block model using inverse distance squared ("IDS"). The IDS methodology was selected due to the paucity of data available for the estimation within each rock unit.

Mineral resource estimates are reported in Table 4.4 and have been classified as Inferred by SRK.

Table 4.4: Inferred Mineral Resource estimates for Kananga

Rock Type	Mt	%TCu	%TCo
RATGR	0.6	2.63	1.19
DSTRAT	0.6	4.39	0.70
RSF	0.5	2.67	1.15
RSC	2.9	1.90	1.59
SDB	2.3	1.39	1.12
Total	**6.9**	**2.07**	**1.29**

Tilwezembe

Hard copies of geological sections covering the Tilwezembe Pit and Tilwezembe East orebodies were scanned and digitised capturing Gécamines' interpretations of the geology and structure within the deposits. A total of 20 geological east-west sections at 25m spacing were captured with 10 sections each covering Tilwezembe pit from X-16580m to X-166125m and Tilwezembe East from X-16627m to X-166425m. Figure 4.10 is a section through the Tilwezembe East deposit.

The Tilwezembe orebody is confined within a breccia and hosted by a series of dolomites and shales. Seven lithologies have been defined and modelled in Tilwezembe Pit and 10 in Tilwezembe East. These lithologies consist of four varieties of strongly brecciated dolomites and shales in Tilwezembe Pit and six varieties in Tilwezembe East, hosting the copper and cobalt mineralisation.

At the time of the modelling work, detailed drillhole data was unavailable for Tilwezembe East. Instead, like at Kananga, a drillhole database was created by measuring the depths and gleaning the data annotated on the section plots. Data for %Mn was not annotated on the section plots.

Variograms parameters modelled for Tilwezembe Pit were utilised in the estimation for Tilwezembe East. Grades were estimated into a combined orebody model consisting of six dolomitic lithologies using the composite data gleaned from the section plots.

SRK visited the site of the Tilwezembe Pit again in April 2006 and found that mining operations had ceased and the contractor had removed all his equipment. SRK is of the opinion that the orebody has been mined out in the Tilwezembe Pit and has therefore excluded it from the inventory of mineral resources.

The Mineral Resources in Tilwezembe East as at April 2006 are shown in Table 4.6 and have been classified as Indicated by SRK.

Table 4.6: Indicated Mineral Resources for Tilwezembe East as at April 2006 [1]

Deposit	Mt	%TCu	%TCo
Tilwezembe East	5.7	1.49	1.04
Total	**5.7**	**1.49**	**1.04**

1 The surface topography has not been surveyed since August 2004.

Mineral Resources and Reserves

The Mineral Reserves stated as a recoverable Mineral Reserve Estimate (in compliance with the Guidance Note for Mining, Oil and Gas Companies—March 2006 issued by AIM) to which grade adjustment factors, extraction factors, and processing plant recovery factors have to be applied.

The following modifying factors have been applied:

- Grade adjustment factor 95%
- Extraction Ratio 95%
- Processing Plant Recovery Factors 91% Cu and 85% Co

Table 4.7: GEC—Summary of Mineral Reserves and Mineral Resources as at April 2006

	Gross					Attributable					
Category	Ore	Grade (%)		Contained Metal (kt)		Ore	Grade (%)		Contained Metal (kt)		Operator
	(Mt)	Cu	Co	Cu	Co	(Mt)	Cu	Co	Cu	Co	
Mineral Reserves:											
Probable											
KOV	140	4.83	0.47	6,757	658	105	4.83	0.47	5,068	493	DCP
Total Reserves	**140**	**4.83**	**0.47**	**6,757**	**658**	**105**	**4.83**	**0.47**	**5,068**	**493**	DCP
Mineral Resources:											
Indicated											
KOV	172	5.09	0.49	8,755	843	129	5.09	0.49	6,566	632	DCP
Tilwezembe	6	1.49	1.04	89	62	5	1.48	1.03	72	50	DCP
Total Measured+Indicated Resources	178	4.97	0.51	8,844	905	134	4.96	0.51	6,638	682	DCP
Inferred											
Kananga	7	2.07	1.29	143	89	5	2.07	1.29	107	67	DCP
Total Inferred Resources	7	2.07	1.29	143	89	5	2.07	1.29	107	67	DCP
Total Resources	**185**	**4.85**	**0.54**	**8,994**	**999**	**139**	**4.85**	**0.54**	**6,745**	**749**	DCP

Mining

Geohydrology

The KOV Open Pit is currently flooded with an increasing water level as the incoming water flow exceeds the current pumping rate of about 2500 m^3/h. The volume of water in the pit is estimated to be around 12 million cubic metres. Before mining can commence the water needs to be pumped and the silt removed. In addition, once mining operations are started, boreholes will have to be commissioned to control groundwater seepage into the pit.

The initial pit dewatering is based on the provision of three stand alone systems with a capacity of 2,400 m^3/h; each system will entail three pumps in series, the first pump will be mounted on a floating barge with an access platform and the two booster pumps will be skid mounted and positioned along the pipeline on the shore of the current pit lake, the system will pump the water via a 2,700m long 600mm ND dedicated pipeline.

Once the KOV Pit has been dewatered one of the three systems with a capacity of 2,400m^3/h will be removed and re-installed to dewater the Kamoto East Pit.

Cost estimates were completed for the pit dewatering infrastructure. These costs include construction costs and allowances for contingencies and engineering. The total cost of the works has been estimated to be around US$16.43 million.

Once the pit has been dewatered then prevention of groundwater entering the pit is required and therefore the commissioning of the boreholes around the pit will be necessary. The groundwater control system will entail 45 boreholes to abstract the groundwater before entering the pit for the first 5 years of mining operation. The total capacity of these boreholes will be 8,300m^3/h. This will increase to 59 boreholes with a total capacity of 9,000m^3/h at the end of LoM.

The boreholes of Phases 1 and 2 will be positioned on the northern wall of the pit generally along the conveyor belt, the ones for Phase 3 will be installed at the bottom of the pit once the pit has been dewatered and the boreholes of Phase 4 will cover an arch on the top north-western section of the pit.

The KOV Pit requires protection against stormwater runoff and sediments from the old Kolwezi and Musonoi Pits which currently drain into the KOV Pit. The major contribution of solids into the pit comes from these sources. It is proposed to construct two dams with a capacity of 125,000m^3 each on the eastern side of the pit to contain and minimize stormwater runoff and solids entering the pit.

Geotechnical Investigation

SRK undertook a slope stability assessment to feasibility level for the KOV and Kamoto East open pits at Kolwezi in the DRC. A drilling programme was initiated in December 2005 to obtain geological, geotechnical and geohydrological information but at the time of this study the geotechnical holes have not been completed. Due to the limited information, the geotechnical slope angles determined as part of this study can be classed as inferred slopes. This relates to a pre-feasibility level and not a feasibility level study. The completion of the geotechnical boreholes will enable a more thorough feasibility study.

The geotechnical analysis consisted of empirical analysis to determine the overall slope angles and numerical analysis to determine the sensitivity of the slopes to dewatering the pit, and the effect of dewatering KOV only on the stability of the "bridge" between the pits.

Based on this study the recommended overall slope angles are as follows:

- 22° for the KOV north slope;
- 30° for the KOV south slope;
- 30° for the Kamoto east north slope;
- 35° to 40° for the Kamoto east north slope.

The numerical analyses show that dewatering of the pit pools will de-stabilize the slopes to some degree but that the expected extent of failure will be localised. The destabilising effect of dewatering can be ameliorated by dewatering the pit slopes in unison with the pit pools or to a lesser extent, by dewatering of the pit pools over a period of six months or longer. Due to the flat slope angle beneath the in pit crusher, instability of the slope beneath the crusher is not expected. Based on the numerical analysis carried out to investigate the stability of the "bridge" between KOV and Kamoto east, it is concluded that dewatering of the KOV pit without dewatering the Kamoto East pit will not cause shearing of the "bridge" into the KOV pit.

Mine Design

SRK with Bateman were commissioned in early 2005 to conduct a pre-feasibility study of the KOV, Kananga and Tilwezembe open pit mines in the Kolwezi area in the DRC. The Base Case developed in the pre-feasibility study considered the three mines supplying ore to the KZC concentrator at a rate of 220ktpm (2.64Mtpa). In addition to the Base Case, further options at different production rates were considered in which a Direct Leach Process was used for the recovery of copper and cobalt. Based on the results of the pre-feasibility study, SRK completed a Feasibility Study of the KOV open pit for a production rate of 400ktpm (4.8Mtpa) feeding ore to a Direct Leach Plant. The mining study included open pit optimisation, practical pit design and production scheduling to convert the Mineral Resources to Reserves and to determine the capital and operating cost to the level of accuracy of ±10 to 15%.

An optimisation study for Kananga was done as part of the pre-feasibility study to determine the open pit footprint for longterm planning purposes based on the Inferred Mineral Resources that were estimated at the time.

A study was also done for Tilwezembe East along the same lines as for Kananga. The optimisation study showed that once mining has been established that a production rate of 60ktpm can be maintained for more than 7 years. As Tilwezembe is situated 27km away from Kolwezi the transport of ore is more costly than for Kananga although a railway line extends to the mine which will allow ore to be transported by rail to the KZC processing plant.

Mine planning for KOV open pit was carried out for the LoM with a plant capacity of 4.8Mtpa using the practical pit designs for each of the selected seven cuts or pushbacks. An average RoM grade was targeted at 5.2%Cu with a tolerance of ±1%Cu to allow for the plant feed to be as uniform as possible.

The only restriction that was applied to the design of the practical pit was to limit mining to 50m from the existing waste crusher and the waste conveyor. A safe distance of 500m from the Kamoto Hoisting Shaft was also maintained in the mine planning.

The priority for the pit design was the ramping system, in order to maintain access to the ore crusher, waste crusher and waste dumps in the north-west and south side of the pit during the mining operation.

Production scheduling was carried out on an annual basis allowing for de-watering, silt removal and pre-stripping during the first two years.

The mineral reserves within each practical designed pit were determined after inclusion of 5% mining dilution and 5% mining losses. The mineral reserves were further classified as Probable Reserves in accordance with the Guidelines of the SAMREC Code and presented in Table 5.6.

Table 5.6: Probable Mineral Reserves within the Designed Practical Cuts (CoG; 1.0%Cu)

	Ore			Waste	Total Rock	Stripping
Pushback	Mt	Cu%	Co%	Mt	Mt	Ratio
Cut 1	18.2	5.31	0.43	187.8	206.0	10.3
Cut 2	33.6	5.04	0.57	417.5	451.1	12.4
Cut 3	17.6	4.27	0.44	250.7	268.3	14.2
Cut 4	16.9	4.40	0.38	159.3	176.2	9.4
Cut 5	12.4	3.92	0.68	54.4	66.8	4.4
Cut 6	17.5	4.90	0.36	125.2	142.7	7.2
Cut 7	23.7	5.33	0.38	228.3	252.0	9.6
Total	**139.9**	**4.83**	**0.47**	**1,423.2**	**1,563.1**	**10.2**

In the pit design it was endeavoured to limit the total waste to 64Mtpa to minimise trucking of waste rock to the dumps in the northwest and to the south of the pit. In some years, due to a short haul distance or no access from the top benches to the crushers, waste rock is hauled to the dumps instead of being conveyed.

For scheduling purposes two in situ copper grade categories were defined; a higher grade ore at a CoG of 4%Cu and a lower grade ore between 1%Cu and 4%Cu. After year 13 with the excavation of Cut 5 (containing the lower grade ore compared to the other Cuts), the lower grade ore has to be stockpiled and then blended with the higher grade mined from Cut 4 and Cut 6.

The smoothed LoM schedule is presented in Figure 5.1.

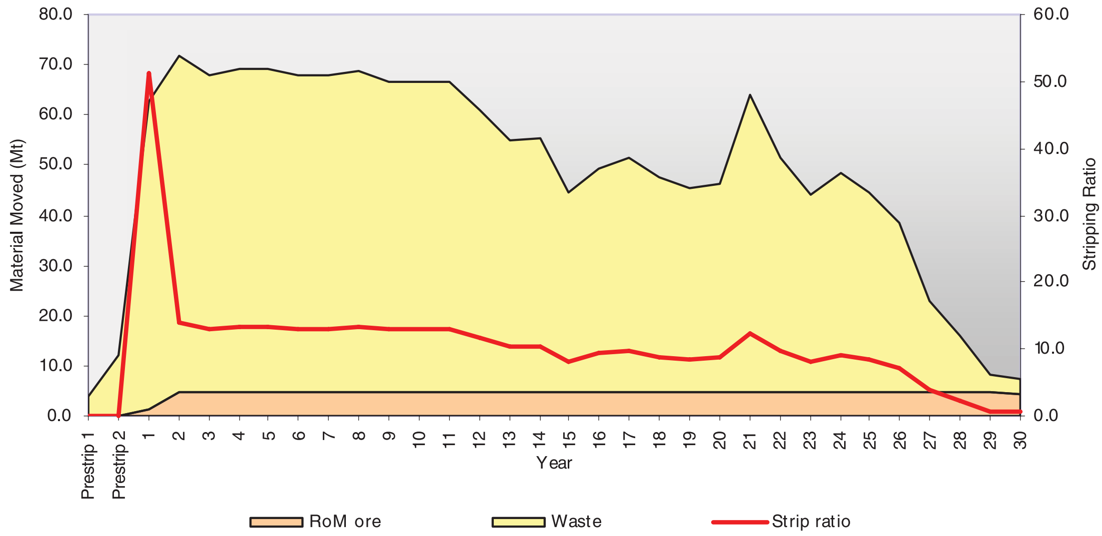

Figure 5.1: Total Rock Movement for the Life of Mine

The production scheduling was used for estimation of the required truck hours, the mining fleet estimation as well as the operating and capital cost estimation. The equipment selection was made based on the production requirements and is summarised in Table 5.8.

Table 5.8: Mining, Support and Utility Equipment

Item	Peak Requirements
Primary Mining Fleet	
Blast hole drills	4
Hydraulic face shovels	5
Electric haul trucks	19
Major Support Equipment	
Front End Loader, 190t	2
Bulldozer, 65t	7
Rubber Wheel Dozer, 45t	4
Graders, 24t class – 4.2 m blade	5
Rock Breaker	2
Secondary drills (including grade control and geotechnical drilling)	3
Water Carts, 100tonne	3
Wheel Loader, Road Building, 50ton	1
Road Building Tippers	2
Diesel Bowser, 12 000 ℓ	2
Low-bed	1
Utility Fleet:	
Light Delivery Vehicles (LDV),	20
Road Stone Crusher	1
Lighting Plants	7
Busses	3

From the detailed mining equipment schedule and quotations received from equipment suppliers, capital costs were derived for the primary, main support and utility support equipment as shown in Table 5.10. The replacement and expansion capital requirements over the LoM amounts to US$335.03 million.

Table 5.10: Summary of Initial Capital Expenditure

Item	Budget Capital Estimate (US$ million)	Assembly (5%) (US$ million)	Import Duty (2%) (US$ million)	Total (US$ million)
Initial Capex				
Trucks	64.87	3.24	1.30	69.41
Shovels	50.79	2.54	1.02	54.35
Drills	13.48	0.67	0.27	14.42
Front End Loader	6.77	0.34	0.14	7.24
Track Dozer	5.71	0.29	0.11	6.11
Rubber Wheel Dozer	3.15	0.16	0.06	3.38
Graders	1.85	0.09	0.04	1.98
Water Carts	2.00	0.10	0.04	2.14
Secondary drill	2.40			2.40
Road Building Tippers	1.60	0.08	0.03	1.71
Other Secondary Equipment	1.35			1.35
Utility Fleet	2.12			2.12
Other	2.36			2.36
Sub Total	**158.44**	**7.92**	**3.17**	**168.96**
Contingency of 10%				**16.90**
Total				**185.86**

Table 5.12 summarises the mining operating cost of the project determined from first principles, by expressing the cost as a function of the production outputs.

Table 5.12: Average LoM Mining Operating Cost

Description	Operating Cost (US$/tonne)
Expressed per tonne of RoM mined	13.52
Expressed per tonne of total material moved	1.21

Krupp Industrietechnik GmbH, Germany, ("Krupp") prepared an estimate for the primary ore crusher and conveyor system from the KOV Pit to the processing plant. The Krupp scope of work included the design, fabrication, limited factory assembly, corrosion protection, transport, site erection and commissioning of the ore handling system.

The proposed project consisted of the following components:

- RoM tip with static grizzly, receiving hopper and apron feeder;
- Primary crushing plant c/w scalping screen and jaw;
- Conveyors complete with sacrificial, overland, stacking and reclaiming;
- 50,000-tonne longitudinal stockyard with stacker and bucket wheel reclaimer.

The capital expenditure estimate for the ore crushing and conveying system is US$48.3 million.

In the early 1990's Krupp supplied and installed an overburden, or waste removal system in the KOV pit for Gécamines. The system was abandoned in the late nineties during the time of the unrest and political instability in the DRC. The gyratory crushing plant and a section of the conveyor was cold commissioned, but was never installed in the lower part of the KOV pit. It is still situated at the side of the pit, some 2.5 km away, in pre-assembled condition. The jaw crushing plant, together with the complete conveyor system and spreader, were installed and commissioned, but only operated for a short period of time. It has as such not handled a significant quantity of material.

This waste removal system forms part of the infrastructure of the KOV open pit mine, and is included in the JV agreement between GEC and Gécamines. It is a valuable asset in the Project, and essential part of the planned mining of KOV.

Krupp advised that the system is capable of handling 8,000tph of waste material on a sustainable basis. This confirmed reports given previously to GEC by Gécamines. This translates to a tonnage of 64Mtpa of waste removal capacity, based on operating on a 24 hour per day, and 7 days per week operation.

The price submitted by Krupp for refurbishment of the complete waste handling system is estimated at US$27.1 million, based on an exchange of US$1 = ZAR6.50.

Mineral Processing

The New Processing Facility study is based on feeding 4.8Mtpa of KOV ore to the new leach plant plus 0.9Mtpa of both Tilwezembe and Kananga ores to the KZC concentrator with the concentrates being fed to the new leach plant.

For this CPR, the use of the KZC concentrator and the mined ore from both Tilwezembe and Kananga has been excluded for the financial model. The concentrates to be produced from these two mines will be approximately 216kpta, or less than 5% of the leach plant throughput. The copper grade will be higher than the KOV material whilst the cobalt grade will be considerable higher than the KOV material.

The KOV pit is currently flooded and as such fresh bulk samples of ore are not available for pilot plant test work. This has resulted in the necessity of using previously mined ore (from 1998) for the bulk test work to be undertaken at Mintek. Metallurgical drilling of the ore body has been undertaken with samples being dispatched to SGS Lakefield.

The pilot plant test phase at Mintek as well as the metal recovery test work has still to be completed and thus cannot be reported at this time. The preliminary results indicate that an overall leach recovery of >90% for Cu with >83% for Co is likely, depending upon lithology. Acid consumption will be between 100 and 260kg/t, again depending upon lithology. The average acid consumption is likely to be approximately 150kg/t, but subject to additional test work.

The New Processing Facility to be built for processing the KOV ore is expected to consist of the following unit processes:

- RoM Crushing, Conveying and Milling
- Sulphide Flotation Recovery
- Dewatering of Oxide Tailings
- Sulphide Oxidation
- Sulphur burning plant for acid production
- Agitated Atmospheric Oxide and Pressure Sulphide Leaching in sulphuric acid solution
- Thickening and Filtration of leach residue
- Leach residue repulsed and pumped to tailings dam
- PLS clarification
- Copper Solvent Extraction
- Copper Electro-winning
- Raffinate recycle to leach
- Copper removal from cobalt bleed stream with secondary Solvent Extraction
- Cobalt stream purification with lime
- Cobalt precipitation and recovery with lime
- Claiming of cobalt hydroxide to oxide

It is expected that test work will validate the design assumptions, but if not the design can be modified to incorporate the metallurgical test work results.

The basic design criteria upon which the design is based are regarded as reasonable, subject to the successful completion of the test work. From these design criteria, a full set of preliminary Process Flow Diagrams and Plant Layout Drawings have been completed and a detailed mechanical equipment list has been developed to enable the project to be adequately costed.

The current capital budget is for $US580.6 million. It has been stated in the Bateman Report that the sulphide flotation and leaching circuit has been deferred. This deferment is for 12 months only and not eliminated from the process flow sheet. The sulphide circuit is to be commissioned one year after the main plant is commissioned. The amount to be included (as per Bateman's) in the future capital model is $US61.3 million.

Material Contracts

The Kolwezi Concentrator (KZC) is currently running at a significantly reduced capacity compared to nameplate (nominally 4Mtpa), due to a lack of funds and proper repair and replacement of equipment. The KZC facilities were initially designed to simultaneously process two very different types of ores, i.e. oxide ores (malachite) with a siliceous gangue, and mixed ores (malachite/chalcocite) with a dolomitic gangue.

Currently KZC operates at a reduced availability treating between 1,000 to 2,000tpd. Milling, thickening and filtration are the current bottlenecks within the concentrator, the other being the infrequent delivery of RoM ore to the primary crusher and reagents for flotation.

The Shituru Refinery remains under the ownership and management of Gècamines. Under the terms of the Joint Venture Agreement with GEC, DCP have first right to treat concentrate through the plant to produce copper and cobalt cathode. It is proposed that concentrate will either be sold to third parties or the concentrate will be shipped from KZC by road and/or rail and treated under a toll treatment agreement at Shituru. GEC has the right to take over the management of the refinery if deemed appropriate.

It should be noted however, that due to the poor state of repair of the Shituru facility, less than half of the throughput name plate capacity is achievable and recoveries are significantly lower than indicated.

The lime plant is situated at Katontwe which is 6km north-west of Likasi and is accessible by both road and rail. The plant has been in operation since 1929 when production of lime commenced. The operation has been extended with the addition of two rotary kilns and cement grinding since commissioning. The limestone quarry that is located one kilometer from the lime plant.

The New Processing Facility will utilise approximately 200ktpa of crushed limestone and 85ktpa of burnt lime. This means that in excess of 300kt of quality limestone will be mined per annum for the New Processing Facility.

The Katontwe Lime quarry and Process Plant has the capability of providing the required quantity of limestone and burnt lime to the New Processing Facility, subject to the injection of capital funds for mining equipment and plant refurbishment. The quality of the energy source (coal) should be upgraded to improve the kiln availability.

Tailings Disposal

This section presents a review of the tailings disposal facilities proposed and highlights any potential risk(s) and subsequent consequences, and to make appropriate comments regarding such potential risks.

It is noted that the tailings dam and return water dam will be unlined facilities.

This is based upon the assumption that both the tailings and the slurry water will not impact on the environment through pollutant generation, either separately or jointly. In the terms of reference, SRK draws attention to the requirement to review the physical and chemical characteristics of the tailings material. At the time of report generation, the physical and chemical properties of the tailings material have not been established. This requirement is now under way (June 2006).

From the description of the sites, it is noted that the Far West Site will be easier to manage, as the topography is flatter and more uniform than the Far East site.

SRK has allowed for the capex in phases, mainly for earthworks and filter drain quality considerations. The total envisaged capex for the tailings dam complex is US$31.79 million.

Engineering Infrastructure

The DCP infrastructure at Kolwezi examined during the site visit included the roads, airport, KOV site, Diesel workshops, fuel storage, KZC plant, waste handling, the Kolwezi substation, Gécamines workshops and Luilu Substation.

- The DCP infrastructure ranges in condition from good to poor. The infrastructure has been accessed and the need for professional refurbishment is understood. A reasonable capital expenditure provision has been made for this work.

- A long term power supply risk has been identified both in the potential shortage of power available for distribution and the lack of a SNEL supply agreement. Solutions are being sought by the DCP team. Their approach has a high probability of success.
- The general management of the electrical supply infrastructure at both the RO and Kolwezi Substations is poor. Their lack of funds and management skills are recognised. Capital expenditure has been allocated to assist in certain areas. Solutions are required that would influence management and discipline in areas outside DCP's control.
- The operating cost allowed after the various infrastructural upgrades is not clearly listed. The amounts allocated appear too low. It is recognised that these amounts are small compared to the total operational costs of DCP and hence do not pose a significant risk. A more detailed list is recommended.
- Dust generation management for the local roads will be required on an on-going basis, particularly in the villages, to ensure adequate road safety.
- The long term maintenance and repair of the roads, particularly the regional road remains significant risk, as the control of key factors affecting the road such as traffic density, overload and weather cannot be controlled by DCP.
- No fatal flaws were noted in the existing infrastructure and the future plan.

Human Resources

GEC anticipates that 31 out of some 54 senior management positions would be filled by expatriates during the early stages of the development and operation. Recruitment is likely to be conducted throughout the world. During construction, most of the staff would be on a rotation basis. However, as married accommodation and facilities become available, this would fall away.

There is a population of approximately 400,000 in Kolwezi, which has a long history of mining. GEC plans to recruit the maximum number of skilled staff possible from among the workers at Gécamines and from the area surrounding Kolwezi and in the greater DRC. DCP and Gécamines have established a joint team to handle worker expectations and their lack of pay for some time.

GEC has established remuneration levels that it believes will attract, retain and motivate workers. Jobs will be graded according to complexity and skill required. A "gate-wage" concept has been adopted which does away with allowances for food, housing and goods.

One of the key personnel within GEC and DCP that will be involved in the KOV Project is Jim Gorman, PrEng, BSc (Mining)—Acting CEO and COO. Jim is a mining engineer who has 37 years of extensive production, technical and executive management experience in the mining industry. Jim held executive management positions at Rossing Uranium in Namibia and Palabora Mining in South Africa. He was General Manager at Kansanshi Mine in Zambia, responsible for construction through to full production. He joined GEC in 2006.

Environmental Management

Environmental control measures have been planned and will be implemented to limit negative impacts but strict compliance with applicable international practices may not be possible at all times. A particular example is water quality, with discharges during the wet season exceeding acceptable salinity levels. Incremental impacts related to this, will, however, be limited in the current highly degraded environment in which the mine will operate. With adequate attention to environmental controls conditions can be improved with time and to this end GEC intends to implement the Project in a manner that is compliant with the environmental guidelines set out in the World Bank Pollution Prevention and Abatement Handbook and the IFC's Performance Standards on Social and Environmental Sustainability (which were formerly known as IFC Safeguard Policies), and will reflect these standards and policies, with any modifications as are appropriate for the GEC's circumstances.

SRK has compiled a draft EIS and EMP for Global Enterprises Corporate Ltd (GEC) in accordance with the principles of acceptable environmental practice and in terms of the applicable legislation of the Democratic Republic of the Congo (DRC) and in accordance with the requirements set out in Schedule IX of the Mining Environmental Guidelines of the DRC for the mining of the KOV open pit and the establishment of the new plant and tailings disposal facilities and associated works. In addition SRK has compiled a conceptual EIS and EMP for mining at Tilwezembe and Kananga, with ore from these two areas to be processed in the existing Kolwezi Concentrator from which concentrate will be railed to the refinery at Shituru.

All make-up water for the project will be obtained from a series of dewatering boreholes surrounding the KOV pit. The purpose of these boreholes is to limit the inflow of water into the pit. These boreholes will pump water to the Luilu River. The installed pumping capacity will initially be 8,300m^3/hr increasing to 9,000m^3/hr. Takeoff points will be provided for the plant and for the Luilu Village. The plant will only require approximately 470m^3/hr and therefore the majority of this water will be discharged to the river. This water is considered to be of generally good quality.

For the envisaged new direct leach plant diversion drains will be provided to limit the amount of runoff entering the plant area from the adjacent external catchment. All polluted runoff and washdown water originating at the plant site will be diverted by a system of drains to two 150,000m^3 lined pollution control dams at the bottom end of the plant site. Water collecting in these dams will be recycled for use in the plant. These dams have only been sized provisionally. It is still necessary to finalise the site lay-out and corresponding catchment areas, as well as to agree on the containment philosophy (number of hours average or peak rainfall to be contained, before release to the environment/river). At this stage it is probable that in times of heavy rain it may be necessary to discharge some of this water.

Where compliance with prescriptive DRC regulatory limits may be considered unnecessary or impractical for the specific site constraints and conditions, a risk based approach will be followed. This will enable planners to assess the potential for the proposed operation to impact on the environment and to develop solutions which address the real issues rather than simple compliance with the regulations.

A large section of the Musonoi Village will need to be relocated in year 14. It is therefore recommended that a provisional amount of US$30 million be allowed for this relocation. No social development plan has yet been compiled. However, it is proposed that an amount, still to be decided by the shareholders, will be allocated for this programme.

In terms of article 294 of the Mining Code, mines are required to provide financial security to cover the rehabilitation costs of the operations. It is understood that this refers specifically to the closure rehabilitation costs and does not include operational costs. The closure cost for this project amounts to US$9.53 million. The Regulations specify a set of formulae in terms of which this amount can be provided in instalments.

Technical Economic-Input Parameters

The valuation of the KOV Project as presented herein, has *inter alia* been based on the LoM plan and resulting production profile and associated revenue stream from saleable products operating costs and capital expenditure profiles (collectively referred to herein as TEPs) as provided to SRK by GEC, reviewed, and adjusted where appropriate by SRK. The generation of a LoM plan requires substantial technical input and detailed analysis and is critically dependent upon assumptions of the long-term commodity prices and their impact on the following: cut-off grades; potential expansion or contraction of the Mineral Resource and Mineral Reserve Base and the return on capital expenditure programmes.

The TEPs for the KOV Project have been correctly incorporated in the cashflow projections in the financial model ("FM") compiled by GEC and include:

- Commodity production profiles;
- Total Working Costs profiles as previously defined; and
- Capital Expenditure profiles.

The TEPs for the KOV Project are given in Table 12.1. All expenditures are stated in calendar years and in 1 January 2006 constant money terms. The capital and operating costs are all given in US Dollars. Any component of the costs which was derived from a foreign-currency denominated quote has been converted to US Dollars at the exchange rates ruling on 1 January 2006.

Table 12.1: GEC—KOV TEPs at 1 January 2006 (Real Terms)

Year	Tonnage Milled	Saleable Products: Copper Cathode	Saleable Products: Cobalt Salt	Real Expenditures: Total Working Costs	Real Expenditures: Capital	Real Expenditures: Total Expenditure
	(Mt)	(kt)	(kt)	(US$m)	(US$m)	(US$m)
2006	0.0	0.0	0.0	5.1	172.4	177.4
2007	0.0	0.0	0.0	17.1	373.0	390.1
2008	0.0	0.0	0.0	27.6	319.1	346.7
2009	0.9	34.1	2.1	128.6	358.7	487.3
2010	3.9	157.0	10.1	309.1	54.4	363.5
2011	4.7	240.8	19.9	369.0	28.4	397.4
2012	4.8	251.6	20.8	378.0	22.5	400.6
2013	4.8	221.0	18.7	364.2	13.8	378.0
2014	4.8	186.1	19.6	351.7	15.3	367.0
2015	4.8	201.4	26.9	367.1	43.0	410.1
2016	4.8	228.9	26.1	386.1	76.3	462.5
2017	4.8	233.3	25.7	387.9	17.8	405.6
2018	4.8	235.4	23.2	384.9	48.9	433.8
2019	4.8	238.5	20.8	382.9	17.7	400.6
2020	4.8	208.4	19.2	365.5	114.5	480.0
2021	4.8	183.5	18.3	352.0	25.8	377.8
2022	4.8	186.5	16.7	348.5	19.6	368.1
2023	4.8	183.5	25.7	355.9	60.0	415.8
2024	4.8	183.5	24.0	353.9	52.9	406.8
2025	4.8	183.5	25.7	357.0	10.4	367.3
2026	4.8	183.5	19.2	346.9	22.6	369.5
2027	4.8	183.5	16.7	341.8	42.8	384.6
2028	4.8	183.5	14.7	334.1	73.5	407.6
2029	4.8	214.9	11.4	349.1	12.4	361.5
2030	4.8	210.1	11.4	341.6	34.8	376.4
2031	4.8	208.4	15.5	342.4	15.7	358.1
2032	4.8	207.0	18.7	348.4	50.4	398.7
2033	4.8	205.7	17.1	343.8	11.6	355.4
2034	4.8	235.0	15.1	349.6	17.8	367.4
2035	4.8	240.2	15.1	335.5	9.7	345.2
2036	4.8	237.2	15.1	326.1	9.4	335.5
2037	4.8	233.7	17.5	319.2	9.1	328.3
2038	4.8	219.1	19.4	350.7	9.0	359.7
Total	139.1	6,118.5	550.2	10,421.1	2,163.2	12,584.2

The production profile of the copper and cobalt recovered per Table 12.1 is shown in Figure 12.1.

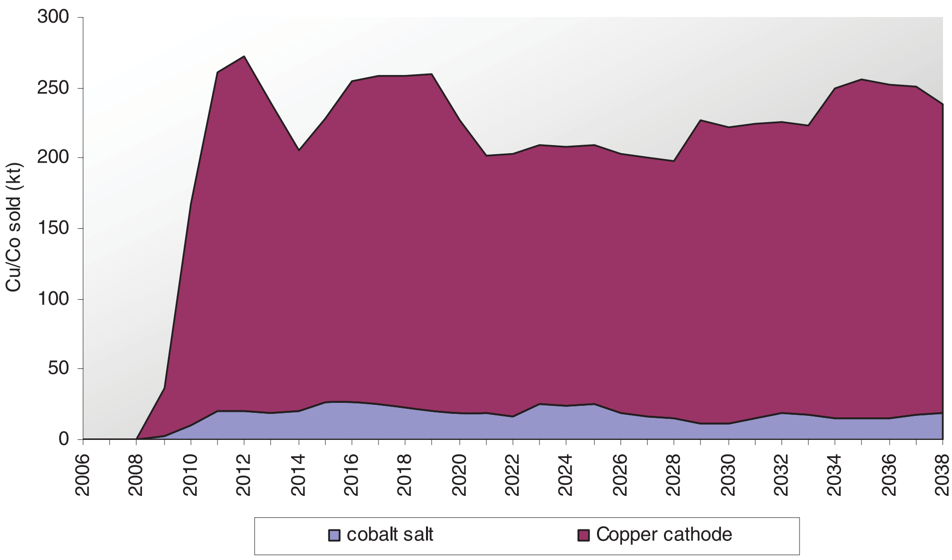

Figure 12.1: Production profile—recovered copper and cobalt

Material Properties Valuation

The summary valuation for GEC is based on a sum of the parts approach using:

- The discounted cashflow ("DCF") technique applied on a post-tax pre-finance basis on real cash flows for the KOV project. This is based on the DFS as provided by GEC including the resulting Technical Economic Parameters ("TEPs") (Section 13);
- For the Kananga and Tilwezembe projects, an average value has been derived from a number of different valuation techniques:
  - The value of contained copper and cobalt in declared resources, modified by the weighted effect of a range of technical issues;
  - The DCF value for KOV converted to a US$/t of equivalent copper produced and applied to the copper-equivalent contained in each deposit;
  - Comparable transactions for Cu-Co projects in the DRC during the past 1 to 2 years, converted to a US$/t of copper-equivalent produced; and
  - Future exploration expenditure.

No value has been attached to the Material Contracts.

The NPVs for KOV at a range of discount factors in relation to the WACC are shown in Table 13.3;

Table 13.3: KOV—NPV at various discount factors

Discount Factor (%)	NPV (US$m)
0.0%	8,789.9
2.0%	5,944.0
4.0%	4,105.8
6.0%	2,881.0
8.0%	2,040.5
10.0%	1,447.9
12.0%	1,019.7

Note that the above NPV's relate to cashflows attributable to the KOV Project and directly to DCP, and do not represent the quantum attributable to GEC.

SRK has assigned the average of the numbers in Tables 13.11 and 13.12 as the values for Kananga and Tilwezembe, US$37m and US$55m respectively. Note that these values relate to the gross in-situ Cu-equivalent tonnage and not the quantum attributable to GEC.

Assigning a value to GEC is not simply a case of taking a 75% share of the DCP valuation. In terms of the JVA, GEC funds Gécamines' share of the capital requirement and records this as a loan. The loan together with interest at an agreed rate, is then repaid from future cashflows before any amounts are paid to Gécamines.

This process is correctly incorporated into the FM and is the basis on which a portion of the KOV value has been assigned to GEC (Table 14.1).

The summary valuations ascribed to GEC and DCP set out in Table 14.1 are based on a sum of the parts approach, as described in this CPR.

Table 14.1: Summary Valuation for DEC and GEC

Property	Units	Value Attributable to DCP	GEC
KOV (DCF @ 10%)	(US$m)	1,447.9	1,085.9
Kananga	(US$m)	37.0	27.7
Tilwezembe	(US$m)	55.0	41.4
Total Asset Valuation	**(US$m)**	**1,539.9**	**1,155.0**
Adjustments:			
Estimated net (debt)/cash at 1 July 2006	(US$m)	26	26
Equity Value	**(US$m)**	**1,513.9**	**1,129.0**

Concluding Remarks

SRK has conducted a comprehensive review and assessment of all material issues likely to influence the future operations of GEC and DCP as these relate to the KOV Project. The Kananga and Tilwezembe properties have been treated as exploration projects for valuation purposes. In terms of the JVA with Gecamines, DCP has the right to use or access to the Processing Facilities. As these facilities do not relate directly to the DFS for KOV, they have been examined on the basis of material contracts and excluded from the valuation process.

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An Independent Competent Persons' Report on the Material Properties of Global Enterprise Corporate Limited

Our Ref: GEC AIM cpr final draft—rev 1 26 June 2006

1 INTRODUCTION

1.1 Background

SRK Consulting (South Africa) (Proprietary) Limited ("SRK") has been commissioned by the directors of Global Enterprises Corporate Limited ("GEC") to prepare an independent competent persons' report ("CPR") on the material assets and liabilities associated with the following Cu-Co projects located near Kolwezi in the Katanga Province of the Democratic Republic of Congo (the "Material Properties"):

KOV, a mining project;

Kananga, an exploration project; and

Tilwezembe, an exploration project.

The exploitation rights to the Material Properties are held in a joint venture vehicle DCP SARL (DRC Copper and Cobalt Project, "DCP"). GEC has a 75% interest in DCP, with the remaining 25% held by Gécamines (*la Générale des Carrières et des Mines*). Key information pertaining to the Material Properties has been summarised in Table 1.1.

Table 1.1 Summary Table of Assets

Asset	Holder	Status	Licence expiry date	Licence area	Comments
KOV	DCP	Development	16 Feb 2036	8.49km^2	Feasibility study completed, due to commence construction once financing secured.
Kananga	DCP	Exploration	16 Feb 2036	11.04km^2	Pre-feasibility study completed, additional drilling and testwork underway. Intermittently mined by Gécamines.
Tilwezembe	DCP	Exploration	16 Feb 2036	7.64km^2	Pre-feasibility study completed, additional drilling & testwork underway. Intermittently mined by Gécamines.

These have been valued by SRK using valuation techniques appropriate to the stage of development of each project.

1.2 Competent Persons' Report—Requirement, Compliance and Sources of Data

1.2.1 Requirement

SRK was approached by GEC to prepare a CPR on the Material Properties in accordance with the admission requirements of the Alternative Investment Market ("AIM") of the London Stock Exchange in anticipation of a listing of GEC on AIM during 2006. SRK has been informed that a copy of this CPR will be filed with AIM and included in the prospectus lodged by GEC in support of its admission to AIM (the "Admission Document").

1.2.2 Compliance

This report has been prepared under the direction of the Competent Person (the "CP") who assumes overall professional responsibility for the document (Section 1.7). The report, however, is published by SRK, the commissioned entity, and accordingly SRK assumes responsibility for the views expressed herein. Consequently all references to SRK mean the CP and vice-versa.

This CPR has been prepared in accordance with the Guidance Note for Mining, Oil and Gas Companies issued by AIM on 16 March 2006.

1.2.3 Sources of Data

Details of data/information used to prepare this CPR are as follows:

- Gécamines Western Proposal—pre-feasibility study, Report No M3037, Bateman Metals (as lead consultant) 15 June 2005;
- DRC Copper/Cobalt Project SARL—Definitive Feasibility Study ("DFS"), compiled by GEC, May 2006 (with input from a consortium of independent consultants);
- Financial Evaluation Model for the KOV Project, as part of the Definitive Feasibility Study for the KOV Project, compiled by Global Enterprises Corporate, May 2006;

1.2.4 Site Visits

Inspection visits of the Material Properties were made by members of the SRK team as follows:

- V Simposya (Geology, Resources) Dec. 2004, 2005, 2006;
- J Venter (Rock Engineering) Dec 2004;
- HG Waldeck (Mining, Reserves) May 2003, May 2006;
- B Read (Environment) Jan to Apr 2006;
- M Hobbs (Infrastructure, Power, Services) May 2006;
- K Stanford (Tailings disposal) Mar 2006.

Mr G Cunningham (Metallurgy, Process Plant) worked full time in Kolwezi for one year in 1999/2000 and was therefore familiar with the conditions in the area.

1.3 Effective Date and Valuation Date

The effective date (the "Effective Date") of this report is deemed to be 1 July 2006, and is co-incident with the Valuation Date. The cashflow projections commence on 1 July 2006, consequently SRK does not foresee any material change to the Technical-Economic Inputs, (TEPs) and the resultant valuation of GEC between the Effective Date and the date of the planned admission of the Company to AIM.

1.4 Verification, Validation and Reliance

SRK has made use of certain technical, financial and legal information in preparing this report. The technical information as provided to and taken in good faith by SRK has not been independently verified by means of re-calculation. SRK has however:

- Conducted a review and assessment of all material technical issues likely to influence the future performance of the Material Properties:
  - Inspection visits to the Material Properties, surface structures and associated infrastructure at Kolwezi undertaken between December 2004 and May 2006;

 - A review of the pre-feasibility study for the Material Properties compiled in June 2005;
 - A review of the DFS compiled by GEC in May 2006;
 - Discussion and enquiry with key personnel on site and GEC corporate offices;
 - Report on the estimation and classification of Mineral Resources and Mineral Reserves as developed by SRK for the DFS as at April 2006;
 - Report on the production forecasts contained in the Life-of-Mine ("LoM") plan, Mineral Reserves and mining budgets as developed by SRK for the DFS;
 - an examination of historical information provided by GEC; and
- Satisfied itself that such information is both appropriate and valid for valuation as reported herein. SRK considers that with respect to all material technical-economic matters it has undertaken all necessary investigation, both in terms of level of investigation and level of disclosure, to satisfy the reporting requirements of the Guidance Note for Mining, Oil and Gas Companies issued by AIM in March 2006 ("AIM Mining Guidance").

SRK's approach in the Mineral Resource and Mineral Reserve estimations and classifications is detailed in Section 4 of this CPR.

SRK has performed all necessary validation and verification of the information provided by GEC in order to place an appropriate level of reliance on such information.

1.4.1 Technical Reliance

SRK places reliance on the contributors to the DFS that all technical information provided to SRK is both valid and accurate for the purpose of compiling this CPR. The information with respect to Mineral Resources and Mineral Reserves has been prepared under the direction of SRK.

1.4.2 Financial Reliance

In consideration of all financial aspects relating to the Material Properties and the valuation thereof, SRK has placed reliance on GEC that the following information is accurate as at 1 July 2006:

- Unredeemed capital balances;
- Assessed losses;
- Opening balances for debtors, creditors and stores;
- Working capital and taxation logic;
- Balance sheet items, specifically cash on hand, debt and mark to market value of derivative instruments (currency and commodity hedges).

1.4.3 Legal Matters

In relation to legal aspects relating to the valuation of the Material Properties, SRK has assumed that the following are correct as at 31 May 2006:

- "a statement by the Directors of any legal proceedings that may have an influence on the rights to explore for minerals, or an appropriate negative statement" is suitably disclosed elsewhere in the Admission Document;
- the legal ownership of all mineral and surface rights has been verified; and
- no significant legal issue exists which would affect the likely viability of a project and/or on the estimation and classification of the Mineral Reserves and Mineral Resources as reported herein.

1.5 Valuation Basis

The summary valuation for GEC is based on a sum of the parts approach using:

- The discounted cashflow ("DCF") technique applied on a post-tax pre-finance basis on real cash flows for the KOV project. This is based on the DFS as provided by GEC including the resulting

Technical Economic Parameters ("TEPs") incorporated into the financial model ("FM"). The post-tax pre-finance cash flows presented for KOV incorporate the commodity price projections as shown in Table 11.1.

- For the Kananga and Tilwezembe projects, an average value has been derived from a number of different valuation techniques:
  - The value of contained copper and cobalt in declared resources, modified by the weighted effect of a range of technical issues on the risk or likelihood of the resources being mined. The valuation is based on a concensus opinion as supplied by GEC for the long-term copper price of US$1.10/lb and cobalt price of US$10.00/lb;
  - The DCF value for KOV converted to a US$/t of equivalent copper produced (applying the above prices to the total copper and cobalt produced) and applied to the copper-equivalent contained in each deposit;
  - Comparable transactions for Cu-Co projects in the DRC during the past 1 to 2 years, converted to a US$/t of copper-equivalent produced; and
  - Future exploration expenditure.

No value has been attached to the Material Contracts.

1.6 Limitations, Reliance on Information, Declarations, Consent and Copyright

1.6.1 Limitations

GEC has confirmed in writing to SRK that to its knowledge the information provided by it was complete and not incorrect, misleading or irrelevant in any material aspect. SRK has no reason to believe that any material facts have been withheld.

The achievability of LoM Plans, budgets and forecasts are neither warranted nor guaranteed by SRK. The forecasts of for the KOV project as presented and discussed herein have been developed by SRK based on information supplied to it by GEC, and cannot be assured; they are necessarily based on economic assumptions, many of which are beyond the control of GEC. Future cash flows and profits derived from such forecasts are inherently uncertain and actual results may be significantly more or less favourable.

This report includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur, SRK does not consider them to be material.

1.6.2 Reliance on Information

SRK believes that its opinion must be considered as a whole and that selecting portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in the report. The preparation of such a report is a complex process and does not lend itself to partial analysis or summary.

SRK's value for the Material Properties is effective at 1 July 2006 and is based on information provided by GEC throughout the course of SRK's investigations, which in turn reflect various technical-economic conditions prevailing at the date of this report. In particular, the value is based on expectations regarding the commodity prices prevailing at the date of this report.

These and the underlying TEPs can change significantly over relatively short periods of time. Should these change materially the value could be materially different in these changed circumstances. Further, SRK has no obligation or undertaking to advise any person of any change in circumstances which comes to its attention after the date of this CPR or to review, revise or update the CPR or opinion.

1.6.3 Declarations

SRK will receive a fee for the preparation of this report in accordance with normal professional consulting practice. SRK will receive no other benefit for the preparation of this report. Neither

SRK nor any of its employees and associates employed in the preparation of this CPR has any pecuniary or beneficial interest in GEC or in the Material Properties. SRK considers itself to be independent.

In this CPR, SRK provides assurances to the Directors of GEC that the TEPs, including production profiles, operating expenditures and capital expenditures, of the KOV Project as provided to SRK by GEC and reviewed and where appropriate modified by SRK are reasonable, given the information currently available.

1.6.4 Consent

SRK consents to the issuing of this report in the form and content in which it is to be included in documentation distributed to the directors of GEC, and in the Admission Document.

Neither the whole nor any part of this report nor any reference thereto may be included in any other document without the prior written consent of SRK as to the form and context in which it appears.

1.6.5 Copyright

Copyright of all text and other matter in this document, including the manner of presentation, is the exclusive property of SRK. It is a criminal offence to publish this document or any part of the document under a different cover, or to reproduce and/or use, without written consent, any technical procedure and/or technique contained in this document. The intellectual property reflected in the contents resides with SRK and shall not be used for any activity that does not involve SRK, without the written consent of SRK.

1.7 Qualifications of Consultants

SRK is a subsidiary of the international group holding company, SRK Global Limited (the "SRK Group"). The SRK Group comprises 500 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. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgment issues. The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, CPRs 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 companies and their projects, providing mining industry consultancy service inputs. SRK also has specific experience in commissions of this nature.

This CPR has been prepared based on a technical and economic review by a team of consultants sourced from the SRK Group's offices in South Africa. These consultants are specialists in the fields of geology, resource and reserve estimation and classification, open pit mining, rock engineering, metallurgical processing, hydrogeology and hydrology, tailings management, infrastructure, environmental management and mineral economics.

The individuals who have provided input to this CPR, who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

- Victor Simposya, PrSciNat, MSAIMM, BScMinSci, MSc;
- Peter Terbrugge, PrSciNat, MGSSA, MAEG, MSAIMM, ARSM, DIC, MSc;
- Brian Read, PrSciNat, MSAIAE, MGSSA, NatDip, BSc, BSc(Hons);
- Gordon Cunningham, PrEng, MSAIMM, MMMASA, BEng;
- HG (Wally) Waldeck, PrEng, FSAIMM, AMAMMSA, BScEng, MBA;
- Martin Hobbs, PrEng, BSc(Eng), BComm;
- Kenneth Stanford, PrTech, TMSAICE, NatDip(T4); and
- Andrew McDonald, CEng, FSAIMM, MIMMM, MSc, MBL.

The Qualified Person with overall responsibility for reporting of Mineral Resources in this CPR is Mr Victor Simposya PrSciNat (SACNASP), who is a Principal Geologist with SRK. Mr. Simposya is a resource geologist with 20 years experience in the mining industry and has been responsible for the reporting of Mineral Resources on various properties in Southern Africa and internationally during the past 5 years.

The Qualified Person with overall responsibility for the CPR and for reporting of Mineral Reserves is Mr. HG (Wally) Waldeck PrEng (ECSA), who is a Partner with SRK. Mr Waldeck is a mining engineer with 34 years experience in the mining industry and has supervised numerous due-diligence reviews in Southern Africa and internationally during the past 10 years.

2 GEC, DCP AND THE MATERIAL PROPERTIES

2.1 Introduction

This section gives an overview of GEC, DCP and the Material Properties including description, location, historical development and historical production statistics. A brief description of the country setting is included.

2.2 Company and Operating Structure

2.2.1 GEC

Global Enterprises Corporate Limited ("GEC"), a company registered in the British Virgin Islands, is 100% owned by Nikanor plc. Following execution of a Memorandum of Understanding on 5 May 2004, GEC and Gécamines entered into a Joint Venture Agreement ("JVA") on 9 September 2004 in relation to the rehabilitation and exploitation of three brownfield Cu-Co mines within Gécamines' "Group West" operational zone located near Kolwezi in the Katanga Province of the Democratic Republic of Congo ("DRC") (see Figure 2.2). The JVA comprises the mines of KOV, Kananga and Tilwezembe, the processing facilities of the Kolwezi Concentrator and an existing electro-refinery at Luilu (the "Existing Electro-Refinery"), and certain rights in relation to the Shituru refinery in Likasi.

2.2.2 DCP

DCP SARL (DRC Copper and Cobalt Project, "DCP") is a joint venture vehicle established by Gécamines and GEC under the laws of the DRC for the exploitation of the KOV, Kananga and Tilwezembe deposits. DCP, a company registered in the DRC, is held 75% by GEC and 25% by Gécamines.

Three exploitation permits for the KOV, Kananga and Tilwezembe open pit mines were transferred from Gécamines' existing mining title Exploitation Permit No 525 (*Permis d'Eploitation*, "PE"). The three exploitation permits provide DCP with the exclusive right to conduct exploration, development and exploitation activities with respect to copper and cobalt (as well as related mineral substances) within the relevant mining zones (Figures 2.2 and 2.3).

The historical development of DCP is summarised in Table 2.1.

Table 2.1: DCP—historical development

Date	Activity
May 2004	Gécamines and GEC signed Memorandum of Understanding.
September 2004	Gécamines and GEC entered into Joint Venture Agreement.
October 2004	Ratification of JVA by Gécamines Board.
December 2004	Presentation by Gécamines and GEC to Economy and Finance Ministers ("ECOFIN"); recommendation by ECOFIN to Council of Ministers.
May 2005	Completion of a pre-feasibility study on the KOV, Tilwezembe and Kananga projects.
July 2005	Approval by Council of Ministers.
October 2005	Approval of the JVA by Decree (No 05/114) of the President of the DRC.
October 2005	Approval of the establishment of DCP SARL by Decree (No 05/115) of the President of the DRC.
December 2005	Ratification of JVA by DCP SARL.
February 2006	PE 4960, PE4961 and PE4963 granted to DCP.
February 2006	PE 4960, PE4961 and PE4963 registered to DCP at the Mining Registry.

A key commercial term of the JVA involves the lease or granting of certain rights to DCP with respect to following processing facilities (together the "Processing Facilities"):

- The Kolwezi Concentrator ("KZC");
- The Existing Electro-Refinery (together with waste sites and certain infrastructure together "Luilu facilities")
- Certain buildings and workshops (such as those at SKM) ("KOV facilities");
- Treatment of concentrates at the Shituru hydro-metallurgical treatment plant in Gécamines' Central Group operational zone around the town of Likasi.

In return for the use of the KZC, the Luilu facilities and the KOV facilities, a percentage of net sales receipts (2% for the first four years and 1.5% thereafter) will be payable to Gécamines.

The JVA also gives DCP a right of first refusal on surplus production of acid and/or lime from any of Gécamines' neighbouring plants.

The lease/granting of these rights has been treated as a material contract, although it will not form part of the valuation of GEC. A brief description of the Processing Facilities is presented in Section 6.12 of this CPR.

2.3 Country Setting

2.3.1 History

The Belgian Congo was established as a Belgian colony in 1907 and gained its independence as the Democratic Republic of the Congo in 1960. Following parliamentary elections, Patrice Lumumba was elected as Prime Minister and Joseph Kasavubu as president. The early years of the DRC were marred by political and social instability. Lt. General Joseph Mobutu seized power and declared himself president in a November 1965 coup. He subsequently changed his name to Mobuto Sese Seko and changed the name of the country to the Republic of Zaire. After 32 years in power, Mobutu's regime was toppled by a rebellion led by Laurent Kabila in May 1997. He renamed the country the Democratic Republic of Congo, but in August 1998 his regime was challenged by an insurrection backed by Rwanda and Uganda. Troops from Zimbabwe, Angola, Namibia, Chad and Sudan intervened to support the Kabila regime and a cease-fire was signed in July 1999. Laurent Kabila was assassinated in January 2001 and his son Joseph Kabila was named head of state. In October 2002, the new president was successful in negotiating the withdrawal of Rwandan forces occupying eastern DRC. A transitional government was set up in July 2003, from which time Joseph Kabila has remained as president and is joined by four vice presidents representing the former government, former rebel groups and the political opposition. The transitional government plans to hold a series of elections in July 2006 to determine the presidency and the National Assembly seats.

2.3.2 Climate

The climate in the DRC varies from tropical rain forest in the Congo River basin to tropical wet-and-dry in the southern uplands to tropical highland in the eastern areas above 2,000m in elevation.

The KOV project is located in the southern uplands of the DRC, where the tropical climate is characterised by distinct wet and dry seasons. The wet season (summer) starts in October and ends in April, with the heaviest rainfall occurring from November to January. The dry season occurs from May to September (winter). The average air temperature remains fairly constant (17°C to 22°C) throughout the year. Although there is no distinct winter and summer temperature regime, average temperatures peak during September and October at 30°C. The coldest month is July with an average daily minimum temperature of 10°C.

The vegetation in the area is deciduous tropical dry woodland generally referred to as Miombo Woodland. Miombo Woodland are characterised by woodland interspersed with broad grassy, seasonally waterlogged areas. Trees seldom grow to heights exceeding 20m, with the majority probably less than 8m high.

2.3.3 Regional Infrastructure

The DRC has considerable hydroelectric power generating capacity, which is controlled and distributed by the national power utility, Societe Nationale de Electricite ("SNEL").

Kolwezi lies along the transcontinental railroad system and has access to both east and west coast ports of Tanzania and Angola, as well as South Africa. Lubumbashi, some 300km south east of Kolwezi, is the commercial and industrial centre of the Katanga Province and hosts an international airport.

The existing infrastructure around Kolwezi (e.g. buildings, water lines, workshops and roads) is in a very bad state of repair. Cellular phones work in the area, although coverage is patchy.

2.3.4 Mining History

Up until recent years, all copper production in the DRC was through the State-owned Gécamines. Gécamines used to be the world's largest cobalt producer and one of the world's major copper producers during the 1970's and 1980's. Around 1987, Gécamines was producing about 470kt of copper and about 35kt of cobalt per annum, employing up to 23,000 people.

2.4 KOV

2.4.1 Description and Location

The KOV mining project comprises the four large orebody fragments of Kamoto-East, Oliviera and Virgule (hence the name KOV) and FNSR. The open pit operations of KOV and Kamoto East are situated at Latitude 10°42'S and Longitude 25°25'E, some 5km west of the town of Kolwezi in the Katanga Province in the south-eastern part of the DRC (Figure 2.2).

From the aerial photo of the KOV and Kamoto East pits (Figure 2.4), it can be seen that the existing opencast mine workings are filled with water. A programme to dewater the pits and the surrounding slopes is planned to start in the fourth quarter of 2006.

The KOV pit is in an area that is highly disturbed by past mining activities. The old Musonoi pit, located immediately east of KOV, has been partly back-filled. Extensive accumulations of waste materials are present to the north and south of KOV, and the abandoned and flooded Oliviera pit is situated to the north-east. The Kamoto East and Kamoto Nord pits are located to the south and south west of KOV. Some underground mining occurred in the Kamoto underground section.

2.4.2 History

Mining at the Musonoi pit commenced in 1943 and appears to have ceased production by 1984. The Kamoto East pit started in 1959 and continued operations to 1985. The Kamoto Nord pit operated from 1964 to 1983. The main KOV pit started in 1983.

Metallurgical records indicate that the ore from the Kamoto East orebody was delivered to the plant from 1960 to 1985, while ore production from the KOV pit commenced in early 1985 and carried on, at declining production rates, through to 2000. During that time, some 38Mt of ore was delivered to the plant at an average grade of 5.8%Cu and 0.5%Co (Table 2.2).

Table 2.2: KOV—Historical Production

Period	Source	Plant Feed	Head Grade (%)		Contained Metal (kt)	
		(Mt)	Cu	Co	Cu	Co
1960 – 1985	Kamoto East	20	5.97	0.53	1,194	106
1983 – 2000	KOV	18	5.56	0.48	1,001	86
Total		**38**	**5.78**	**0.51**	**2,195**	**192**

2.4.3 Existing KOV Infrastructure

A significant asset within the existing infrastructure at KOV is a KRUPP waste conveyor system, complete with in-pit crushers and a large spreader, capable of handling 64Mt of waste per year. This conveyor system operated for only 6 months in 1997. Thyssen-Krupp engineers inspected the system in March 2006 and reported that the conveyor system was in very good condition.

2.5 Kananga

2.5.1 Description and Location

The Kananga project is situated at Latitude 10°40'S and Longitude 25°28'E some 5km north of the town of Kolwezi.

The deposit has not been adequately drilled but is believed to continue largely uninterrupted over a strike length of about 6km.

The ore is mainly oxide in nature with very little sulphide material in the mineralogy. Predominant copper minerals are malachite and pseudo-malachite associated with the cobalt mineral, heterogenite. The host rock is both dolomitic and siliceous. Copper and cobalt head grades are reasonably well defined using both current and historical records from the Gècamines geology database and from head grades of ore processed in the KZC concentrator. The resulting oxide concentrate was leached and refined at the 'Old' Luilu and Shituru refineries, located at Kolwezi and Likasi respectively, producing a "B" grade copper cathode and cobalt metal.

The Kananga pit is located close to the Dilala River and wetland and is also within 20m of the Lubumbashi-Lobito railway line.

A panorama photo of the Kananga Pit is shown in Figure 2.5 taken in May 2006.

2.5.2 History

Gècamines conducted open pit mining at Kananga using contract mining from around mid-2004. Latterly, organised mining by Gècamines has stopped and has been replaced by predominantly artisanal mining within the existing pit and along strike. The existing open pit is located at the western extremity of the orebody and had been mined to a depth of approximately 20m during SRK's visit to site in December 2004. Present mining operations have been confined to a small hill about 20m to 30m above the surrounding areas. The hill has subsequently been removed, as can be seen in Figure 2.5.

GEC has been unable to locate any records of historical production from Kananga.

2.6 Tilwezembe

2.6.1 Description and Location

The Tilwezembe project is located at Latitude 10°47'S and Longitude 25°42'E some 27km south east of Kolwezi, close to the Lualaba River on the road to Likasi. Mining has taken place intermittently since 1999.

A rail siding and contractors yard was established close to the site. Due to a shortage of rolling stock and locomotives, a stockpile of ore had been created close to the siding.

The deposit has not been adequately drilled but is believed to continue largely uninterrupted over a strike length of about 6km.

The ore is mainly oxide in nature with very little sulphide material in the mineralogy. Predominant copper minerals are malachite and pseudo-malachite associated with the cobalt mineral, heterogenite. The host rock is both dolomitic and siliceous. Copper and cobalt head grades are reasonably well defined using both current and historical records from the Gècamines geology database and from head grades of ore processed in the KZC concentrator. The resulting oxide concentrate was leached and refined at the 'Old' Luilu and Shituru refineries, located at Kolwezi and Likasi respectively, producing a "B" grade copper cathode and cobalt metal. The Tilwezembe deposit is very rich in cobalt, but also contains a high proportion of manganese that requires an adjustment to the processing method for this ore.

An aerial photo of the Tilwezembe open pit is shown in Figure 2.6.

2.6.2 History

Gècamines conducted open pit mining at Tilwezembe using contract mining on and off for a period of about 7 years. Latterly, organised mining by Gècamines has stopped and has been replaced by predominantly artisanal mining within the existing pit and along strike. The existing open pit is located at the western extremity of the orebody and had been mined to a depth of approximately 60m at the time of SRK's visit to site in December 2004. Mining operations at Tilwezembe have been confined to a hill above the elevation of the surrounding areas and only a few benches had been developed. In April 2006, SRK noted that the mining operations had ceased (Figure 2.6) and the contractor has removed all his equipment.

GEC has been unable to locate any records of historical production from Tilwezembe.

2.7 Regulatory Environment

SRK has been supplied with Chapter 3—Legal Aspects of the Feasibility Study for KOV (prepared by Djunga & Risasi and Linklaters) with respect to GEC's legal title to the mineral rights on the Material Properties on which the relevant part of the description below is based.

2.7.1 DRC Law in respect of Mining Title

DRC introduced the current Mining Code (Law No. 007/2002) (the "Code"), on 11 July 2002.

The Code was supplemented by the Mining Regulations (Decree No. 038/2003 of 26 March 2003) ("MR").

The right of ownership of the deposits of mineral substances constitutes in principle a right that is separate and distinct from the rights resulting from the surface area. However, subject to any rights of third parties over the surface, the holder of an exploitation licence has the right, pursuant to Articles 64 and 283 of the Code, to use the land surface necessary for his activities and in particular to build installations and infrastructures required for its mining exploitation, and to establish inside or outside his demarcated perimeter means of communication and transport of any type. The exploitation licence also entails the right to exploit artificial deposits (i.e. stockpiles and tailings) located within the mining perimeter covered by the licence.

The period of validity of new exploitation permits granted under the Code (*permis d'exploitation*) ("PE") is 30 years. The term of validity of a PE that derives from a Concession issued pursuant to the legal regime applicable prior to the enactment of the Code, however, expires on the original expiry date (Articles 336 of the Code and 580(c) of the MR). However, it is renewable several times for durations of fifteen years.

In terms of DRC property law (Law No. 73/020 of July 20, 1973), the soil and sub-soil are the exclusive and inalienable property of the State. Rights to use the land can be obtained pursuant to a grant of concession (*concession ordinaire ou perpétuelle*) by the State under the general principles of property law; pursuant to a lease from the holder of a concession or pursuant to a grant of rights to the minerals or timber located on the land.

In terms of the Code, any occupation of land depriving the rightful occupants of enjoyment of the surface rights, any modification rendering the land unfit for cultivation, will cause the holder of the mining rights, at the request of the rightful holders of the surface rights, to pay fair compensation, corresponding either to the rent or the value of the land at the time of its occupation, plus fifty per cent. Land means the ground on which the individuals have always carried out or are effectively carrying out any activity. However, the usual occupants of the land may, in agreement with the holder, continue to exercise their right to cultivate the land provided that the work in the fields does not hinder the mining activities. The owner of the surface rights may then no longer continue to construct buildings on it. The simply passing through the land by the holder does not entitle the owner to any compensation if no damage results therefrom (the Code, Article 281).

The PE entitles the holder to use the underground water and water courses within the permit area for the requirements of the mining exploitation in compliance with the requirements set forth in the environment plan to be submitted for the Project and approved by the *Direction chargée de la Protection de l'Environnement Minier* ("DPEM") and subject to the authorisation of the Governor of the province (Articles 64 and 283 of the Code).

The MR require the holder of a PE that is obtained pursuant to the transformation of a pre-existing mining right to submit an Environmental Adjustment Plan ("EAP") for approval (Article 408 of the MR). Since the MR require that all exploitation activities be undertaken in compliance with the relevant approved plan for the protection of the environment (Article 404 of the MR), failure to deliver the EAP may lead to suspension of works decided by the Minister in accordance with Articles 292 of the Code and 570 of the MR.

Once an EAP is approved, the holder of the PE will be required to put in place a financial guarantee as security for the performance of the rehabilitation obligations as determined in the EAP which must be acceptable to the DPEM. This security must be maintained until certification of satisfaction of the obligations has been obtained. The amount of the security as well as any other sums which may be provisioned by the titleholder for rehabilitation of the site are deductible in determining taxable income up to 0.5% of the turnover for the tax year during which the provision is made.

In terms of the Code a legal entity incorporated pursuant to Congolese law and which has its registered administrative office in the DRC and whose corporate purpose is mining activities is eligible for mining rights irrespective of the percentage equity interest held by an individual of foreign nationality or a legal entity incorporated pursuant to foreign law (Code, Article 23).

The holder of a mining exploitation title will be subject to the mining royalties due to the Treasury (at a rate of 2% for non-ferrous metals) on the amount of sales minus the costs of transport, analysis concerning the quality control of the commercial product for sale, insurance, and costs relating to the sale transaction (Code, Articles 240 and 241). Liability for mining royalties starts upon commencing of exploitation. Such royalties are due upon sale of the product.

The transfer of a PE does not relieve the initial holder from its obligations regarding rehabilitation of the environment (Article 186 of the Code). Liability for damages deriving from works prior to the transfer is joint and several for both the former and the new title holder. The former holder is required, however, to inform the new holder of any significant dangers or disadvantages resulting from exploitation, insofar as it is aware of them. Failing which, in case of any environmental liability arising prior to the transfer of the PE, the new holder will have the option to cancel/terminate the transfer or to recoup a portion of the transfer price. The new holder can also request, at the expense of the former title holder, the former title holder to eliminate the dangers or to suppress the inconveniences which may be caused to third parties (Article 280 of the Code).

2.7.2 GEC—Current Status

GEC and Gécamines entered into a JVA in relation to the rehabilitation and exploitation of the KOV, Kananga and Tilwezembe mines near Kolwezi. The JVA was structured for consistency with the provisions of the Code and included the following key elements:

- Obligation to pay customs duties, taxes (30% tax on profits and 10% turnover tax on local sales) and royalties (due to the Public Treasury at 2% of sales proceeds for non-ferrous metals, net of transport, insurance and sales costs);
- Obligation to pay surface area fees (US$5/ha/yr, plus a surface area tax of US$0.08/ha/yr, for the area covered by the PE);
- Obligations with respect to the protection of the environment; and
- Leasing by Gécamines (as Lessor) of the Processing Installations to the JVA (as Lessee), including obligations of the Lessee to maintain asset condition, meet taxes, observe all laws and make investments to ensure development.

In terms of the JVA, GEC is required to provide/procure financing necessary for the rehabilitation of the mines and related processing installations and infrastructure in order to bring them up to anticipated production levels and implement the necessary construction and upgrade works.

The three exploitation permits (PE4960, PE4961 and PE4963) grant DCP the exclusive right to carry out, within the areas over which they have been granted (see Figure 2.3), exploration, development, construction and exploitation works in connection with the mineral substances for which the permits were granted (i.e. copper, cobalt and associated mineral substances) (see Table 2.3). By virtue of these PE's, DCP is entitled to use the land to build installations and facilities required for mining exploitation. DCP's three PE's stem from Gécamines' PE No525, which itself originates from Gécamines' concession No23 granted under the former mining legislation. DCP's three PE's are valid until the anticipated expiry for Concession No23 (i.e. 3 April 2009), but can be renewed several times for further periods of 15 years (Article 67 of the Code). The three exploitation permits granted to DCP were registered in the Mining registry in March 2006.

Table 2.3: Mineral and Surface Rights at the Material Properties

Mining Property	Exploitation Permit No	Rights granted	Held By	Area of Title	Valid Until
KOV	PE4961	Cu, Co and associated minerals + Use of surface	DCP	10 blocks, 8.49km^2	03/04/2009; renewable
Kananga	PE4960	Cu, Co and associated minerals + Use of surface	DCP	13 blocks, 11.04km^2	03/04/2009; renewable
Tilwezembe	PE4963	Cu, Co and associated minerals + Use of surface	DCP	9 blocks, 7.64km^2	03/04/2009; renewable

The renewal of a PE may only be refused for a limited number of reasons expressly set out in the Code, including in particular: failure to pay surface charges, failure to demonstrate adequate remaining resource, insufficient financial capability of the titleholder; and failure to update environmental documentation.

The expiry date of the JVA, viz. 3 April 2039, is designed to reflect the renewal mechanics in the Code on the assumption that DCP's PE's are extended twice (for periods of 15 years each) following the expiry of the initial term.

Although Gécamines has not yet had an EAP approved for its PE525, contrary to the requirements of the Code, GEC is completing an Environmental Impact Study ("EIS") and an Environmental Management Plan ("EMP") as part of the DFS for the DCP Project. The EIS and EMP should satisfy the requirements of an EAP set by the Code and the MR. Now that the DFS has been completed, GEC will now finalise the EIS/EMP and submit these for approval as the EAP's for the three PE's.

GEC has not yet put in place the financial fund guaranteeing the performance of its rehabilitation obligations in terms of the EMP. This security can take the form of a cash deposit with a financial institution approved by the Central Bank or such other form of security described in the MR. The treatment of contributions to the rehabilitation fund adopted by GEC is not according to the Code, but GEC has assured SRK that the process followed has been discussed and accepted by the Mining Ministry.

In addition to approval of the EIS and EMP, DCP will also be required to obtain the following authorisations:

- Authorisation of the Governor of Katanga province;
- A licence for import/export activities;
- Post and telecommunication licence for radio equipment; and
- Various authorisations with respect to its local and expatriate workforce.

The areas of the three exploitation permits and their locations relative to Kolwezi for the KOV, Kananga and Tilwezembe projects are shown in Figure 2.3.

2.8 SRK Comments

Information provided to SRK indicates that a business organisation formed in terms of SARL (*société par actions à responsabilité limitée*) should have at least seven shareholders. To satisfy this requirement, GEC has included in its Deed of Incorporation five nominee shareholders who collectively hold a 0.5% share in DCP on behalf of GEC, with GEC holding 74.5%.

Formation of DCP has been authorised by the President of the DRC and the three PE's for KOV, Kananga and Tilwezembe have all been registered with the Mining Registry. According to the local law opinion provided by Djunga & Risasi on or about the date hereof, DCP has legitimate and secure tenure to the mineral rights of the three projects and is authorised to exploit the KOV, Kananga and Tilwezembe orebodies.

Figure 2.1: GEC—Locality Plan of Assets of JVA in DRC

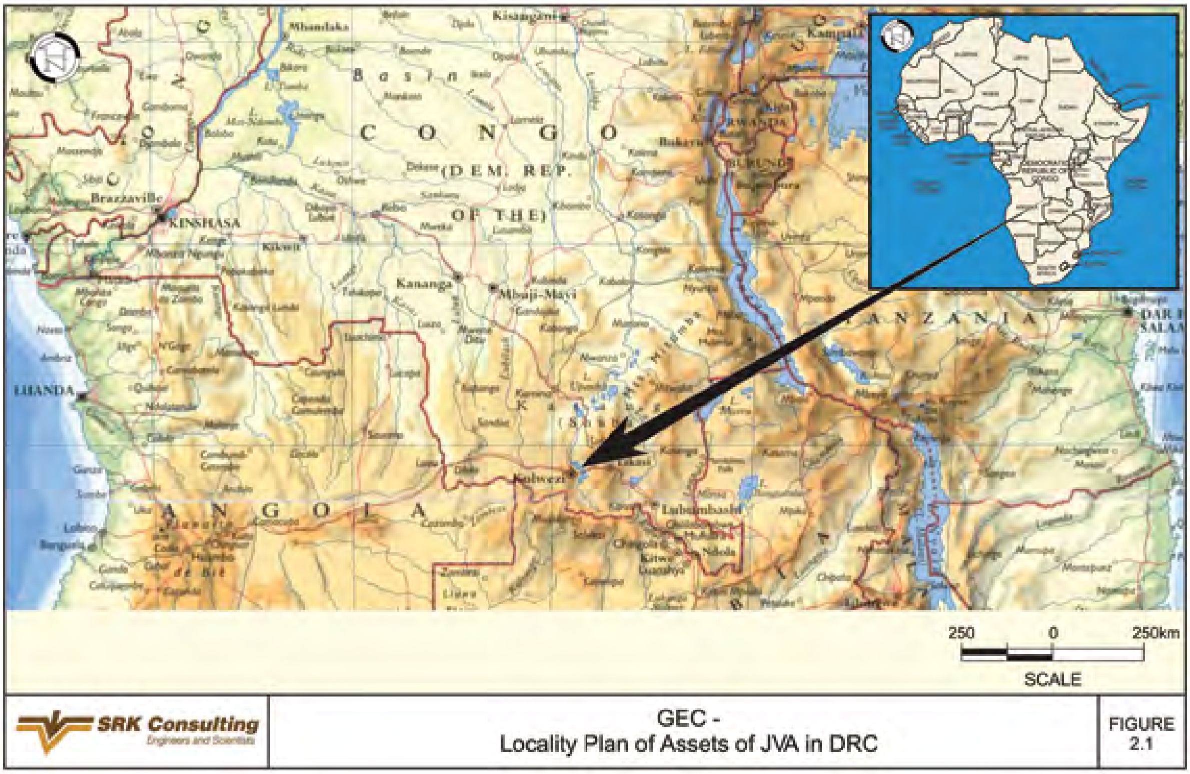

Figure 2.2: GEC—Exploitation Permits for the KOV, Kananga and Tilwezembe project areas

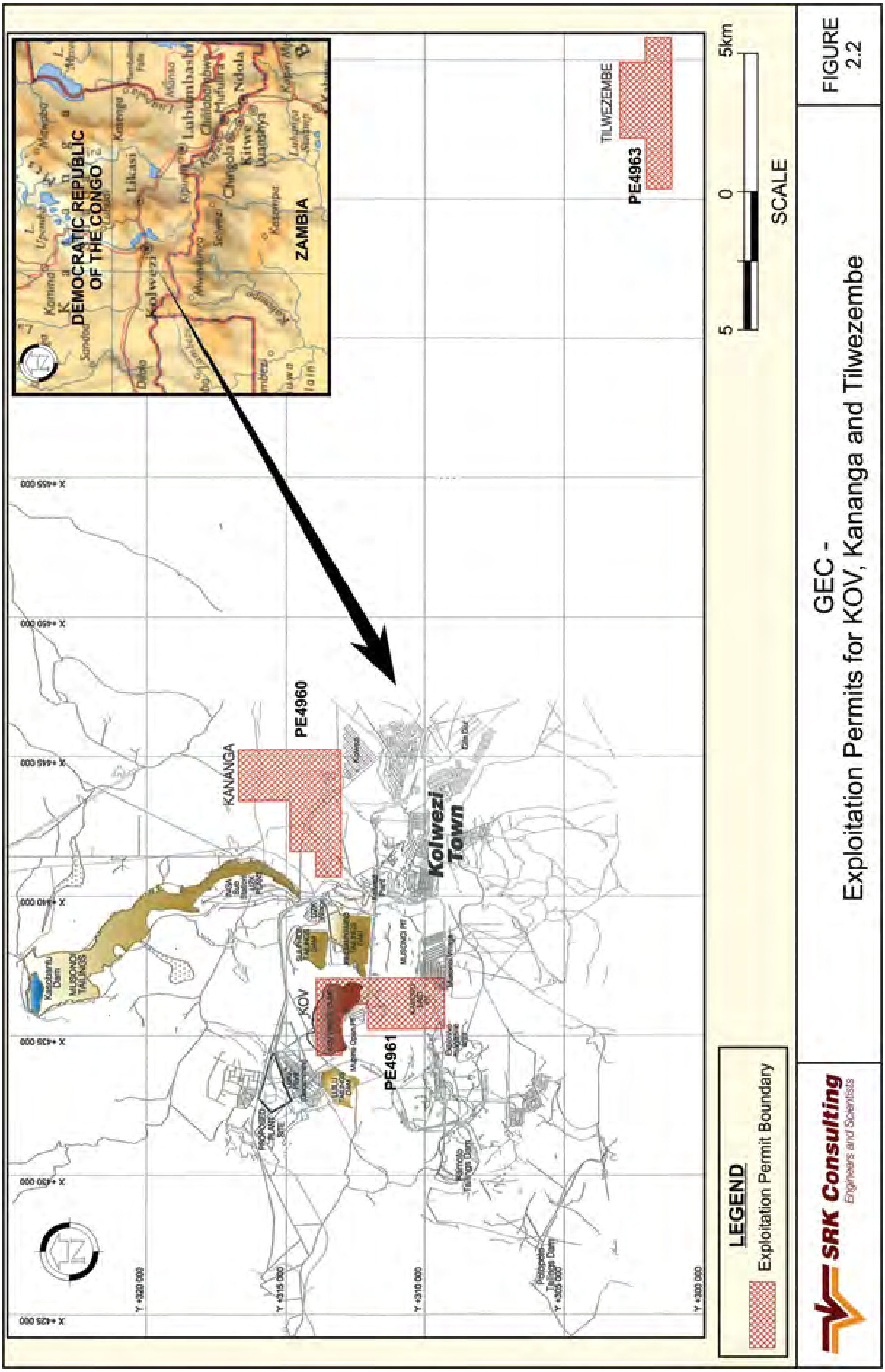

Figure 2.3: Aerial photo of the KOV and Kamoto-East Pits

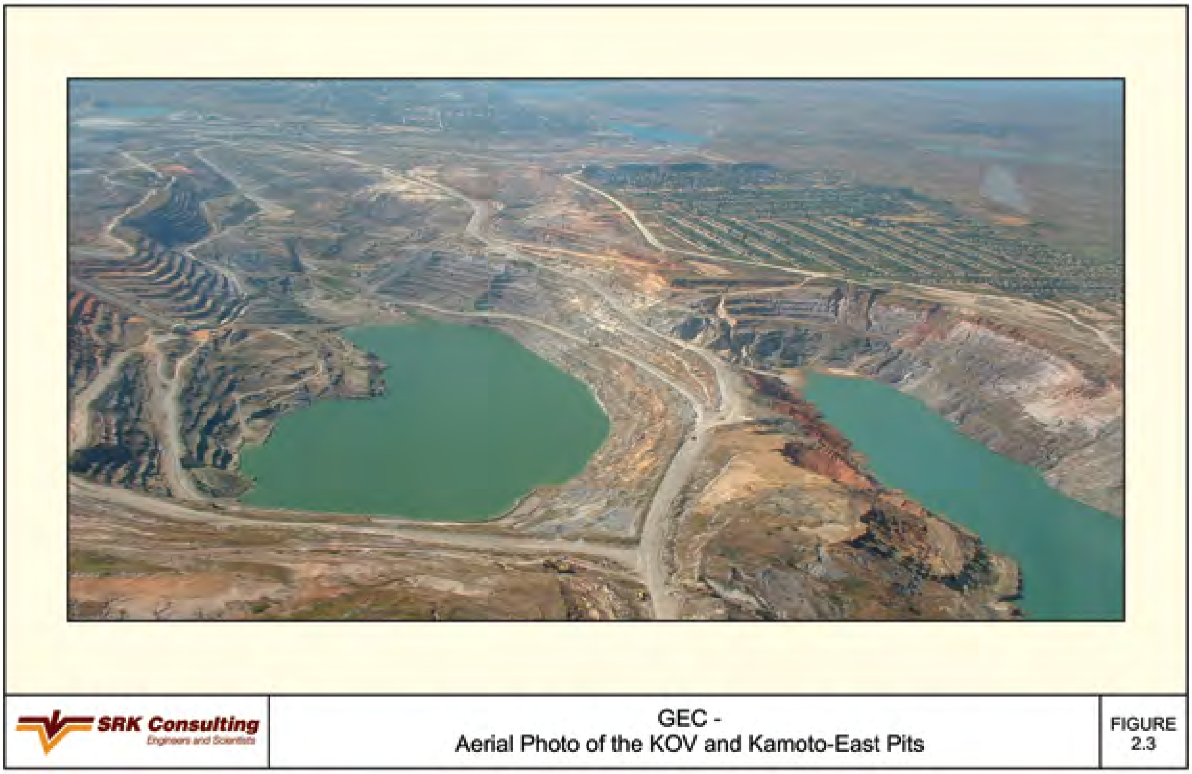

Figure 2.4: Panorama Photo of the Kananga Pit

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Panorama Photo of the Kananga Pit

FIGURE
2.4

Figure 2.5: Aerial Photo of the Tilwezembe Pit

3 GEOLOGY

3.1 Introduction

This section gives a brief summary of the regional geological setting around Kolwezi and describes the geology of the KOV, Kananga and Tilwezembe deposits.

3.2 Regional Geology

The KOV, Kananga and Tilwezembe orebodies are located at the western end of the Katangan Copperbelt, one of the great metallogenic provinces of the world, and which contains some of the world's richest copper, cobalt and uranium deposits, (Figure 3.1).

These deposits are hosted mainly by metasedimentary rocks of the Late Proterozoic Katangan System, a 7,000m thick succession of sediments with minor volcanics, volcanoclastics and intrusives. Geochronological data indicate an age of deposition of the Katangan sediments of ~880Ma and deformation during the Katangan Orogeny at <650Ma. This deformation led to the formation of the NW-SE trending Lufilian Arc, which extends from Namibia on the west coast of Africa through to Zambia, lying to the south of the DRC. Within the DRC, the zone extends for more than 300km from Kolwezi in the northwest to Lubumbashi in the southeast.

Stratigraphically, the rich copper and cobalt deposits found in Zambia and the DRC occur localised in the Roan Supergroup ("Roan"). The Roan occurs at the base of the Katanga succession, unconformably overlying the basement rock of Kibaran age (Mid-Proterozic). The Roan is separated from the overlying rocks of the Upper and Lower Kundelungu Supergroups by a conglomerate, The Grand Conglomerat. The Lower Kundelungu is composed of sandstones and shales with a basal conglomerate while the Upper Kundelungu consists essentially of sediments and is separated from the Lower Kundelungu by a conglomerate, the Petit Conglomerat.

Within the Lufilian Arc are large scale E-W to NW-SE trending folds with wavelengths extending for kilometres. The folds are faulted along the crests of the anticlines through which rocks of the Roan Supergroup have been diapirically injected into the fault zones, squeezed up fault planes and over thrust to lie above rocks of the younger Kundelungu. The overthrusted Roan lithologies occur as segments or "fragments" on surface which are intact units preserving the original geological succession within each of the fragments. A fragment could be of hundreds of metres aligned across the fault plane.

In the Katangan Copperbelt, the mining for copper and cobalt occurs in these outcropping to sub-outcropping fragments.

3.3 General Stratigraphy

The generalised stratigraphy of the Katangan System is shown in Figure 3.2. The Roan has been correlated across the Katangan Copperbelt into four main formations or groupings, R1 to R4. The divisions between each of the R series are often marked by an unconformity. The main orebody lithologies belong to the R2 Formation, but R3 and R4 Formations are also known to contain mineralisation. Within each of the R series are sub-divisions identifying the different lithological units. Rocks belonging to the Roan Supergroup are described briefly below from the oldest to the youngest:

Breche Heterogene or Heterogeneous Breccia (BH): This breccia is composed of angular and sometimes well rounded rock fragments of all the various rock types of the Roan Group. The fragments vary in size from a few millimetres to several centimetres in diameter while the matrix is made up of finer-grained sandy particles of the same material as the fragments.

Breche RAT or Brecciated RAT (B RAT): A reddish-pink brecciated rock with calcite and silica veinlets and is at times well mineralised with specular haematite, occurring as veinlets.

Roches Argilleuses Talceuse (RAT): The RAT is considered the boundary between the R2 and R1 units and consists of an upper RAT Grises (R2) and a lower RAT lilas (R1). Both are massive but sheared in places, silty or sandy, dolomitic rocks. Mineralisation in the form of malachite and black oxides occurs associated with the upper RAT.

Dolomie Stratifie or Stratified Dolomite (D Strat): This is a well bedded to laminated, argillaceous dolomite, which forms the base of the traditional "Lower Ore Zone" in Gécamines' nomenclature. The mineralisation consists of copper and cobalt oxides.

Roches Siliceuses Feuilletées Foliated (Laminated) and Silicified Rocks (RSF): This is a grey to light brown thinly bedded laminated and highly silicified dolomites. The unit is generally well mineralised with copper and cobalt oxides. Together with the D Strat, the RSF comprise the Lower Orebody.

Roches Silicieuses Cellulaires or Siliceous Rocks with Cavities (RSC): Vuggy and infilled massive to stromatolitic silicified dolomites. Copper mineralisation is almost absent in this rock and these were therefore regarded as barren. However, the infillings are enriched in wad (manganese oxide) and heterogenite (cobalt oxide) and this rock is the target of artisanal activity.

Schistes De Base or Basal Schists (SDB): Reddish-brown to grey silty and nodular dolomite to siltstone. This unit is well mineralised with copper and cobalt in varying amounts and forms the Upper Orebody.

Shales Dolomitiques Superieurs or Upper Dolomitic Shales (SDS): Yellowish, cream to red bedded laminated dolomitic siltstones and fine-grained sandstones. The rock is sparsely mineralised with malachite.

Calcaire a Minerais Noirs or Calcareous Unit with Black Minerals (CMN): A slightly banded and laminated light grey to grey silicified dolomite mineralised with black oxide of iron, manganese and cobalt. The unit bears some similarities with the RSC.

Dipeta (R3): Greyish to dark red or brown stratified shales and micaceous schist.

Mwashya (R4): altered stratified greyish siliceous dolomitic rock with oolitic horizons and a few bands of light yellow talcose schist. Nodules of hematite often occur.

3.4 Geology of the Deposits

The surface topography consists of flat-lying plains underlain by rocks of the Lower Kundelungu which rarely crop out as the flat plains are covered by overburden, deep soil and vegetation. Rocks of the Lower Roan occur in three discrete localities in sheared contact with the Lower Kundelungu formations. The Lower Roan sub-outcrops due to a thick sequence of overburden. The Lower Roan is overlain by a sequence of fine to coarse-grained saprolite which in turn is capped by a pisolitc iron crust.

Structurally, the Lower Roan fragments dip steeply to the north and strike in a NW-SE direction. The contacts with the Lower Kundelungu are brecciated and the depth extent of the Lower Roan fragments remains unknown.

3.4.1 Mineralisation

Primary mineralisation, in the form of sulphides, within the Lower Roan is associated with the D Strat and RSF for the Lower Orebody and the SDB and SDS for the Upper Orebody and is thought to be syn-sedimentary in origin. Typical primary copper sulphide minerals are bornite, chalcopyrite, chalcosite and occasional native copper while cobalt is in the form of carrolite. The mineralisation occurs as disseminations or in association with hydrothermal carbonate alteration and silicification.

Supergene mineralisation is generally associated with the levels of oxidation in the sub-surface sometimes deeper than 100m below surface. The most common secondary supergene minerals for copper and cobalt are malachite and heterogenite. Malachite is the main mineral mined within the confines of the current KOV pit.

The RSC, a lithological unit stratigraphically intermediate between the Upper and Lower Orebody host rocks contains relatively lower copper mineralisation. The RSC contains appreciable copper mineralisation near the contacts with the overlying SDB formation and the underlying RSF formations. The middle portion of the RSC, considered to be "sterile" by Gécamines, normally contains relatively less copper mineralisation and is sometimes not sampled. The mineral potential of the RSC is less well known.

The RSC has been observed to be well mineralised in supergene cobalt hydroxide, heterogenite, which occurs as vug infillings, especially near the surface.

3.4.2 KOV

There are three main individual "fragments" hosting mineralised Lower Roan lithologies within the KOV pit area. These are Kamoto East, Oliveira and Virgule, from which the name KOV is derived. A fourth and smaller fragment, the FNSR, is a remnant of the Musonoi West fragment mined to the east of KOV pit. The FNSR lies below and is sub-parallel to the Virgule orebody.

Other fragments within the area are OEUF and Variante. The OEUF consists mostly of hangingwall lithologies occurring above the Virgule fragment and the Variante lies below the Virgule and Oliveira fragments but outcrops towards the east in the Musonoi West area. Lower Roan lithologies have been identified in the Variante but current investigations indicate poor copper and cobalt mineralisation within these lithologies. Gécamines' interpretations of the relationship between the fragments are shown in Figure 3.3. Two other sections through the fragments based on Gécamines' interpretations of the drill hole results are shown in Figure 3.4.

Within each of the mineralised fragments, the succession of lithologies is intact although in the FNSR fragment the Lower Roan lithologies occur overturned.

The fragments that make up the KOV orebody occur in an east-west striking synclinal structure consisting of a steeply dipping southern limb and a shallow dipping northern limb named the Kamoto East and Virgule orebody respectively, while the Oliveira fragment occurs as a shallower dipping orebody in faulted contact with and below the Virgule orebody.

3.4.3 Kananga

Based on the interpretation by Gécamines, the Kananga orebody forms the northern limb of the Kananga-Dilala syncline. The deposit plunges to the south. The Kananga orebody outcrops and forms a ridge with a NNE strike. The ridge falls quite rapidly towards the south and has been cut to form part of the embankment for the Lobito railway line which runs parallel to the ridge between 10m to 20m away for most of the strike length of the orebody.

The deposit is divided by the Musonoi River into two hills called Kananga East and Kananga West. Mining activities have taken place on the western edge of the Kanaga East hill.

The geology and extent of the Kananga area covered by drilling is shown in Figure 3.5.

3.4.4 Tilwezembe

Based on the exploration work undertaken, the interpretation by Gécamines indicates that the Tilwezembe deposit is located in an NE-SW anticlinal structural lineament, which extends further to the east hosting four other known copper and cobalt deposits (Kisanfu, Myunga, Kalumbwe and Deziwa).

The mineralisation is hosted in the Neoproterozoic Lower Mwahsya Formation (or R4) consisting of strongly brecciated siliceous dolomites and shales with bands of hematite and oolites within the dolomites. Structurally, the Roan rocks constitute the anticline core overlying the younger Kundelungu Formation.

Mineralisation generally occurs as infilling of fissures and open fractures associated with the brecciation. The typical mineralisation consists mainly of copper minerals (chalcopyrite, malachite and pseudomalachite) and cobalt minerals (heterogenite, carrolite and spherocobaltite) and manganese minerals (psilomelane and manganite).

A section through the Tilwezembe deposit based on Gécamines' interpretations of the drill hole results is shown in Figure 3.6.

3.5 Exploration Programmes

3.5.1 Kananga

GEC has compiled an exploration programme for the Kananga deposit which is scheduled to involve the following main steps during the next one to two years:

- Drilling programme—resource confirmation and extension along strike;

- Geological modelling;
- Metallurgical test work;
- Evaluation of Processing Facilities;
- Feasibility study.

3.5.2 Tilwezembe

GEC has compiled an exploration programme for the Tilwezembe deposit which is scheduled to involve the following main steps over the next two to three years:

- Drilling programme—resource confirmation and extension along strike;
- Geological modelling;
- Metallurgical test work;
- Feasibility study.

3.5.3 Exploration Budget

GEC's budget to conduct exploration-related activities at the Kananga and Tilwezembe orebodies for the three years 2006 to 2008 is summarised in Table 3.1.

Table 3.1: GEC—Exploration Budgets for Kananga and Tilwezembe

Project / Activity	Units	2006	2007	2008
Kananga:				
Resource drilling	US$m	1.0	1.0	
Geological modelling	US$m	0.1	0.2	
Metallurgical testwork	US$m		0.5	
Evaluate Processing Facilities	US$m		0.5	
Feasibility Study	US$m		1.0	1.0
Tilwezembe:				
Resource drilling	US$m		1.0	0.5
Geological modelling	US$m		0.1	0.2
Metallurgical testwork	US$m		0.2	0.3
Feasibility Study	US$m			1.0
Total	**US$m**	**1.1**	**4.5**	**3.0**

The exploration budgets set out in Table 3.1 for the various aspects appear to be reasonable, based on the planned activities incorporated into the exploration programmes.

Figure 3.1: GEC—Regional Geology of the West Katangan Copperbelt

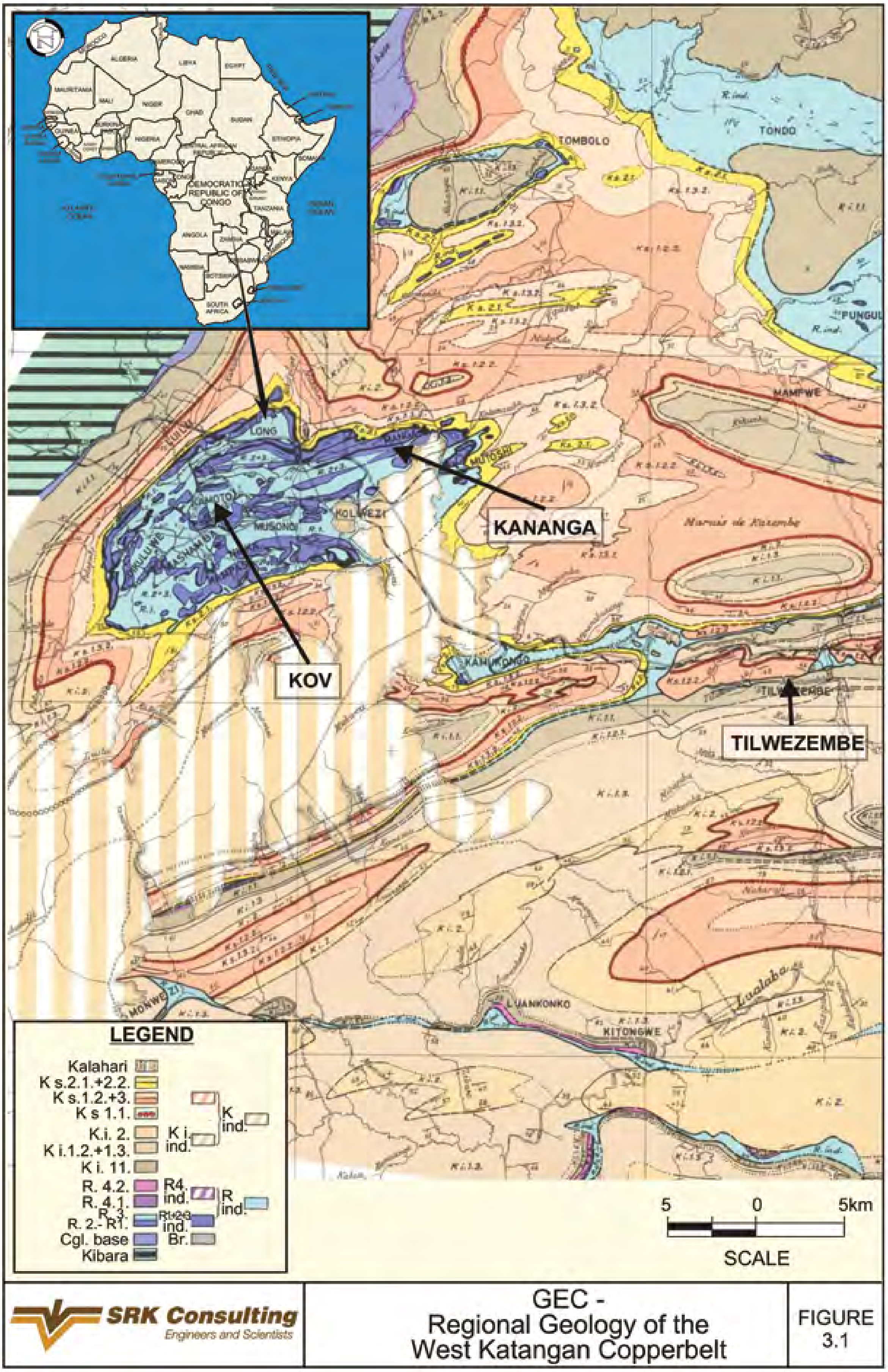

Figure 3.2: GEC—Generalised Stratigraphy of the Katangan System

KATANGA SYSTEM STRATIGRAPHY

System	Series	Formation	Local Name			Description	Thickness (m)
KATANGA	UPPER KUNDELUNGU					Sediments	30 - 50
	LOWER KUNDELUNGU					Sediments, sandstones and shales	200 - 500
	ROAN	R4-1and 2	MWASHYA			Shales, siltstone, sandstone to dolomites	50 - 100
		R3-2	DIPETA			Shales and sandy schists	1000?
		R3-1	RGS			Roches Greseuse Superieur	100 -200
		R2-3	CMN			Calcaire a Minerais Noirs	130
		R2-2	SDS			Schistes Dolomitic Superieur	50 -80
		R2-1	SDB	UOB	COMBINED OREBODY	Schistes De Base	10 -15
			RSC	IOB		Roches Siliceuses Cellulaire	12 -25
			RSF	LOB		Roches Silicieuses Feuilletees	5
			D STRAT			Dolomie Stratifiee, argilitic dolomite	3
			RAT grises			Roches Argilleuses Talceuse	2 -5
		R1	RAT 2			Roches Argilleuses Talceuse	190
			RAT 1			Roches Argilleuses Talceuse	40
			POUDINGUE			Unknown formation, transgression conglomerate	?

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GEC - Generalised Stratigraphy of the Katangan System

FIGURE 3.2

Figure 3.3: GEC—Local Geology of the KOV orebody (ex Gécamines)

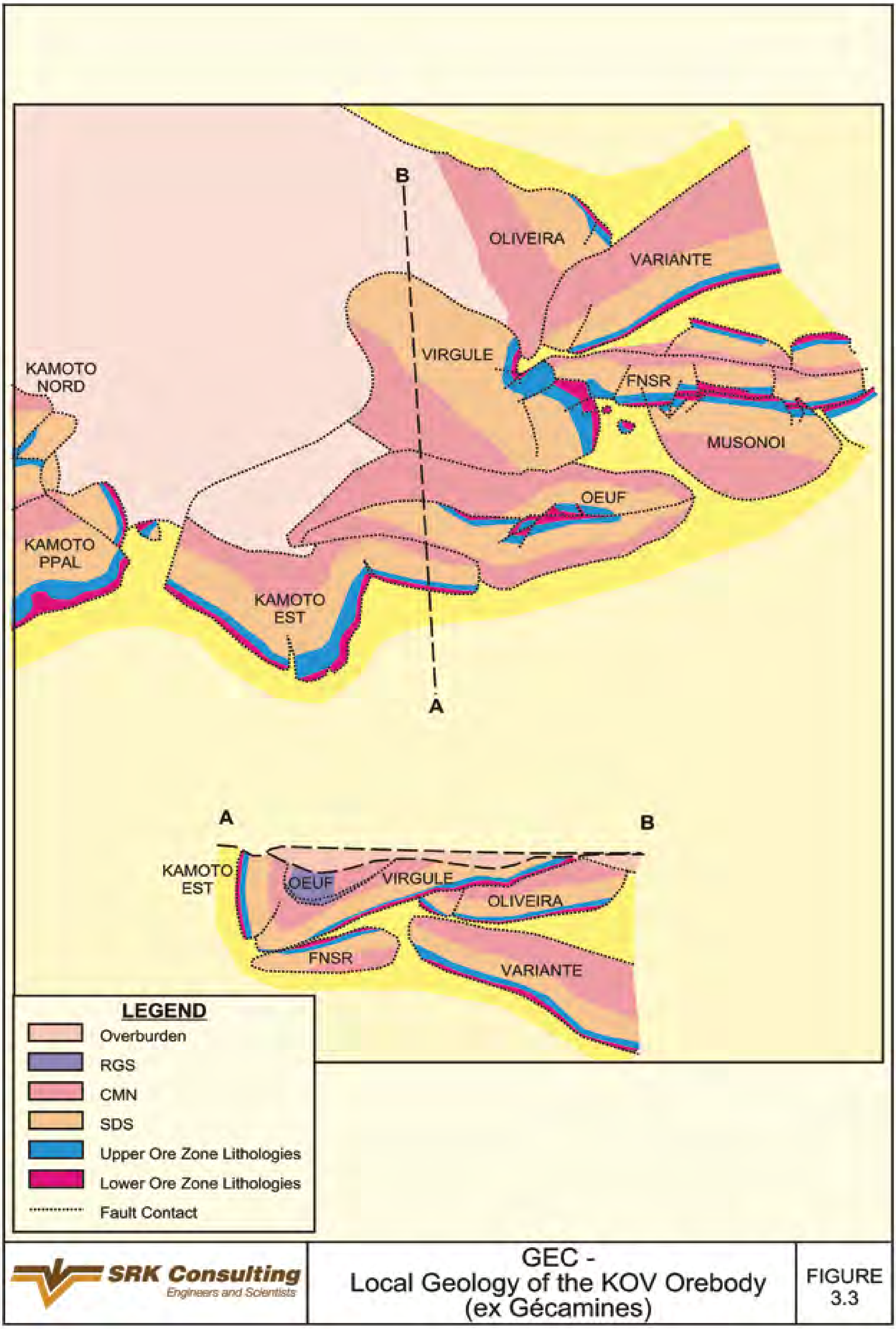


Figure 3.4: GEC—Interpreted sections through the KOV orebody (ex Gécamines)

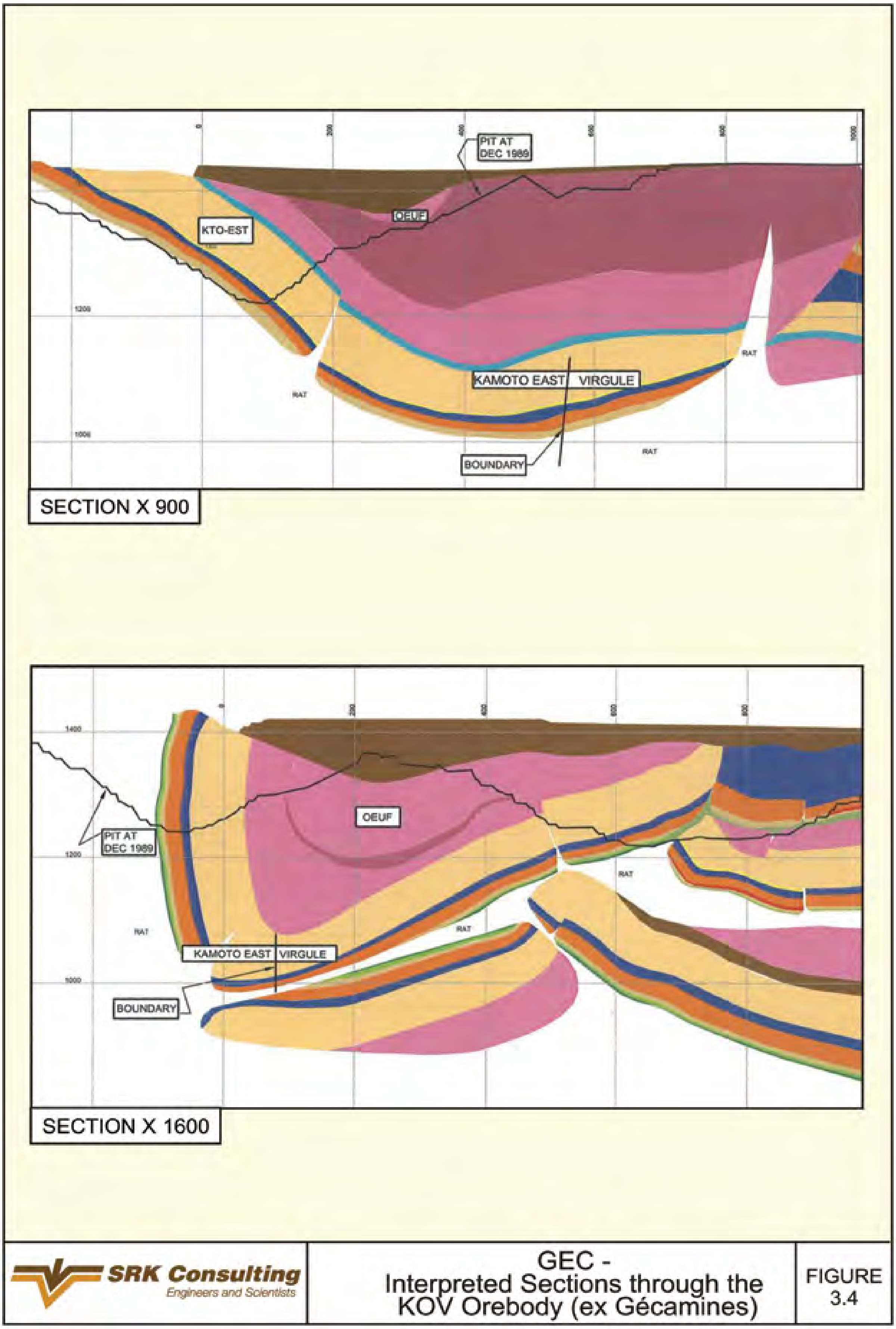


Figure 3.5: GEC—Simplified Geology of the Kananga deposit including borehole locations (ex Gécamines)

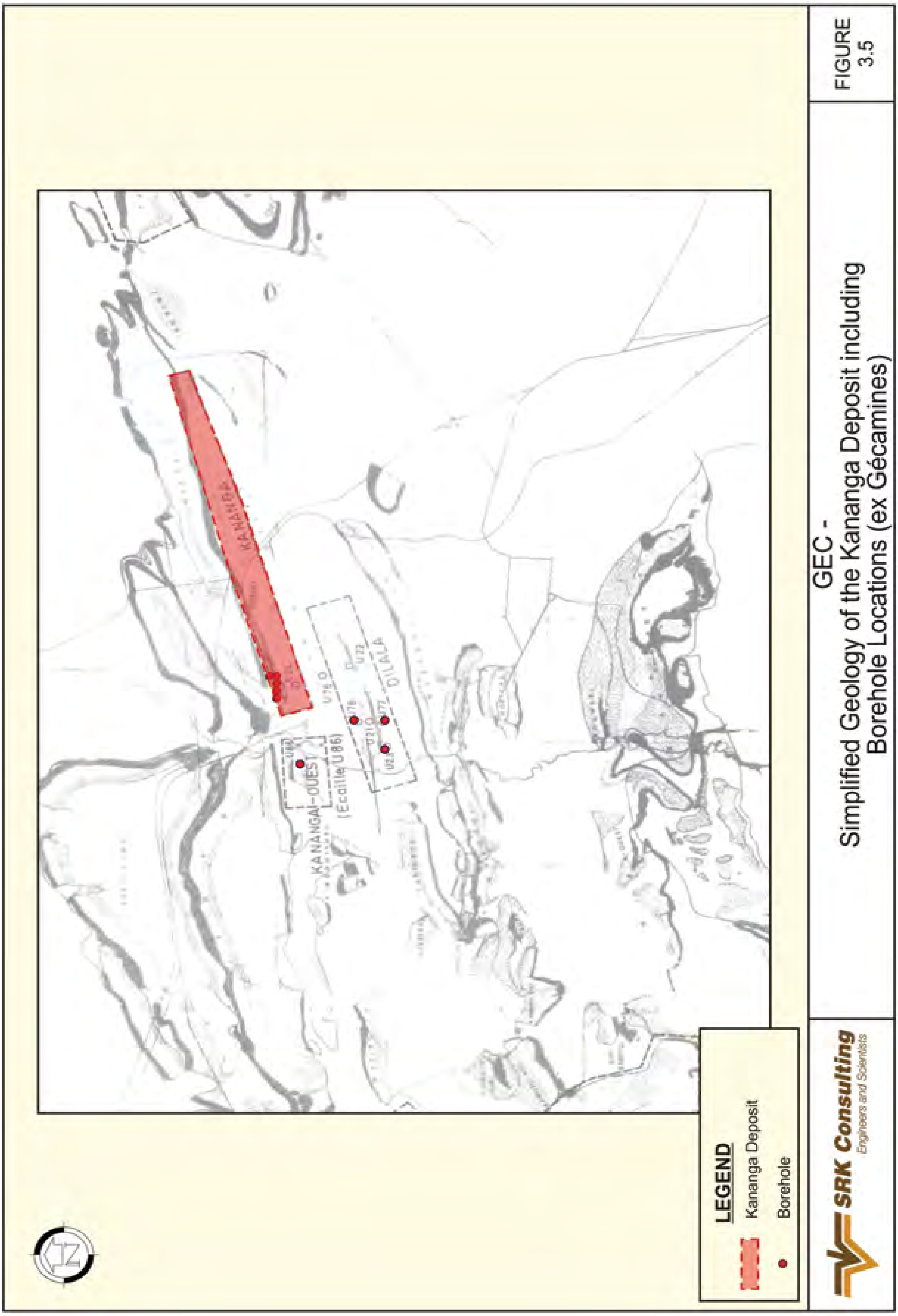
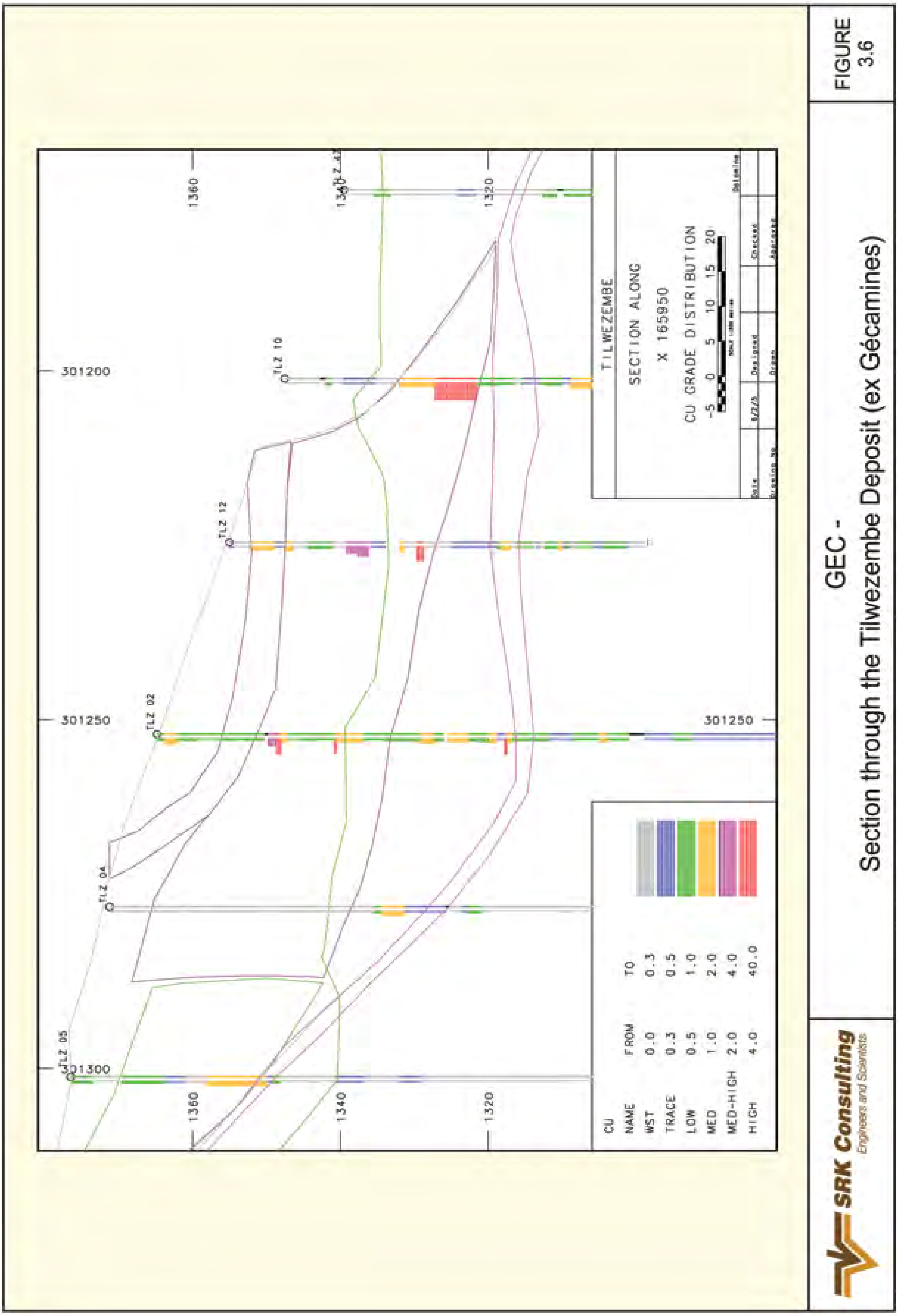


Figure 3.6: GEC—Section through the Tilwezembe deposit (ex Gécamines)



4 MINERAL RESOURCES AND RESERVES

4.1 Introduction

This section summarises the methods used to derive and classify the Mineral Resource and Mineral Reserve estimates for the KOV, Kananga and Tilwezembe properties.

4.1.1 Disclaimer

The underlying geological data for the resource model generated in this report was accepted in good faith and SRK cannot accept any liability either direct or consequential for the validity of this information.

4.2 Site Visits

As part of the pre-feasibility study, SRK undertook a visit to the site over a period of two weeks in early December 2004 primarily to view and familiarise itself with the assets and to collect geological information. Visits were made to the Kamoto East and KOV pits, the Kananga and Tilwezembe operations. Selected cores from each of the deposits were also inspected during the visit.

Since then, SRK has maintained a presence in Kolwezi to conduct the KOV Feasibility Study and as part of the current drilling investigations being conducted by GEC.

4.3 Core Inspection

Selected cores from previous drilling campaigns were inspected at the core shed in Kolwezi. The following observations were made from the core inspection:

- Some of the core boxes are partially rotten;
- Some of the core had been moved around and was no longer in sequence;
- Some of the cores inspected were still intact but there were signs of deteriorations especially for the semi-consolidated cores;
- The wooden depth markers are still in the core boxes but are no longer legible;
- In certain holes, especially from Kananga and Tilwezembe, the depths were marked on paper, most of which had faded although some are still legible;
- For certain sections of the cores, only half cores were available and appeared to have been split by diamond saw;
- Core recoveries in the cores seen from the KOV drillholes were low in the hangingwall lithologies where in some cases, there was no core recovered at all. Within the ore lithologies core recovery varied but was generally much better than in the hangingwall units;
- The cores seen from the drillholes at Kananga and Tilwezembe also displayed low core recoveries;
- The mineralisation in the ore lithologies from KOV consisted mostly of malachite, but other copper minerals seen included bornite, chalcosite, pseudo-malachite and chrysocolla. For cobalt the common minerals included heterogenite, sphaerocolbaltite and carrolite.

4.4 KOV

The information used in the generation of the KOV orebody models consisted of:

- **Surface Topography Plans**—on four 1:1000 scale sheets consisting of contours and spot heights.
- **Geological Cross Sections**—seventeen cross-sections on a 1:2000 scale and at a 100m spacing from X-600 to X-2200 across the deposit. The sections contained projections of drillholes annotated with lithological units, Gécamines geological interpretation and the latest pit surface topography as at December 1989 for the Kamoto Pit and December 1991 for the KOV pit. Also on the sections were tabulations of composite orebody intersections in each drillhole.

- **Drillhole Logs**—copies of drillhole logs for 176 holes pre-fixed KOV and KTO within the KOV pit area. This did not include KTO drillhole logs specific to the Kamoto East orebody which were provided by Gécamines in September 2005. For each drillhole there were several sections in the log, each section was of multiple pages. The main sections were collar page, laboratory report, sample logs and summary of intersections. Down the hole surveys were contained in the collar report. Excluding the geological logs, all the sections were captured in Microsoft Excel worksheets.
- **Bench Plans**—bench plans at 10m intervals from 1330m to 960m elevation and on a 1:1000 scale were also collected for the Kamoto East and KOV pits. There were 47 copies on A0 size, on average 2 plans per bench.

After a thorough review SRK opted for the use of the geological information for the modelling in preference over the bench plans on the basis of:

- The bench plans contain information derived from the interpretations of geological sections;
- There are 47 bench plots to capture information from compared to 17 section plots;
- The basis for the geological interpretations are the drillhole information, which are drilled along the section lines;
- The bench plans contain outlines of ore zones with annotated tonnages and grades of and %total copper grade and acid soluble copper, but the drillhole database acquired from Gécamines has limited acid soluble copper assays and therefore the source for these grades on the bench plans cannot be verified.

4.4.1 Historical Drilling

The drillhole logs indicate that exploration drilling commenced in the early 1940's initially targeting the Kamoto East orebody and initial holes were prefixed KTO. Although the target was to drill into the Kamoto East orebody, a substantial number of KTO holes were later drilled into the present day KOV pit.

Later in the 1980's, another drilling campaign commenced to define the KOV orebodies and this campaign continued into the early 1990's. The drilling was carried out along section lines spaced about 100m apart. Where feasible, drillholes were spaced about 100m along these section lines. The holes drilled in this campaign were prefixed KOV.

Most of the drillholes within the Kamoto East and the KOV pit areas were drilled vertically, with only a few being inclined. Kamoto East drilling was problematic due to the steepness in the dip of the strata resulting in the majority of the holes intersecting the near surface expression of the Kamoto East orebody with only the inclined holes providing intersections at depth.

In general, the majority of the drillhole intersections in Kamoto East were within the areas that have been mined out and there are very few orebody intersections below the current pit bottom.

4.4.2 GEC Drilling Programme

GEC commenced a drilling programme in December 2005 primarily to collect samples for metallurgical testwork and also to confirm the general orebody intersections and grade. The programme is for 8 metallurgical and 15 confirmatory holes and is being managed by SRK.

The objective of the programme was:

- To collect material for metallurgical testwork from orebody intersections;
- To maximise on the core recovery within the orebody intersections;
- To confirm the orebody intersections and tenor of mineralisation and allow for the understanding of the grade distributions within the total orebody package;
- To provide comparative data on the quality of the assays for copper and cobalt values; and
- To provide material for density determinations to be done across the spectrum of lithologies and mineralised zones.

The holes were to provide intersections within each of the orebody zones, Kamoto East, Virgule, Oliveira and FNSR and were sited for a wide coverage of the deposit.

Drill holes into the Kamoto East have, however, been abandoned due to access problems exacerbated by the poor geotechnical conditions in the near surface rocks.

By end May 2006, a total of 7 holes had been completed. The orebody intersections in these holes are comparable to the Gécamines holes.

4.4.3 Drillhole Database

The drillholes have been collected over a period spanning more than 50 years from KOV. The logging has evolved over that period into a Gécamines standardised logging system on which some of the newer drillholes in KOV were captured. The system is a manual recording of the salient features of the core and includes at least 6 separate sections describing the collar information, the contact information, the laboratory report, the sample log, the orebody intersection summary report and the down-the-hole borehole survey for inclined holes. Each section may be more than one A4 sheet depending on the depth of the hole and the amount of information available.

Not all the cores within the orebody lithologies were sampled for analysis. The lithologies sampled were the Upper orebody host rocks, the SDB and the Lower orebody rocks, the RSF, DSTRAT and the RATGR. The basis for the selection of the cores for sampling was the presence of visible copper mineralisation. Core lengths within these lithologies deemed to be barren of copper were not sampled and an entry was made in the sample log for that interval with the comment “steriles” or barren. Significant cobalt mineralisation also occurs within these lithologies and some of the ‘sterile” zones could possibly contain cobalt mineralisation.

The RSC, intermediate between the two Upper and Lower orebodies, does not contain significant copper mineralisation and, as rule, was not sampled by Gécamines except in cases where copper mineralisation was visible. The RSC has been shown to contain cobalt mineralisation within the Katangan Copperbelt.

The stratigraphic sucession from the BOMZ through to the bottom of the RATGR is referred to as the package in this report, and consitities the Upper, Intermediate and Lower ore zones. The true thickness of the combined lithologies varies between 40m and 50m.

As a summary, the down-the-hole survey forms part of the collar sections and the details of the down the hole surveys were not captured. Information from the remaining sections were captured for each hole.

In the sample log, the details recorded include amongst others the sample interval depth, sample number, the core recovery, and any visible mineralisation information captured includes lithology, core recovery, core angles, descriptions of the rock types and mineralisation and alterations. Assay data is added to the log upon receipt of the laboratory results of the analyses of a particular sample.

The drillhole logs were captured in Microsoft Excel worksheets. The information captured from the log sheets for each drillhole included the collar, laboratory report, sample logs and summary of intersections.

4.4.4 Graphical Data Capture

The modelling of the KOV orebodies was undertaken from the geological information presented in the hard copies of 17 sections plots from X-600 to X-2200 at 100m spacing. The geological and structural interpretations on the sections were done by Gécamines and the models have been based on this interpretation.

The sections were scanned and digitised capturing all the relevant geological information, including lithological units, structures, surface and pit topographies and drillholes traces.

A vertical line defining the boundary between Kamoto East and Virgule orebodies is plotted on each of the sections. SRK projected the boundaries from each of the sections onto a plan view to show the orientation of the boundary between the two orebodies as shown on the Gecamines sections (Figure 4.1). The boundary has resulted in the identification of four orebody zones, i.e. Kamoto East, Virgule, Oliveira and FNSR.

4.4.5 Data Analyses

There are a total of 215 drillholes that have orebody intersections in the database of which 64 are KTO holes and 151 are pre-fixed KOV. The distribution of the holes are shown in Figure 4.1. Some of the holes drilled did not intersect the Roan stratigraphy and have therefore been excluded from the drillhole database for resource modelling.

4.4.6 Sampling and Assaying

Cores from the orebody intersections were sampled for chemical analyses. The lengths of core sampled varied and it is SRK's understanding that this was a consequence of the sample recovered within each run. In the Gécamines logging sheet is a column for percentage recovery where values ranging from 1% to 100% are entered to describe the amount of core recovered in the sample length. Core recoveries are only recorded for cores that were sampled.

The core sampled was cut along the longitudinal axis with one half of the core sent for laboratory analyses and the other half retained in the core boxes. The lithologies sampled were the Upper Orebody host rocks (lower SDS and SDB) and the Lower Orebody rocks (RSF, DSTRAT and the RATGR) and portions of the RSC deemed to be mineralised. In some cases the entire RSC was not sampled at all. SRK understands that the visibility of copper mineralisation in the core was used as the criterion for sampling the core. Core lengths deemed to be barren of copper were not sampled and an entry was made in the sample log for that interval with the comment "steriles" or barren. It is possible, in SRK's view, that the unsampled cores could contain finely disseminated copper mineralisation not visible to the naked eye. There is a further possibility, especially in the RSC, for the "sterile" zones to contain cobalt mineralisation. In Figure 4.2, in drillholes KOV 426 and KOV 427, the entire RSC is sampled and there is good grade copper mineralisation (2-3%) within the mid-RSC. In drillhole KOV 428, the mid-portion of the RSC was sampled. The partial or selective sampling, although common in the RSC, was also evident in the other Roan lithologies.

Due to this pre-selection during sampling, the assay database is incomplete and impacts on the accuracy of the estimates generated from such data during the mineral resource estimation process.

The assay database describes the sample in terms of the length of sample, depths (FROM and TO) of intersection and the amount of core recovered in that sample length. There was no systematic approach to sample lengths as indicated by the variations in the sample lengths in the database. The minimum sample taken was 0.5m and the maximum sample was 2.5m. The sample lengths were also a consequence of the sample recovered within the run.

The sample database contains assay data for the following:

- %TCu—the percentage total copper content of the sample;
- %Cuox—the percentage of the copper in the sample due to oxide copper. In the modelling, this is reported as %ASCu. Less than 50% of the samples were analysed for %ASCu;
- %Cu mal—the proportion of the copper due to malachite. Only a few samples contain values on this column;
- %TCo—the percentage total cobalt content of the sample;
- %CaO soluble—the relative proportion of soluble calcium oxide in the sample. Less than 30% of the total database were assayed for CaO.

4.4.7 Summary of Database Issues

SRK has reviewed the database and the following issues with data quality are highlighted:

- Poor core recovery within the orebody varying between 65-75%, the worst recovery being in the RSC lithology;
- The cutting of the total copper grade to 12%, if above 12%. However, not all the samples with total copper grades above 12% in the database have been cut;
- The RSC formation in contact with the SDB and the RSF was sampled, but near the middle of the formation, selectively sampling was on the basis of visible copper mineralisation;

- The orebody zones are sampled for total copper and total cobalt, but assays for acid soluble copper and cobalt and for calcium oxide are limited;
- Samples with zero core recovery, mostly from the earlier drilling in Kamoto East, are shown with high total copper values; presumably from the analyses of the sand collected in the absence of core.

4.4.8 Grade Distribution and Statistics

The grade distribution is discussed with respect to the lithologies within the package and based on the available sampling information. Figure 4.3 is a South-North section looking west showing the structural setting of the Kamoto East, Virgule, FNSR and Oliveira orebody zones and the total copper grade distributions in the drillholes.

Generally and within each of these orebody zones, higher grade copper mineralisation is localised within the SDB and RSF lithologies which constitute the upper and lower orebody formations. The DSTRAT and RATGR also make up the Lower orebody but the grades are slightly lower compared to the RSF. Overall the copper grade distribution pattern within the package going from the top to the bottom of the succession is:

- higher grades at the top(SDB);
- lower grades in the middle(RSC);
- then higher grades in the upper portion of the Lower orebody(RSF); and
- a tail off near the bottom of the orebody contact.

The copper grade distribution within the RSC is misleading as a large proportion of the lithology has not been sampled. The lateral grade distributions in the package in each of the orebody zones, the drillholes were composited over the orebody package, defined from the top of BOMZ to the bottom of the RATGR formation. Unsampled sections were not included in the composites.

Composites were made over the mineralised portions of the package intersected in each drillhole and plotted on plan views to illustrate the lateral copper grade distributions in the package of each of the orebody zones. Section views showing %TCu grade profiles in the various lithological units and orebody types are shown in Figure 4.4 for the Kamoto East and Virgule orebodies and Figure 4.5 for Oliveira and FNSR orebodies. The figures show:

- There is a relatively higher grade (red and magenta) zone in the central Kamoto East area;
- There is a north-west south-east trending high grade zone near the current KOV pit sump surrounded by lower grade intersections in the Virgule orebody;
- There is limited information within the FNSR to conclude on the grade distribution;
- The Oliveira orebody is relatively of moderate grade with a halo of lower grades towards the limits of the defined orebody.

The average sample length in the database is about 2.5m. The drillhole data was composited at 2.5m intervals within the package of lithologies and the statistics from the composite file are reported for the package and within each of the orebodies intersected in Table 4.1. The statistics all show a minimum value of zero. Genuine values of zero total copper grades are recorded in the logs and these have been carried forward. These should be distinguished from the non-sampled samples indicated by absent data entries.

Table 4.1: Statistics by orebody type the Combined Composited KOV Database

Orebody	Variable	Minimum	Maximum	Mean	Standard deviation
FNSR	%TCu	0.29	12.00	**5.70**	3.36
	%TCo	0.01	4.39	**0.42**	0.65
Oliveira	%TCu	0.00	16.59	**4.82**	2.94
	%TCo	0.00	5.72	**0.64**	0.64
Kamoto East	%TCu	0.00	17.30	**5.44**	3.47
	%TCo	0.00	3.05	**0.36**	0.40
Virgule	%TCu	0.00	15.70	**5.01**	3.45
	%TCo	0.00	10.19	**0.40**	0.60

The assay database is incomplete for %ASCu and %CaO as less than 50% of the samples have assays for %ASCu and less than 20% of the samples have assays for %CaO. Geostatistical modelling of these two variables would lead to misleading results. SRK have excluded these two variables in the grade estimation and resource reporting processes downstream.

The %ASCu and %CaO grade values have been used to provide for a feel of the distribution of oxide-sulphide and dolomitic-siliceous ore types. For the oxide-sulphide, the ore is considered oxide when the proportion of %ASCu to the %TCu is higher, and sulphide with low %ASCu content relative to the %TCu. As an exercise, SRK have used a ratio of 0.5 as the division between the two ore types: when the %ASCu to %TCu ratio is less than 0.5, the ore is considered sulphide and when the ratio is higher than 0.5 the ore is considered oxide.

Gécamines established a relationship between the %CaO and the %TCu to determine whether the ore is dolomitic or siliceous. The threshold values for the categorisation are:

Siliceous ores	%TCu / %CaO>15
Dolomitic ores	%TCu / %CaO<=15

However, the determinations of the ore type could not be duplicated from the drillhole database due to limited %CaO assays in the database: less than 30% of the samples intersecting the orebody were assayed for %CaO compared to %TCu.

4.4.9 Density Determinations

For the conversion of volumes into tonnes, Gécamines assigned density values based on an empirical criterion obtained from standardised values generated from the exhaustive data available from all Gécamines operations within the Katangan Copperbelt. The criterion was based on the categorisation of the ore type into either dolomitic or siliceous based on the relationship between %TCu and %CaO described above.

Generally the oxide ores have been assigned values of 2.2t/m^3, the mixed ores 2.4t/m^3 and the sulphide ores 2.6t/m^3.

In the models generated for KOV, SRK has used a density of 2.2t/m^3 based on the inference that the majority of the ore is oxide. This inference is based on the visual inspection of the mineralisation in the cores and the observations in the field, where the predominant copper mineral is malachite. Orebody intersections from the current drilling also seem to confirm that.

From the current GEC drilling, SRK have undertaken density determinations on selected cores within the orebody intersections of the completed drillholes. The determinations were done on site by Archimedes Principle, which compares the weight of a sample in air to the volume of water displaced when the same sample is immersed in water. A total of 67 determinations were done and the average density value for the package realised was 2.1t/m^3. The coefficient of variation, which is the ratio of the standard deviation over the average value, varies within a 10% band which indicates strong consistency in the values.

The results of these determinations are consistent with the values from Gécamines and confirm the density of the ore lithologies in the model.

4.4.10 Geological Modelling

For modelling purposes, the lithological units within each of the fragments were modelled as separate entities and a bounding string was used to indicate the limit of the each fragment.

Each of the digitised sections were imported into Datamine and modelled by lithological units as 3-dimensional wireframe surfaces. For each lithological unit, a 3-dimensional block model was generated from the adjacent surfaces. Parent blocks of dimensions 100m x 100m x 10m sub-celled to 25m x 25m x 10m in the X, Y and Z directions were filled above each of the generated surfaces.

A lithological block model was generated by superimposing the two stratigraphically successive lithologies models and selecting the blocks lying in between the surfaces for the particular lithology.

A similar approach was undertaken until a geology model for each fragment was generated. These individual fragment models were then added together in a particular sequence that ensured that the contact relationships as observed in the sections were maintained.

The surface topography was generated from a combination of the digitised contours and spot heights and the pit profile digitised from the section plots. Whereas every effort was made to reproduce the spot height elevations as accurately as possible, the quality of the copies of the spot height plans was very poor.

The surface topography was draped over the generated geological model and all the blocks lying above the surface were removed from the block model.

4.4.11 Grade Estimation

Grades were estimated into the block model using geostatistical techniques.

Omni directional variograms were computed and modelled for the package and within each orebody zone for %TCu and %TCo. The variograms modelled indicated that samples were correlatable over distances of 300m to 600m for %TCu and 200m to 300m for %TCo.

The variograms were not well structured with the nugget effect constituting about 50% to 90% of the total variance.

One of the main factors contributing to the structure of the variograms is the incomplete or selective sampling of the lithologies resulting in an incomplete package in places and the use of such data. Other issues affecting the quality of the variograms were inconsistent cutting of high copper grades and the quality and representativeness of the assays themselves.

Samples used in the grade estimation were sourced within a search ellipsoid of dimensions equivalent to about twice the variogram range. A minimum of 5 and a maximum of 20 samples were used to estimate a block.

4.4.12 Estimation and Classification of Mineral Resources

Grades were estimated into each of the orebody zones using the respective composites data files and the omni-variogram models for each zone. Blocks above the surface topography were then removed from the model and the inventory of mineral resources computed.

The terms and definitions used to present the statement of mineral resources are those given in the SAMREC Code. The bases for the classification include:

- The quantity and quality of the data used in the generation of the mineral resources;
- The unavailability of assays in portions of the package due to selective sampling on the basis of visible copper mineralisation;
- Poor core recovery within the orebody, varying between 65-75%,
- The inconsistent cutting of the copper grades- the cutting of the total copper grade if above 12% to 12%. However, not all the samples with total copper grades above 12% in the database have been cut;
- The relatively incomplete assays for %ASCu and %CaO compared to the %TCu data;
- The unavailability of density data for any of the lithologies within each of the deposits investigated;
- That there was a history of mining within each of the pits;
- The unavailability of historical reconciliation production records.

The mineral resources have been classified into the Indicated Mineral Resource category and are shown in Table 4.2.

Table 4.2: Updated Indicated Mineral Resources for the KOV Project, dated April 2006

Orebody	Mt	%TCu	%TCo
Kamoto	34	5.39	0.33
Virgule	64	5.09	0.41
Kamoto + Virgule	**98**	**5.19**	**0.38**
Oliveira	57	4.73	0.68
FNSR	17	5.65	0.44
Total Indicated	**172**	**5.09**	**0.49**

The total inventory is 172Mt at average total copper and total cobalt grades of 5.09% and 0.49% respectively.

Gécamines estimates for the Kamoto East, Virgule and Oliveira orebodies of 151Mt @ 5.21%TCu and 0.44%TCo, is comparable to SRK's estimates of 155Mt @ 5.02%TCu and 0.49% TCo for the same orebodies. The comparative data excludes the estimates for FNSR. The FNSR orebody within the KOV pit area is a remnant from the mining in the Musonoi Pit to the east of the KOV pit and was therefore not included in Gécamines' KOV pit estimates.

4.5 Results of the current drilling

The seven holes in the completed GEC drilling programme intersected orebody lithologies of similar tenor and thickness to the holes drilled by Gecamines. Of particular importance was that the holes were sampled throughout the sucession of the package. The sampling and assying of the RSC lithology indicated relatively lower copper grades with good cobalt values. Overall, the grade distribution in the RSC is consistent with the general observations from the Gecamines drilling: good copper values near the contacts and low copper values in the middle of the lithology. Figure 4.6 is a section through the Oliveira orebody showing the copper grade distribution of the new hole VRDG14 compared with the existing holes.

4.6 Kananga

4.6.1 Information Collected

Copies of borehole logs, plans, sections and reports were collected from the Geology offices in Likasi for the three orebodies. The list of information collected and used in the geological modelling and resource estimation process is detailed below by orebody.

The following sets of information were received for the Kananga orebody:

- 1;1000 scale plans T1490 and T1500 with contours of surface topography;
- Seven sections showing drillhole trace and geological interpretations of the orebody lithologies at 1:500 scale. Each section contains one drillhole;
- 1:500 scaled bench plans at 10m intervals from 1320 to 1410m elevation;
- Collar information and hole orientation available for 10 holes, KNG01 to KNG11 (excluding KNG08);

Except for the collar files, no borehole logs were provided for Kananga. On the geological sections provided the trace of the borehole down the section was plotted, annotated with the average intersected grade with each rock unit. In the absence of the actual sample logs, a composite drillhole log was generated by measuring along the trace of the drillhole, the depths and length of sample, and assigning the annotated composite grades for %TCu and %TCo to the sample length.

Assay information generated from the annotated section plots was only available for 6 drillholes.

4.6.2 Data Capture

The graphical data on sections was used to generate 3-dimensional models of each of the three orebodies within the project.

The orebody has been defined over a 300m strike length based on the seven drillholes with assay data spaced on average about 50m apart supplemented by surface trenching information. The orebody is sub-vertical and dips steeply towards the south.

Except for one, the drillholes intersecting the Kananga orebody were drilled southwards at various inclinations and based on the geological interpretations, the drillholes are sub-parallel to the dip of the strata and can be interpreted to be sampling the same unit down the hole. One hole KNG 05 was drilled northwards and cuts across the stratigraphy providing the only complete orebody intersection within the Kananga orebody. The drillhole coverage for Kananga is shown in Figure 4.7, while Figure 4.8 is a section through the Kananga orebody showing the orientation of the drillholes with respect to the orebody structure.

At each section line position, the lithological units that host the Upper (SDB), the "barren" intermediate zone (RSC) and Lower orebody (RSF, DSTRAT and RATGR) have been defined as a closed envelope graphically. Above the SDB is the SDS unit and below the RATGR is the RAT unit, considered the hangingwall and footwall waste rock units respectively.

Each of the envelopes generated for each of the rock types were linked across the length of the defined orebody area to form a wireframe defining a volume for that rock type. The edges of the wireframes terminate at the drillhole positions KNG 01 in the west and KNG 04 in the east.

4.6.3 Block Modelling

A block model was generated for each of the rock types within the Kananga orebody by filling blocks of dimensions 5m x 5m x 5 m in X, Y and Z directions inside the respective wireframes. The limit of the block model in the Z direction is 1200masl, up to the extent of the extrapolated geological sections.

The individual rock block models were added together to generate an ore body model for Kananga. Blocks of waste were added outside of the ore models and blocks above the surface topography were removed from the final geological model.

4.6.4 Grade Estimation

As stated above, the database for Kananga was generated from the composite drillhole assay data annotated on the sections. There are a total of 55 composites with grade value intervals gleaned from the sections plots. Table 4.3 presents the statistical averages of the thickness and grade values within each rock unit against the number of samples. The average elevation column has been added to show the centre of gravity of the data with respect to the general surface topography of 1393masl in the orebody area. The majority of the composites are occurring below 50m of the surface.

Table 4.3: Statistics by rock type from the gleaned composite data, Kananga

		Rock Type					
Grade	**Variable**	**RAT GR**	**DSTRAT**	**RSF**	**RSC**	**SDB**	**BRECHE**
	True Width, m	5	4	4	20	10	
	Avg. elev, mamsl	1371	1337	1350	1285	1282	1322
	No. records	10	9	9	13	11	3
%TCu	Minimum	0.60	1.21	1.03	0.40	0.70	0.90
	Maximum	6.84	13.82	4.42	3.25	2.00	1.58
	Mean	**2.58**	**4.39**	**2.83**	**1.82**	**1.39**	**1.14**
	Stand dev	2.01	3.96	1.20	0.84	0.34	0.31
%TCo	Minimum	0.09	0.10	0.23	0.10	0.20	0.50
	Maximum	2.90	1.44	2.07	4.55	3.09	2.56
	Mean	**1.21**	**0.66**	**1.13**	**1.33**	**1.34**	**1.40**
	Stand dev	0.81	0.47	0.58	1.46	1.02	0.86

The %TCu and %TCo grades were estimated into the block model using inverse distance squared ("IDS"). The IDS methodology was selected due to the paucity of data available for the estimation within each rock unit. Each rock block model was estimated separately using the samples of the respective rock type. To ensure that all the blocks in the model were estimated, the minimum number of samples for estimation was set at one and all the samples were sourced and used from the expanded search neighbourhood covering the entire model extent.

Mineral resource estimates are reported in Table 4.4 and have been classified as Inferred by SRK.

Table 4.4: Inferred Mineral Resource estimates for Kananga

Rock Type	Mt	%TCu	%TCo
RATGR	0.6	2.63	1.19
DSTRAT	0.6	4.39	0.70
RSF	0.5	2.67	1.15
RSC	2.9	1.90	1.59
SDB	2.3	1.39	1.12
Total	**6.9**	**2.07**	**1.29**

4.7 Tilwezembe

4.7.1 Information Collected

The following sets of information were received for the Tilwezembe Pit and Tilwezembe East orebodies:

- 1:500 scale plan of contours and spot heights showing the Tilwezembe pit topography as at August 2004;
- 1:1000 surface contour plan for Tilwezembe East
- 1:250 scale geological sections annotated with drillhole composite intersections spanning the Tilwezembe pit and East orebodies at intervals of 25m across, except for a zone of about 200m between the pit and the east orebody;
- 1:250 scale bench plans at 5m intervals for the east orebody and 1:500 scale bench plans within the pit area
- Borehole logs for Tilwezembe pit containing the collar and assay files. All the holes were drilled vertical;
- collar data for the Tilwezembe East drillhole data.

Sample data was not available for Tilwezembe East. Instead the drillhole database was generated by measuring the depths along the profile of the drillhole on the section plots and gleaning the composite grade values annotated alongside the trace of the drillhole (copies of actual assay data were made available much later).

Lithological information was unavailable in the borehole logs for the Tilwezembe data.

At Tilwezembe the modelling was split into Tilwezembe Pit and Tilwezembe East. Tilwezembe Pit is the area of current mining activity which has been drilled on a spacing of 25m and Tilwezembe East forms the eastern extension of the pit orebody and has limited exploration drilling restricted to two parallel fences running along the crest of the ridge. Figure 4.9 shows the drillhole data coverage and the surface topography in the Tilwezembe Pit and Tilwezembe East.

4.7.2 Graphical Data Capture

Hard copies of geological sections covering the Tilwezembe Pit and Tilwezembe East orebodies were scanned and digitised capturing Gécamines' interpretations of the geology and structure within the deposits. A total of 20 geological east-west sections at 25m spacing were captured with 10 sections each covering Tilwezembe pit from X-16580m to X-166125m and Tilwezembe East from X-16627m to X-166425m. Figure 4.10 is a section through the Tilwezembe East deposit.

Seven lithologies were identified and modelled in Tilwezembe Pit. The lithologies as defined by Gécamines consist mostly of variations of dolomites.

4.7.3 Data Analyses

Examination of the geological and the orebody interpretations on the sections revealed that there was no apparent lithological control on the mineralisation. This was tested by the statistical analyses of the data falling within each of the lithologies. Seven lithologies were identified and modelled in the Tilwezembe Pit. Drillhole data falling within each of these lithologies were extracted and studied to try and establish a relationship between each of the lithologies and the copper and cobalt grades. The assay data displayed a strong variance within each lithology suggesting that there was no definite lithological control of the mineralisation.

All the drillhole data above the breccia horizon were combined and composited at 5m intervals. Statistics from the composite data are indicated in Table 4.5.

Table 4.5: Statistics from the composited data—Tilwezembe Pit

Category	%TCu	%TCo	%Mn
No. Samples	617	617	594
Minimum	0.06	0.04	0.00
Maximum	35.00	13.92	11.60
Mean	**1.17**	**0.98**	**1.14**
Standard deviation	2.51	1.39	1.31

Omni directional variograms were computed for each of the grade variables in Tilwezembe pit and modelled with ranges of just about 65m, 45m and 46m for the %TCu, %TCo and %Mn variables. This indicates that there is no spatial continuity of grades beyond 50m away from any two drillhole spacing. Attempts were made to compute directional variograms at varying spread angles to provide an understanding of any preferred directional or spatial continuity, but the structures were not well developed for modelling.

4.7.4 Geological Modelling

The Tilwezembe orebody is confined within a breccia and hosted by a series of dolomites and shales. Seven lithologies have been defined and modelled in Tilwezembe Pit and 10 in Tilwezembe East. These lithologies consist of four varieties of strongly brecciated dolomites and shales in Tilwezembe Pit and six varieties in Tilwezembe East, hosting the copper and cobalt mineralisation. These lithologies are discontinuous and contained in strongly brecciated country rock.

For modelling purposes and where a particular lithology is not continuous across sections, the extent of any lithology was limited to half the distance beyond the last section line.

A background model of the breccia and country rock was generated. The individual lithological models were then superimposed onto this model to define the extent of the ore hosting lithologies.

4.7.5 Grade Estimation

In view of the apparent lack of relationship between the grade and lithological units, grades were estimated into a combined block model of all the ore hosting lithologies for %TCu, %TCo and %Mn using the respective composite samples and variograms parameters. Blocks were estimated using a minimum of 5 and a maximum of 20 samples, within a search neighbourhood equivalent to two times the variograms range.

4.7.6 Tilwezembe East Grade Estimation

At the time of the modelling work, detailed drillhole data was unavailable for Tilwezembe East. Instead, like at Kananga, a drillhole database was created by measuring the depths and gleaning the data annotated on the section plots. Data for %Mn was not annotated on the section plots.

Variograms parameters modelled for Tilwezembe Pit were utilised in the estimation for Tilwezembe East. Grades were estimated into a combined orebody model consisting of six dolomitic lithologies using the composite data gleaned from the section plots. A minimum of 5 and a maximum of 20 samples were sourced from a search neighbourhood of three times the range of variograms or each respective grade variable.

4.7.7 Computation and Classification of Mineral Resources

A surface topography dated August 2004 was draped over the block model and the mined portion was removed from the resource model for Tilwezembe Pit. During the site visit in December 2004, the Tilwezembe Pit was being mined by a contractor and the pit was only a few benches above the modelled orebody bottom. SRK visited the site again in April 2006 and found that mining operations had ceased and the contractor had removed all his equipment. SRK is of the opinion that the orebody has been mined out in Tilwezembe Pit and has therefore excluded it from the inventory of mineral resources.

The Mineral Resources in Tilwezembe East as at April 2006 are shown in Table 4.6 and have been classified as Indicated by SRK.

Table 4.6: Indicated Mineral Resources for Tilwezembe as at April 2006[1]

Deposit	Mt	%TCu	%TCo	%Mn
Tilwezembe East	5.7	1.49	1.04	
Total	**5.7**	**1.49**	**1.04**	

1 The surface topography has not been surveyed since August 2004.

4.8 Summary of Mineral Resources

The terms and definitions used to present the statement of mineral resources are those given in the SAMREC Code. The Mineral Resources as presented in Table 4.7 are inclusive of the declared Mineral Reserves.

4.8.1 Mineral Reserves

The Mineral Reserves stated as a recoverable Mineral Reserve Estimate (in compliance with the Guidance Note for Mining, Oil and Gas Companies—March 2006 issued by AIM) to which grade adjustment factors, extraction factors, and processing plant recovery factors have to be applied.

The following modifying factors have been applied:

- Grade adjustment factor 95%
- Extraction Ratio 95%
- Processing Plant Recovery Factors 91% Cu and 85% Co

Table 4.7: GEC—Summary of Mineral Reserves and Mineral Resources as at April 2006

Category	Gross Ore	Gross Grade (%)		Gross Contained Metal (kt)		Attributable Ore	Attributable Grade (%)		Attributable Contained Metal (kt)		Operator
	(Mt)	Cu	Co	Cu	Co	(Mt)	Cu	Co	Cu	Co	
Mineral Reserves:											
Probable											
KOV	140	4.83	0.47	6,757	658	105	4.83	0.47	5,068	493	DCP
Total Reserves	**140**	**4.83**	**0.47**	**6,757**	**658**	**105**	**4.83**	**0.47**	**5,068**	**493**	DCP
Mineral Resources:											
Indicated											
KOV	172	5.09	0.49	8,755	843	129	5.09	0.49	6,566	632	DCP
Tilwezembe	6	1.49	1.04	89	62	5	1.48	1.03	72	50	DCP
Total Measured+Indicated Resources	178	4.97	0.51	8,844	905	134	4.96	0.51	6,638	682	DCP
Inferred											
Kananga	7	2.07	1.29	143	89	5	2.07	1.29	107	67	DCP
Total Inferred Resources	7	2.07	1.29	143	89	5	2.07	1.29	107	67	DCP
Total Resources	**185**	**4.85**	**0.54**	**8,994**	**999**	**139**	**4.85**	**0.54**	**6,745**	**749**	**DCP**

4.9 SRK Comments

4.9.1 KOV

The terms and definitions used to present the statement of mineral resources are those given in the SAMREC Code. The bases for the classification include:

- The quantity and quality of the data used in the generation of the mineral resources;
- The unavailability of assays in portions of the package due to selective sampling on the basis of visible copper mineralisation;
- Poor core recovery within the orebody, varying between 65-75%,
- The inconsistent cutting of the copper grades- the cutting of the total copper grade if above 12% to 12%. However, not all the samples with total copper grades above 12% in the database have been cut;
- The relatively incomplete assays for %ASCu and %CaO compared to the %TCu data;

- The unavailability of density data for any of the lithologies within each of the deposits investigated;
- The unavailability of historical reconciliation production records.

The mineral resources have been classified into the Indicated Mineral Resource category. The total inventory is 172Mt at average total copper and total cobalt grades of 5.09% and 0.49% respectively.

The seven holes in the completed GEC drilling programme intersected orebody lithologies of similar tenor and thickness to the holes drilled by Gecamines. Of particular importance was that the holes were sampled throughout the succession of the package. The sampling and assying of the RSC lithology indicated relatively lower copper grades with good cobalt values. Overall, the grade distribution in the RSC is consistent with the general observations from the Gecamines drilling.

4.9.2 Kananga

The Cu and Co grades were estimated into the block model using inverse distance squared ("IDS"). The IDS methodology was selected due to the paucity of data available for the estimation within each rock unit. Each rock block model was estimated separately using the samples of the respective rock type. To ensure that all the blocks in the model were estimated, the minimum number of samples for estimation was set at one and all the samples were sourced and used from the expanded search neighbourhood covering the entire model extent.

Mineral resource estimates have been classified as Inferred by SRK and amount to 6.9Mt at an average grade of 2.07%Cu and 1.29%Co.

4.9.3 Tilwezembe

At the time of the modelling work, detailed drillhole data was unavailable for Tilwezembe East. Instead, like at Kananga, a drillhole database was created by measuring the depths and gleaning the data annotated on the section plots. Data for %Mn was not annotated on the section plots. Variograms parameters modelled for Tilwezembe Pit were utilised in the estimation for Tilwezembe East. Grades were estimated into a combined orebody model consisting of six dolomitic lithologies using the composite data gleaned from the section plots.

The remaining Mineral Resources in Tilwezembe East have been classified as Indicated by SRK and amount to 5.7Mt at an average grade of 1.49%TCu and 1.04%TCo.

Figure 4.1: GEC—Drillhole plan over KOV and Kamoto East

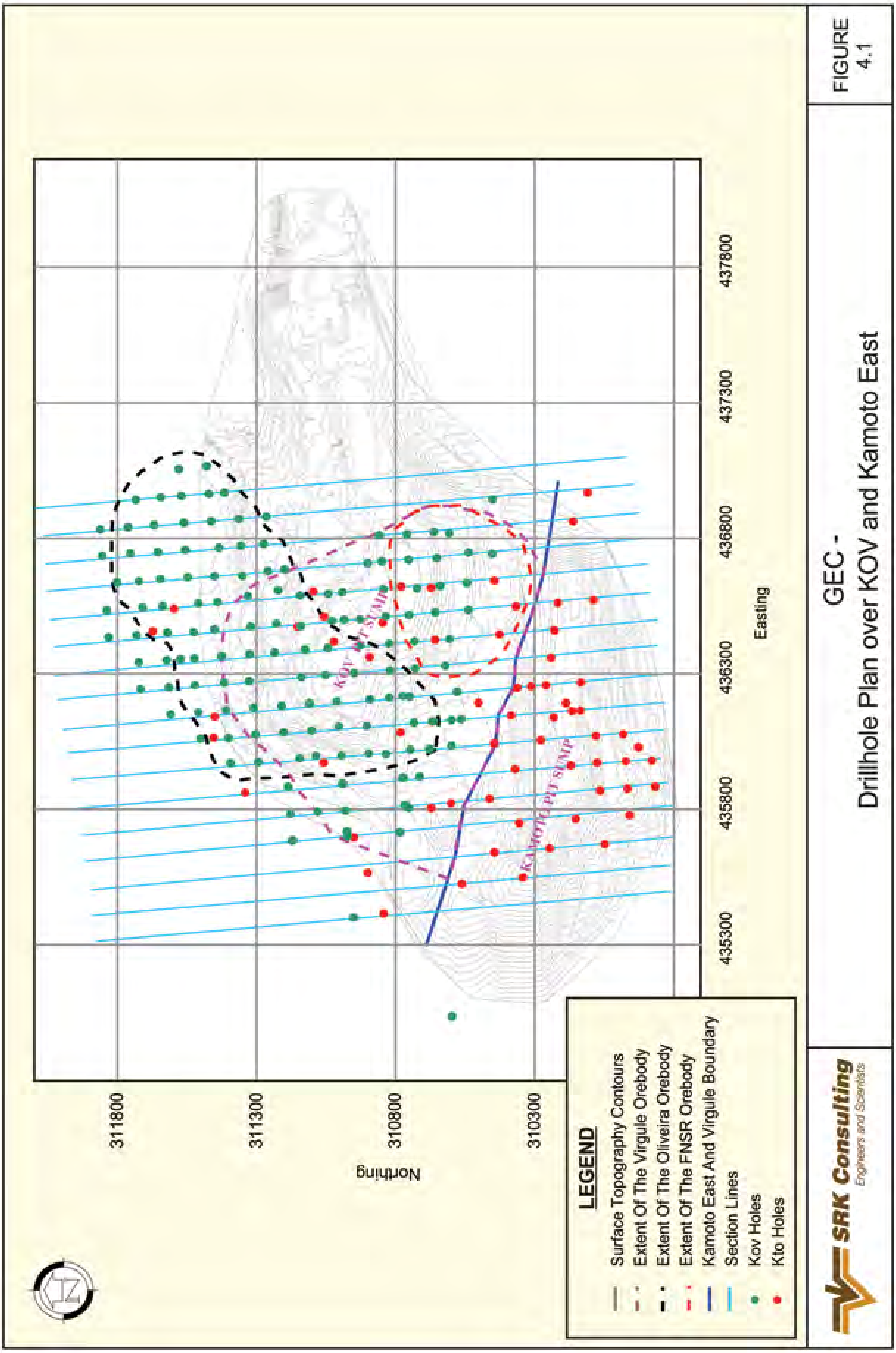


Figure 4.2: GEC—Section showing orebody lithologies and drillhole copper grade distributions

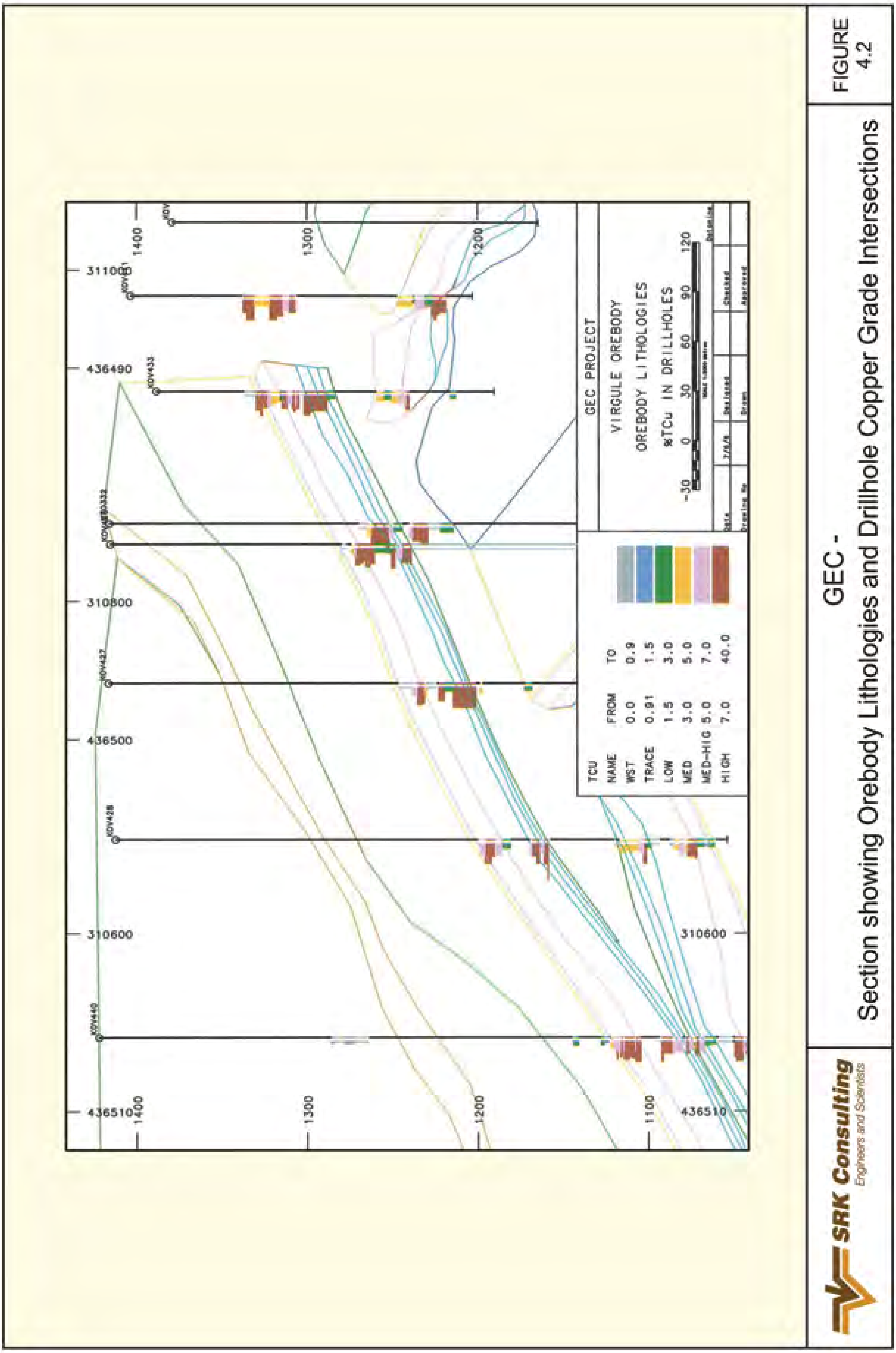


Figure 4.3: GEC—Section showing structural setting of orebodies and drillhole copper grade intersections

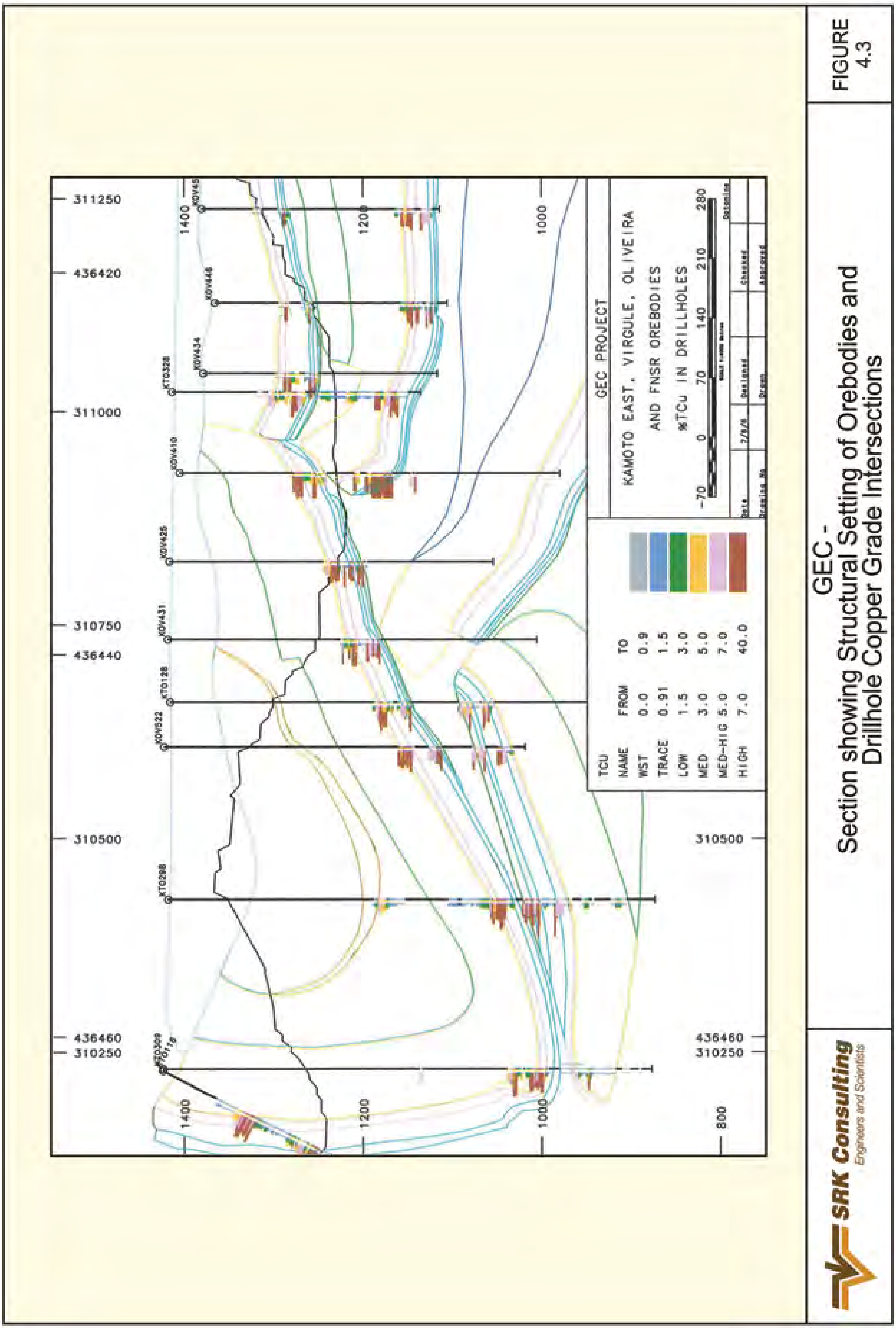


Figure 4.4: GEC—Examples of %TCu grade distribution in Kamoto Virgule

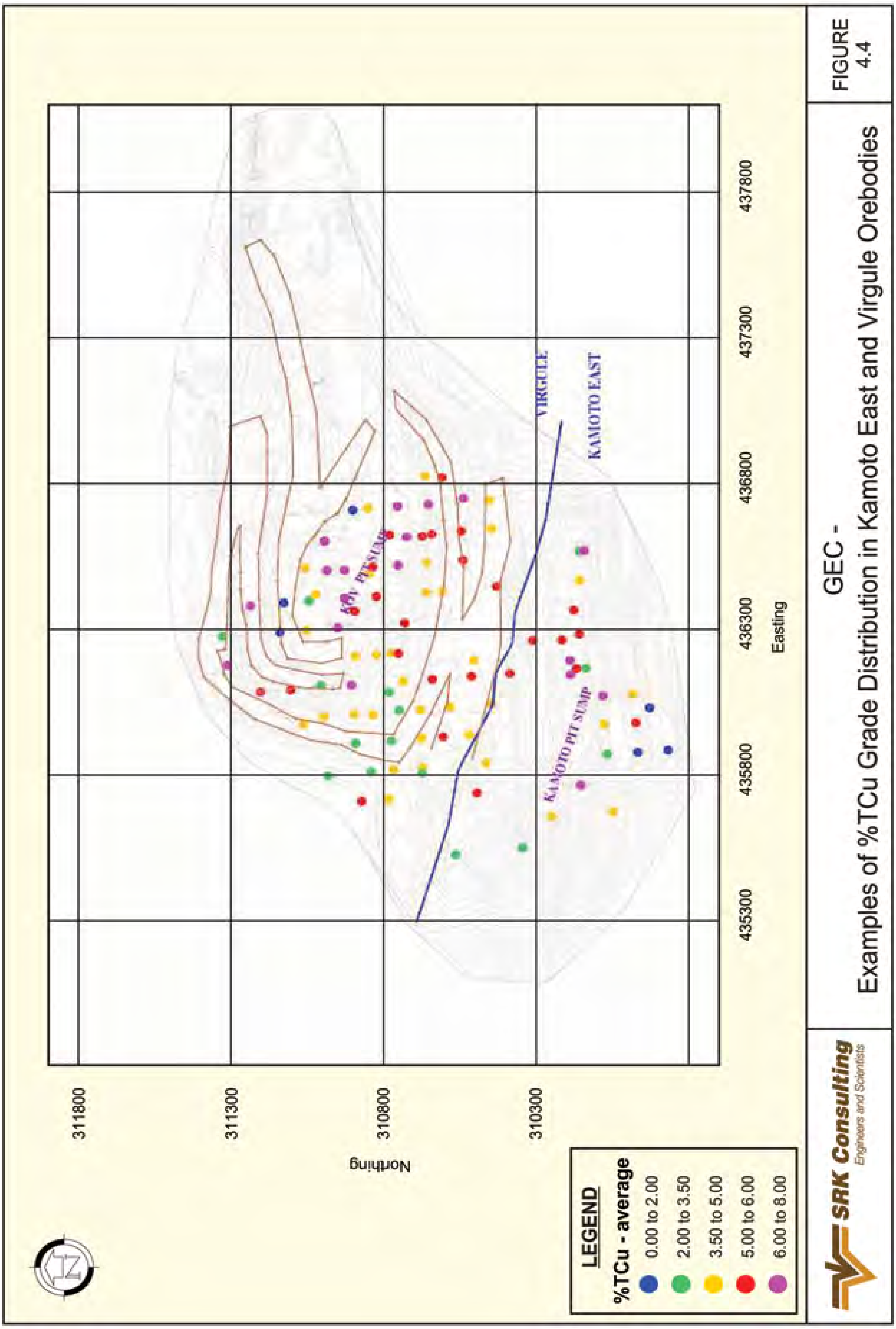


Figure 4.5: GEC—Examples of %TCu grade distribution in Oliveira and FNSR

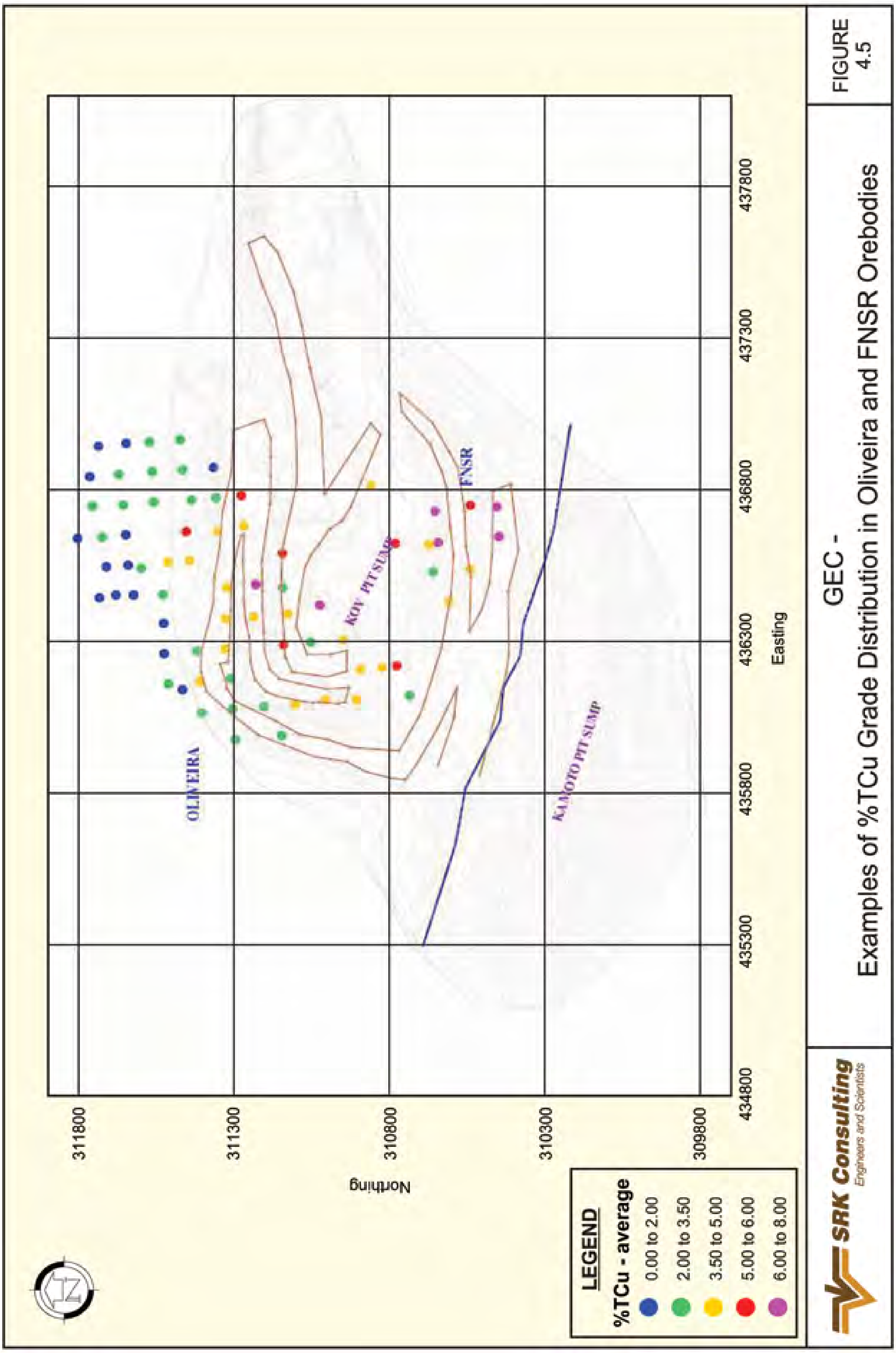


Figure 4.6: GEC—Comparative drillhole logs for Gecamines vs GEC

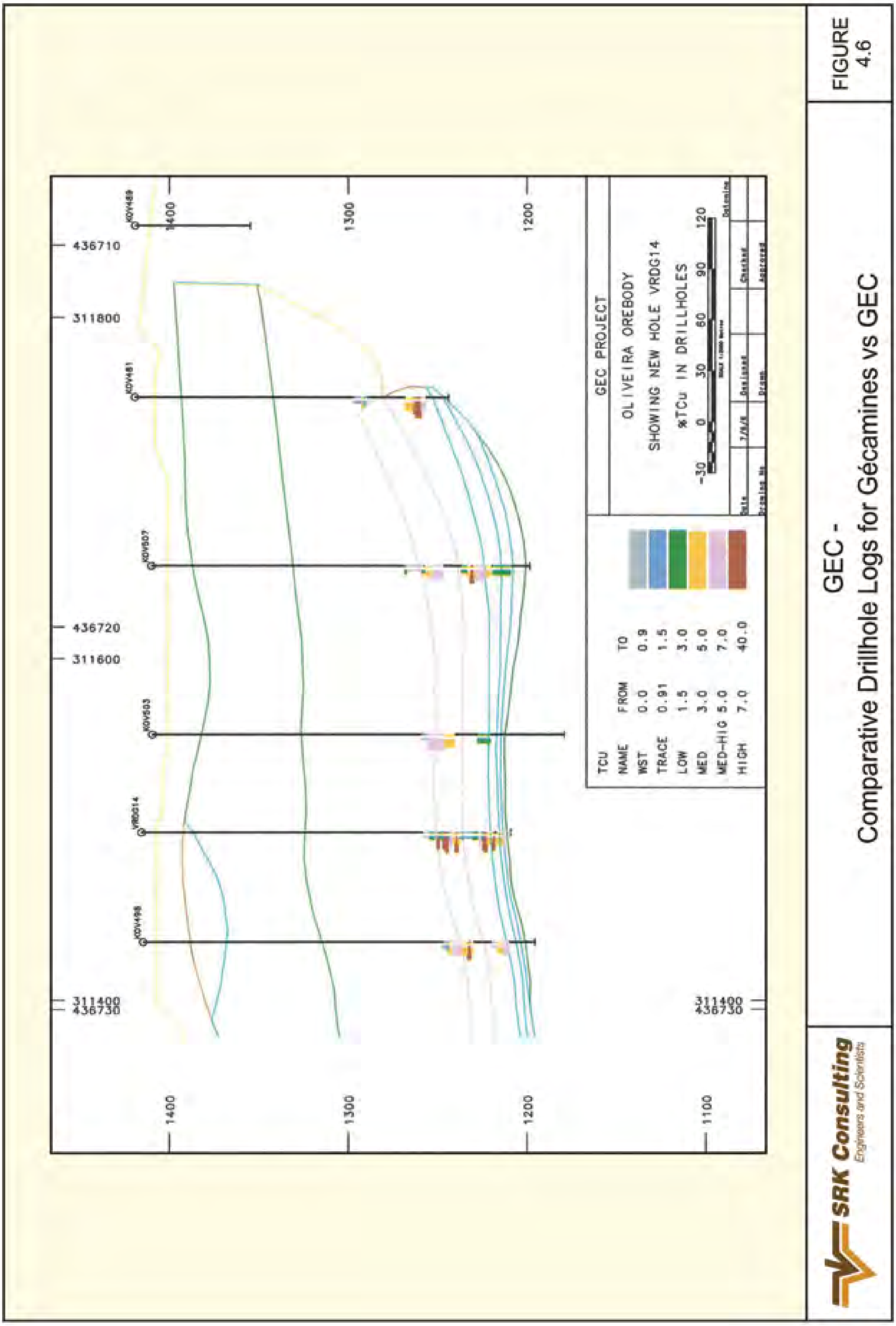


Figure 4.7: GEC—Kananga drillhole coverage

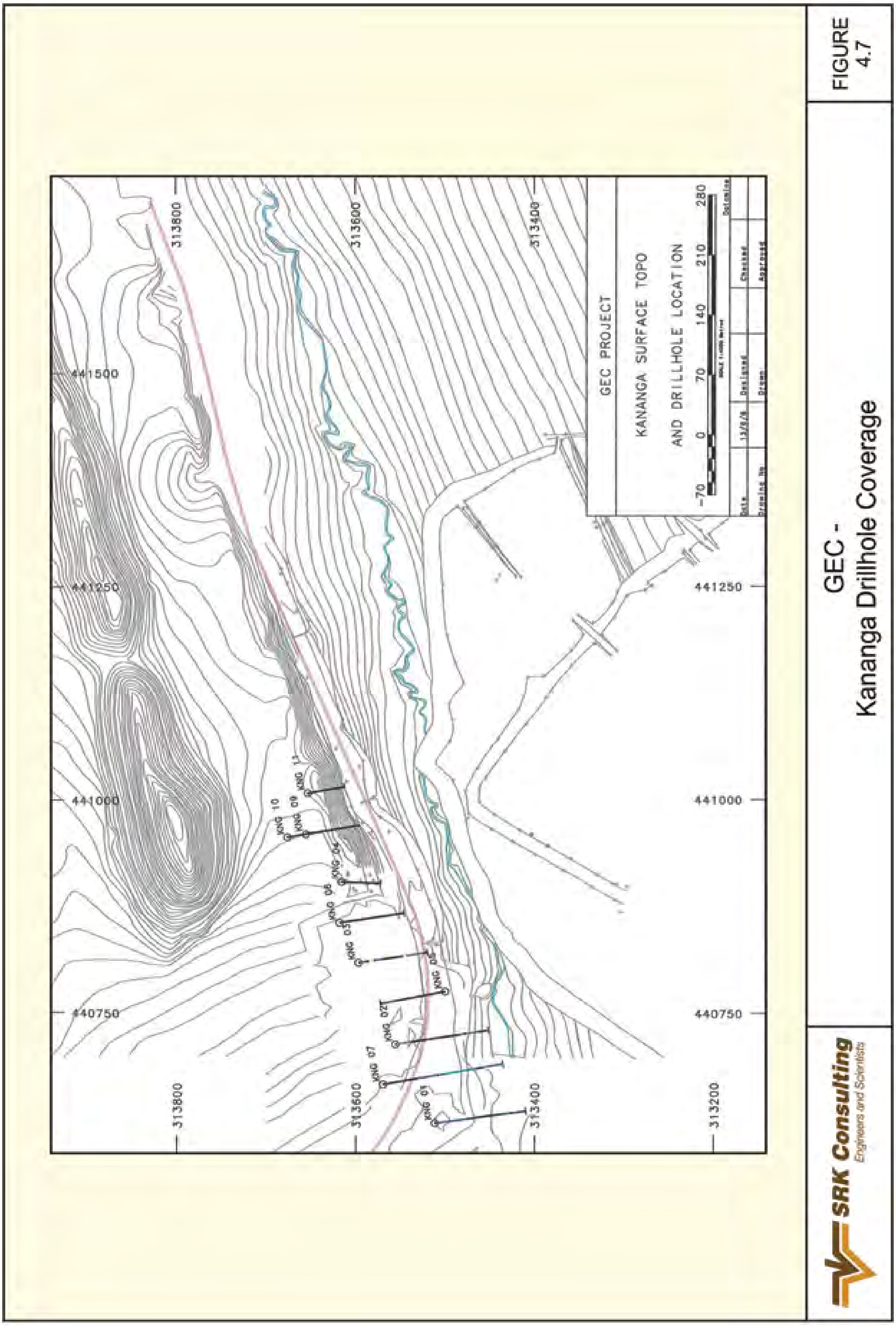


Figure 4.8: GEC—Typical section through Kananga orebody

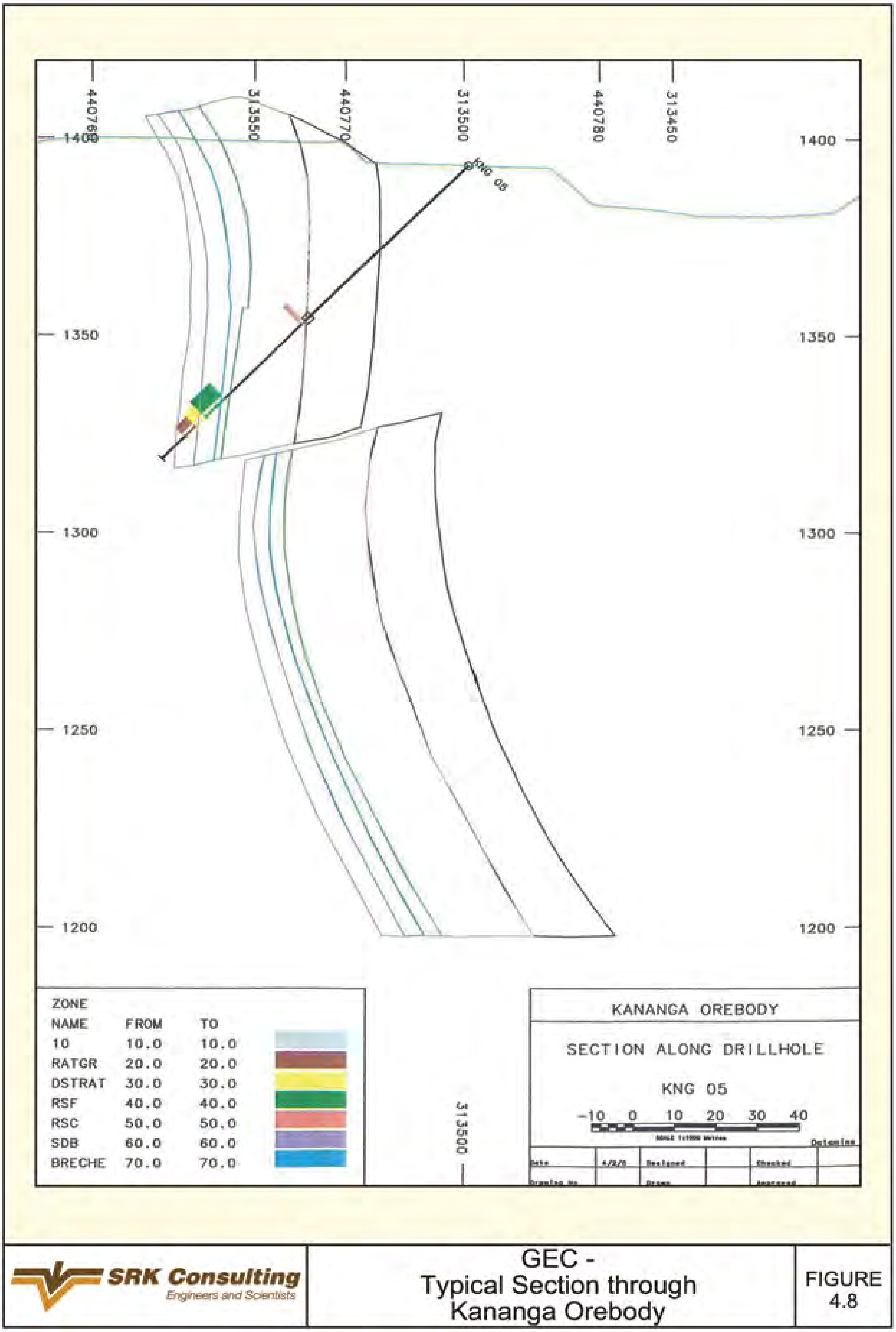


Figure 4.9: GEC—Tilwezembe drillhole coverage

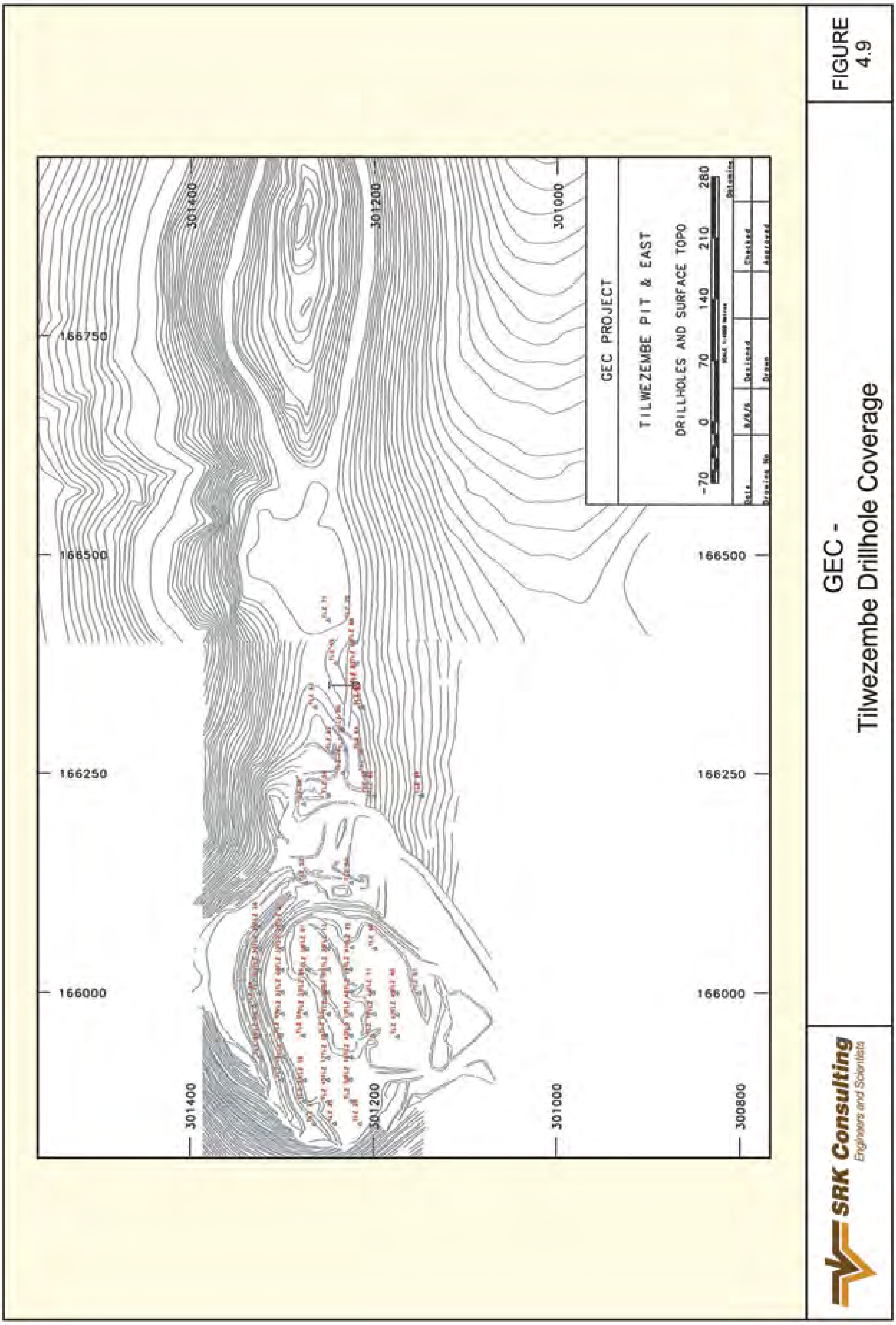
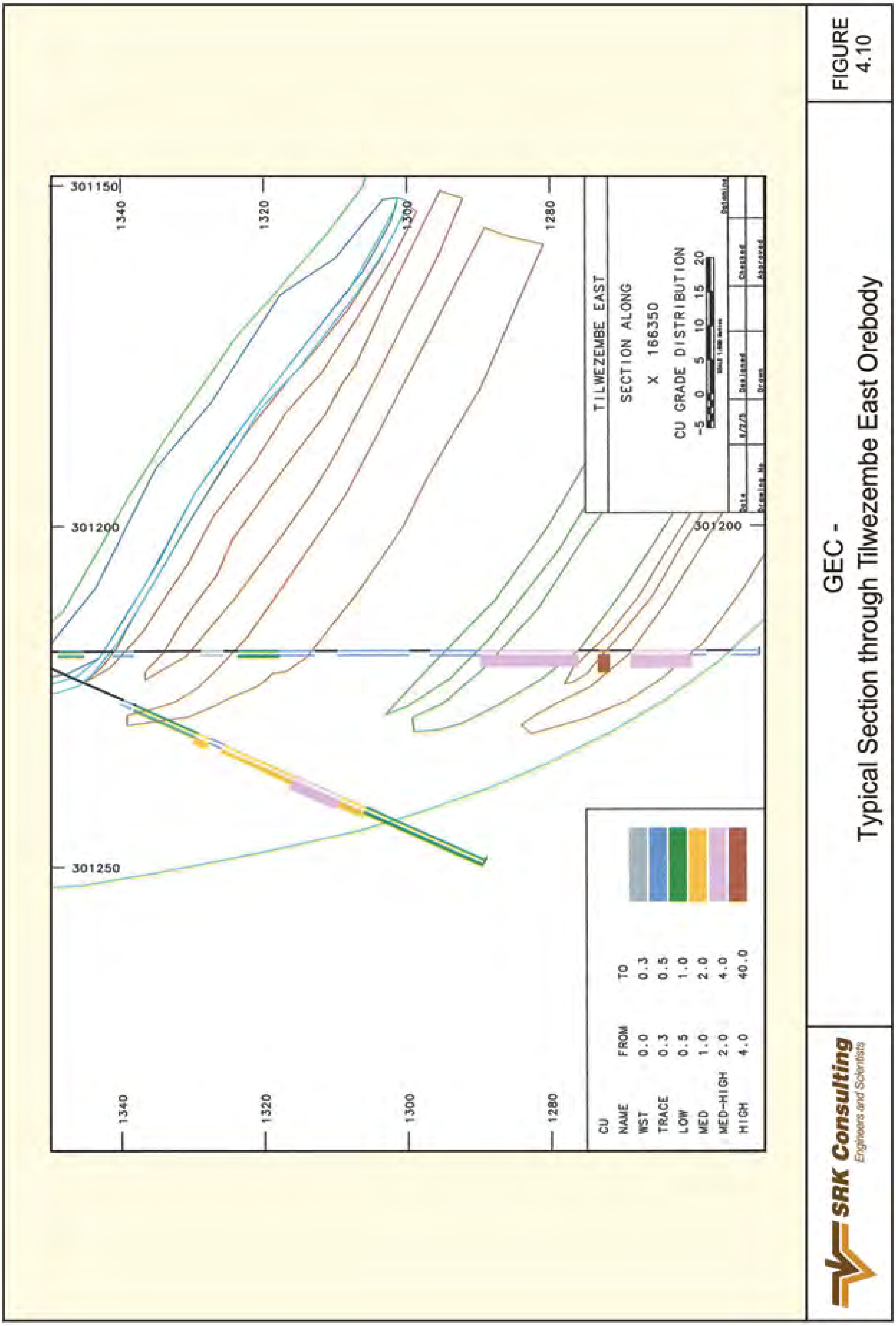


Figure 4.10: GEC—typical sections through Tilwezembe East orebody



5 MINING

This section includes discussion and comment on the mining engineering aspects of the LoM Plan as developed for the KOV Open Pit. Based on the LoM Plan, the mining equipment requirements were determined together with the labour complement as well as the capital and operating costs. The de-watering of the KOV pit has also been addressed.

5.1 Geohydrology

5.1.1 Pit Dewatering

The KOV Open Pit is currently flooded with an increasing water level as the incoming water flow exceeds the current pumping rate of about 2500 m^3/h. The volume of water in the pit is estimated to be around 12 million cubic metres. Before mining can commence the water needs to be pumped and the silt removed. In addition, once mining operations are started, boreholes will have to be commissioned to control groundwater seepage into the pit.

This section describes the proposed works to dewater the pit, installation of the dewatering borehole network and the mechanical and electrical equipment for the boreholes to control the groundwater and stormwater for protection against runoff and siltation.

Cost estimates of the new infrastructure for a project horizon of 5 years have been estimated and are tabulated in a summary format. The cost of additional new infrastructure required to cater for the increasing flows and heads as the pit enlarges and deepens are also shown.

The capacity of the pit dewatering system is based on three sources of water as shown in Table 5.1:

- the volume of water to be pumped out of the pit of about 12 million cubic metres during a period of about 6 months;
- groundwater inflow into the pit (geohydrological investigations indicated that the current groundwater inflow into the pit is currently about 2,800m^3/h and that it will increase to 3,800m^3/h when the pit is empty at level 1,220m amsl); and
- stormwater runoff into the pit.

Table 5.1: Initial Pit Dewatering: Summary of Discharge Flows

	Flow Discharge (m^3/h)	
Water Source	**Initial**	**After 6 months**
Groundwater inflow	2,800	3,800
Pit water	2,900	2,900
Stormwater runoff	500	500
Pumping Capacity	6,200	7,200

The initial pit dewatering is based on the provision of three stand alone systems with a capacity of 2,400 m^3/h; each system will entail three pumps in series, the first pump will be mounted on a floating barge with access platform and the two booster pumps will be skid mounted and positioned along the pipeline on the shore of the current pit lake, the system will pump the water via a 2,700m long 600mm ND dedicated pipeline.

The first pumpstations will be mounted on a floating barge anchored to the shore by means of rope steel cables, the floating barge will house a pumpset. The barge pump of each system will be supplemented by two fixed booster pumpstations, these two booster pumps will be mounted on a 16 m long by 8m wide concrete base, these pumpsets and pipework will be exposed to the elements, the base will also provide space for the container with the respective MCC for the three pumps, the 7.5MVA transformers will be installed in an adjacent bay.

Once the KOV Pit has been dewatered one of the three systems with a capacity of 2,400m^3/h will be removed and reinstalled to dewater the Kamoto Pit. It is expected that at this stage the bridge between the pits will be lower and that access ramp to the Kamoto Pit is in place. An additional section of pipeline and the other two systems will remain in order to cater for the groundwater inflow from the horizontal drains and stormwater runoff.

Cost estimates were completed for the pit dewatering infrastructure as shown in Table 5.2. These costs include construction costs and allowances for contingencies and engineering.

Table 5.2: Initial Pit Dewatering Cost Estimate

Items	Amount (US$ million)
600ND Pipelines to Musonoi	5.13
Pumpstations for Kov	5.37
Electrical Works for Kov	2.17
600ND Pipeline to Kamoto	0.63
Relocation of Pumpstation to Kamoto	0.49
Electrical Works for Kamoto	0.50
Engineering	2.14
Total Cost in US$	**16.43**

The total cost of the works has been estimated to be around US$16.43 million. The cost of the items for KOV Pit will be incurred during Year 0 whereas the cost of the items for Kamoto will be spent after Year 2.

The operating cost or power cost of pumping was estimated to be US$0.76million, this cost is based on a total volume to be pumped of 26 million cubic metres (12 million of water in the pit plus an additional of 14 million for groundwater ingress) at a power tariff of US$0.035/kWh.

A programme for the pit dewatering indicates that the total duration from commencement of design and equipment procurement to handover of plant will be 15 months. The programme is based on the assumption that longlead items such as pumpsets, pipes and electrical equipment will be sourced directly as the procurement of these items are in the critical path of the programme and that the construction of civil works and installation of mechanical and electrical plant will be tendered.

SRK envisages that once the pumping equipment has been procured and installed, the actual pit dewatering will commence in Month 16 and should be completed by end of Month 21 when the removal of silt and mud can commence. The pumping equipment will then be moved to the Kamoto East pit for dewatering to be started.

5.1.2 Groundwater Control

Once the pit has been dewatered then prevention of groundwater entering the pit is required and therefore the commissioning of the boreholes around the pit will be necessary.

The groundwater investigations and modelling has established a number of borehole phases with respective number of active boreholes as well as flow discharges

The groundwater control system will entail 45 boreholes to abstract the groundwater before entering the pit for the first 5 years of mining operation. The total capacity of these boreholes will be 8,300m^3/h. This will increase to 59 boreholes with a total capacity of 9,000m^3/h at the end of LoM.

The boreholes of Phases 1 and 2 will be positioned on the northern wall of the pit generally along the conveyor belt, the ones for Phase 3 will be installed at the bottom of the pit once the pit has been dewatered and the boreholes of Phase 4 will cover an arch on the top north-western section of the pit.

Two pipeline reticulations about 6 km long each will collect the borehole water and convey it to a 500m^3 pressed steel break pressure tank. From the break pressure tank, twin 800mm ND steel pipelines will convey the water to the Luilu River running generally along the railway line in the western direction, around the eastern boundary of the Existing Electro-Refinery and then parallel to the national railway line to the Luilu River. The length of the pipelines will be about 8.3km. These twin pipelines can feed along their route the Existing Electro-Refinery, the Luilu Village and the proposed New Processing Facility.

The power required for the boreholes for the initial 5 years will be 8,010kW at 6.6kV (about 10MVA) and the power required for Years 15 and 20 will be 14,000kW (about 18MVA).

Cost estimates were completed for the groundwater control infrastructure. These costs include construction cost and allowances for contingencies and engineering (Table 5.3).

Table 5.3: Groundwater Control Cost Estimates

	Amount (US$ million)			
Items	**Year 0**	**Year 5**	**Year 10**	**Year 15**
Collector Pipelines	4.28	0.83	0.94	0.00
BH Pumps	17.42	3.15	6.08	1.90
Break Pressure Tank	0.30	0.00	0.00	0.00
800ND Pipelines to Luilu	13.68	0.00	0.00	0.00
Electrical Reticulation	3.95	0.71	1.38	0.43
Engineering	5.95	0.70	1.26	0.35
Total Cost in US$	**45.58**	**5.39**	**9.66**	**2.69**

5.1.3 Stormwater Control

The KOV Pit requires protection against stormwater runoff and sediments from the old Kolwezi and Musonoi Pits which currently drain into the KOV Pit. The major contribution of solids into the pit comes from these sources.

It is proposed to construct two dams with a capacity of 125,000m^3 each on the eastern side of the pit to contain and minimize stormwater runoff and solids entering the pit. These two dams will normally operate sequentially allowing the pumping of the decanted stormwater runoff into the Musonoi River via one of the dewatering system with a capacity of 2,400m^3/h, the settled solids will be removed with earthwork equipment once the dam has been empty and the runoff has been diverted into the other dam.

The capacity of the dams have been defined for no spillage into the pit according with the last 50 years of rainfall record with the stated pumping capacity and to store the expected annual volume of solids into the one of the two dams. The volume of the two dams will also cater for the maximum monthly runoff volume into the two dams without any pumping.

The dam will also receive the groundwater inflow into the pit from the horizontal drains and stormwater runoff within the pit. It is envisage that two systems with a capacity of 1,000m^3/h each of rugged design and trailer mounted will be installed to pump into the dam from where it will be pumped into the Musonoi via one of the initial pit dewatering system.

In addition part of Musonoi Village and other areas appear to drain into the Kamoto Pit. By Year 5 when the pits consolidate it will be necessary to contain the stormwater runoff from these areas by constructing a dam, pumpstation and pipeline to discharge into the channel running on the southern top of Musonoi and Kolwezi Pits and draining into the Musonoi River. This requires some additional topographical survey in order to define the catchment area and the necessary works.

The total cost of the works has been estimated to be around US$11.19 million (Table 5.4).

Table 5.4: Stormwater Control Cost Estimates

	Amount (US$ million)	
Items	**Year 0**	**Year 5**
Dam, Pit Stormwater Protection and Silt Trap	2.90	0.00
Pumpstations	1.85	1.85
Pipelines	0.24	0.40
Musonoi Village	2.50	0.00
Engineering	1.12	0.34
Total Cost in US$	**8.61**	**2.58**

5.2 Geotechnical Investigation

SRK undertook a slope stability assessment to feasibility level for the KOV and Kamoto East open pits at Kolwezi in the DRC.

5.2.1 Principal Objectives

The programme objectives were as follows:

- To prepare a feasibility report on the open pit geotechnics, based on the information available at the time of writing;

- To recommend slope angles for the open pits;
- To comment on the effect of the "rapid drawdown" pore pressure condition on the stability of the pit slopes as the KOV pit is dewatered;
- To comment on the risk of failure of the "bridge" between the KOV and Kamoto East pits as dewatering commences; and
- To comment on the geotechnical issues regarding the proposed storm water control dam wall to the east of the KOV pit.

5.2.2 Available Information

The information available at the time of the study consisted of the following:

- Various drawings of the open pits;
- Laboratory test results of the rock and soil units in the open pits. These were evaluated and discarded due to a lack of confidence in the results;
- The geological report by Armand Francois dated 1973; and
- An on site assessment of the geotechnical situation by SRK in December 2004.

A drilling programme was initiated in December 2005 to obtain geological, geotechnical and geohydrological information but at the time of this study the geotechnical holes have not been completed. Due to the limited information, the geotechnical slope angles determined as part of this study can be classed as inferred slopes. This relates to a pre-feasibility level and not a feasibility level study. The completion of the geotechnical boreholes will enable a more thorough feasibility study.

The geotechnical analysis consisted of empirical analysis to determine the overall slope angles and numerical analysis to determine the sensitivity of the slopes to dewatering the pit, and the effect of dewatering KOV only on the stability of the "bridge" between the pits.

5.2.3 Results

Based on this study the recommended overall slope angles are as follows:

- 22° for the KOV north slope;
- 30° for the KOV south slope;
- 30° for the Kamoto east north slope;
- 35° to 40° for the Kamoto east north slope.

The numerical analyses show that dewatering of the pit pools will de-stabilize the slopes to some degree but that the expected extent of failure will be localised. The destabilising effect of dewatering can be ameliorated by dewatering the pit slopes in unison with the pit pools or to a lesser extent, by dewatering of the pit pools over a period of six months or longer. Due to the flat slope angle beneath the in pit crusher, instability of the slope beneath the crusher is not expected.

Based on the numerical analysis carried out to investigate the stability of the "bridge" between KOV and Kamoto east, it is concluded that dewatering of the KOV pit without dewatering the Kamoto East pit will not cause shearing of the "bridge" into the KOV pit. The dewatering will however lead to localised instability of the KOV south slope.

5.2.4 Recommendations

Based on the work carried out during this study, the following is recommended:

- Slope angles of between 22° and 35° should be adhered to as the weak and friable nature of the rock formations in the KOV and Kamoto east pits make steeper slope angles hazardous;
- Ramp access to the pit bottom is currently under threat as both of the ramps have become unstable and are showing signs of severe slope instability;
- The geotechnical site investigation program for the feasibility study to be completed as the borehole cores will provide the only tangible information for more detailed design;

- The dewatering of slopes in the KOV pit should be considered a priority as slope dewatering is the most effective tool available to improve stability conditions;
- A slope monitoring program should be initiated as soon as possible to determine slope movement rates and vectors and give early warning of potential slope instability during pit pool dewatering and mining operations; and
- The damming of surface water run-off in the east of the KOV pit is feasible from a geotechnical point of view on condition that the dam foundations are adequately designed as soft rock units perpendicular to the proposed dam wall may act as water conduits and cause piping underneath the dam wall.

5.3 Mine Design

5.3.1 Introduction

SRK with Bateman were commissioned in early 2005 to conduct a pre-feasibility study of the KOV, Kananga and Tilwezembe open pit mines in the Kolwezi area in the DRC. The Base Case developed in the pre-feasibility study considered the three mines supplying ore to the KZC concentrator at a rate of 220ktpm (2.64Mtpa). In addition to the Base Case, further options at different production rates were considered in which a Direct Leach Process was used for the recovery of copper and cobalt. Based on the results of the pre-feasibility study, SRK completed a Feasibility Study of the KOV open pit for a production rate of 400ktpm (4.8Mtpa) feeding ore to a Direct Leach Plant. The mining study included open pit optimisation, practical pit design and production scheduling to convert the Mineral Resources to Reserves and to determine the capital and operating cost to the level of accuracy of ±10 to 15%.

5.3.2 Kananga

An optimisation study for Kananga was done as part of the pre-feasibility study to determine the open pit footprint for longterm planning purposes based on the Inferred Mineral Resources that were estimated at the time. The study indicated that the main railway line will have to be relocated if the exploratory drilling confirms the geological model and additional resources. The optimisation study also indicated that a production rate of 60ktpm at Kananga can be maintained for more than 8 years.

5.3.3 Tilwezembe

A study was also done for Tilwezembe East along the same lines as for Kananga. The optimisation study showed that once mining has been established that a production rate of 60ktpm can be maintained for more than 7 years. As Tilwezembe is situated 27km away from Kolwezi the transport of ore is more costly than for Kananga although a railway line extends to the mine which will allow ore to be transported by rail to the KZC processing plant.

5.3.4 Open Pit Optimisation

The open pit optimisation was carried out using Whittle-4X. The purpose of the open pit optimisation is to define the optimised limit of the orebody resources that is economically mineable based on the available mining, processing, and operating cost as well as metal price parameters. Mining constraints are also input into the optimisation.

The input parameters for the optimisation included the following:

- **Pit slope angles**: The overall slope angle included the ramping system and for the NW to NE slope, a slope angle of 22° was defined and for the other areas an angle of 30° was used. These angles were based on the recommendations from the Geotechnical Investigation.
- **Mining cost**: This cost item includes drilling, blasting, loading, hauling, crushing and conveying of waste rock, and pit dewatering. A mining cost of $\$1.30/\text{tonne}_{\text{rock}}$ was used for the optimisation.
- **Mining dilution and mining recovery**: It was assumed that five percent waste will be diluted with ore during the mining operation. The mining recovery was assumed to be 95%.

- **General and Administration Cost**: This includes on-mine costs other than mining and processing.
- **Processing cost**: The total processing cost to process ore in the Direct Leach Plant was estimated to be $27.32/ore tonne.
- **Processing recovery**: The processing recovery for copper and cobalt at 91% and 85% was provided by the client.
- **Off-Mine Costs including Selling cost:** Cu US$401.14/tonne Cu, Co US$534.46/tonne Co.
- **Product price:** Cu US$2,646/tonne (120c/lb), Co US$8,818/tonne (US$4.00/lb salts)

5.3.5 Optimisation Results

The open pit optimisation was carried out for four scenarios in which the influence of different restrictions were considered including the railway line in the North, the Waste Crushing and Conveying System to the East and the Musunoi Village as well as the Kamoto Hoisting Shaft to the South.

The results of the optimisation (Table 5.5) showed that the additional tonnage that can be added to the project when the restrictions are moved or relocated for all the above scenarios justifies mining the additional tonnage at a lower overall stripping ratio.

Table 5.5: Summary of Optimisation Results

Scenario	RoM Tonnage (Mt)	%Cu	%Co	Waste Tonnage	Strip Ratio	Tonnage Increase
I	84.9	4.82	0.47	1,146.5	13.51	—
II	97.3	4.72	0.49	1,182.4	12.15	12.4
III	104.0	4.85	0.45	1,314.7	12.65	6.7
IV	150.5	4.84	0.48	1,507.5	10.01	53.2

To carry out the practical pit designs for the mining sequences, pit-shells that are of a practical size were selected for the base case and the practical pits were designed accordingly.

5.3.6 Mine Planning

Mine planning was carried out for the LoM with a plant capacity of 4.8Mtpa using the practical pit designs for each of the selected seven cuts or pushbacks. An average RoM grade was targeted at 5.2%Cu with a tolerance of ±1%Cu to allow for the plant feed to be as uniform as possible.

The only restriction that was applied to the design of the practical pit was to limit mining to 50m from the existing waste crusher and the waste conveyor. A safe distance of 500m from the Kamoto Hoisting Shaft was also maintained in the mine planning.

The priority for the pit design was the ramping system, in order to maintain access to the ore crusher, waste crusher and waste dumps in the north-west and south side of the pit during the mining operation.

Production scheduling was carried out on an annual basis allowing for de-watering, silt removal and pre-stripping during the first two years.

5.3.7 Practical Pit Design

The following considerations were incorporated into the practical pit design:

- **Design Criteria**—The practical pits were developed using the pit-shells as selected from the different optimsations. The objective was to include ramps, bench and berm configurations in order to calculate the ore and waste tonnage within the practical pits. These designs and the resulting reserves would then form the basis for the production scheduling.
- **Ramp**—The main ramps were designed 35m wide and at a 10% gradient. For the last few benches in the pit bottom, the ramp width was reduced to a minimum width of 25m.
- **Batter Angle**—It was assumed that an 80° batter angle would result from the normal blasting practices.

- **Inter-ramp slope Angle**—In the NW to NE wall the inter-ramp slope is designed at 25° and in all other areas a 35° slope angle has been selected for the pit design in agreement with the recommended slope angles.
- **Bench Height**—Initially, 10m benches were selected for both ore and waste rock based on the mining practice employed by Gécamines when mining the KOV Open Pit. After the selection of a suite of larger mining equipment, the bench height was increased to 15m that is considered better suited to the size of equipment and without compromising dilution and grade control when loading ore.

5.3.8 Pit Design

Maintaining the ramp access to the waste crushers, waste dumps and to the ore crusher was considered a priority in the pit design to utilise the full capacity of the waste crushers and conveying system. In order to minimise the haulage distance to the final ore and waste destinations, several ramping system are designed for each pit expansion. This results in utilization of more than one road for rock haulage from a loading station to a dumping point. This ensures the continuity of the ore and waste rock haulage during the life of mine.

During the preparation of the production schedule it was aimed to limit the vertical advance in each cut to 60m (4 benches). In most periods this was achieved for the wider benches. A number of smaller benches on the west side of the pit and in the upper elevations with a small hill-side bench geometry, have been excavated in one period in the same pushback. The vertical advance rate per year with the seven cut configuration is 47m per year (3 benches). This rate of advance is considered in line with the practice in large open pits.

5.3.9 Mineral Reserves

The mineral reserves within each practical designed pit were determined after inclusion of 5% mining dilution and 5% mining losses. The mineral reserves were further classified as Probable Reserves in accordance with the Guidelines of the SAMREC Code and presented in Table 5.6.

Table 5.6: Probable Mineral Reserves within the Designed Practical Cuts (CoG; 1.0%Cu)

	Ore			Waste	Total Rock	Stripping
Pushback	Mt	Cu%	Co%	Mt	Mt	Ratio
Cut 1	18.2	5.31	0.43	187.8	206.0	10.3
Cut 2	33.6	5.04	0.57	417.5	451.1	12.4
Cut 3	17.6	4.27	0.44	250.7	268.3	14.2
Cut 4	16.9	4.40	0.38	159.3	176.2	9.4
Cut 5	12.4	3.92	0.68	54.4	66.8	4.4
Cut 6	17.5	4.90	0.36	125.2	142.7	7.2
Cut 7	23.7	5.33	0.38	228.3	252.0	9.6
Total	**139.9**	**4.83**	**0.47**	**1,423.2**	**1,563.1**	**10.2**

5.3.10 Production Schedule

The total capacity of the waste crushing and conveying system has been estimated to be 64Mtpa. In the pit design it was endeavoured to limit the total waste to 64Mtpa to minimise trucking of waste rock to the dumps in the northwest and to the south of the pit. In some years, due to a short haul distance or no access from the top benches to the crushers, waste rock is hauled to the dumps instead of being conveyed.

For scheduling purposes two in situ copper grade categories were defined; a higher grade ore at a CoG of 4%Cu and a lower grade ore between 1%Cu and 4%Cu. After year 13 with the excavation of Cut 5 (containing the lower grade ore compared to the other Cuts), the lower grade ore has to be stockpiled and then blended with the higher grade mined from Cut 4 and Cut 6.

The smoothed LoM schedule is presented in Table 5.7 and the total rock movement per period is presented in Figure 5.1.

Table 5.7: Life of Mine Schedule

Period	RoM Ore (Mt)	RoM Ore Grade %Cu	RoM Ore Grade %Co	Silt Removal (Mt)	Waste (Mt)	Stripping Ratio	Total Ore & Waste (Mt)
Prestrip 1				2.0	4.0		6.0
Prestrip 2				2.0	12.0		14.0
1	1.2	4.62	0.30	1.0	61.4	51.2	63.6
2	4.8	4.37	0.30	1.0	66.8	13.9	72.6
3	4.8	5.64	0.50	1.0	63.0	13.1	68.8
4	4.8	5.76	0.51		64.5	13.4	69.3
5	4.8	5.06	0.46		64.5	13.4	69.3
6	4.8	4.26	0.48		63.0	13.1	67.8
7	4.8	4.61	0.66		63.0	13.1	67.8
8	4.8	5.24	0.64		64.0	13.3	68.8
9	4.8	5.34	0.63		62.0	12.9	66.8
10	4.8	5.39	0.57		62.0	12.9	66.8
11	4.8	5.46	0.51		62.0	12.9	66.8
12	4.8	4.77	0.47		56.2	11.7	61.0
13	4.8	4.20	0.45		50.3	10.5	55.1
14	4.8	4.27	0.41		50.5	10.5	55.3
15	4.8	4.20	0.63		39.6	8.2	44.4
16	4.8	4.20	0.59		44.7	9.3	49.5
17	4.8	4.20	0.63		46.9	9.8	51.7
18	4.8	4.20	0.47		42.7	8.9	47.5
19	4.8	4.20	0.41		40.4	8.4	45.2
20	4.8	4.20	0.36		41.5	8.7	46.3
21	4.8	4.92	0.28		59.0	12.3	63.8
22	4.8	4.81	0.28		46.7	9.7	51.5
23	4.8	4.77	0.38		39.2	8.2	44.0
24	4.8	4.74	0.46		43.8	9.1	48.6
25	4.8	4.71	0.42		39.7	8.3	44.5
26	4.8	5.38	0.37		33.8	7.0	38.6
27	4.8	5.50	0.37		18.2	3.8	23.0
28	4.8	5.43	0.37		11.2	2.3	16.0
29	4.8	5.35	0.43		3.5	0.7	8.3
30	4.4	5.06	0.49		3.1	0.7	7.5
Total	**140.0**	**4.83**	**0.47**	**7.0**	**1,423.2**	**10.2**	**1,570.2**

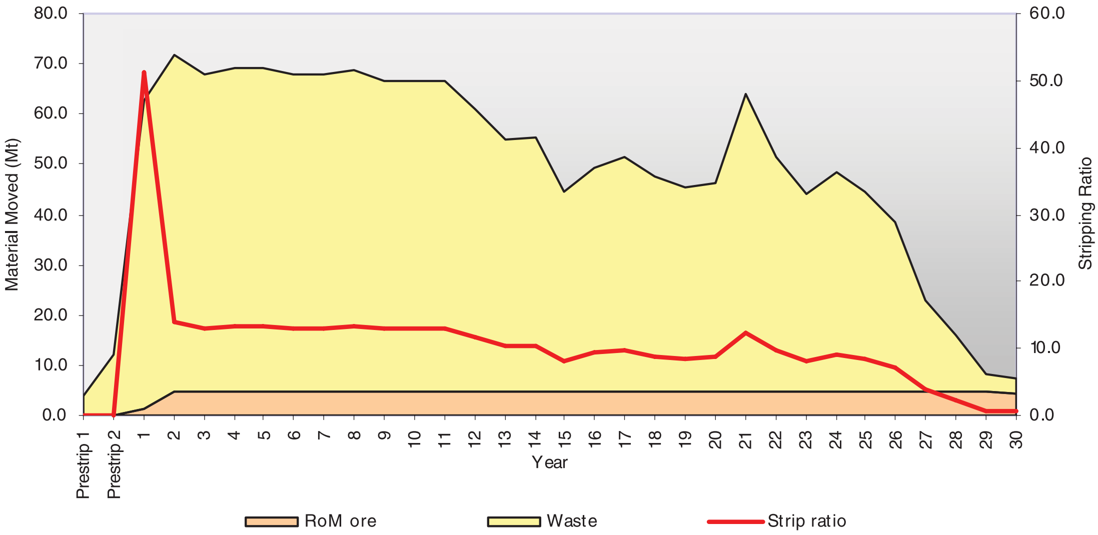


Figure 5.1: Total Rock Movement for the Life of Mine

The production scheduling was used for estimation of the required truck hours, the mining fleet estimation as well as the operating and capital cost estimation.

5.4 Mining Operations

The philosophy underlying the study was to aim for a conventional truck and shovel operation that can achieve the lowest possible mining cost. Capital and Operating costs were prepared on the basis of the mining fleet being owner operated, with direct hired management, technical services, maintenance services and operating personnel employed on a continuous basis.

Equipment suppliers would be expected to participate in an alliance agreement in regards to offering guaranteed norms and life expectancies on all major machine components for the life of the equipment.

The following mining and support activities are included under Mining Operations:

- Pre-stripping,
- Drilling of blast holes, including secondary activities,
- Blasting activities,
- Loading and hauling of the waste and ROM ore,
- Stockpiling and re-handling of the ROM feed to the crusher,
- Construction and maintenance of pit, haul roads and waste dump areas,
- Dust Suppression of haul roads,
- Building and maintaining the waste dumps,
- Silt removal,
- Geological (infill) and Geotechnical drilling.

5.4.1 Pre-stripping

Allowance is made during the first two years at the start of the project to do pre-stripping at a rate of 6.0Mtpa for the first year and 14Mtpa for the second year. This time will also be used for the de-watering of the current KOV pit, removal of silt from the pit and the commissioning of the existing waste handling system.

5.4.2 Drilling

The planned bench height used for the KOV Feasibility Study is 15m. The planned blast hole diameter is 251mm with a 1.5m sub drill. The planned blasthole diameter is based on the bench height, expected rock conditions and the scale of operations that will lead to an adequate distribution of explosives in the ground.

Two drilling patterns are defined based on the material characteristics:

- Medium / hard material with a burden of 7.8m and a spacing of 9.0m,
- Soft material with a burden of 10.5m and a spacing of 12.0m.

Large rotary electric drills with a single pass drilling range of 20m will be employed because of low unit cost and longevity considerations.

5.4.3 Blasting

Blasting will be conducted by a blasting team employed by the mine. The maximum top feed size to the crusher is 1 000mm. It is further assumed that 100% of the blast holes will be wet. The holes will be charged with emulsion type explosives.

The average powder factor estimated is 0.56 kg/m^3 and is based on the use of emulsion type explosives with the assumption that 40% of the blasted material will be "soft" and 60% of the blasted material "medium to hard".

Secondary rock breaking will be done with a mobile hydraulic hammer and provision is also made for secondary blasting. Airblast and vibration monitoring is recommended to be done with every blast because of its importance when responding to environmental issues.

5.4.4 Loading and Hauling

The size of the primary loading and hauling equipment was determined by considering a range of shovels (including electric rope shovels and hydraulic shovels) and haul trucks with capacities ranging from 220t to 340t. Based on the equipment matching and the high level cost analysis SRK recommend the use of electric haul trucks with a payload capacity of 280t and hydraulic shovels capable of loading the 280t trucks in four to five passes as primary hauling and loading equipment.

The maximum capacity of the waste handling system is 64Mtpa. Waste is transported to the waste dumps when the waste handling system's capacity is reached.

5.4.5 Stockpiling

Stockpiling of RoM ore is simulated during the mine scheduling process to ensure that a constant feed of 4.8Mt of RoM ore per annum is delivered to the primary ore crusher. Additional stockpiling may be required as part of the grade control process to smooth variation in the head grade.

Capacity exists in the stockpiling area and a Front End Loader is available to load RoM ore from the stockpile as required.

5.4.6 Mining Calendar

In order to calculate operating hours the following assumptions were made:

- A seven day work week schedule will be used for the project,
- The mechanical availability of the primary equipment is 85%,
- The utilisation of the primary equipment is 75%,
- Secondary equipment hours are approximately 65% of primary hours.

The result is 5,184 operating hours per annum being used for the simulation of the primary equipment. The anticipated effect of a five month long rainy season is dealt with by adding an additional 20% to the numbers of primary equipment required.

5.5 Mining Equipment

Equipment selection was made based on the production requirements and is summarised in Table 5.8.

Table 5.8: Mining, Support and Utility Equipment

Item	Peak Requirements
Primary Mining Fleet	
Blast hole drills	4
Hydraulic face shovels	5
Electric haul trucks	19
Major Support Equipment	
Front End Loader, 190t	2
Bulldozer, 65t	7
Rubber Wheel Dozer, 45t	4
Graders, 24t class – 4.2 m blade	5
Rock Breaker	2
Secondary drills (including grade control and geotechnical drilling)	3
Water Carts, 100tonne	3
Wheel Loader, Road Building, 50ton	1
Road Building Tippers	2
Diesel Bowser, 12 000 ℓ	2
Low-bed	1
Utility Fleet:	
Light Delivery Vehicles (LDV),	20
Road Stone Crusher	1
Lighting Plants	7
Busses	3

This utility fleet does not include workshop equipment such as mobile cranes, tyre handler, service trucks and fork lifts that has been allowed for under Surface Infrastructure.

5.6 Maintenance of Mining Equipment

The maintenance facilities were determined based on the maintenance philosophy set for the project, which incorporates the following features:

- The primary mining and major support equipment will be maintained by the mine, under the supervision of a Original Equipment Manufacturer (OEM) representative,
- All maintenance work would be carried out by the mine's personnel according to the specified requirements,
- Guarantees on major equipment components could provide capped cost on major operating cost, managing on a risk sharing basis,
- All special tools and facilities to be provided by the mine,
- Procurement of machines could be done in a way to ensure the life cycle cost are captured and guaranteed over the expected life of the machine.

A detailed Continuous Maintenance Monitoring System (CMMS) has been adopted by GEC for preventative maintenance scheduling as well as recording machine and component status.

Preventative and planned maintenance principles will be used to replace major components before failure occurs and replaced by new or reconditioned items to original OEM standards. Condition monitoring will be extensively used to support this strategy.

5.7 Labour

In this part of the Feasibility Study mainly first level mining labour (operators) was catered for, including the direct supervisory personnel. Further to the above, allowance was also made for the maintenance crews and maintenance operators for maintaining the complement of mobile equipment. The mining labour needed for the Mineral Resource Management functions and some Engineering Development functions are also included in the labour costs.

Based on a seven day work week and three shift per day, four operating teams are required. The operators were allocated with an additional 10% allowance for leave and training absenteeism.

Some of the non-critical supporting activities only work on a day shift or a two shift basis. This was encompassed in the Labour Cost Model.

The peak labour requirements are shown in Table 5.9.

Table 5.9: Peak Labour Requirements

Category	Peak Numbers
Mining Operators	273
Local Artisans	32
Local Engineering Supervisors	7
Local Mining Supervisors	13
Local Managers	2
Expatriate Artisans	16
Expatriate Engineering Supervisors	4
Expatriate Mining Supervisors	5
Expatriate Managers	2
Total	**354**

5.8 Capital Expenditure

From the detailed mining equipment schedule and quotations received from equipment suppliers, capital costs were derived for the primary, main support and utility support equipment described in Table 5.8. Table 5.10 summarises the initial capital expenditure for mobile mining equipment as defined above, while Table 5.11 gives the replacement and expansion capital requirements for the LoM.

Table 5.10: Summary of Initial Capital Expenditure

Item	Budget Capital Estimate (US$ million)	Assembly (5%) (US$ million)	Import Duty (2%) (US$ million)	Total (US$ million)
Initial Capex				
Trucks	64.87	3.24	1.30	69.41
Shovels	50.79	2.54	1.02	54.35
Drills	13.48	0.67	0.27	14.42
Front End Loader	6.77	0.34	0.14	7.24
Track Dozer	5.71	0.29	0.11	6.11
Rubber Wheel Dozer	3.15	0.16	0.06	3.38
Graders	1.85	0.09	0.04	1.98
Water Carts	2.00	0.10	0.04	2.14
Secondary drill	2.40			2.40
Road Building Tippers	1.60	0.08	0.03	1.71
Other Secondary Equipment	1.35			1.35
Utility Fleet	2.12			2.12
Other	2.36			2.36
Sub Total	**158.44**	**7.92**	**3.17**	**168.96**
Contingency of 10%				**16.90**
Total				**185.86**

Table 5.11: Summary of Replacement and Expansion Capital Expenditure

Replacement and Expansion Capex	Budget Capital Estimate (US$ million)	Assembly (5%) (US$ million)	Import Duty (2%) (US$ million)	Total (US$ million)
Sub Total	285.96	13.3013.30	5.32	304.57
Contingency of 10%				30.46
Total				**335.03**

5.9 Operating Expenses

The blasting cost is based on a quotation for the delivery of a down-the-hole service received from AEL. The following cost items were considered in calculating the operating cost for the primary, main support and utility support equipment:

- Fuel,
- Tyres,
- Maintenance and Repairs including lubricants,
- Ground engaging Tools,
- Power consumption,
- Labour as discussed above.

The price for diesel fuel was fixed at US$0.86 per litre. The fuel consumption of the equipment was determined by using the consumption as indicated by the relevant Suppliers.

The cost of tyres was calculated using the tyre life estimator curves provided by the relevant Suppliers, to derive the cost per hour rate.

Table 5.12 summarises the mining operating cost of the project determined from first principles, by expressing the cost as a function of the production outputs. A breakdown of the cost into different components is shown in Figure 5.2.

Table 5.12: Average LoM Mining Operating Cost

Description	Operating Cost (US$/tonne)
Expressed per tonne of RoM mined	13.52
Expressed per tonne of total material moved	1.21

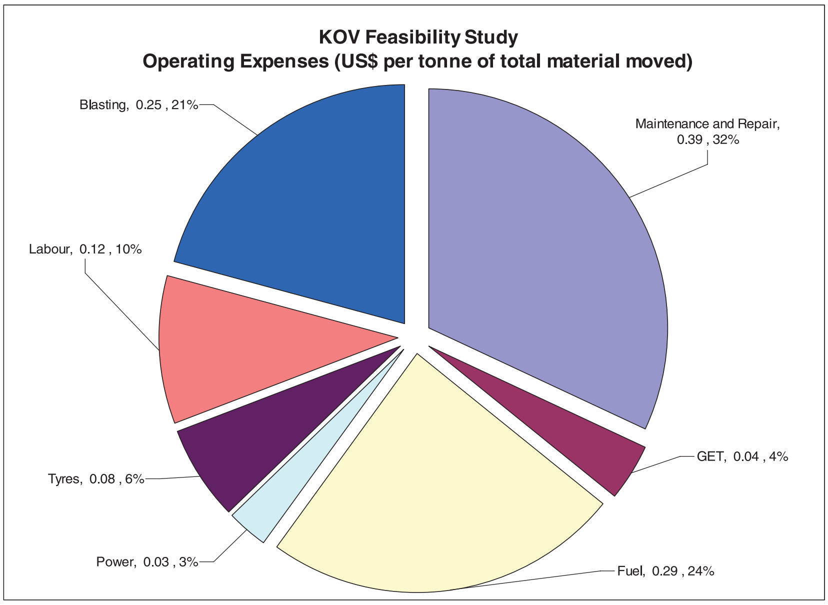


Figure 5.2: Breakdown of Mining Operating Cost

5.10 Ore Crusher and Conveyor System

5.10.1 Proposed Ore Crushing and Conveying System

Krupp Industrietechnik GmbH, Germany, ("Krupp") prepared an estimate for the primary ore crusher and conveyor system from the KOV Pit to the processing plant. The Krupp scope of work included the design, fabrication, limited factory assembly, corrosion protection, transport, site erection and commissioning of the ore handling system.

The proposed project consisted of the following components:

- RoM tip with static grizzly, receiving hopper and apron feeder;
- Primary crushing plant c/w scalping screen and jaw;
- Conveyors complete with sacrificial, overland, stacking and reclaiming;
- 50,000-tonne longitudinal stockyard with stacker and bucket wheel reclaimer.

5.10.2 Design Criteria and Specifications:

• Life span	40 years;
• Capacity required	400ktpm;
• Capacity required	4.8Mtpa;
• Theoretical hours available	8,760 hrs;
• Utilisation	70%;
• Availability	80%;
• Actual hours available	4,906hrs;
• Actual capacity	979tph;
• Design capacity	1,200tph;
• Stockyard—Width & Length	25m x 300m;
• Stockyard—Live storage capacity	50 000t;
• Stockyard—Reclaim capacity	750tph

5.10.3 Capital Expenditure Estimate

The capital expenditure estimate for the ore crushing and conveying system is summarised in Table 5.13.

Table 5.13: Summary of Ore Crushing and Conveying System Capital Estimate

Item	Total (US$ million)
RoM Tip and primary crushing only	8.8
Conveyors only	22.5
Stockyard only	8.1
Design, supply, erected & commissioned—RSA	39.4
Risk (not included in total)	3.9
Shipping (Estimate)	8.9
Total: Design, supply, erected & commissioned—DRC Kolwezi	**48.3**

5.10.4 Refurbishment of the Waste Crushing and Conveying System

In the early 1990's Krupp supplied and installed an overburden, or waste removal system in the KOV pit for Gécamines at Kolwezi in the DRC. The system was abandoned in the late nineties during the time of the unrest and political instability in the DRC. The gyratory crushing plant and a section of the conveyor was cold commissioned, but was never installed in the lower part of the KOV pit. It is still situated at the side of the pit, some 2.5 km away, in pre-assembled condition. The jaw crushing plant, together with the complete conveyor system and spreader, were installed and commissioned, but only operated for a short period of time. It has as such not handled a significant quantity of material.

This waste removal system forms part of the infrastructure of the KOV open pit mine, and is included in the JV agreement between GEC and Gécamines. It is a valuable asset in the Project, and essential part of the planned mining of KOV.

In early January 2006 GEC approached Krupp Materials Handling, a division of ThyssenKrupp Engineering (Pty) Ltd, ("TKE"), in South Africa, to conduct an audit on the entire overburden removal system in order to assess the condition of the equipment and establish budget prices for the re-commissioning of the system. TKE was also requested to submit budget prices for hoppers and loading bridges that can handle 300-tonne tip trucks to replace the existing hoppers and loading bridges designed for 150-tonne trucks.

Summary of findings by ThyssenKrupp Engineering

In summary it can be said that Krupp found the complete system in remarkably good condition. They also confirmed that they are in a position to replace parts damaged by the years of standing idle in the sun, in a relatively short time, and are prepared to give GEC a performance guarantee after completion of these repairs.

Capacity of the system

Krupp advised that the system is capable of handling 8,000tph of waste material on a sustainable basis. This confirmed reports given previously to GEC by Gécamines. This translates to a tonnage of 64Mtpa of waste removal capacity, based on operating on a 24 hour per day, and 7 days per week operation.

Cost of refurbishment

The price submitted by Krupp for refurbishment of the complete waste handling system is estimated at US$27.1million, based on an exchange of US$1 = ZAR6.50. This price includes the replacement of the following:

- Complete electrical and instrumentation system;
- All belting as well as troughing, transition, impact and return idlers;
- Packing of all equipment into containers in Johannesburg;
- Basic refurbishment of all electric motors, gearboxes, hydraulic pumps and drives;
- Provision for the replacement of all missing components; and
- Installation, erection and commissioning.

Krupp's price excludes the following:

- All civil work required for the preparation and installation of the second crushing plant and the in-pit conveyor and the required foundation for the proposed extension hoppers and loading bridges;
- Transport of equipment from Johannesburg to Kolwezi;
- Import duties and taxes payable in the DRC;
- Storage and handling of equipment on site;
- All cranage, scaffolding and lifting equipment;
- Accommodation and subsistence for Krupp employees;
- All costs related to site induction and medical examination; and
- All traveling costs from Johannesburg to Kolwezi.

The total cost of the refurbishment for the entire waste handling system, including the costs to cover the above exclusions, is summarised in Table 5.14.

Table 5.14: Summary of Capital Estimate of Refurbishing Waste Handling System

Item	Total Cost (US$million)
Equipment supply, installation and commissioning	27.1
Exclusions	2.9
Contingencies	1.6
Total Capital Cost—Waste Handling System	**31.6**

Programme for refurbishment

GEC obtained the following programme from Krupp for the refurbishment works:

• Place order and finalise supply agreement:	1 month
• Krupp procurement of all parts and equipment	6 months
• Packing and paperwork for despatch	1 month
• Transport to Kolwezi	2 months
• Installation	6 months
• Inspection and cold commissioning	1 month
• Hot commissioning	2 weeks
• Total programme:	18 months

5.11 SRK Comments

5.11.1 Mine Design

The practical pit design for the project was carried out using the pit shells resulting from the optimisation process on the current geological block model. A Production Schedule was also prepared for the life of mine with two years of pre-stripping and 30 years of ore production.

The following concluding remarks can be made:

- The mining operations of the project are based on a conventional truck and shovel open pit mining method. The planned bench height is 15m and it is assumed that all the material will be drilled and blasted. The truck hours simulated in the mining production schedule forms the major input in determining the numbers of primary equipment required for the project. The secondary and utility equipment fleets is derived from the primary fleet.
- The mining capital estimate is based on the number of mobile equipment required over the life of mine of the project, the estimated useful economic life of the equipment and the equipment cost as indicated by quotes received from Original Equipment Manufacturers (OEM's). The operating expenses are determined by the operating hours of the mining equipment, the operating cost per hour, the cost of blasting and the mining labour cost based on a seven (7) day work week with four operating crews.

The following recommendations apply to the further more detailed study of the project:

- The topography surface used in the study is dated and not accurate. This needs to be updated with a new and current survey to ensure that the volumetric calculation is accurate.
- It will also assist in the design of the infrastructure around the pit area including the roads outside the pit exits to the ore and waste dumping locations.
- The optimisation is sensitive to the slope angles and a detailed geotechnical investigation is required for more accurate slope angles.
- That the concession boundaries be extended to cater for the final pit and waste dumping areas.
- Discussions with preferred Original Equipment Manufacturers (OEM's) be initiated in order to form alliance agreements that will attempt to address the risks regarding equipment purchase lead times and equipment maintenance.
- That a process of recruitment and training of equipment operators, maintenance personnel and other essential personnel be initiated.

5.11.2 Waste Handling System

The Krupp proposal is an important aspect of the DCP Project. It is recommended that the proposal be accepted and that GEC and DCP should enter into final negotiations with Krupp as soon as the finance for the project is obtained.

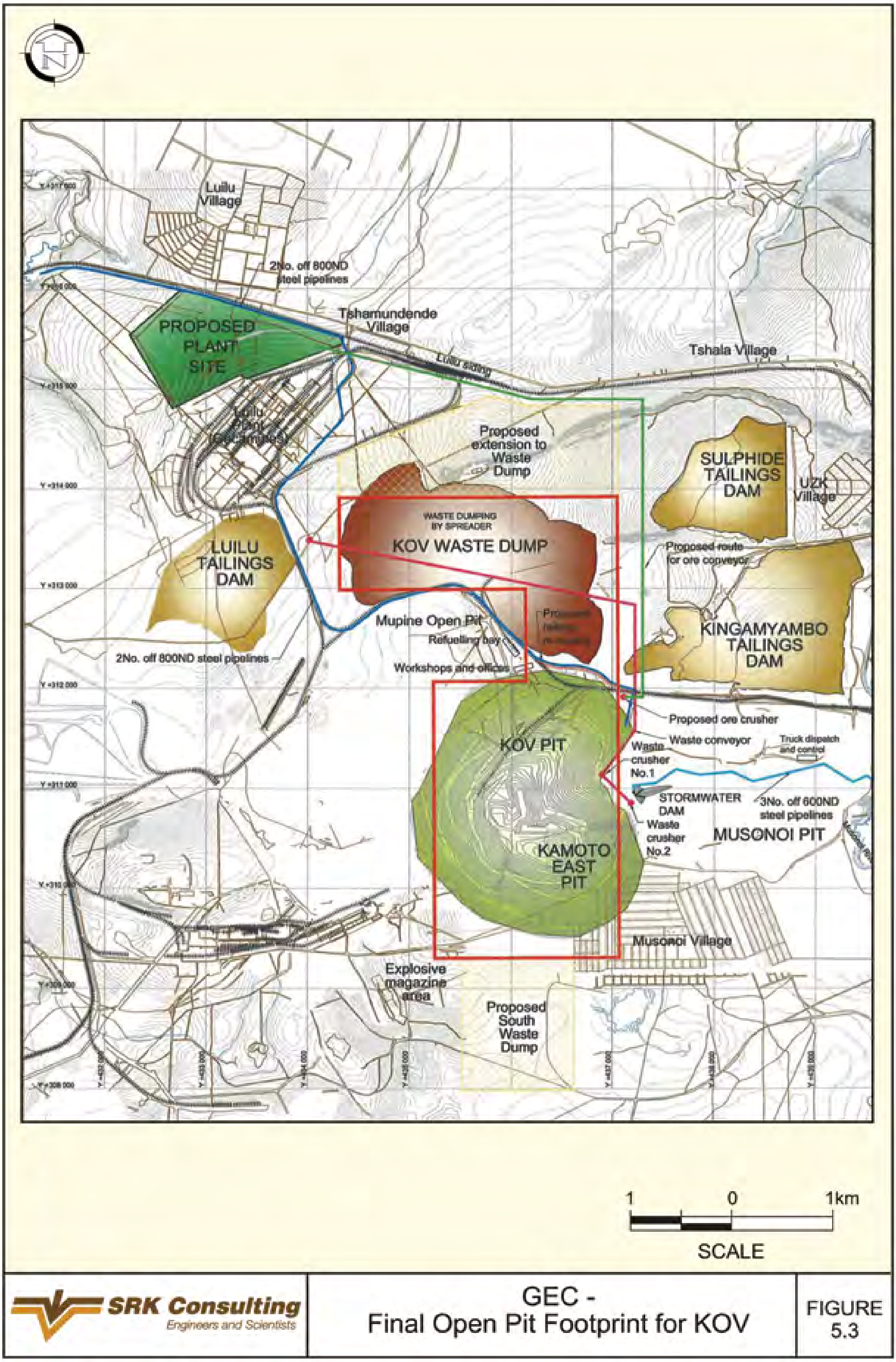
As DCP would require re-benching of the KOV pit to start by about mid 2008, the placement of a contract with Krupp should be in January 2007.

5.12 Risks and Opportunities

The following risks and opportunities were identified by SRK that may impact on the project:

- The lead time for the purchase of mobile mining equipment, especially primary haul trucks and shovels, could be up to 18 months.
- The scarcity and low availability of tyres for mobile mining equipment is a concern and could negatively influence the purchase lead times.
- In order to achieve the planned mechanical availability in the remote location of the project specific attention should be given to maintenance management and the availability of spares.
- The proposed mobile mining equipment is high technology equipment and the skills of the available local labour compliment may not be suited to the operation of this equipment. Focus on the training of operators is essential.
- The rock characteristics (including density) of the proposed mining area are currently poorly defined. Improved knowledge in this regard will improve the estimates for blasting costs and the operating costs of mining equipment.
- The final footprint of the KOV open pit and waste dumping areas will extend beyond the current concession boundaries (Figure 5.3) and poses a risk to the LoM of the project. GEC is presently negotiating to have the boundaries extended.
- The final footprint of the KOV pit extends over portion of the Musonoi village (Figure 5.3) after Year 15 of the project. While provision has been made for the relocation of the affected residents, this will need to be carefully and timeously planned.
- The possibility of utilising re-furbished equipment during the initial stages of the project is being investigated at present that will allow preparations for de-watering and mining to be initiated in the near future.

Figure 5.3: Final open pit footprint for KOV



6 MINERAL PROCESSING

This section includes discussion and comment on the metallurgical processing aspects associated with the Material Properties and Material Contracts. Specifically, detail is given on the process metallurgy and process engineering aspects relating to plant capacity, availability and metallurgical performance as incorporated in the LoM plan for the KOV Project.

6.1 Introduction

The New Processing Facility study is based on feeding 4.8Mtpa of KOV ore to the new leach plant plus 0.9Mtpa of both Tilwezembe and Kananga ores to the KZC concentrator with the concentrates being fed to the new leach plant.

For this CPR, the use of the KZC concentrator and the mined ore from both Tilwezembe and Kananga has been excluded for the financial model. The concentrates to be produced from these two mines will be approximately 216kpta, or less than 5% of the leach plant throughput. The copper grade will be higher than the KOV material whilst the cobalt grade will be considerable higher than the KOV material.

The new leach plant has been designed to accept this material and as such has been costed, but the benefits have been ignored in this review.

6.2 Metallurgical Test Work

The KOV pit is currently flooded and as such fresh bulk samples of ore are not available for pilot plant test work. This has resulted in the necessity of using previously mined ore (from 1998) for the bulk test work to be undertaken at Mintek. Metallurgical drilling of the ore body has been undertaken with samples being dispatched to SGS Lakefield.

The KOV deposit is an oxide deposit consisting of five lithological units, namely:

- SDB, surface material, consists mainly of oxidised material, called '**SDB Ox**', but also contains a sulphide fraction, called '**SDB Sulph**';
- **'RSC'**;
- **'RSF'**;
- **'D Strat'**;
- **'Rat Grise'**, deepest material.

Mintek was contracted to complete test work to evaluate the proposed Bateman flow sheet and determine operating conditions for the recovery of copper and cobalt from the material. The test program has been divided into two phases, i.e. copper recovery (Phase 1) followed by impurity removal on a bleed stream from Phase 1 to result in a pure cobalt product (Phase 2). Phase 1 includes:

- Comminution;
- Mineralogical investigation, using X-Ray Diffraction (XRD) and Scanning Electron Microscope (SEM) techniques;
- Laboratory test work to determine optimum leach conditions;
- Laboratory test work to obtain data for primary copper solvent extraction (SX);
- Piloting: leach, Cu SX, Cu electro-winning (EW).

Mintek has received 31 bulk bags, containing a total of 27 tonnes of material. Included was 1 tonne of Fines, which is a mixture of the different material. The material had been mined some 8 years ago, so some weathering has probably occurred. The material destined for Mintek was handpicked, thus it may be slightly higher in copper than the run-of-mine ore, due to the striking green colour of the copper-rich malachite. Preliminary results only are available.

The metallurgical drilling consists of a total of eight holes with a number of intersections of the different lithologies. Of the eight holes, only four have been completed and reported. The data consists of an arithmetic average of all the reported results, although each intersection is of a varying distance. SGS Lakefield has received the metallurgical drill core and has been contracted to complete the head grade analysis and basic metallurgical characterisation of the ore (namely leach rate, recoveries, etc.). Preliminary results only are available.

6.2.1 Experimental Conditions and Results

Grade of Samples and Blends

Material of the 6 different strata, as well as the Fines, was well blended and sampled to obtain a head grade analysis. After initial test work, 4 tonnes of material was blended, according to the following ratio:

• SDB Ox	8
• SDB Sulph	1
• RSC	13
• RSF	4
• D Strat	2
• Rat Grise	2

The analysis of the individual strata, calculated blend and analysed blend are shown in Table 6.1.

Table 6.1: Bulk Sample—Head grade analysis

Metal[a]	Units	Blend	Calculated blend	SDB Ox	SDB Sulph	RSC	RSF	D Strat	Rat Grise	Fines
Cu	%	6.9	7.1	8.12	12.7	5.4	7.8	6.5	10.9	7.5
Co	%	0.47	0.53	0.43	0.20	0.81	0.27	0.11	0.21	0.4
Mg	kg/t	13.5	12.8	10	11.9	4.7	8.7	58	41	17
Al	kg/t	23.5	28.6	63.9	57.8	5.1	17.7	26.4	50	32
Si	kg/t	306	319	277	264	367	334	241	251	235
Ca	kg/t	4.6	5.3	2.7	2	1.7	1.4	52.8	1.7	13
Ti	kg/t	2.0	2.5	5.1	4.7	0.8	1.8	2.1	3.8	2.6
V	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Cr	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Mn	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Fe	kg/t	6.9	7.2	6	7.8	6	7.0	6.2	21	10
Ni	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Zn	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Pb	kg/t	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.62	0.8	<0.5

[a] detection limit = 0.05% or 0.5 kg/t

The SGS results for the recently drilled metallurgical core are summarised in Table 6.2.

Table 6.2: Metallurgical Drilling—Head Grade Analysis (SGS)

2006 Drill Hole Evaluation

		Copper			Cobalt		
Virgule Deposit	**No. of Samples**	**Total %**	**% Acid Sol**	**% insol**	**Total %**	**% Acid Sol**	**% insol**
Met 001	NOT COMPLETE						
Met 002	NOT COMPLETE						
Met 003	42	5.45	4.79	0.66	0.25	0.22	0.03
Met 004	51	4.78	4.53	0.25	0.38	0.36	0.02
Met 005	NOT COMPLETE						
Met 006	44	3.66	3.44	0.22	0.67	0.60	0.07
Met 007	NOT COMPLETE						
Met 008	47	6.15	5.95	0.20	0.84	0.76	0.08
Arithmetic Average		**5.01**	**4.68**	**0.33**	**0.54**	**0.49**	**0.05**
SRK Revised		5.90			0.47		
FNSR Deposit							
Met 001	NOT COMPLETE						
Met 002	NOT COMPLETE						
Met 003	36	5.31	4.77	0.54	0.10	0.09	0.01
Met 004		Not Intersected					
Met 005	NOT COMPLETE						
Met 006		Not Intersected					
Met 007	NOT COMPLETE						
Met 008	41	5.17	4.79	0.38	0.39	0.36	0.03
Arithmetic Average		**5.24**	**4.78**	**0.46**	**0.25**	**0.23**	**0.02**
SRK Revised		5.65			0.44		
Olivera Deposit							
Met 001	NOT COMPLETE						
Met 002	NOT COMPLETE						
Met 003		Not intersected					
Met 004		Not Intersected					
Met 005	NOT COMPLETE						
Met 006							
Met 007	NOT COMPLETE						
Met 008							
SRK Revised		4.73			0.68		

The data presented in Table 6.2 are preliminary only, as drilling is ongoing and analytical results are still being finalised. Analysing the results, the leach efficiencies for Virgule seem to be 93% for Cu and 91% for Co with FNSR being 91% for Cu and 92% for Co. These results confirm that the oxide ores will leach readily.

Comminution

Drop-weight test data for the five primary samples listed above from the KOV deposit were processed by JKTech using standard methodologies. The data were analysed to determine the JKSimMet Comminution parameters. These parameters are used, together with equipment details and operating conditions to predict the SAG/Autogeneous Mill performance. The same data is used for ore type characterisation for the crusher model.

The data has been reported but has not been included in the current design process conducted by Bateman Minerals and Metals.

Mineralogy

X-Ray Diffraction ("XRD") evaluation was conducted on the six KOV samples and it was determined that the main copper bearing mineral is malachite, with cuprite, kolwezite, heterogenite, brochantite and vonbezinngite also being identified. The cobalt bearing minerals are heterogenite and kolwezite. The Scanning Electron Microprobe ("SEM") evaluation has still to be finalised, but the preliminary results are available and shown in Table 6.3 below.

Table 6.3: Mineralogical results on 6 KOV Samples by SEM

Mineral Abundance: Mass %
Samples: Various KOV ore samples.

Sample:	Product	**SDB-Oxide**	**D-Strat**	**RSC**	**RSF**	**RAT-Grise**	**SDB-Mix**
	Lablel	**MBR/06/297**	**MBR/06/296**	**MBR/06/295**	**MBR/06/294**	**MBR/06/293**	**MBR/06/294**
	Mass Flow	**100**	**100**	**100**	**100**	**100**	**100**
Fraction:	Name	**Bulk**	**Bulk**	**Bulk**	**Bulk**	**Bulk**	**Bulk**
	Average Particle Size (µm)	8.9	7.8	11.2	10.1	8.9	9.3
	Mass Flow Distribution (Mass %)	100.0	100.0	100.0	100.0	100.0	100.0
Mineral (Mass %)	**Mineral**	**Bulk**	**Bulk**	**Bulk**	**Bulk**	**Bulk**	**Bulk**
	Chalcocite±Covellite	**0.5**	**3.8**	**1.1**	**1.0**	**0.9**	**14.0**
	Bornite	0.0	0.2	0.0	0.0	0.0	0.1
	Pyrite	0.0	0.0	0.0	0.0	0.0	0.0
	Carrollite	0.0	0.1	0.1	0.0	0.1	0.5
	Other Cu-sulphides	0.0	0.1	0.1	0.1	0.0	0.4
	Total Sulphides:	**0.5**	**4.3**	**1.3**	**1.2**	**1.0**	**15.0**
	Pseudomalachite	0.2	0.0	0.4	0.1	0.4	0.1
	Malachite±Cuprite	**10.0**	**5.3**	**6.9**	**10.6**	**16.4**	**1.7**
	Heteroginite	0.2	0.0	0.3	0.1	0.1	0.0
	Kolwezite	0.2	0.0	0.0	0.1	0.0	0.0
	Chrysocolla/ CuSilinterface	2.5	0.7	1.9	3.4	1.9	1.3
	Total Cu- & Co-oxides:	**13.2**	**6.0**	**9.5**	**14.2**	**18.8**	**3.1**
	Quartz	***36.8***	***41.9***	***82.7***	***72.7***	***38.2***	***33.4***
	Mica	***38.5***	***4.2***	***0.4***	***3.5***	***16.0***	***34.3***
	Chlorite	1.2	13.3	1.6	2.8	15.7	1.5
	Kaolinite/Clay	5.1	0.4	0.3	1.3	2.1	4.2
	Total Silicate Gangue:	**81.7**	**59.9**	**85.0**	**80.3**	**71.9**	**73.4**
	Dolomite	***0.1***	***25.0***	***0.3***	***0.1***	***0.2***	***4.3***
	Magnesite	0.0	0.5	0.0	0.0	0.2	0.0
	Total Carbonates:	**0.1**	**25.5**	**0.3**	**0.1**	**0.4**	**4.3**
	Fe-oxides/hydroxides	3.5	3.1	3.6	3.8	6.5	3.4
	Fe(Co)-oxides/ hydroxides	0.0	0.0	0.0	0.0	0.0	0.0
	MnCoCuhydroxide1	0.0	0.0	0.0	0.0	0.0	0.0
	Rutile	0.7	0.3	0.1	0.2	0.5	0.5
	Total Oxide Gangue:	**4.1**	**3.4**	**3.7**	**3.9**	**7.0**	**3.9**
	Other	0.3	0.9	0.3	0.3	0.9	0.3
	Total:	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Acid Soluble Leach

Initial test work was aimed at determining the leaching efficiency of copper and cobalt, as well as the acid consumption of the different strata. This work was performed on material that had been milled to a size of 80%-75 micron. The data produced is summarised in Table 6.4.

Table 6.4: Acid Soluble Leach Test Results

	Units	SDB Ox	SDB Sulph	RSC	RSF	D Strat	Rat Grise
Acid Consumption:(1)							
Total	kg/t ore	164	123	102	181	221	198
Cu	kg/t ore	120	71	77	114	77	163
Co	kg/t ore	6	1	13	4	1	3
Gangue(2)	kg/t ore	37	52	13	4	1	3
Final pH		1.5	1.6	1.5	1.8	1.6	1.5
Final redox	mV	342	322	342	349	314	343
Leaching afficiency:(1)							
Cu	%	96	36	92	94	76	97
Co	%	89	18	93	83	61	76
SO_2 consumption	kg/t	4.9	0	4.7	0	0	3.8

1 based on head and residue analysis

2 Gangue acid consumption was calculated as: Total – Cu – Co = Gangue Acid Consumption (GAC)

Results show >90% copper leaching for SDB Ox, RSC, RSF and Rat Grise. Poor copper leaching (36%) was achieved with the SDB Sulph sample, while average copper leaching of 76% was obtained for D Strat. Between 83% and 93% cobalt leaching was achieved with SDB Ox, RSC and RSF. Poor cobalt leaching (18%) was obtained with SDB Sulph.

These results are preliminary and require optimisation.

Leach Optimisation Tests

A final blend of the different strata was based on the leaching efficiencies for copper and cobalt, as well as the acid consumption, as determined during initial laboratory tests. The ratios for the final blend were:

- SDB Ox 8
- SDB Sulph 1
- RSC 13
- RSF 4
- D Strat 2
- Rat Grise 2

Four tonnes of material was blended in this ratio to provide enough material for further laboratory scale optimisation tests and the pilot plant.

Optimisation of the leaching conditions was done on the blended sample varying grind size, temperature and solids content. The effect of variations in these parameters is presented below:

- **Grind Tests**

The same leaching method was followed on the blended material as during the initial leaches on the different strata. Conditions and results for the leaches performed on the different sized material are set out in Table 6.5.

Table 6.5: Leach, grind size: conditions and results

Item		80%-150 μm	80%-106 μm	80%-75 μm
Solids content	%	30	30	30
Temperature	°C	62	60	63
Final pH		1.50	1.58	1.57
Final redox	mV	336	341	346
Leaching efficiency:[a]				
Cu	%	90	91	92
Co[b]	%	>90	>90	>90
Leach liquor concentration:				
Cu	g/L	30.3	29.8	27.4
Co	g/L	1.9	1.8	1.7
H_2SO_4 consumption:[a]				
Total	kg/t ore	178	211	210
Cu	kg/t ore	96	97	99
Co	kg/t ore	8	8	6
Gangue	kg/t ore	73	106	106

[a] based on head and residue analysis

[b] cobalt in the final leached residue was below the detection limit of 0.05%.

Similar leaching efficiencies were obtained for the different sized material, i.e. 90-92%Cu. Co leaching of 100% was calculated. This is due to the fact that cobalt in the solid residue, after leaching, was below the detection limit of the analytical method used, i.e. <0.05%Co. It can be safely said that >90% of the cobalt present in the feed material, was leached.

These tests showed that there was no advantage, in terms of the copper and cobalt leached, in milling to the finer grind of 80%-75μm. Similar leaching efficiencies were obtained with the grind size of 80%-150μm. The acid consumption increased with a finer material, from 178kg H_2SO_4/t ore for 80%-150μm to 210 and 211kg H_2SO_4/t ore for the 80%-106μm and 80%-75μm, respectively.

Further optimisation test work was done on the coarser grind, i.e. 80%-150 μm.

- **Temperature**

The effect of temperature during leaching was investigated. These tests were done on material milled to 80%-150µm and at a solids content of 20%. The lower solids content in the leach was chosen, to result in a reduced final leach liquor composition (specifically the copper concentration) that will be suitable as feed for the downstream copper solvent extraction (SX) unit operation. Conditions and results for these tests are summarised in Table 6.6.

Table 6.6: Leaching, effect of temperature

Item		24 °C	49 °C	79 °C
Solids content	%	20	23	20
Final pH		2.2	1.6	1.7
Final redox	mV	308	338	349
Leaching efficiency:[a]				
Cu	%	88	89	92
Co	%	86	100[b]	100[b]
Leach liquor concentration:				
Cu	g/L	16	20	18
Co	g/L	1.0	1.4	1.2
H_2SO_4 consumption:[a]				
Total	kg/t ore	137	189	166
Cu	kg/t ore	94	95	99
Co	kg/t ore	7	8	8
Gangue	kg/t ore	36	86	59

[a] based on head and residue solids

[b] cobalt in the final leached residue was below the detection limit of 0.05%

Results showed that an increase in temperature, from 24°C to 79°C, showed a slight improvement in the leaching efficiencies of both copper and cobalt. An increase in acid consumption (total and gangue) was noted with an increase in temperature. A variation in the acid consumption values for the same material was picked up during the test program and this is being investigated.

- **Acid consumption**

The acid consumption of the blended sample was found to be higher than that expected, based on the acid consumptions of the individual strata, as shown in Table 6.7.

Table 6.7: Leach: acid consumption check

Sample	Test no	Final pH	Total H_2SO_4 consumption (kg/t)
Blend: 80%-150 µm	1	1.56	144
	2	1.55	136
Blend: 80%-75 µm	1	1.56	144
	2	1.53	147
SDB Ox	1	1.52	156
	2	1.56	158
RSC	1	1.55	128
	2	1.50	132
D Strat	1	1.50	244
	2	1.54	261

Total acid consumptions for the blended material varied between 136 and 147kg/t. This correlates well with the calculated value, based on initial leaches performed on the different strata, of 139kg/t. Only D Strat exhibited high acid consumptions, i.e. between 244 and 261kg/t. This is expected, due to the high dolomitic content of the material.

Further tests are underway, to provide a better understanding of variations in the acid consumption values.

Outstanding Test Work

The pilot plant test phase at Mintek as well as the metal recovery test work has still to be completed and thus cannot be reported at this time.

6.2.2 Leach: Gécamines Method

Gécamines have developed a standard method for determining acid consumptions and leaching efficiencies on ores. The main differences between the Gécamines Standard method and the method followed at Mintek are outlined in Table 6.8. A few leaches were performed, using the Gécamines Standard method.

Table 6.8: Leach: Gécamines standard method vs Mintek method

Parameter	Gécamines Standard method	Mintek method
Initial liquor	Water containing 2.5 g/L Fe^{2+}	water
	pH 1.5	Natural pH of water
pH controlling agent	500 g/L H_2SO_4	8 M (784 g/L) H_2SO_4
Reducing agent	200 g/L SMBS	SO_2 gas

Very similar results, in terms of both copper and cobalt leaching efficiencies and acid consumption, were obtained with the two different methods. Results for the leaches performed with the Gécamines method are summarised in Table 6.9.

Table 6.9: Leach, Gécamines standard method: results

	Leached (%)[1]		
	Cu	Co	H_2SO_4 consumption (kg/t ore)
Acid leach:			
SDB Sulph	51	19	130
RSC	93	93	104
D Strat	78	100[2]	301
Reductive leach:			
SDB Sulph	62	27	
RSC	93	100[2]	
D Strat	78	100[2]	

1 based on head and residue analyses

2 Co in final solid residue analysed below detection limit of 0.05%

6.2.3 Test Work Summary

The above metallurgical test work has been performed by Mintek (on bulk samples) and SGS Lakefield (on drill core). The test programmes are on going and only preliminary results have been released for review.

These preliminary results indicate that an overall leach recovery of >90% for Cu with >83% for Co is likely, depending upon lithology.

Acid consumption will be between 100 and 260kg/t, again depending upon lithology. The average acid consumption is likely to be approximately 150kg/t, but subject to additional test work.

6.2.4 Continuing test work

No test work has been planned at this stage to confirm the flotation concentrate performance in the future refinery. It is recommended that such test work be instigated at some stage to confirm the suitability of the process. This is not expected to be a concern as this concentrate has historically been process through the 'Old' Luilu Refinery.

6.3 Metallurgical Process—KOV

Leaching of copper/cobalt oxide ores has been conducted at Kolwezi for many years. Ore and flotation concentrates have been processed through the 'Old' Luilu Refinery since 1960. The technology that was employed was state of the art at the time, but has been superceded by newer technologies such as Solvent Extraction. The 'Old' Luilu Refinery does not form part of this project in any way and is not a DCP asset. The Existing Electro-Refinery was constructed in 1975, but was never commissioned and is not planned to be part of the New Processing Facility.

The New Processing Facility to be built for processing the KOV ore is expected to consist of the following unit processes:

- RoM Crushing, Conveying and Milling

- Sulphide Flotation Recovery
- Dewatering of Oxide Tailings
- Sulphide Oxidation
- Sulphur burning plant for acid production
- Agitated Atmospheric Oxide and Pressure Sulphide Leaching in sulphuric acid solution
- Thickening and Filtration of leach residue
- Leach residue repulsed and pumped to tailings dam
- PLS clarification
- Copper Solvent Extraction
- Copper Electro-winning
- Raffinate recycle to leach
- Copper removal from cobalt bleed stream with secondary Solvent Extraction
- Cobalt stream purification with lime
- Cobalt precipitation and recovery with lime
- Claiming of cobalt hydroxide to oxide

The above process has not been confirmed by completed laboratory test work nor pilot plant runs, but is a substantially proven process at other copper oxide leaching operations internationally. It is expected that test work will validate the design assumptions, but if not the design can be modified to incorporate the metallurgical test work results.

The simple conceptual process description is defined below with a basic flow sheet, as developed by Bateman Minerals and Metals. This is based on other successful copper oxide processing plants.

6.3.1 RoM Crushing, Conveying and Milling

KOV pit is approximately 6km from the chosen refinery site. Thus there are three options for crushing and milling the ore and delivering it to the downstream leach section:

- Transport of RoM from the pit to the refinery; crushing and milling at the refinery.
- Primary crushing at the pit, and an overland conveyor to the refinery.
- Primary crushing and milling at the pit, and pumping milled slurry to the refinery.

The first option has been discounted because of the logistical difficulties and cost involved. Of the other two options, conveying requires higher capex and lower opex; pumping vice-versa. Only conveying has been considered to-date, but this will be revisited before detailed design is completed.

6.3.2 Oxide Leach Process

KOV ore is predominantly oxide and, by world standards, is high-grade with an average copper grade of 4.83%Cu, whereas there are plants elsewhere in the world operating successfully with a feed grade of less than 1%.

Oxide ores are easy to leach in sulphuric acid, as is done at ‘Old’ Luilu, but acid consumption is a major operating cost. Hence it was normal to concentrate the ore before leaching, in order to remove much of the acid consuming mineral fraction (gangue) in the flotation tails. The downside was of course the high metal loss to the tails. At Kolwezi Concentrator (KZC), some 30% of the copper and 50% of the cobalt is lost (and resides on the tailings deposits).

With present day high copper prices, however, it pays to leach the ore directly without the concentrator step. At many copper plants, this leach is performed directly on prepared piles of crushed ore (i.e. heap leach). Obviously this is cheaper in terms of capex and opex than an agitated leach similar to 'Old' Luilu and hence is suitable for low-grade ore. However, the downside is lower overall copper recovery and a very slow reaction rate which limits the practical size of the operation.

In the course of the 2005 pre-feasibility study, it was confirmed that, despite the acid cost, there was a major financial benefit to be gained by leaching the KOV ore directly in an agitated leach, rather than processing it first through KZC. On copper ores, this process is being successfully operated elsewhere.

Thus the agitated tank leach was selected in principle. The feed to the leach will be milled ore, but test work is required to determine the optimum feed size and other design criteria.

6.3.3 Cobalt Leaching

Whilst copper oxides and bi-valent cobalt (Co^{2+}) dissolve relatively easily in the sulphuric acid agitated leach, it is known that the tri-valent Co^{3+} species must be reduced in order to dissolve it. Thus a variety of reducing agents are used in Cu/Co leach circuits to achieve this. At the 'Old' Luilu Refinery, copper powder and sodium metabisulphite ($Na_2S_2O_5$) are used. However they are expensive and represent a significant operating cost. In the proposed new refinery, the reducing agent will be sulphur dioxide (SO_2) gas obtained from the sulphuric acid plant. This has been shown by Gecamines[1] to be an effective reagent, although fume extraction and off-gas scrubbing are required to control fugitive gas emissions.

6.3.4 Sulphide Recovery

KOV ore is mainly oxide but it is known that there are sulphides present and that the ratio of sulphides to oxides will increase in later years as the pit is mined to greater depths.

Sulphides will not dissolve in a normal atmospheric leach, so most oxide leach plants make some provision to recover the sulphide fraction to avoid a loss of copper. A secondary reason for recovering the sulphides is that the sulphide processing route can convert the sulphur to sulphate and ultimately sulphuric acid, thus saving considerably on the overall plant acid requirement. This sulphide recovery route can be:

- Floating the leach feed to recover sulphides, followed by fluid bed roasting of the sulphides.
- Floating the leach feed followed by pressure oxygen leaching of the sulphides.
- Floating the atmospheric leach residue to recover undissolved sulphides, followed by roasting.
- Floating the atmospheric leach residue, followed by pressure leach.

All of these routes have been successfully utilised at various plants. However, in this case, it was decided to adopt the second option above because difficulties have been reported with post-leach flotation caused by fine gypsum etc. and because it will be easier to phase in the possible future addition of extra sulphides, especially pyritic sulphides, from external sources.

The choice of pressure leach, as opposed to roasting, is based on similar successful processes internationally.

6.3.5 Liquid/Solid Separation

In a leach/SX circuit, a fundamental principle is that the barren solution after SX (i.e. raffinate) is recycled to leach to utilize the acid that is regenerated in SX. Thus the solution circuit is "closed" apart from a bleed stream that is taken off for impurity removal and, in this case, cobalt recovery. This means that the amount of fresh water brought into the circuit is critical as it dictates the bleed stream volume. The primary sources of water input in this balance are:

- Moisture content of the ROM and concentrates
- Mill water addition
- Wash water applied to leach residue liquid/solid separation.

In order to address this water balance problem, some plants have resorted to milling in raffinate. This has given problems in the mill with corrosion of mill balls and even the mill itself, despite being ceramic or rubber-lined, due to the acid content and copper cementation effect. Thus, in this case, it was decided to mill in water and adopt the most effective means of de-watering the milled slurry, recycling the water to the mill. This led to the selection of horizontal vacuum belt filters, which are the most suitable filters for dewatering on a large scale. However, belt filters are expensive in terms of opex and capex and need careful control and monitoring. An essential part of the current test work programme will be to demonstrate their suitability to the KOV ore in particular.

Similarly, post-leach dewatering is a critical operation and, in this case, residue wash efficiency is very important in order to minimise the loss of entrained valuable metal in the leach residue sent to tails. Again, the horizontal belt filters have been selected for this study, in preference to the commonly used CCD thickener circuit. From previous experience, it is known that the capex and opex of these two techniques are similar, but it is suggested that this aspect be reviewed again later when all the test work results are known.

6.3.6 Copper Solvent Extraction

The leach/SX/EW flow sheet concept was initiated in the 1960's, but became more widely used in the 1980's and 1990's as the technology was proven and subsequently improved with the development of better SX reagents, particularly solvents with a high selectivity for copper over iron. It was demonstrated that high-purity electro-won cathode copper could be produced by this route that can be sold at a premium over electro-refined cathode. This is in contrast to earlier years when electro-won cathode always sold at a discount because of its poor quality.

Thus the selection of the SX process using LIX®984N as the extractant was virtually an automatic choice. It should be noted nevertheless that this will be one of the largest copper (or any other metal) SX/EW plants in the world. It is proposed to process 250ktpa copper, whereas Skorpion zinc is rated at 150ktpa zinc. Thus, at the detailed design stage, it is recommended to employ a specialist SX Technology Provider, such as BSX.

A "conventional" circuit configuration has been included in the design, incorporating 2 parallel identical SX trains. However, in recent years attempts have been made to re-configure such circuits with a view to reducing operating costs. Essentially this involves running one train as a primary high-tenor circuit, with the second train operating on lower tenor wash solutions. The claimed operating cost reductions come from:

- Increased recycling of high-acid, high-grade raffinate directly to leach.
- Production of lower copper tenors in the low-grade raffinate used as wash water, reducing the soluble copper losses.
- Decreased acid tenor in the low-grade raffinate and decreased cost of neutralising the bleed stream.

Certain operations have recently adopted this approach and it is suggested that the concept be explored further during detailed design, possibly by BSX.

6.3.7 Electro-winning Technology

Bateman has designed several copper electro-winning plants and could do so again without external technological input. However, on a refinery of this size (250ktpa), the stripping and handling of almost 6,400 cathodes per day represents a major task. In small refineries, stripping can be accomplished by manual labour. In medium sized plants, a semi-automated stripping machine is normally employed, in which the stripping is mechanised but the machine is controlled by an operator. In large plants, such as this proposed New Processing Facility, it is best to fully automate the whole process to cope with the high volume. However, an automated stripping machine is a highly specialised piece of equipment, as is the automated cathode handling crane. This type of automated process was employed in Namibia, where the specialised technology was provided by Umicore Engineering from Belgium.

For copper EW, there are two companies in the world who specialise in this technology incorporating the use of stainless steel cathode blanks:

- Kidd Creek (Falconbridge, Canada)
- Xstrata (MIM, Australia)

Bateman has worked successfully with Xstrata in the past and recommends employing them here as Technology Providers. This will involve expenditure in the form of a Licensing Fee and know-how Agreement in addition to the capex for the stripping machines, cranes and cathodes supplied via Xstrata, but are considered worth the expenditure in order to have the advantage of world leading proven technology.

For the purpose of this study, it has been assumed that Xstrata will act as Technology Providers, but this aspect must also be re-visited in the value engineering phase. At 250ktpa, the KOV project is almost double the size of the largest Xstrata EW plant to-date.

6.3.8 Cobalt Purification and End-Product

The end-product for cobalt has been debated at length with the owners. There are various possibilities for the final product, namely:

- Standard-grade cobalt cathode (>99.3%)
- High-grade cobalt cathode (>99,8%, vacuum de-gassed)
- Cobalt carbonate salt
- Cobalt sulphide salt
- Cobalt oxide/hydroxide salt
- Cobalt sulphate

Ultimately, because of the high tonnage of cobalt involved in this project, it was decided to go for a high-purity salt route, initially at least. This route will require the least capital expenditure and also avoids the need for:

- Cobalt SX with Cyanex 272—which is a very expensive reagent; prone to upsets due to pH variation and oxidation (e.g. manganese), and
- Electro-winning of cobalt—which is a difficult process to operate successfully, especially on a large scale.

Production of carbonate was discounted because of the expense and logistical difficulty involved in sourcing the required amount (150tpd) of sodium carbonate reagent.

For similar reasons the production of sulphide, which requires the expensive sodium hydrosulphide (NaHS) reagent, was discounted.

Cobalt sulphate can be produced in a crystalliser plant, using steam as the heating medium. This may be a possibility on a limited scale, when the amount of steam available is confirmed during detail design.

For oxide/hydroxide production, a large amount of burnt lime is required. However, this reagent is required in any event for the cobalt circuit and the plant will inevitably be set up for bulk handling and slaking of burnt lime. Thus the oxide/hydroxide production route was selected for design purposes—to be confirmed later during value engineering.

6.3.9 Manganese Removal

The major impurity in the cobalt bleed stream, apart from iron which is easily removed by neutralising with limestone slurry, is manganese. Manganese is present in most cobalt ores and in this case is present at high levels in the Tilwezembe ore concentrate from KZC, so a dedicated manganese removal step is required, even for a low-grade cobalt product route.

Amongst others, Gecamines, Congo Minerals and CSIRO have demonstrated that an air/SO_2 mixture is effective as an oxidising reagent that will allow the precipitation of manganese without significant co-precipitation of cobalt. MINTEK have also investigated this technique on similar Cu/Co studies, and will research it further during the current test work campaign.

In the current flow sheet, the addition of air/SO_2 is included in the impurity removal stage aimed at residual iron, aluminium and arsenic removal, by adding Milk of Lime (MOL) to pH4.1.

6.3.10 Conceptual Process Flowsheet

The conceptual process flow sheet as developed by Bateman Minerals and Metals is shown in Figure 6.1. This is based on the description detailed above.

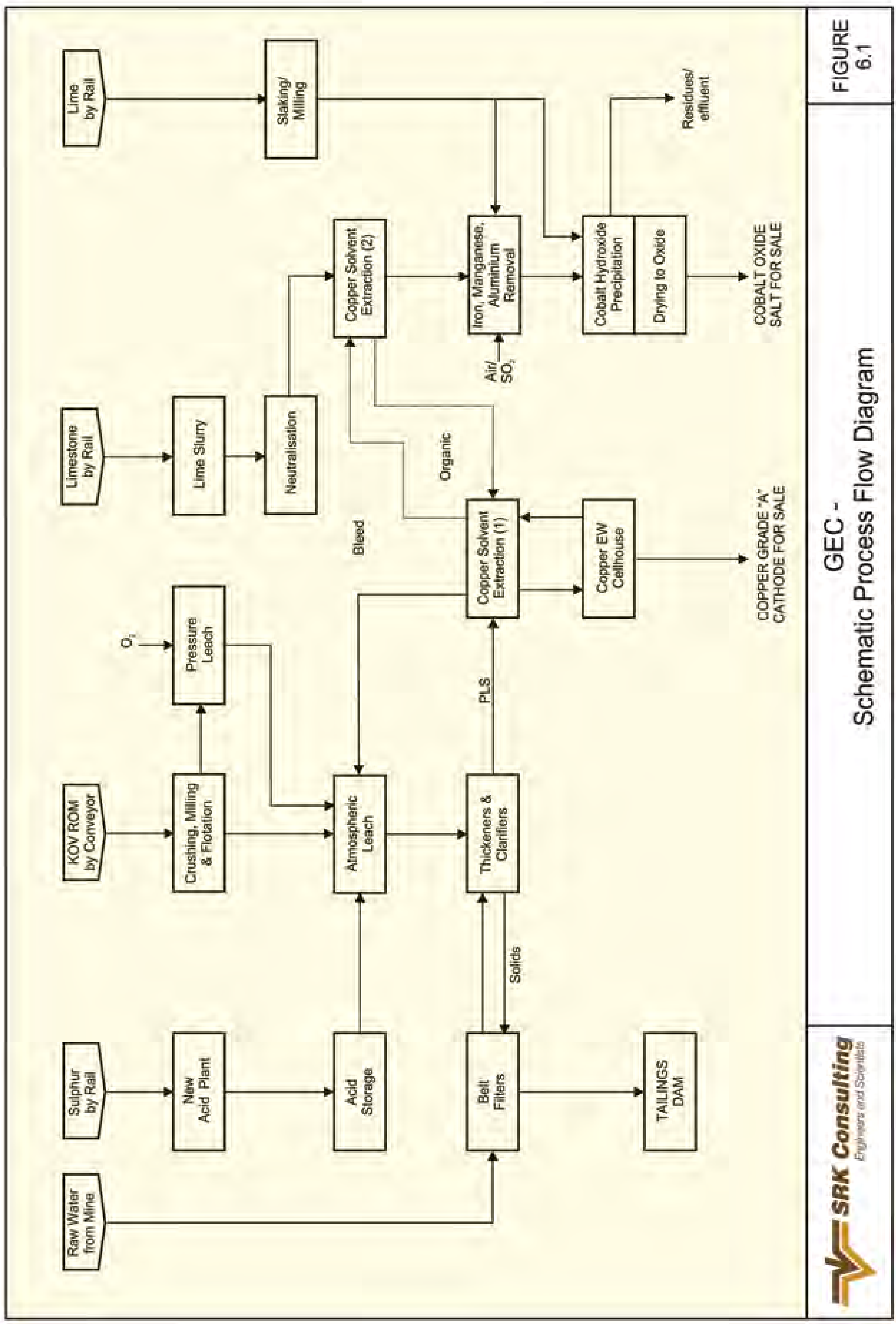
The basic design criteria upon which the design is based are regarded as reasonable, subject to the successful completion of the test work. From these design criteria, a full set of preliminary Process Flow Diagrams and Plant Layout Drawings have been completed and a detailed mechanical equipment list has been developed to enable the project to be adequately costed.

The details have included the sulphide flotation and pressure oxidation sections as well as the treatment of KOV.

The selected process route is regarded as reasonable and is similar to technology used elsewhere in the world, but will need to be optimised subject to the results of the ongoing test work, currently being undertaken. The detailed design is awaiting the test work being completed at Mintek, and thus the overall project is based on previous design work.

Whilst this is far from ideal, the design is sufficiently robust to be regarded as reasonable.

Figure 6.1: Schematic Process Flow Diagram



6.3.11 Plant Location

It has been stated by DCP that the siting of the New Processing Facility next to the 'Old' Luilu Refinery, is not ideal but is the only large area available for this project. There is a potential risk that pollution of the environment (or other incidents) from the 'Old' Luilu Refinery being attributed to the New Processing Facility. The JV agreement is reported by DCP to exclude any historical pollution as a result of the plant location, but this could be of concern to the project and a detailed base line environmental study is required with on-going monitoring.

6.3.12 Tailings Disposal

It is stated by Bateman that the new tailings dam to retain the leach residue from the New Processing Facility will be located some 13 km from the refinery site. This facility will be a new construction. There will be double pipelines to the dam with a maintenance access way. This is regarded as adequate and appropriate.

6.4 Process Performance

The process performance as detailed in the design criteria can be summarised as below:

- Plant Availability; 93%
- Plant Operation; 24 hours per day, 365 days per year
- Copper Recoveries; 92.9% based on historical calculated values from the 'Old' Luilu Refinery
- Cobalt Recoveries; 84.9% based on historical calculated values from the 'Old' Luilu Refinery

These performances are comparable to that being achieved from the current preliminary test work results, but the final reports are awaited when the test work is completed.

The capital programme currently calls for the deferment of the sulphide flotation circuit and leaching plant from the flow sheet for a period of one year. The mineralogical review indicated above shows that there is potentially a significant proportion of the copper mineralogy in the form of sulphide minerals, which will not leach in the oxide circuit. Considering that 13% of the anticipated mineral deportment remains as sulphide, the delay in the commissioning of the sulphide circuit for only one year will not be significant and will allow training of the staff to be staggered.

The current Bateman flowsheet has recoveries of 91% for copper and 88% for cobalt during the leach with the SX-EW recovery being 100% as well as the cobalt precipitation. The copper recovery is regarded as possibly low when considering the results achieved above. The cobalt recovery is somewhat higher than would be expected. It is SRK's view that the refinery cobalt recovery be reduced to approximately 85%, as indicated above. Copper recovery could be increased to 93% but subject to the remaining test work results. SRK has applied a process recovery of 85% for Cobalt and 91% for Copper in the model.

6.5 Process Infrastructure

The following process infrastructure will be required for the new leach plant:

- Raw Water and Potable Water supply
- Plant Fire Protection
- Steam, both high pressure and low pressure
- Demineralised Water supply
- Cooling Water supply
- Oxygen Plant (for the Pressure Leach)
- Compressed Air
- Fuel Supply
- Accommodation and Transport
- Process Buildings and Facilities
- Non-Process Buildings and Facilities
- Laboratory
- Office Automation and Communications

- Earthworks and Terracing
- Plant Roads and Stormwater control
- Fencing
- Rail Sidings
- Weighbridges
- Waste water and Sewage treatment Facilities
- Tailings Dam
- Power Supply
- Off-site Infrastructure

These items have been included in the cost estimate for the new leach plant and as such are regarded as a fair and reasonable approach.

6.6 Refinery Manpower

The manpower complement for the Refinery as proposed by Bateman is 468, with 37 expatriate employees and the remaining local employees. The spread of staff is as below:

- Process G&A— 11 international and 32 national
- Operations— 186 national
- Maintenance— 11 international and 74 national
- Administration:
  - General— 1 international and 1 national
  - Security— 1 international and 58 national
  - SHE— 4 international and 8 national
  - HR— 2 international and 23 national
  - Accounting— 2 international and 8 national
  - Procurement— 1 international and 21 national
  - Engineering Utilities— 4 international and 20 national

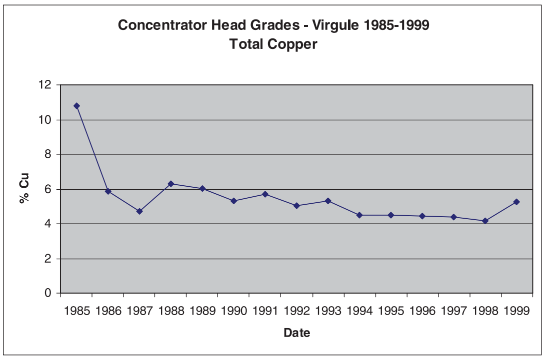
These numbers of employees seem fair and reasonable for the refinery management, including the head office function.

The operating skills are expected to be available within the Kolwezi community as Kolwezi Concentrator and the 'Old' Luilu Refinery have operated in the past. All employees will need to be retrained as the new refinery is significantly different from the previous operations. Training will be a critical issue on the refinery.

6.7 Historical Metallurgical Performance

No historical metallurgical performance data have been made available and any data would be regarded as of little value to the new refinery, as the process route through the 'Old' Luilu Refinery, with oxide flotation at KZC, was different to what is envisaged now.

Head grade information provided (Figure 6.2) is for the KOV deposit and should be comparable to what will be expected when mining recommences.



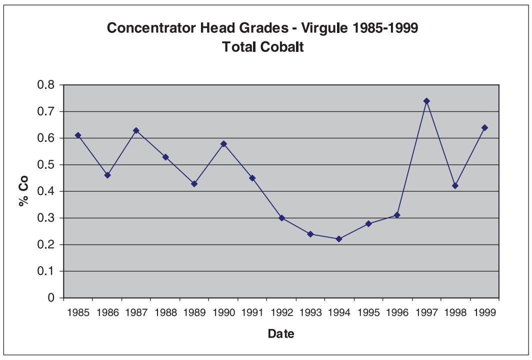


Figure 6.2: Historical head grades from KOV

6.8 Product Quality

It is expected that the refinery will produce approximately 250ktpa of Grade A copper cathode and 27.5ktpa of cobalt as a hydroxide or oxide salt. The quality has been based on similar operations, but has not been confirmed by test work on the particular ores from KOV.

6.9 Capital Expenditure

The current capital budget is for $US580.6 million, excluding the following items from the basic flow sheet presented earlier—

- Sulphide flotation and associated reagent plant— $US10.9 million
- Pressure Leach on sulphide concentrates— $US23.3 million
- Potable water and Oxygen Plant— $US11.0 million

It has been stated in the Bateman Report that the sulphide flotation and leaching circuit has been deferred. It has been stated by GEC that this deferment is for 12 months only and not eliminated from the process flow sheet. The sulphide circuit is to be commissioned one year after the main plant is commissioned. The amount to be included (as per Bateman's) in the future capital model is $US61.3 million, which is some 35.6% more than the take out price detailed in the current budget, which caters for the continued fees associated with a longer period on site plus the contingency, and is regarded as reasonable. It is stated that deferring this capital will assist the cash flow and enable a more phased approach to training of the staff—this is fair and reasonable.

In addition, a saving of $US10.8 million has been included to allow for procurement from 'low cost' centers such as India and China.

The basic criteria for this capital cost estimate has been the detailed equipment list, vendor quotations and an exchange rate of R6.50 = $US1.00. This exchange rate has had a significant effect upon the overall capital expenditure. It is stated by Bateman that the costing is based in May 2006 money terms and is accurate to -5% +15%.

Table 6.10 summarises the capital for the Refinery, based on area costing. The capital cost is regarded as reasonable. A contractor contingency 0f $US56.5million has been included in the estimate.

Table 6.10: Process Plant Capital Expenditure Estimate

Description	Capital Expenditure (US$m)
Milling, etc	44.16
Leach	80.61
Copper SX	48.20
Acid Plant	2.83
Copper EW	113.54
Cobalt	17.17
Utilities	6.87
Infrastructure	24.46
E&I and Piping	21.80
Sub-total	**359.66**
Office, Indirect, Fees and Other Costs	113.13
Acid Plant	51.32
Contingency	56.50
Total Plant Capital	**580.60**

6.10 Operating Costs

The accuracy of the costing exercise is stated as -5%+15% and is said to have been based in May 2006 money terms.

The fixed cost portion of the refinery operating cost includes labour, General and administration, maintenance and operating supplies/spares, laboratory and mobile equipment. The variable cost portion includes freight, power, reagents and consumables, Sulphur, limestone and quicklime, grinding media and liners, solvent extraction, EW cell house and royalties. The costs have been detailed and summarised on per tonne of copper, per pound of cobalt and per tonne RoM basis.

The operating costs as reported by Bateman, but modified for KOV only, are detailed in Table 6.11, for the refinery operating at steady state under normal controls with expected tonnage being achieved.

Table 6.11: Process Plant Operating Cost Estimate

	Annual Operating Cost (US$million)		
Description	**Total**	**Attributable - Cu**	**Attributable - Co**
Power	26.14	24.89	1.25
Water	0.01	0.01	—
Consumables	85.14	66.02	20.00
Labour (fixed)	9.54	8.37	1.16
Maintenance & operating spares (fixed)	17.38	15.66	1.72
Laboratory (fixed)	7.51	6.82	0.69
Mobile equipment (fixed)	1.96	1.79	0.17
Freight	—	—	—
General and Admin (fixed)	3.04	2.74	0.30
Environmental management	1.00	0.90	0.10
Miscellaneous	3.75	3.38	0.39
Royalties, licence fees	—	—	—
Total	**155.46**	**130.57**	**25.77**
Split:			
Fixed	40.43	36.28	4.15
Variable	115.03	94.29	21.62

Analysing the above, a total cost of $US40.43million can be regarded as the fixed cost portion of the costs or about 26% of the total costs. The remainder is a variable cost portion.

The above cost estimate has been based on processing KOV material and concentrates from KZC. Removing the KZC effects on the refinery, the refinery operating costs for the KOV project alone will be US$32.39/t of ROM feed or US$563/t of copper cathode and US$0.54/lb of cobalt. These costs are are considered to be reasonable.

The costs are detailed on a first principles basis on unit consumptions with appropriate costs applied to the consumable. The methodology is appropriate and is based on other operations, as the test work results are not available for this KOV ore.

6.11 Programme

The programme for the construction of the refinery from date of award is slightly over three years, including full commissioning. This is regarded as fair and reasonable for a plant of this complexity. The build up after commissioning has been modified to 24 months to achieve full name plate capacity and this is now regarded as fair and reasonable.

6.12 Material Contracts

6.12.1 Kolwezi Concentrator

The Kolwezi Concentrator (KZC) is currently running at a significantly reduced capacity compared to nameplate (nominally 4Mtpa), due to a lack of funds and proper repair and replacement of equipment. The KZC facilities were initially designed to simultaneously process two very different types of ores, i.e. oxide ores (malachite) with a siliceous gangue, and mixed ores (malachite/chalcocite) with a dolomitic gangue.

Since the concentrator first began operation in 1940, the source of ore has changed, as well as the modernisation of mining equipment and the construction of new concentrators and metallurgical plants. It was then decided only to process oxide ores at KZC, i.e. those with a siliceous or dolomitic gangue, and these in variable proportions.

In the late 1980's, early 1990's an upgrade programme was instituted, which would improve production to nameplate capacities. Due to continued civil unrest, force majeure was declared by the engineering company in 1991, and the project was abandoned. Thus incomplete construction and incomplete installation were left "as is". Purchased equipment was locked in containers and has been there ever since the time of the abandonment.

Since then KZC has been processing less and less. Maintenance has been neglected largely due to the lack of funds. The plant has since then been "self maintained", i.e. sections of the plant were cannibalised to keep other sections running, homemade "make-shift" equipment were used as substitutes, and certain streams were re-routed or diverted. This perseverance has ensured that KZC at least keeps operating.

Currently KZC operates at a reduced availability treating between 1,000 to 2,000tpd. Milling, thickening and filtration are the current bottlenecks within the concentrator, the other being the infrequent delivery of RoM ore to the primary crusher and reagents for flotation.

Figure 6.3 shows the historical production data for the KZC plant. It shows a steady decline in production from full capacity in 1986.

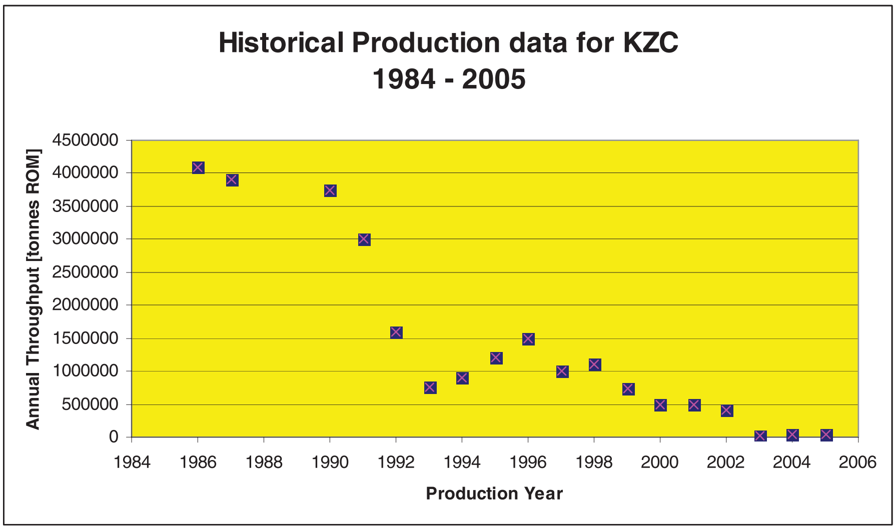


Figure 6.3: Historical production for KZC

Metallurgical recovery at KZC has typically been 70% for copper and 50% for cobalt.

The concentrator is divided into 2 distinct sections; Section 1 is used for the treatment of siliceous ores and section 2 for the treatment of dolomitic ores. The difference between the types of oxide ores is determined by the ratio of Cu to CaO (soluble). A ratio higher than 15 is typical of siliceous ores and lower than 15 is typical of dolomitic ores. Both sections undergo 3-stage crushing, with the primary and secondary crushing in open circuit and the tertiary crushing in closed circuit.

Crushed ore (i.e. mill feed) is stored in a covered stockpile that provides a buffer between crushing and milling. Both sections undergo 2 stages of milling with the rod mills being used as the primary milling stage and ball mills as the secondary milling stage. Primary milling is in open circuit, while secondary milling is in closed circuit with classifying cyclones.

The milled ore is then conditioned before being pumped or gravity fed to their respective flotation sections. The flotation process routes in section 1 and section 2 are different due to the possibility of treating dolomitic and mixed non-dolomitic ores in section 2. Section 1 is setup for the treatment of siliceous ores with roughing, scavenging and cleaning flotation cells. Section 2 has additional sulphide pre-float roughers and cleaners. Section 2 is split into two identical lines for the flexibility of treating siliceous, dolomitic or mixed non-dolomitic ores.

Concentrate produced in both sections is thickened and filtered separately. Siliceous and dolomitic concentrates (filter cake) is stored separately on open storage pads. Filter cake is then transported to the refinery by rail.

Tailings from both sections are pumped to a central tailings pump station from where the tailings flow to tailings disposal. KZC tailings are currently discarded into the Musonoi River system and eventually accumulate in the river and on the river banks. A new tailings dam site has been identified for future disposal.

Reagents

Oxide copper flotation has a complicated suite of reagents compared to that of pure sulphide copper flotation. The reagent suite as well as the dosage rates for copper oxide flotation has been based on many years of research and development. The current reagent regime as employed by KZC is indicated in the table below.

Table 6.12: Reagent Consumptions

Reagent	Ore	
	Siliceous Ore	Dolomitic Ore
	(g/t)	(g/t)
Rinkalore 10	1200	60
Gas Oil	400	260
Tall Oil	50	0
Sodium Carbonate	600-700	0
Sodium Silicate	700-1000	250
Sodium Hydrosulphide	0	2500-4000
PNBX	0	200-250
41G Frother	0	60
Separan AP45	3	3
Magnafloc 351	2	2
Sulphuric Acid	300	300

6.12.2 Shituru refinery

The Shituru Refinery remains under the ownership and management of Gècamines. Under the terms of the Joint Venture Agreement with GEC, DCP have first right to treat concentrate through the plant to produce copper and cobalt cathode. It is proposed that concentrate will either be sold to third parties or the concentrate will be shipped from KZC by road and/or rail and treated under a toll treatment agreement at Shituru. GEC has the right to take over the management of the refinery if deemed appropriate.

A simplistic diagram of the Shituru "Hydro" flow sheet is shown in Figure 6.4.

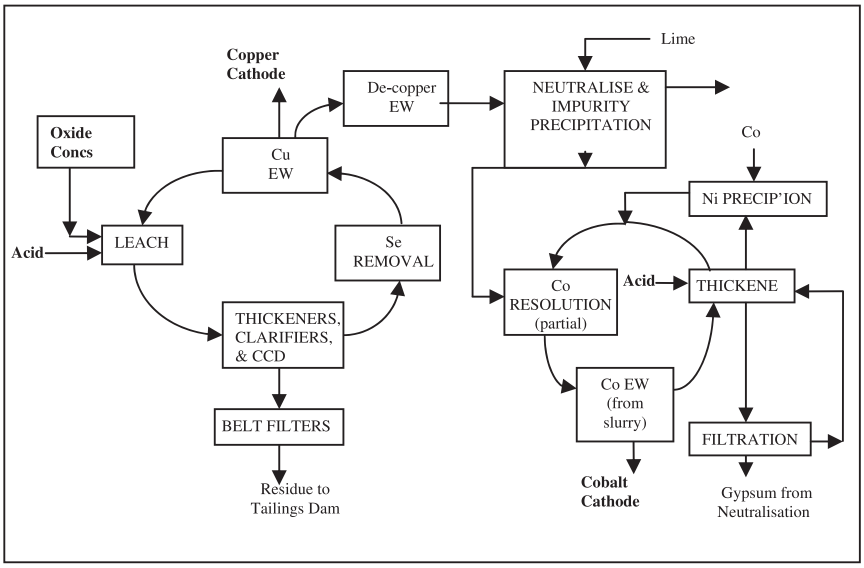


Figure 6.4: Simplified diagram of Shituru "Hydro" flow sheet

The main features of note in this flow sheet are:

- As with all old copper EW plants, with no SX between leach and EW, the grade of copper cathode produced is low—well below the LME Grade A standard. As such, it sells at a discount. It is, however, possible to refine the cathode to wire-bar standard in the adjacent Thermo facility reverberatory furnaces.

- The leach reactors are air-agitated pachucas, rather than the mechanically agitated vessels used nowadays. The downside of this is slightly reduced extraction efficiency.
- Sulphuric acid is supplied from sulphur-burning acid plant(s) located on the same site as the refinery.
- Lime is a major and critical reagent. It is supplied from the Kantontwe lime plant near Likasi.
- The roaster was a relatively recent addition to permit some sulphide concentrate treatment and reduce the acid requirement. It is currently inoperable but it is not required for the Tilwezembe/ Kananga concentrate.
- The belt filters on the leach residue stream were added recently to improve overall recoveries of copper and cobalt, by reducing soluble losses.
- The cobalt circuit is complex and inefficient, resulting in cobalt losses in excess of current day standards.
- The cobalt EW plant is unique in the world, producing cobalt from slurry. The grade of cobalt produced is well below current "high-grade" standards but can be refined in arc furnaces in the Thermo section. Cathode sells at a discount.
- The Thermo furnace operations involve extra costs and losses of metal.
- Power is in short supply at Shituru and it is likely that increased throughput especially that of electro-won cobalt would be constrained.

The stated name-plate capacities and historical parameters of key components of the process are illustrated in Table 6.13.

Table 6.13: Principal Parameters of Shituru plant

Section	Capacity
Oxide Concentrate Reception	1 200 – 1 500 tpd
Sulphide Concentrate Roaster	120 t/pd
Leach Plant Capacity	30 000 tpm
Oxide Concentrates Grade	
Cu	10 - 25%
Co	1.5 - 2%
Copper EW Capacity	135 000 tpa
Cobalt EW Capacity	7 200 tpa
Acid Consumptions	3.5 t/t Cu & 8.5 t/t Co
Lime Consumption	10 t/t Co
Recovery of Copper	85%
Recovery of Cobalt	65%

It should be noted however, that due to the poor state of repair of the Shituru facility, less than half of the throughput name plate capacity is achievable and recoveries are significantly lower than indicated.

6.12.3 Lime Plant

The lime plant is situated at Katontwe which is 6km north-west of Likasi and is accessible by both road and rail. The plant has been in operation since 1929 when production of lime commenced. The operation has been extended with the addition of two rotary kilns and cement grinding since commissioning. The limestone quarry that is located one kilometer from the lime plant.

The New Processing Facility will utilise approximately 200ktpa of crushed limestone and 85ktpa of burnt lime. This means that in excess of 300kt of quality limestone will be mined per annum for the New Processing Facility.

A resource of 15Mt of limestone grade material is claimed by Gécamines. Claims of a further 50Mt resource were made by the operators, but not substantiated. The local geology suggests that the claim may be accurate, but will necessitate acquisition of at least one neighboring property to access them. A bankable feasibility study will require a far more detailed inspection and further investigation will be required to confirm a reserve.

The supply of limestone is unlikely to be problematic to the DCP Project.

The mobile quarry plant is both inadequate for the planned volumes and at more than twenty years old is now obsolete. A dual set of problems exists here with; in the first instance, the existing fleet will need to be increased and replaced. In the second instance, considerable effort will need to be put into overburden removal. In summary, this lime operation will require a considerable investment in mobile equipment.

The lime production plant is well preserved considering its vintage and also well maintained considering the slender budget available. During an inspection by Bateman, a number of items were identified as problem areas with the major ones being:

- The electrostatic precipitator filters;
- The refractory linings;
- The crushing plant.

Some 2,000t of storage is available with another 300t of emergency storage. This is considered inadequate for the requirement of DCP and additional storage of at least 2,000t is necessary.

Kiln Production Capacity

The following capacities were obtained from the operation:

Unit	Design capacity (tpd)	Actual capacity(tpd)
Kiln1	100	100
Kiln 2	180	150
Kiln 3	250	180
Total	**530**	**430**

The refurbishment of all three kilns is essential to maintain the current capacity as stated. This capacity is really only adequate only for the New Processing Facility and no other projects in the area.

The production cost for the different products from Katontwe are estimated in the following table. These costs are based on data obtained from the Katontwe and are not considered to be very reliable and only a qualified indication at best.

	Cost (US $t)	
	Current	Projected
Limestone quarried	10,6	4,3
Limestone crushed	20,2	8,7
Burnt lime	93,0	67,0

The largest cost component is the supply of energy to the kilns. Coal is sourced locally and is of inferior quality and needs to be upgraded.

The potential production capacities (per annum) and achieved production figures as provided by Gécamines are detailed in the following table (tonnes):

	Capacity	Achieved
Limestone	780,000	494,994 (1979)
Quick Lime	150,000	123,509 (1981)
Hydrated Lime	7,000	
Cement	150,000	
Overburden		550,445 (1981)

The current production (per annum) is considerably lower than achieved and any increased production will require considerable investment.

Conclusion

The Katontwe Lime quarry and Process Plant has the capability of providing the required quantity of limestone and burnt lime to the New Processing Facility, subject to the injection of capital funds for mining equipment and plant refurbishment. The quality of the energy source (coal) should be upgraded to improve the kiln availability.

This plant does not form part of the DCP Project, but is an essential source of two of the major reagents to be used in the process.

6.13 Comments, Risks and Opportunities

6.13.1 KOV

- Metallurgical test work is based on samples which were mined 8 years ago as fresh samples are not available due to the flooding of the pit,
- No test work seems to have been planned on the KZC concentrates,
- Metallurgical design currently based on previous experience,
- Location of the new Refinery is next to the old refinery and the possibility of pollution is considerable,
- Capital Cost changes as a result of the findings from the metallurgical test work,
- Delayed implementation of the sulphide circuit from the capital expenditure,
- Operating costs are not based on the metallurgical test work but similar operations,
- The refinery cobalt recovery in the Bateman flowsheet is 88% whereas the current testwork is indicating less than 85%. It is recommended that the recovery be reduced to 85% until the test work confirms the recovery. SRK has applied a process recovery of 85% for Cobalt in the model,
- Copper recovery could be increased to 93% but subject to the remaining test work results. SRK has used a process recovery of 91% for Copper throughout in the modelling,

7 TAILINGS DISPOSAL

This section presents a review of the tailings disposal facilities proposed and highlights any potential risk(s) and subsequent consequences, and to make appropriate comments regarding such potential risks.

7.1 Information Supplied

The following information was supplied:

- A LoM Plan (105Mt of tailings over a period of 25 to 27 years);
- A monthly generation of 380ktpm with a ramp-up period of 2 years;
- The directive to consider three possible tailings disposal sites; and
- Gécamines topographical survey plans obtained from the Kolwezi Mine Survey Office.

Other available information was the latest Google Earth Satellite photographic image of the Kolwezi mining area.

7.2 Design Criteria

It is noted that the tailings dam and return water dam will be unlined facilities. This is based upon the assumption that both the tailings and the slurry water will not impact on the environment through pollutant generation, either separately or jointly. In the terms of reference, SRK draws attention to the requirement to review the physical and chemical characteristics of the tailings material. At the time of report generation, the physical and chemical properties of the tailings material have not been established. This requirement is now under way (June 2006). The SRK report makes mention of the lack of such information and has adopted assumptions based on earlier involvements in copper tailings disposal.

SRK has adopted a realistic conservative approach in this regard. From past experience, SRK is confident that if all the environmental requirements are met, and that the envisaged tailings stream is similar to the current concentrator deposits noted at Kolwezi, then the proposed tailing dam philosophy will work.

7.3 Site Selection and Capacity Evaluation

The site selection also addressed potential Zones of Influence in terms of South African risk management requirements (SABS 0286:1998). The SA Code of Practice was developed as a consequence of the Merrispruit Tailings Dam failure. The risk overview is sufficient for this level of study, however, once the final site has been chosen a detailed site inspection within the surrounding Zone of Influence ("ZoI"), must be carried out as well as a follow-up desktop study and inclusion in the detailed design report for the future tailings dam complex. ZoI protocols will also be necessary in the site's code of practice and operating manual.

Details of the anticipated capacities of both the Far East and Far West Sites were investigated. Both sites are capable of accommodating the anticipated 105Mt as given in the LoM Plan. From the description of the sites, it is noted that the Far West Site will be easier to manage, as the topography is flatter and more uniform than the Far East site.

Mention is again made for using assumed materials characteristics of the envisaged tailings product. A large data-base covering various aspects of copper tailings is quoted in the included references from various sources. The design details which follow on, have been based on the aforegoing data-base (information source) as well as the experience in tailings dam design. The design details briefly address all general aspects of the tailings dam.

The use of tropical soils as a structural material is also dealt with, ie in the starter wall and return water dam wall. Cognisance must be taken of SRK's observances in this regard. (Past experience has shown that when these observances with regard to tropical soil handling and usage have been breached, that walls constructed from tropical soils have failed. It has also been noted that post failure risk is significantly increased as well as large expenditures on remediation, which could have been avoided if sound observances were followed in the first instance).

A water balance is included on the tailings dam complex. As a significant amount will, as a consequence of rainstorm events, spill into nearby river systems, it is essential that the discharge meets environmental standards. Because mainly of discharge and accidental spillages, SRK does not recommend the Far East site, if possible, as the river systems in close proximity the Far East discharge into Lake Nzilo. The Far East Site and surrounding rivers form a "greenfields" area.

7.4 Quantities and Feasibility Costing

SRK has allowed for the capex in phases, mainly for earthworks and filter drain quality considerations. The total envisaged capex for the tailings dam complex is US$31.79 million.

Opex is split into two sections, namely "daily, weekly and monthly" requirements as well as other "operating costs" which do not occur as frequently as the aforegoing, neither can they be accurately quantified beforehand. These items are better budgeted for on an annual basis in a lump sum form to be spent on an as-and-when basis.

For the purpose of this study it is recommended that:

- "daily, weekly, and monthly" opex be based on a tonnage disposal rate of US$0.29/t at a throughput rate of 380ktpm. This amounts to US$1.33 million p.a.;
- other "operating costs" considered as opex, should be covered initially by an annual budget estimate of US$0.4 million;

The closure and aftercare costs associated with the tailings dam complex are contained in the environmental section (Section 11).

It should be noted that the tailings disposal report (in the water balance section) recommends that a sum of US$0.3 million be set aside for the possible construction of a clean water dam, as the existing fresh water dam in the basin of the return water dam is likely to be inundated.

7.5 Risk Assessment

This highlights the need for a comprehensive detailed design in order to reduce the risk of any potential problem not being thoroughly investigated. In particular, the following aspects will need to be determined to enable detailed design to occur: the tailings characteristics (physical and chemical), site founding conditions, properties of local construction materials, geohydrological and geochemical considerations as well as local hydrological conditions.

7.6 Conclusions

- From the site selection exercise, the Far West Site is considered as the first choice, then the Far East, then the Kakifulwe River.
- The LoM tonnage can be accommodated.
- The pumping distance to the Far East is about 6km longer than that to the Far West, which is the worst case scenario.
- A phased construction CAPEX of about US$31.79 million is estimated for all 5 phases.
- Part of a pine forest will be lost if the Far West Site is utilised—but it is not considered a major problem.
- The handling and use of tropical soils must be carefully observed—especially the construction window of only 4 months for the construction of large earth walls, namely the starter wall and return water dam wall.
- An envisaged construction programme is detailed.
- An up to date topographical survey is required for the detailed design which is currently under way.

8 ENGINEERING INFRASTRUCTURE

This section includes discussion and comment on the infrastructure-related aspects at and around the town of Kolwezi and the associated support infrastructure to DCP necessary for the execution of the LoM plans.

The DCP infrastructure at Kolwezi examined during the site visit included the roads, airport, KOV site, Diesel workshops, fuel storage, KZC plant, waste handling, the Kolwezi substation, Gécamines workshops and Luilu Substation.

8.1 Primary Access

8.1.1 Roads

The condition of the road infrastructure can be summarised as follows:

- Local roads

  The local roads service both the local community and mine traffic. These roads, constructed of in part asphalt and in part gravel, pass through villages, towns and mining operations. They are generally in very poor condition due to potholes, erosion and surface failure. Storm water drainage is inadequate and local drainage channels do not function.

- Haul roads

  The KOV haul road to the KZC Concentrator is in reasonable condition and can be upgraded to accommodate the heavier haul requirements.

- Regional roads

  The main regional access road to the site is via Lubumbashi. The road section to Nguba is in particularly poor condition. The condition of the bridges is reported to restrict transport loads to 65 tonne. The repair of the Lualaba River barge is required to prevent further limitation in load capacity. The barge repair is in progress.

The necessity for road upgrade has been identified in the DFS and substantial steps have been described for the road improvements. This encompasses the local and haul roads and the regional road to Nguba. These steps will accommodate the short/medium term only. Further design work and costing is required to improve the costing for the roads. Longer term steps will require heavier duty road construction and sealed roads.

Provision has been made for ongoing maintenance.

Dust generation management will be required on an on-going basis, particularly in the villages, to ensure adequate road safety.

The road management will be a challenge in terms of optimising upgrading and maintenance costs, safety and production. The DCP approach is reasonable for achieving this end and bold in its approach of improving the regional roads. The long term maintenance and repair of the roads, particularly the regional road remains a significant risk, in that the control of key factors affecting the roads such as traffic density, overload and weather cannot be controlled by DCP.

8.1.2 Rail

DCP has a substantial rail infrastructure on the mine linked with the SNCC rail network (Figure 8.1). Part of the network is electrified. This network will be used for the supply of raw materials to DCP (diesel, sulphur, limestone), the transportation of materials on the mine (copper and cobalt ore) and the delivery of final products from the mine (concentrate, copper and cobalt). The network is important due to its location/layout and its independence from the road infrastructure.

The track will probably be used for the transportation of ore from Tilwezembe and Kananga to KZC Plant. Road haulage is only anticipated for any ore from Kananga destined for treatment at the KZC Plant. This will minimise ore haulage along roads which will improve operational safety. The back-up option of road hauling ore is not recommended due to the operational risk of hauling in the vicinity of children and vehicles being unsighted due to dust.

The network is in poor condition but can easily be brought up to standard by replacing damaged equipment, re-ballasting, realigning tracks and drainage improvements etc. Some extensions to the network have been identified such as the spur to the New Processing Facility.

Weigh scales are to be to be added for materials management.

The DCP fleet includes numerous rail locomotives in a poor state of repair. Provisions have been allowed for the refurbishment of selected locomotives. Verification of these amounts is required. However the quantum involved does not constitute a substantial risk.

The funds estimated for the rail upgrade are reasonable for the purpose.

8.1.3 Airport

Kolwezi airport is a substantial airport suitable for medium sized aircraft. The airport allows rapid access into the region. The runway (1,750m x 30m) will require some maintenance in the short term.

8.2 Workshop and Support Facilities

Three major workshop facilities exist at and around DCP.

The first is the existing DCP owned diesel workshop near the KOV mine. This infrastructure is directly available for the mining operations. The facility includes a substantial diesel machinery workshop (two buildings with 22 maintenance bays), wash bay area, spares storage, offices, and fitter shop. It is in good condition and equipped with excellent facilities. The workshops needs to be fenced and the security improved. These workshops will be used up to the point when larger 300 ton trucks are purchased and a larger workshop is built closer to the mine. They will possibly also be used in parallel with the new workshop.

The Gécamines West workshops are located directly adjacent to the KZC plant. This facility is owned by Gécamines and, although without a substantial order book/workload, is an excellent resource with substantial technical competence. The facility can be fed with engineering repair and refurbishment jobs to ensure that it survives to support selected areas of the future DCP operations. It was noted that the Lubumbashi and other regional engineering workshop capability is very limited.

The SKM and GEC owned railway workshop is located at Kamoto. This facility was not inspected but could act as a good support facility for railway machinery.

8.3 Fuel Supply and Storage

The rail linked diesel fuel storage facility consists of six tanks each with a capacity of 1.5 million litres. These tanks can be secured and refurbished to supply the fuel for the entire mine and other regional operators. The upgrade costs estimated for this purpose are adequate. The fuel considered includes diesel, petrol, light fuel oil and Jet A1 fuel.

The management and operation of this facility can be sub-contracted to a suitable third party.

The facility is in reasonable condition and has significant strategic value for fuel management in Kolwezi.

8.4 Water Supply

The regional geology results in sedimentary layers with high permeability which necessitates dewatering operations for mining operations. Water is generated in from the dewatering operations at KOV and other catchment sites. This water is in excess of the planned consumption. A water balance has been constructed for the DCP future operations. Water is treated as required and distributed to mining operations, housing estates and Kolwezi town. Excess water is discharged into local rivers.

A water shortage is not anticipated; however a 20 Ml water reservoir will be constructed at the New Processing Facility site to overcome any supply interruptions.

Potable water treatment will be effected as required. Provision has been made for these water pumping supply and treatment costs.

8.5 Electrical Supply

8.5.1 RO Substation

SNEL (Socie'te' Nationale d'Electricite') provides a single feed 120kV powerline to supply the Kolwezi substation from the RO Substation. The RO substation has a capacity of 200MVA and supplies the major users in the area including Kisenge, Kadi, Luilu and Kisantu. This major distribution substation is part of the broader electrical Southern Network and has alternative feeds from Nzulo, Tenke and Nseke. It is a reliable, professional installation.

Management weaknesses were evident in the security, maintenance bay and spares control. Influence should be applied to SNEL to ensure the ongoing reliability of the installation.

Power shortages are occurring in the region which is resulting in regular load shedding in the Southern Region. These shortages are reportedly resulting from units being removed from operation due to lack of maintenance funds. At the same time power is being exported to Zimbabwe and South Africa.

While World Bank funding may become available for upgrading the electrical network in the Congo, the short term power supply shortage could be overcome by reduction in power export. DCP have indicated the possibility of funding some power generation equipment provided that a long term power supply is guaranteed.

The RO substation has capacity for supplying the proposed New Processing Facility.

8.5.2 Kolwezi substation

The Kolwezi substation is located in the Gecamines workshops and is owned by Gecamines. It is equipped with lightning arrestors on the incoming feed and HT power factor correction. The HT switchgear (Reyrolle and Evershed & Vignoles Ltd) is very old. The three single phase 33MVA transformers reduce the voltage to 11kV for local distribution. A spare transformer is stored on site.

The substation is not equipped with fire detection, fire suppression and transformer dividers.

The storm water drainage is blocked and overgrown. The oil soak pits need restoration. The cable trenches are run down with missing and broken covers.

The security fence is absent in areas and should be upgraded.

Periodic testing of switchgear and earthing is undertaken annually.

The current status of the Kolwezi substation results in a medium to high risk of extended power interruption to the KZC plant and other users. It is recommended that the substation be reviewed in detail. Improvements in fire detection, suppression, equipment upgrades and general management are recommended.

8.5.3 DCP Distribution

The existing electrical infrastructure is operational but generally requires improved maintenance and modernisation. The Low Tension, ("LT") distribution, particularly at KZC is in poor condition.

8.5.4 Electrical supply in general

A supply agreement for the future power requirements is not concluded. Discussions are being held between DCP and SNEL to conclude a long term supply agreement. This agreement is necessary to reduce the risk of non supply.

The risk of power interruptions or of an ongoing area wide power shortage effecting DCP production is significant.

There is a medium to high risk of a major electrical failure resulting in power interruptions in excess of three days at the Kolwezi substation.

The solutions to these risks are complex. DCP has approached these challenges positively by appointing an experienced electrical expert (Bernoit Munanga) to focus on the solutions and conclude a supply agreement with SNEL.

The DFS covers the electrical distribution in detail. Capex provision is allowed and adequate for the proposed upgrades and new installations.

8.6 Capital Costs

The capital estimates for the work described above are summarised in Table 8.1. The amounts in Table 8.1 do not include any contingency provisions.

Table 8.1: Infrastructure Capital Estimates

Item	Capital Expenditure (US$million)
Electrical	
KOV Substation Upgrade	3.4
New Processing Faclity Substation	20.4
Luilu transmission line	2.7
KOV transmission line	2.5
EPCM Costs	2.1
Workshops	
Existing workshops	6.7
New workshops	6.3
Roads	
Internal	3.5
Haul roads	0.3
External roads	36.9
External bridges	3.2
Rail	
Rail rehab costs	5.8
New refinery spur	1.4
Fuel Facility upgrade	0.8
Total	**96.1**

8.7 Operating Costs

The opex for the infrastructure is not clearly listed. The amount noted for roads, dewatering and other operations appears to be too low. A more detailed breakdown is recommended.

8.8 Service Plant and Logistics

The process defined for the New Processing Facility will require the supply of large quantities of sulphuric acid at between 100 and 260kg/t or ore, as indicated in the test work detailed previously. This will require between 1,333tpd and 3,467tpd of pure sulphuric acid, which is approximately 426 and 1,100tpd of pure Sulphur. This quantity of acid will require a full sulphuric acid plant on site with a large store for containing the Sulphur stocks necessary to cover delivery surges. This will be a logistical problem in receiving this quantity of material on the existing rail network in Central Africa, even considering the lowest indicated quantity of sulphur. The logistics have the advantage that the imports and exports are balanced which should ease the movement of rolling stock.

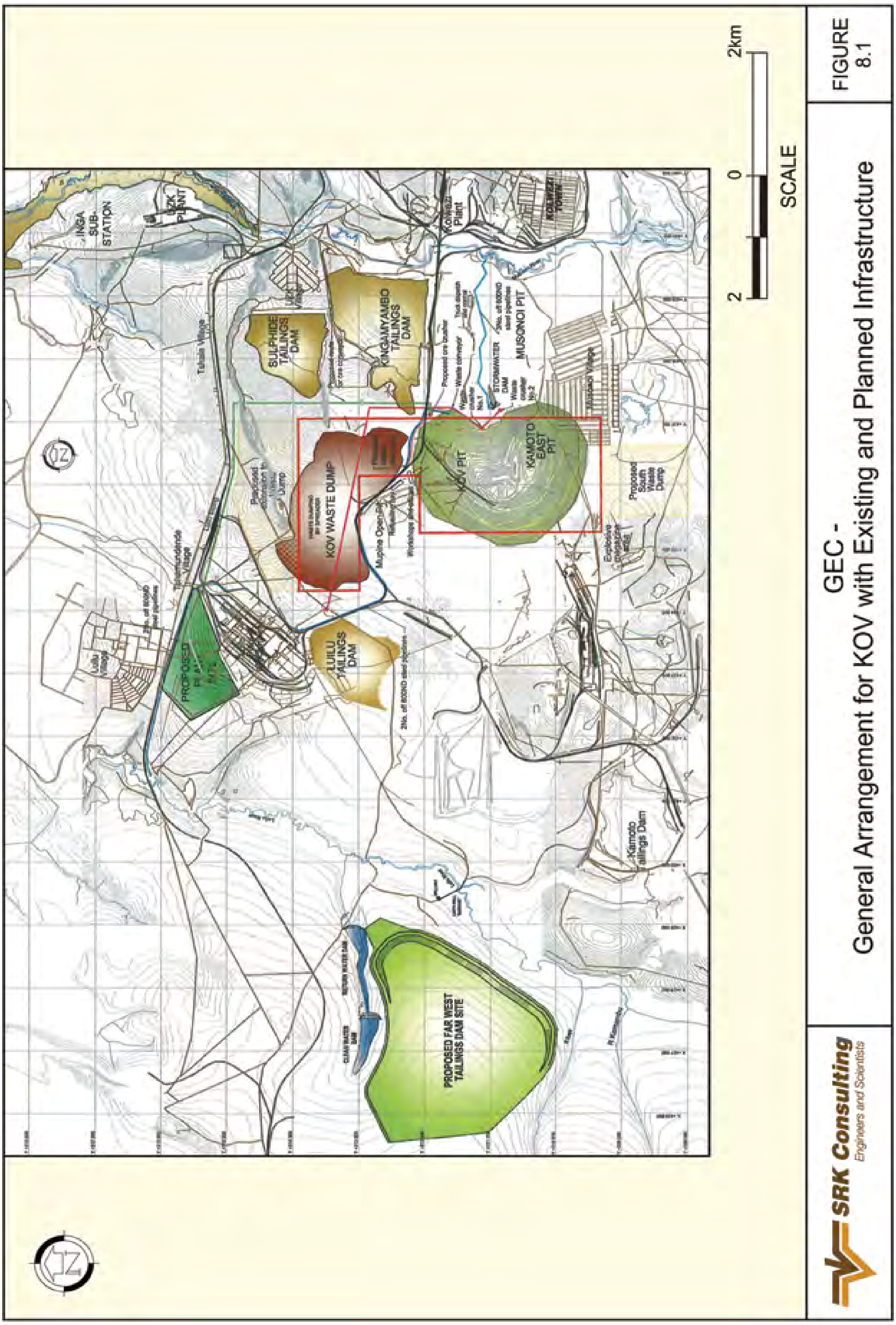
In addition, the plant will require the consumption of approximately 550tpd of crushed limestone and 240tpd of burnt lime. Lime in all forms is to be sources from a local independent quarry and treatment plant at Katontwe, near Likasi.

The above reagents represent 10 wagons per day, every day from a port for the Sulphur and 18 wagons per day from the local limestone quarry. This will be in addition to all other requirements for the project, such as diesoline, other consumables, etc.

8.9 SRK Comments

- The DCP infrastructure ranges in condition from good to poor. The infrastructure has been accessed and the need for professional refurbishment is understood. A reasonable CAPEX provision has been made for this work.
- A long term power supply risk has been identified both in the potential shortage of power available for distribution and the lack of a SNEL supply agreement. Solutions are being sought by the DCP team. Their approach has a high probability of success.
- The general management of the electrical supply infrastructure at both the RO and Kolwezi Substations is poor. Their lack of funds and management skills are recognised. CAPEX has been allocated to assist in certain areas. Solutions are required that would influence management and discipline in areas outside DCP's control.
- The OPEX allowed after the various infrastructural upgrades is not clearly listed. The amounts allocated appear too low. It is recognised that these amounts are small compared to the total operational costs of DRC and hence do not pose a significant risk. A more detailed list is recommended.
- Dust generation management for the local roads will be required on an on-going basis, particularly in the villages, to ensure adequate road safety.
- The long term maintenance and repair of the roads, particularly the regional road remains significant risk, as the control of key factors affecting the road such as traffic density, overload and weather cannot be controlled by DCP.
- No fatal flaws were noted in the existing infrastructure and the future plan.

Figure 8.1: General Arrangement for KOV with existing and planned infrastructure



9 HUMAN RESOURCES

This section includes discussion on the human resources related aspects associated with the KOV Project.

9.1 Labour Legislation

9.1.1 Transfer of mining operations

In terms of the DRC Labour Code (law 015/2002 of 16 October 2002), in case of substitution or change of the legal position of the employer, all employment contracts existing on the date of such change will remain in effect as between the new operator and the individual employees. In this respect, Gécamines has provided to GEC a list of employees in relation to the assets covered by the JVA. DCP and Gécamines have agreed that on completion of the DFS, DCP will submit a list of the Gécamines employees it wishes to employ for the Material Properties.

9.1.2 Priority to Congolese employees

Mining companies are free to recruit their employees, provided that priority is given to Congolese personnel with equal qualification in terms of education and experience.

9.1.3 Employment of foreigners

Expatriates living and working in the DRC are required to obtain a *visa d'établissement de travail* issued by the *Direction Générale des Migrations* and a work permit issued by the *Commission Nationale de l'Emploi des Etrangers.* SRK understands that DCP is busy with these formalities.

9.1.4 Conditions of employment

The Labour Code prescribes minimum conditions of employment on such matters as hours of work, wages, leave entitlements and notice of termination of employment. It provides for the lodging of workers, medical services and special rules for night work and the protection of women.

The appropriate procedures and rules relating to hiring and firing employees as well as guidelines established for employee benefits must be adhered to.

9.1.5 Professional Training

An operator employing five or more employees must register with the *Institut National de Préparation Professionnelle* ("INPP"). SRK understands that the registration process for DCP with the INPP is underway.

9.1.6 Social security concerns

The law on social security (Decree of 29 June 1961) requires any person who hires one or more workers in the DRC to register with the regional office of the *Institut National de Sécurité Sociale* ("INSS") for each Province in which the company employs workers. The company shall pay monthly social contributions. Under the local security system, each employee must also be registered with the INSS and assigned a permanent affiliation number. Registration with the INSS is compulsory even for expatriate employees already covered by a foreign social insurance scheme. SRK understands that the registration process for DCP with the INSS is underway.

9.2 Manpower Philosophy

GEC has elected to have a flat management strategy with no superfluous positions, both during mine development and operation. Every position will have clearly defined accountabilities. No overlapping of responsibilities will occur.

9.3 Recruitment Philosophy

9.3.1 Expatriates

GEC anticipates that 31 out of some 54 senior management positions would be filled by expatriates during the early stages of the development and operation. Recruitment is likely to be conducted throughout the world. During construction, most of the staff would be on a rotation basis. However, as married accommodation and facilities become available, this would fall away.

9.3.2 Local Employees

There is a population of approximately 400,000 in Kolwezi, which has a long history of mining. GEC plans to recruit the maximum number of skilled staff possible from among the workers at Gécamines and from the area surrounding Kolwezi and in the greater DRC. DCP and Gécamines have established a joint team to handle worker expectations and their lack of pay for some time.

A further complication is the integration into the organisation of as many of the artisanal workers as possible.

9.3.3 Women

GEC plans to employ women into 10% of all roles within the organisation, from truck drivers, plant operators to artisans.

Strict application of standards of behaviour will be practised.

9.4 Remuneration

GEC has established remuneration levels that it believes will attract, retain and motivate workers. Jobs will be graded according to complexity and skill required. A "gate-wage" concept has been adopted which does away with allowances for food, housing and goods.

Work performance will be monitored through key performance measures and adjustments to salaries will be based on measured outputs.

9.5 Operational Management

Key personnel within GEC and DCP that will be involved in the KOV Project are:

Jim Gorman, PrEng, BSc (Mining)—*Acting CEO and COO*

Jim is a mining engineer who has 37 years of extensive production, technical and executive management experience in the mining industry. Jim held executive management positions at Rossing Uranium in Namibia and Palabora Mining in South Africa. He was General Manager at Kansanshi Mine in Zambia, responsible for construction through to full production. He joined GEC in 2006.

Larry Treadgold, CEng, BSc (Hons), FIMMM—*Head of Metallurgy*

Larry has been a metallurgical consultant for more than 20 years and was Copper and Precious Metal Refinery Superintendent for many years at the Ndola Copper Refinery of Roan Consolidated Mines, Zambia. He has also been President of the Micrometallics Division of Noranda Minerals Inc in California. He joined GEC in March 2005.

Simon Tuma-waku, MSc (Mining), DipBA, SA Mine Managers Certificate—*Government liaison*

Simon has 14 years experience in the mining industry. He was Minister of Mines and Hydrocarbons in the DRC for two years and was responsible for the preparation and approval of the New Mining Code. He joined GEC as a special advisor in 2004 and was appointed Chariman in December 2005.

Ben Munanga, BSc (Hons)(Electrical), Post Grad Dip—*Manager Electrical Infrastructure*

Ben is an electrical engineer with extensive experience in electricity generation and supply, having held several executive management positions at Eskom (the South African energy utility). He was Assistant Town Electrical Engineer for Alberton in South Africa and Electrical Maintenance Engineer for Bralima Breweries in Kinshasa, Zambia. He joined GEC in 2006.

Mike Clifford, NatDip, CMA—*Financial Manager*

Mike is an accountant with extensive experience in the mining industry in South Africa, Ghana and Tanzania. During 18 years service with Gold Fields, he held positions of Project Accountant at Tarkwa, Financial and Admin Manager at Buhemba and Financial Manager of Driefontein Consolidated.

Carel Swart, PrEng, BSc (Eng)—*Head of Technical Services*

Carel is an industrial engineer with more than 20 years experience in project development and industrial and mineral processing plants. He has held executive management positions at Art

Precision Works (foundries) and Met-O-Cast Ltd (steel distributors in the UK), as well as project management positions at Iscor Ltd and Bateman Minerals and Metals. He has been with GEC since 2003.

Emile Mota, DEcon.Sci.—*Head of Human Relations*

Emile is a Professor at the University of Lubumbashi and was a former Chief of Staff in the Office of President Laurent Kabila. He has conducted several studies into development projects in the DRC on behalf of the Presidency and authored several publications on industrial and mining development projects in the DRC. He joined DCP in April 2006.

Candidates for the positions of Chief Financial Officer, Head of Mining and Head of Engineering have been identified and GEC expects to make the appointments shortly.

10 ENVIRONMENTAL MANAGEMENT

10.1 Introduction

The following section includes discussion and comment on the environmental management aspects of the Mine. While detailed planning has been undertaken for the proposed mining in the KOV pit and the construction of the proposed new direct leach plant, it is also recognised that prospecting operations are underway at Tilwezembe and Kananga.

Comment is included on the status of environmental legislation applicable to the mine; compliance with legislation; key liabilities and risks over the life of the operation and decommissioning and closure liabilities and risks. This section of the report is based on:

- Discussions with relevant project staff;
- Discussions with the local communities;
- Review of available information environmental and social documentation;
- Closure liability documentation;
- Site visits to inspect surface infrastructure and the proposed sites for new infrastructure.

For the purposes of this section liability and risk are defined as follows: a liability can be assigned a monetary value to be included in the financial model (e.g. costs associated with new storm water drains); and a risk involves too much uncertainty to enable cost predictions to be made (e.g. possibility that more stringent requirements will be imposed by the regulator in the future). Risks or liabilities that would generally be addressed in terms of accepted environmental practice and which do not have significant cost implications have not been discussed. Comments are made relating to the nature of the risk/liability, the level of uncertainty and the manner in which it has been addressed e.g. estimated cost, contingency item or flagged for future consideration.

The criteria used for assessment purposes are those required by legislation and generally accepted practice in the mining industry.

Environmental control measures have been planned and will be implemented to limit negative impacts but strict compliance with applicable international practices may not be possible at all times. A particular example is water quality, with discharges during the wet season exceeding acceptable salinity levels. Incremental impacts related to this, will, however, be limited in the current highly degraded environment in which the mine will operate. With adequate attention to environmental controls conditions can be improved with time and to this end GEC intends to implement the Project in a manner that is compliant with the environmental guidelines set out in the World Bank Pollution Prevention and Abatement Handbook and the IFC's Performance Standards on Social and Environmental Sustainability (which were formerly known as IFC Safeguard Policies), and will reflect these standards and policies, with any modifications as are appropriate for the GEC's circumstances.

10.2 Legislation

The following comments should not be interpreted as an exhaustive legal review by legal specialists, but rather identification of significant legislation affecting environmental planning.

The primary piece of legislation governing mining environmental issues in the Democratic Republic of the Congo (DRC) is the Mining (Code) Act 007/ 2002 of 11 July 2002. The Mining

Regulations (Decree No 038/2003 of 26 March 2003), promulgated in terms of this Act, are of particular relevance. Relevant aspects of these regulations which should be noted are discussed below.

10.2.1 Exploitation (mining) Permit

In terms of Title V of the Decree No 038/2003 the company must apply for an Exploitation (Mining) Permit. In order to apply for such a permit the company must be the title holder of a valid Exploration Permit(s).

10.2.2 Environmental Impact Study (EIS) and Environmental Management Plan (EMP)

The environmental obligations are set out in Title XVIII of Decree No 038/2003. With the exception of temporary quarrying, any mining operation requires an approved Environmental Impact Study (EIS) and of an Environmental Management Plan.

10.2.3 Financial Security Obligation

Articles 410 to 414 set out the obligations in respect of a financial security which must be provided in terms of Article 294 0f the Mining Code. The financial security must provide for the rehabilitation of the environment. The funds of the Financial security shall be made available to the State and managed for the purpose of rehabilitating the site.

10.2.4 Environmental Regulations

A set of Environmental Regulations have been promulgated in terms of the Mining Regulations and are published as separate Schedules to the Mining Regulations. Relevant aspects of these Schedules include:

Schedule II (Financial security)

This Schedule deals with the financial security required in respect of rehabilitation of the environment. The amount of the financial security shall be determined in accordance with the approved Environmental Plan. The financial security shall be maintained until the Titleholder has been issued with an Environmental Clearance Certificate. There are a number of alternative means by which such security can be provided. The total amount of the financial security shall be paid in accordance with a timetable taking into consideration the duration of the mining activities capped at 15 years. The first instalment is only required in year 4 and the quantum of the instalment increases with time such that the major part is only required in the last five years of the 15 year period.

Schedule IX (Contents of EIA and EMP)

This Schedule sets out the contents of the EIS and the EMP and provides detail regarding specific management measures and standards that are required.

10.3 Compliance

10.3.1 Exploitation (mining) permit

The project has three Exploitation Permits (PE 4940, PE 4961 and PE 4963). Further details regarding these permits are provided in Section 2.7.2 of this report.

10.3.2 Environmental Impact Study (EIS) and Environmental Management Plan (EMP)

SRK has compiled a draft EIS and EMP for Global Enterprises Corporate Ltd (GEC) in accordance with the principles of acceptable environmental practice and in terms of the applicable legislation of the Democratic Republic of the Congo (DRC) and in accordance with the requirements set out in Schedule IX of the Mining Environmental Guidelines of the DRC for the mining of the KOV open pit and the establishment of the new plant and tailings disposal facilities and associated works. In addition SRK has compiled a conceptual EIS and EMP for mining at Tilwezembe and Kananga, with ore from these two areas to be processed in the existing Kolwezi Concentrator from which concentrate will be railed to the refinery at Shituru.

10.3.3 Financial security obligation

The Draft EMP for the mining of the KOV open pit and the new plant includes a section in which the environmental costs of the project have been estimated. This estimate is discussed in the relevant section of this report below. This financial security will only need to be provided in 11 instalments starting in the fourth year of the project in accordance with requirements. Furthermore there are a number of options as to how this security can be provided.

10.4 Environmental Issues

10.4.1 General

There are two areas that will potentially be affected by the proposed mining operations, namely the Kolwezi area, adjacent to the town of Kolwezi, and the Tilwezembe area where open pit mining may occur if proved feasible.

Over 110Mt of ore have been mined and processed in the vicinity of the town of Kolwezi during almost a century of mining for primarily copper and cobalt minerals. As a result of mining and metallurgical activities, the area around the town of Kolwezi today has numerous open pits, waste rock, ore and slag dumps, tailings dams, concentrators and other mining-related infrastructure. This has led to extensive impacts on the environment. Rivers in the area, the Luilu and Musonoi Rivers, are extensively contaminated by tailings disposal directly into the river in the case of the Musonoi, and by the failure of a large upstream tailings dam in the case of the Luilu.

It is recognised that water management in the Kolwezi area is complicated by the fact that the area is one with a net water positive environment, with significant seasonal fluctuations, and associated impacts.

A number of sampling exercises have been conducted in the area in the recent past; however, these results are not sufficient for any form of statistical analysis. Data collected on a regular basis (e.g. monthly) over a number of years would be the minimum required). Therefore very little can be concluded regarding baseline water conditions. The full impacts of seasonal water balance and impacts on the environment cannot be comprehensively defined until appropriate time related water monitoring has been undertaken. This would include consideration of the contributions of other activities in the catchment systems. Present water management plans are expected to be further refined and detailed as practical information is generated. The principle of separation of clean and dirty water and recycling of used water will apply.

All make-up water for the project will be obtained from a series of dewatering boreholes surrounding the KOV pit. The purpose of these boreholes is to limit the inflow of water into the pit. These boreholes will pump water to the Luilu River. The installed pumping capacity will initially be 8,300m^3/hr increasing to 9,000m^3/hr. Takeoff points will be provided for the plant and for the Luilu Village. The plant will only require approximately 470m^3/hr and therefore the majority of this water will be discharged to the river. This water is considered to be of generally good quality.

In the Tilwezembe area there has been some mining from an existing pit but further mining will result in the extension of this pit into an area currently unaffected by mining operations. The current pit at Tilwezembe is dry and it is not expected that there will be a need for dewatering, except for rainwater which will collect in the pit.

10.4.2 Water management at the KOV pit

As mentioned above dewatering boreholes will be provided to limit the inflow of water into the pit. One of the primary sources of this groundwater is an aquifer which draws water from the Musonoi River. However, this pit has been drawing groundwater from this aquifer for many years and the impact is not considered to be particularly significant. The drainage of groundwater into the pit is not expected to add any new impacts in respect of any water supply sources currently being used.

Rainfall and groundwater seepage collecting in the pit will be pumped to a set of settling dams before being pumped to the Musonoi River. Although the quality of the water currently in the pit is relatively good, this will change under operational conditions. However, it is expected that this is still likely to be relatively good quality water.

10.4.3 Water management at the concentrator plants

For the envisaged new direct leach plant diversion drains will be provided to limit the amount of runoff entering the plant area from the adjacent external catchment. All polluted runoff and washdown water originating at the plant site will be diverted by a system of drains to two 150,000m^3 lined pollution control dams at the bottom end of the plant site. Water collecting in these dams will be recycled for use in the plant. These dams have only been sized provisionally. It is still necessary to finalise the site lay-out and corresponding catchment areas, as well as to agree on the containment philosophy (number of hours average/peak? rainfall to be contained, before release to the environment/river). At this stage it is probable that in times of heavy rain it may be necessary to discharge some of this water.

Where compliance with prescriptive DRC regulatory limits may be considered unnecessary or impractical for the specific site constraints and conditions, a risk based approach will be followed. This will enable planners to assess the potential for the proposed operation to impact on the environment and to develop solutions which address the real issues rather than simple compliance with the regulations.

At the Kolwezi plant, should it be used to process ore from Tilwezembe or Kananaga, existing water pollution control measures, upgraded as necessary in terms of the envisaged risk based approach, will be used.

10.4.4 Water management at the tailings dams

Decant water from the proposed new tailings dam will flow to a 50,000m^3 return water dam which will be capable of containing several days of process water supply. Surplus water will be discharged to the Luilu River. During the dry season (July) there will be no discharge and during the wet season the discharge will rise to approximately 1,200m^3/hr. The annual average discharge will be approximately 450m^3/hr. The average base flow rate in the Luilu River is 760,000m^3 per annum (this is equivalent to approximately 90m^3/hr).

At this stage there is no definitive estimate of the likely quality of this discharge water. Leach tests on historical samples of oxide ore indicated limited contamination potential. However, it is recognised that the plant proposes to discharge process wastewaters with the tailings, after pH adjustment to pH 10.0. This water will have a very high; as yet undetermined, salt content which is expected to be unacceptable for discharge to the environment. The tailings dam facilities have been designed on the assumption that the water discharged to this dam will be relatively clean. Thus the current status represents an unacceptable, as yet unresolved, situation.

Again, as far as practical, the project will endeavour to comply with the water quality limits set for discharges and consideration of treatment and other management options will be further investigated by the plant which would inherently affect tailings and residue quality received by the new tailings dam.

Although there is no definitive estimate of the likely quality of water seeping from the tailings dam it is considered likely, as noted above, that this will be significantly contaminated. It has been estimated that this water will ultimately reach the Luilu River after a period of approximately 12 to 15 years. It is expected to have a flow rate of approximately 25m^3/hr, representing a limited incremental impact on the Luilu River. Once deposition ceases on closure of the dam and the top of the dam has been re-vegetated the flow rate could be very significantly reduced probably to less than 5m^3/hr.

In the event that the existing Kolwezi plant is used to process ore from Tilwezembe and / or Kananga, the tailings will be disposed of on the existing Kolwezi sulphide tailings dam for which water management facilities will be upgraded as necessary.

10.4.5 Relocation of residents of Musonoi Village

As the pit expands it will ultimately be necessary to relocate a section of the Musonoi Village. This is only expected to happen sometime after year 15. The Musonoi Villagers are already aware of this probability although no details have yet been communicated. Due process will be followed and suitable negotiations will be entered into well in advance of any pending move. The mine will ensure that adequate compensatory measures are provided.

10.4.6 Relocation of residents of Yenge Village

There is a small village, Yenge Village, and a number of very small settlements within the zone of influence of the proposed new tailings dam. These villages and settlements will need to be relocated. This relocation will also have to include finding suitable available land for subsistence farming. Negotiations with this community have not been undertaken. No undue problems are envisaged in this regard but further sociological and archaeological investigation is required.

10.4.7 Air pollution

The KOV pit and, if mining proves feasible at Tilwezembe and Kananga, the pits in those two areas will be sources of noise, vibration and dust. In this respect the KOV pit will impact on the Musonoi village. Should mining prove feasible in the Kananga area the UZK village, which is located close to the area, could be impacted. The Tilwezembe pit is some distance from the nearest significant residential area and problems in this respect are not expected to be as significant as they are for the other two areas. A number of management measures have been put in place to ensure that impacts are kept within acceptable limits. Of particular significance is the proposed 30m high waste rock berm which will be constructed on the land vacated by the relocated section of the Musonoi Village. This will act as a screening berm between the pit and the village.

Historically the un-rehabilitated tailings dams in the area have been regarded as a significant source of dust pollution. Concurrent re-vegetation of tailings deposits will be implemented for this project and during the operational phase a significant part of the top of the dam will be wet at all times thus minimising dust from this source. On closure the top of the dam will be grassed.

Various measures will be incorporated at the plant to ensure that dust and sulphur dioxide levels are kept within perscribed limits.

10.4.8 Loss of land use

In the Kolwezi area, the total area which will ultimately be disturbed by the KOV pit and waste rock dumps will be approximately 1,125ha. As the Mine is an established operation the additional land to be disturbed by the project is limited to an area of approximately 369ha which is approximately one third of the total area. This loss is considered to be of low significance. The total area to be disturbed at Tilwezembe and Kananga can not be established prior to detailed mine planning.

Approximately 100ha of land will be disturbed by the new direct leach plant. This area is already significantly disturbed and the incremental impact is therefore of low significance.

At the tailings dam site a total of approximately 450ha of land which is currently use for charcoal gathering and subsistence farming will be lost. This is some of the less disturbed land in the Kolwezi area. This loss is considered to be of medium significance.

10.4.9 Contamination of ground water resources.

There is a potential for some contamination of ground water from the new plant site. All the stockpiles will be placed on compacted hard base floors which will limit infiltration. Any material such as the sulphur stockpiles will be under roof and on a proper concrete floor. Spill protection measures will be implemented and all contaminated water will be captured in two lined 150,000 m^3 pollution control dams. Seepage from the adjacent Existing Electo-Refinery plant is expected to be impacting on the ground water in this area. Facilities at the new plant are expected to have a limited incremental impact on ground water quality. The groundwater flow direction from this area is north-west towards the Luilu River. A small section of any plume from the plant could impact on shallow wells on the southern edge of Luilu Village. A ground water monitoring system will be put in place to keep track of trends and if necessary corrective measures will be taken.

At the Kolwezi plant and the Kolwezi sulphide tailings dam it is expected that there will be existing ground water contamination and incremental impacts, should these facilities be used, are not expected to be significant.

10.4.10 Ecological issues at Tilwezembe

The botanical survey at Tilwezembe resulted in the identification of endemic species and a possible new plant species that will require specific conservation measures under the guidance of an appropriate botanical expert. Measures for this will need to put in place should mining prove feasible at Tilwezembe.

10.4.11 River and railway line diversions at Kananga

Should mining prove feasible at Kananga it will be necessary to divert the Dilala river and the existing railway line. Conceptual planning for this has been undertaken.

10.4.12 Social issues

A public consultation process was undertaken for the Kolwezi KOV Project. The local communities were given the opportunity to express their concerns and these were taken into account in the development of the EMP.

The main concerns regarding the potential relocation of people revolved around ensuring good communication and proper process as well as ensuring acceptable compensatory measures.

Other issues related to fair employment practices, social up-liftment, pollution and health issues.

In terms of Article 127 of Schedule IX of the Environmental Regulations, the Mine will be required to compile a sustainable development plan. Although this has not yet been done the EMP recommends that monies be budgeted for this.

10.4.13 Decommissioning and Closure

Prior to closure, agreement will be reached between the Mine, the local communities and the Minister regarding possible post mining use of the mine infrastructure. On closure:

- All buildings and infrastructure for which there is no further agreed use or which will revert to Gecamines, such as the Kolwezi concentrator, will be dismantled and the site will be rehabilitated.
- The tops of the tailings dams will be engineered to ensure that all water falling on top of the dam is retained on the dam and the re-vegetation programme will be completed to ensure that all area are adequately covered.
- The waste rock dump re-vegetation programme will also be completed.
- A trench and berm will be constructed around the perimeter of the pits to prevent persons gaining inadvertent access to the pit.
- All rehabilitation work will be maintained for a period of 5 years after which it is expected that an Environmental Clearance Certificate will be issued by the authorities.

10.5 Liabilities

Environmental liabilities and costs are discussed in this section. It should be noted that these cost estimates are best estimates based on the draft EMP. The costs exclude costs associated with possible mining at Tilwezembe and Kananga.

The estimated environmental costs covering the start-up (capital), operational and closure phases are presented in Tables 10.1 to Table 10.3 respectively.

Table 10.1: Environmental Start-up Capital Costs

Item	Years Required	Amount (US$)	Total LoM Cost (US$million)
Tailings dam			
Relocation of Yenge villages	Year 1	1,000,000	1.00
Stripping and stockpiling soil and ferricrete for use in rehabilitation of the tailings dam. In the start-up phase sufficient to be stripped for the first five years.	Year 1	130,600	0.13
Construction of clear water dam immediately upstream of the return water dam	Year 1	350,000	0.35
Stripping and stockpiling soil and ferricrete for use in rehabilitation of the tailings dam. Sufficient to be stripped for years 6 to 10.	Year 5	335,400	0.35
Stripping and stockpiling soil and ferricrete for use in rehabilitation of the tailings dam. Sufficient to be stripped for years 11 to 15.	Year 10	231,500	0.23
Stripping and stockpiling soil and ferricrete for use in rehabilitation of the tailings dam. Sufficient to be stripped for years 15 to 20.	Year 15	106,800	0.11
Stripping and stockpiling soil and ferricrete for use in rehabilitation of the tailings dam. Sufficient to be stripped for years 20 to 25.	Year 20	255,200	0.26
Staff			
Environmental staff costs for 2 years of construction	2 years of construction	170,000 pa	0.34
Environmental monitoring			
Installation of monitoring equipment	Year 1 of construction	20,000	0.02
Monitoring costs	2 years of construction	60,000 pa	0.12
Total			**2.89**

Note: Other environmental start-up costs are included in various other budgets.

A large section of the Musonoi Village will need to be relocated in year 14. It is therefore recommended that a provisional amount of US$30million be allowed for this relocation (Table 10.2).

No social development plan has yet been compiled. However, it is proposed that an amount, still to be decided by the shareholders, will be allocated for this programme.

Table 10.2: Environmental Operating Costs

Item	Years Required	Amount (US$)	Total LoM Cost (US$million)
Tailings dam			
Ongoing placement of ferricrete layer on tailings dam slopes from year 6 on.	Annually from year 6	34,560 pa	0.76
Ongoing placement of soil layer on tailings dam slopes from year 6 on.	Annually from year 6	17,280 pa	0.38
Ongoing re-vegetation of tailings dam slopes from year 6.	Annually from year 6	10,000 pa	0.22
Re-vegetation trials	Annually from year 6 to 15	15,000 pa	0.15
KOV pit and waste rock dumps			
Re-vegetation of sides of waste rock dumps	Annually from year 6	36,800 pa	0.81
Relocation			
Musonoi village	Year 14	30,000,000	30.00
Staff			
Environmental staff costs (salaries, consultants and sundries)	Annually for 27 years	170,000 pa	4.59
Monitoring			
Water monitoring	Annually for 27 years	20,000	0.54
Dust monitoring	Annually for 27 years	20,000	0.54
Noise monitoring	Annually for 27 years	20,000	0.54
Total			**38.53**

In terms of Article 100 of DRC Mining Environmental regulations all buildings and surface infrastructure will have to be demolished on closure, unless the representative of the local communities submits a written request to the Minister and demonstrates that such buildings and infrastructure are necessary for the social-economic development of the territory. At this stage it is not possible to predict what infrastructure may be required following mining. SRK has therefore adopted a conservative approach. It is assumed that all buildings erected by the Mine will need to be removed. Except for the railway loop within the plant site all rails and roads will remain. The existing workshops, stores, fuel depot and magazine which are to be refurbished will also be demolished. The pumps and pipeline from the dewatering boreholes to the Luilu River will remain.

Table 10.3: Environmental Closure Costs

Item	Years Required	Amount (US$)	Total LoM Cost (US$ million)
Plant			
Demolition and rehabilitation of site	28 and 29	4,326,000	4.33
Mine pit and waste dumps			
Demolition of infrastructure including crushers and overland conveyor	28 and 29	585,000	0.58
Construct safety access berm and complete re-vegetation	28 and 29	1,845,000	1.84
Infrastructure			
Demolish workshops, stores, magazine, fuel storage area, sewage plant, pipeline to Musonoi River, and rehabilitate solid waste site. (Note: Borehole pumping scheme and pipeline to Luilu River to remain foe community use.	28 and 29	634,500	0.63
Tailings dam			
Final rehabilitation of tailings dam	28 and 29	1,230,000	1.23
Aftercare			
Monitoring and maintenance for 5 years	30 to 34	910,000	0.91
Total			**9.53**

10.6 Financial Security

In terms of article 294 of the Mining Code, mines are required to provide financial security to cover the rehabilitation costs of the operations. It is understood that this refers specifically to the closure rehabilitation costs and does not include operational costs. The closure cost for this project, set out in Table 10.3, amounts to US$9.53million. The Regulations specify a set of formuale in terms of which this amount can be provided in instalments, as follows:

Year 4 of the operations:	US$0.08million
Year 5 of the operations:	US$0.24million
Year 6 of the operations:	US$0.39million
Year 7 of the operations:	US$0.55million
Year 8 of the operations:	US$0.71million
Year 9 of the operations:	US$0.87million
Year 10 of the operations:	US$1.02million
Year 11 of the operations:	US$1.18million
Year 12 of the operations:	US$1.34million
Year 13 of the operations:	US$1.50million
Year 14 of the operations:	US$1.66million
TOTAL:	**US$9.53million**

10.7 Material Risks and Potential Opportunities to Reduce Liabilities

In SRK's view the potential material risks are the following:

- As the EIS and EMP have not yet been approved by the authorities it is possible that more stringent management measures may be requested before the EIS and EMP are approved.
- Discharges from the tailings dam to the Luilu River, if all process wastewaters are directed to the tailings dam, or from the plant if such wastewaters are not discharged to the tailings dam, could have a significant impact on the river. There is therefore a high risk water treatment costs may have to be incurred. Such treatment costs could require a capital investment of the order of millions of US Dollars and annual operating costs of the order of millions of US Dollars.

- No definitive plans for the relocation of the villages which have to be moved as they are in the zone of influence of the tailings dam have yet been compiled. In particular it has not yet been confirmed whether any sites of cultural significance may be affected or where a suitable alternative site can be found.

11 COMMODITY PRICE PROJECTIONS

The base case Reserve Valuation in section 13 uses the commodity price projections as shown in Table 11.1 below.

Table 11.1: GEC—Forecast Commodity Prices

	Copper price (Real)		Cobalt Price (Real)	
Year	(US$/lb)	(US$/t)	(US$/lb)	(US$/t)
2006	2.40	5,285	15.78	34,794
2007	2.03	4,468	14.06	30,997
2008	1.60	3,525	11.54	25,434
2009	1.46	3,226	11.75	25,904
2010	1.21	2,673	10.50	23,149
2011	1.10	2,425	10.00	22,046
2012	1.10	2,425	10.00	22,046
2013	1.10	2,425	10.00	22,046
2014	1.10	2,425	10.00	22,046
2015	1.10	2,425	10.00	22,046
2016	1.10	2,425	10.00	22,046
2017	1.10	2,425	10.00	22,046
2018	1.10	2,425	10.00	22,046
2019	1.10	2,425	10.00	22,046
2020	1.10	2,425	10.00	22,046
2021 – 2025	1.10	2,425	10.00	22,046
2026 – 2030	1.10	2,425	10.00	22,046
2031 – 2035	1.10	2,425	10.00	22,046
2036 – 2038	1.10	2,425	10.00	22,046

As GEC will be producing a high-purity cobalt salt, SRK has assumed that 85% of the above price projection of US$10.00/lb can be achieved.

SRK understands that GEC has not entered into any forward copper or cobalt sales or hedging contracts.

12 TECHNICAL-ECONOMIC PARAMETERS

12.1 Introduction

The following section includes details on the technical-economic aspects of the LoM plan associated with the KOV Project. Specifically, comment is included on the basis of projections, production schedules, operating costs and capital expenditures, economic projections and price forecasts as presented in the DFS and compiled into a financial evaluation model ("FM") by GEC.

12.2 Basis of Valuation and Technical-Economic Parameters

The valuation of the KOV Project as presented herein, has *inter alia* been based on the LoM plan and resulting production profile and associated revenue stream from saleable products operating costs and capital expenditure profiles (collectively referred to herein as TEPs) as provided to SRK by GEC, reviewed, and adjusted where appropriate by SRK. The generation of a LoM plan requires substantial technical input and detailed analysis and is critically dependent upon assumptions of the long-term commodity prices and their impact on the following: cut-off grades; potential expansion or contraction of the Mineral Resource and Mineral Reserve Base and the return on capital expenditure programmes.

SRK has subjected the FM for the KOV project to detailed review, in the process verifying that the TEPs contained in the FM correlate with those presented in the DFS, that the assumptions used in compiling the model are reasonable and that the calculation methodologies are correct.

Following its independent review and after discussions with GEC, SRK has adjusted where warranted TEPs, entries, calculation methods or assumptions in the FM.

Unless otherwise stated, operating costs comprise:

- **Cash operating cost components**: namely direct mining costs, direct processing costs, direct general and administration costs, consulting fees, management fees, distribution and transportation costs, charges and non-production related sundry income;
- The incremental components, including royalties but excluding taxes paid, required to yield **Total cash costs**;
- The incremental components, including separation benefits, reclamation and mine closure costs (the net difference of the total environmental liability and the current trust fund provision) but excluding non cash items such as depreciation, depletion and amortisation, required together with cash operating costs and total cash costs to yield **Total Working Costs**; and
- **Total costs**: the sum of total working costs, net movement in working capital and capital expenditure.

The LoM capital expenditure programmes for the KOV project are set out in the DFS in sufficient detail for the start-up capital requirements. Ongoing capital needs in the LoM plan after the target start of production in July 2009 are handled in the FM in two ways: estimated replacement/repair cost of mining equipment; and an annual provision equal to 3% of operating costs of any given year.

Environmental provisions have been included in the FM. The treatment of contributions to the rehabilitation fund is not according to the Mining Code, but SRK has been assured by GEC that the process followed in the FM has been discussed and accepted by the Mining Ministry.

A separation benefit equal to 1/50th of the annual employment cost times the number of years in the LoM is included in the last year of the LoM.

12.3 Technical-Economic Parameters

The TEPs for the KOV Project have been correctly incorporated in the cashflow projections in the financial model ("FM") compiled by GEC and include:

- Commodity production profiles;
- Total Working Costs profiles as previously defined; and
- Capital Expenditure profiles.

The TEPs for the KOV Project are given in Table 12.1. All expenditures are stated in calendar years and in 1 January 2006 constant money terms. The capital and operating costs are all given in USDollars. Any component of the costs which was derived from a foreign-currency denominated quote has been converted to USDollars at the exchange rates ruling on 1 January 2006 (Table 12.2).

Table 12.1: GEC—KOV TEPs at 1 January 2006 (Real Terms)

Year	Tonnage Milled	Saleable Products: Copper Cathode	Saleable Products: Cobalt Salt	Real Expenditures: Total Working Costs	Real Expenditures: Capital	Real Expenditures: Total Expenditure
	(Mt)	(kt)	(kt)	(US$m)	(US$m)	(US$m)
2006	0.0	0.0	0.0	5.1	172.4	177.4
2007	0.0	0.0	0.0	17.1	373.0	390.1
2008	0.0	0.0	0.0	27.6	319.1	346.7
2009	0.9	34.1	2.1	128.6	358.7	487.3
2010	3.9	157.0	10.1	309.1	54.4	363.5
2011	4.7	240.8	19.9	369.0	28.4	397.4
2012	4.8	251.6	20.8	378.0	22.5	400.6
2013	4.8	221.0	18.7	364.2	13.8	378.0
2014	4.8	186.1	19.6	351.7	15.3	367.0
2015	4.8	201.4	26.9	367.1	43.0	410.1
2016	4.8	228.9	26.1	386.1	76.3	462.5
2017	4.8	233.3	25.7	387.9	17.8	405.6
2018	4.8	235.4	23.2	384.9	48.9	433.8
2019	4.8	238.5	20.8	382.9	17.7	400.6
2020	4.8	208.4	19.2	365.5	114.5	480.0
2021	4.8	183.5	18.3	352.0	25.8	377.8
2022	4.8	186.5	16.7	348.5	19.6	368.1
2023	4.8	183.5	25.7	355.9	60.0	415.8
2024	4.8	183.5	24.0	353.9	52.9	406.8
2025	4.8	183.5	25.7	357.0	10.4	367.3
2026	4.8	183.5	19.2	346.9	22.6	369.5
2027	4.8	183.5	16.7	341.8	42.8	384.6
2028	4.8	183.5	14.7	334.1	73.5	407.6
2029	4.8	214.9	11.4	349.1	12.4	361.5
2030	4.8	210.1	11.4	341.6	34.8	376.4
2031	4.8	208.4	15.5	342.4	15.7	358.1
2032	4.8	207.0	18.7	348.4	50.4	398.7
2033	4.8	205.7	17.1	343.8	11.6	355.4
2034	4.8	235.0	15.1	349.6	17.8	367.4
2035	4.8	240.2	15.1	335.5	9.7	345.2
2036	4.8	237.2	15.1	326.1	9.4	335.5
2037	4.8	233.7	17.5	319.2	9.1	328.3
2038	4.8	219.1	19.4	350.7	9.0	359.7
Total	139.1	6,118.5	550.2	10,421.1	2,163.2	12,584.2

Table 12.2: GEC—Exchange Rates used to convert foreign-currency denominated costs (at 1 July 2006)

Country	Currency	Foreign Units = US$1.00
South Africa	Rand	6.50

The production profile of the copper and cobalt recovered per Table 12.1 is shown in Figure 12.1.

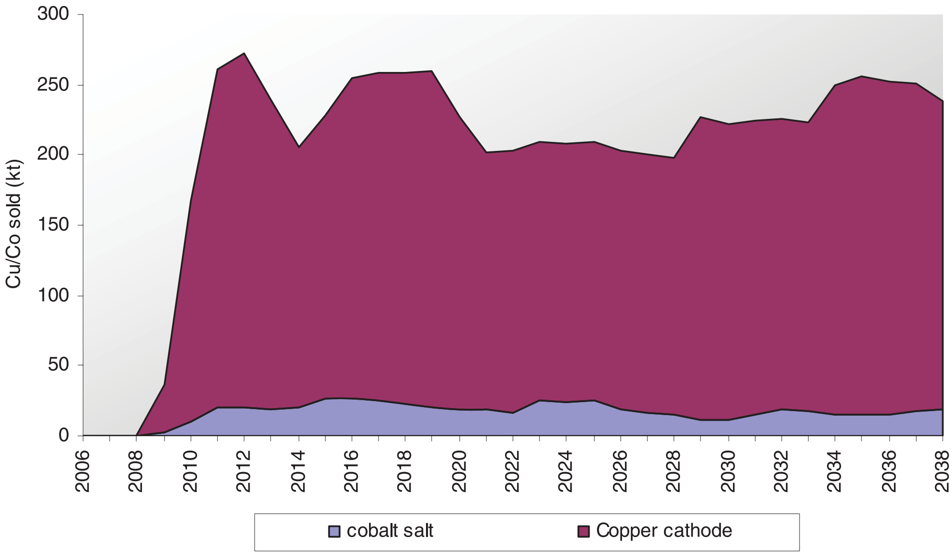


Figure 12.1: Production profile—recovered copper and cobalt

13 MATERIAL PROPERTIES VALUATION

13.1 Introduction

The following section presents discussion and comment on the valuation of the Material Properties. Specifically, comment is included on the methodology used to establish a Base Case including basis of valuation, valuation techniques and valuation results.

13.2 Valuation Methodology

The summary valuation for GEC is based on a sum of the parts approach using:

- The discounted cashflow ("DCF") technique applied on a post-tax pre-finance basis on real cash flows for the KOV project. This is based on the DFS as provided by GEC including the resulting Technical Economic Parameters ("TEPs") (Section 13);
- For the Kananga and Tilwezembe projects, an average value has been derived from a number of different valuation techniques:
  - The value of contained copper and cobalt in declared resources, modified by the weighted effect of a range of technical issues;
  - The DCF value for KOV converted to a US$/t of equivalent copper produced and applied to the copper-equivalent contained in each deposit;
  - Comparable transactions for Cu-Co projects in the DRC during the past 1 to 2 years, converted to a US$/t of copper-equivalent produced; and
  - Future exploration expenditure.

No value has been attached to the Material Contracts.

13.3 Basis of Valuation

13.3.1 KOV

The post-tax pre-finance cash flows for KOV have been extracted from the FM developed by GEC, adjusted where warranted by SRK. The cash flows incorporate the commodity price projections (Table 11.1) and the TEPs (Table 12.1) which are all stated in real constant-money terms. The FM for the KOV project is based on annual cashflow projections ending 31 December, TEPs stated in constant 1 January 2006 money terms and a valuation date of 1 July 2006. The results of this FM are reported in Section 13 and 14.

At the time of writing, no indication of the sensitivity of the Mineral Reserve or LoM plan to commodity price fluctuations was available. In reviewing the FM for KOV, SRK has verified that the following match the relevant entries in the DFS, are correct or have been adjusted by SRK:

- the commodity price forecasts as reflected in Table 11.1;
- all accounting inputs as required for the generation of the FMs in respect of Taxation and Net Movement in Working Capital (debtors, creditors and stores);
- all TEPs;
- the royalty calculations;
- the calculation of real cash flows is in accordance with the fiscal regime of the DRC;
- Calculation formulae and methods; and
- the impact of salvage value on cessation of mining operations is excluded.

Using the FM, as adjusted by SRK where warranted, SRK has derived a Base Case valuation for KOV from which the following are reported:

- Reported a DCF valuation for KOV as at 1 July 2006 which is based on the post-tax pre-finance real cash flows form the FM;
- Performed sensitivity analyses to ascertain the impact of discount factors, commodity prices, total working costs and capital expenditures.

13.4 KOV Valuation: Post-Tax—Pre-Finance Cash Flows

Tables 13.1 and 13.2 present the post-tax pre-finance real cash flows for the KOV Project. Note that these tables are not representative of financial statements as may be customary for determining the consolidated cash flow positions for companies. Further, no account is taken of movements in working capital at the GEC level, or deferrals of tax liabilities between accounting periods, as may be the case in the generation of such financial statements. Each period reflects the financial year from 1 January to 31 December. The KOV valuation is derived from the reported cash flows commencing 1 July 2006.

SRK understands that GEC has not entered into any forward commodity sales or hedging contracts.

Table 13.1: GEC—FM in for KOV Project Real terms (2006 to 2022)

STATISTIC	Financial Year Units Project Year	Totals LoM	2006 1	2007 2	2008 3	2009 4	2010 5	2011 6	2012 7	2013 8	2014 9	2015 10	2016 11	2017 12	2018 13	2019 14	2020 15	2021 16	2022 17
Mining																			
Waste	(Mt)	1,423.2	0.0	4.0	12.0	61.4	66.8	63.0	64.5	64.5	63.0	63.0	64.0	62.0	62.0	62.0	56.2	50.3	50.5
RoM ore	(Mt)	140.0		0.0	0.0	1.2	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8
Processing																			
Plant feed	(Mt)	139.1				0.9	3.9	4.7	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8
Head grade—Cu	(%)	4.84				4.62	4.37	5.64	5.76	5.06	4.26	4.61	5.24	5.34	5.39	5.46	4.77	4.2	4.27
Head geade—Co	(%)	0.47				0.30	0.30	0.50	0.51	0.46	0.48	0.66	0.64	0.63	0.57	0.51	0.47	0.45	0.41
Recovery—Cu	(%)	91%				86%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%
Recovery—Co	(%)	85%				81%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Copper cathode	(kt)	6,118.5				34.1	157.0	240.8	251.6	221.0	186.1	201.4	228.9	233.3	235.4	238.5	208.4	183.5	186.5
Cobalt salt	(kt)	550.2				2.1	10.1	19.9	20.8	18.7	19.6	26.9	26.1	25.7	23.2	20.8	19.2	18.3	16.7
Sales																			
Copper Cathode	(kt)	6,118.5				34.1	157.0	240.8	251.6	221.0	186.1	201.4	228.9	233.3	235.4	238.5	208.4	183.5	186.5
Cobalt Salt	(kt)	550.2				2.1	10.1	19.9	20.8	18.7	19.6	26.9	26.1	25.7	23.2	20.8	19.2	18.3	16.7
Commodity Prices																			
Cu	(US$/lb)		239.7	202.7	159.9	146.3	121.3	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0
Co	(US$/lb)		15.8	14.1	11.5	11.8	10.5	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0
Financial—Real																			
Total Revenue	(US$m)	**25,165.4**	**0.0**	**0.0**	**0.0**	**89.9**	**617.7**	**957.0**	**999.6**	**887.3**	**817.8**	**992.3**	**1043.8**	**1046.8**	**1006.2**	**967.8**	**864.2**	**788.5**	**765.4**
Total Working Costs	(US$m)	**10,421.1**	**5.1**	**17.1**	**27.6**	**128.6**	**309.1**	**369.0**	**378.0**	**364.2**	**351.7**	**367.1**	**386.1**	**387.9**	**384.9**	**382.9**	**365.5**	**352.0**	**348.5**
Mining	(US$m)	1,906.2	0.0	10.2	19.0	64.1	71.1	68.8	68.9	69.6	69.4	68.8	78.0	78.4	77.8	77.8	74.2	73.1	70.6
Processing	(US$m)	4,648.2	0.0	0.0	0.0	32.5	134.0	156.8	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2
General overheads	(US$m)	685.2	5.1	6.8	8.6	21.1	23.0	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2
Marketing	(US$m)	377.5	0.0	0.0	0.0	1.3	9.3	14.4	15.0	13.3	12.3	14.9	15.7	15.7	15.1	14.5	13.0	11.8	11.5
Distribution	(US$m)	1,908.4	0.0	0.0	0.0	6.1	47.9	74.5	77.8	68.6	59.4	66.8	73.9	74.9	74.5	74.2	65.3	58.1	58.3
Environmental Costs	(US$m)	9.5	0.0	0.0	0.0	0.0	0.1	0.2	0.4	0.6	0.7	0.9	1.0	1.2	1.3	1.5	1.7	0.0	0.0
Royalties	(US$m)	850.0	0.0	0.0	0.0	3.5	23.7	32.2	33.6	29.8	27.5	33.4	35.2	35.3	33.9	32.6	29.1	26.5	25.7
Terminal benefits	(US$m)	36.0																	
Working capital movement	(US$m)	0.0																	
Capital Expenditure	(US$m)	**2,163.2**	**172.4**	**373.0**	**319.1**	**358.7**	**54.4**	**28.4**	**22.5**	**13.8**	**15.3**	**43.0**	**76.3**	**17.8**	**48.9**	**17.7**	**114.5**	**25.8**	**19.6**
Project	(US$m)	1,877.3	172.4	373.0	319.1	358.7	54.4	17.8	11.7	3.3	5.1	32.5	65.3	6.6	37.8	6.6	104.0	15.6	9.4
Ongoing	(US$m)	285.9	0.0	0.0	0.0	0.0	0.0	10.6	10.8	10.5	10.2	10.5	11.1	11.1	11.1	11.0	10.6	10.3	10.2
Taxation Liability	**(US$m)**	**3,789.5**	0.0	0.0	0.0	0.0	0.0	0.0	49.7	140.5	133.5	177.4	180.7	190.0	173.0	166.7	124.2	120.4	115.4
Final Net Free Cash—Real	**(US$m)**	**8,791.7**	**-177.4**	**-390.1**	**-346.7**	**-397.4**	**254.2**	**559.6**	**549.4**	**368.8**	**317.3**	**404.8**	**400.6**	**451.1**	**399.4**	**400.5**	**260.0**	**290.4**	**282.0**

Table 13.2: GEC—FM in for KOV Project Real terms (2022 to 2038)

STATISTIC	Financial Year / Units / Project Year	Totals LoM	2023 18	2024 19	2025 20	2026 21	2027 22	2028 23	2029 24	2030 25	2031 26	2032 27	2033 28	2034 29	2035 30	2036 31	2037 32	2038 33
Mining																		
Waste	(Mt)	1,423.2	39.6	44.7	46.9	42.7	40.4	41.5	59.0	46.7	39.2	43.8	39.7	33.8	18.2	11.2	3.5	3.1
RoM ore	(Mt)	140.0	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.4
Processing																		
Plant feed	(Mt)	139.1	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8
Head grade—Cu	(%)	4.84	4.2	4.2	4.2	4.2	4.2	4.2	4.92	4.81	4.77	4.74	4.71	5.38	5.5	5.43	5.35	5.02
Head geade—Co	(%)	0.47	0.63	0.59	0.63	0.47	0.41	0.36	0.28	0.28	0.38	0.46	0.42	0.37	0.37	0.37	0.43	0.48
Recovery—Cu	(%)	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%
Recovery—Co	(%)	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Copper cathode	(kt)	6,118.5	183.5	183.5	183.5	183.5	183.5	183.5	214.9	210.1	208.4	207.0	205.7	235.0	240.2	237.2	233.7	219.1
Cobalt salt	(kt)	550.2	25.7	24.0	25.7	19.2	16.7	14.7	11.4	11.4	15.5	18.7	17.1	15.1	15.1	15.1	17.5	19.4
Sales																		
Copper Cathode	(kt)	6,118.5	183.5	183.5	183.5	183.5	183.5	183.5	214.9	210.1	208.4	207.0	205.7	235.0	240.2	237.2	233.7	219.1
Cobalt Salt	(kt)	550.2	25.7	24.0	25.7	19.2	16.7	14.7	11.4	11.4	15.5	18.7	17.1	15.1	15.1	15.1	17.5	19.4
Commodity Prices																		
Cu	(US$/lb)		110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0	110.0
Co	(US$/lb)		10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0
Financial—Real																		
Total Revenue	(US$m)	**25,165.4**	**926.0**	**895.5**	**926.0**	**803.8**	**758.0**	**719.8**	**735.0**	**723.3**	**795.5**	**853.4**	**819.7**	**852.4**	**865.2**	**857.7**	**895.1**	**894.6**
Total Working Costs	(US$m)	**10,421.1**	**355.9**	**353.9**	**357.0**	**346.9**	**341.8**	**334.1**	**349.1**	**341.6**	**342.4**	**348.4**	**343.8**	**349.6**	**335.5**	**326.1**	**319.2**	**350.7**
Mining	(US$m)	1,906.2	67.1	67.3	68.2	67.0	65.1	60.2	67.3	61.7	57.7	59.8	57.9	55.0	38.9	30.7	21.9	20.6
Processing	(US$m)	4,648.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2	160.2
General overheads	(US$m)	685.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2	22.2
Marketing	(US$m)	377.5	13.9	13.4	13.9	12.1	11.4	10.8	11.0	10.9	11.9	12.8	12.3	12.8	13.0	12.9	13.4	13.4
Distribution	(US$m)	1,908.4	61.3	60.6	61.3	58.5	57.4	56.5	63.7	62.4	63.7	64.8	63.7	70.8	72.2	71.4	71.5	68.3
Environmental Costs	(US$m)	9.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Royalties	(US$m)	850.0	31.2	30.2	31.2	27.1	25.5	24.2	24.7	24.3	26.7	28.7	27.6	28.6	29.1	28.8	30.1	30.1
Terminal benefits	(US$m)	36.0																36.0
Working capital movement	(US$m)	0.0																
Capital Expenditure	(US$m)	**2,163.2**	**60.0**	**52.9**	**10.4**	**22.6**	**42.8**	**73.5**	**12.4**	**34.8**	**15.7**	**50.4**	**11.6**	**17.8**	**9.7**	**9.4**	**9.1**	**9.0**
Project	(US$m)	1,877.3	49.7	42.7	0.1	12.5	32.8	63.7	2.2	24.8	5.8	40.3	1.7	7.7	0.0	0.0	0.0	0.0
Ongoing	(US$m)	285.9	10.2	10.2	10.3	10.1	10.0	9.8	10.2	10.0	9.9	10.1	10.0	10.1	9.7	9.4	9.1	9.0
Taxation Liability	**(US$m)**	3,789.5	154.9	146.4	164.1	128.7	112.8	98.6	109.1	103.7	128.2	138.3	137.3	144.3	154.3	155.1	169.7	172.6
Final Net Free Cash—Real	**(US$m)**	**8,791.7**	**355.2**	**342.3**	**394.6**	**305.6**	**260.6**	**213.6**	**264.5**	**243.2**	**309.2**	**316.3**	**327.0**	**340.8**	**365.7**	**367.1**	**397.1**	**362.3**

13.5 KOV—Net Present Values and Sensitivities

The following section presents the NPV of the nominal cash flow as derived from the FM for KOV.

The various NPV tables include the following:

- NPVs at a range of discount factors in relation to the WACC (Table 13.3);
- NPV sensitivity to sales revenue, total working costs and capital expenditure derived from single parameter sensitivity analysis at the WACC (Table 13.4); and
- NPV sensitivity to sales revenue and total working costs derived from twin parameter sensitivity analysis at the WACC (Table 13.5).

Table 13.3: KOV—NPV at various discount factors

Discount Factor (%)	NPV (US$m)
0.0%	8,789.9
2.0%	5,944.0
4.0%	4,105.8
6.0%	2,881.0
8.0%	2,040.5
10.0%	1,447.9
12.0%	1,019.7

Table 13.4: KOV—Single parameter NPV sensitivity at 10% discount

Sensitivity Range—Revenue	**-30%**	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
Sensitivity Range—Total Working Cost	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**
Sensitivity Range—Capital Expenditure	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**
	(US$m)	**(US$m)**	**(US$m)**	**(US$m)**	**(US$m)**	**(US$m)**	**(US$m)**
Revenue	(284.9)	292.7	870.3	1,447.9	2,025.5	2,603.2	3,180.8
Total Working Cost	2,147.5	1,914.3	1,681.1	1,447.9	1,214.7	981.5	748.3
Capital Expenditure	1,818.8	1,695.2	1,571.6	1,447.9	1,324.3	1,200.7	1,077.0

Table 13.5: KOV—Twin parameter NPV sensitivity at 10% discount (Revenue, Working Costs)

		Revenue Sensitivity						
NPV (US$m)		**-30%**	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
	-15%	65.0	642.6	1,220.3	1,798.0	2,375.7	2,953.4	3,531.0
Total	**-10%**	(51.7)	526.0	1,103.7	1,681.4	2,259.1	2,836.8	3,414.4
Working	**-5%**	(168.3)	409.4	987.1	1,564.8	2,142.5	2,720.1	3,297.8
Costs	**0%**	(284.9)	292.8	870.5	1,448.2	2,025.9	2,603.5	3,181.2
Sensitivity	**5%**	(401.5)	176.2	753.9	1,331.6	1,909.2	2,486.9	3,064.6
	10%	(518.1)	59.6	637.3	1,215.0	1,792.6	2,370.3	2,948.0
	15%	(634.7)	(57.0)	520.7	1,098.3	1,676.0	2,253.7	2,831.4

Note that the above NPV's relate to cashflows attributable to the KOV Project and directly to DCP, and do not represent the quantum attributable to GEC.

13.6 Valuation of Kananga and Tilwezembe

13.6.1 Metal in the ground approach

A valuation of the Resources reported for the Kananga and Tilwezembe projects has been done using an in-situ value of metal in the ground approach. For Inferred Resources, a percentage varying from 2% to 4.5% of the Cu/Co prices at 1 January 2006 is applied on a US$/lb basis on the contained copper and cobalt. For Indicated Resources, a percentage varying from 5% to 10% of the Cu/Co prices has been applied. These percentages are modified by assessing the weighted affect of a range of technical issues on the risk or likelihood of the resources being mined. The issues considered and their relative weightings are set out in Table 13.6.

Table 13.6: Risk-impact matrix for adjusting Copper/Cobalt price percentage

Risk associated with:	High Impact	Medium Impact	Low Impact
Grade not present	0.1	0.5	1.0
Geotechnical aspects affecting mining	0.1	0.5	1.0
Technical mining conditions	0.1	0.5	1.0
Ease of mining and access	0.1	0.5	1.0
Cost of mining	0.1	0.5	1.0
Metallurgical aspects of ore	0.1	0.5	1.0
Process costs	0.1	0.5	1.0
Proximity of existing infrastructure (e.g. rail way lines)	0.1	0.5	1.0
Capital considerations	0.1	0.5	1.0

SRK has used the long-term prices for copper and cobalt, from 2010 onwards, as contained in the FM developed by GEC of US$1.10/lb and US$10.00/lb respectively.

The impact of each of the factors in Table 13.6 was assessed for Kananga and Tilwezembe and together with the percentages of the Cu/Co prices above and the Cu/Co prices in US$/lb were applied to the contained copper and cobalt as given in Table 4.7. Note that the contained copper and cobalt in Table 13.7 is the gross in-situ figure and not the quantum attributable to GEC. The resultant value ranges for the two projects are summarised in Table 13.7.

Table 13.7: Risk-adjusted value ranges for Kananga and Tilwezembe

	Contained Metal (Gross in-situ)			Value Range	
Project	Cu	Co	Weighting	Low	High
	(kt)	(kt)		(US$m)	
Kananga	143	89	0.79	37.9	85.3
Tilwezembe	96	67	0.63	56.3	112.6

13.6.2 DCF value for KOV converted to a US$/t Cu-eq

The DCF value for KOV derived per section 14.5 has been converted to a US$/t of equivalent copper produced and applied to the in-situ copper-equivalent in Kananga and Tilwezembe (Table 13.8). The weighting to get to the copper-equivalent price has used the long-term copper and cobalt prices given above in Section 13.6.1 and factors to reflect the level of confidence in the resource classifications.

Table 13.8: Values for Kananga and Tilwezembe based on KOV DCF value

Project	Contained Cu-equiv metal (gross in situ)	KOV NPV value @ 10% discount per tonne Cu-equiv	Value
	(kt)	(US$/t Cu-eq)	(US$m)
Kananga	182	152.0	27.6
Tilwezembe	336	152.0	51.1

Note that the values in Table 13.8 relate to the gross in-situ Cu-equivalent tonnage and not the quantum attributable to GEC.

13.6.3 Comparable transactions

SRK has examined comparable transactions and/or press announcements regarding Cu-Co projects in the DRC during the last year and extracted a value converted to a US$/t of copper-equivalent produced for each project. A brief description of each project is given here:

- **Kamoto JV**—Katanga Mining announced in a press release dated 11 April 2006 the results of an independent feasibility study prepared on the Kamoto Mine located near Kolwezi. The study covers the first twenty years of production from Proved and Probable Mineral Reserves from which 2.16Mt of copper and 0.109Mt of cobalt are recovered. Using metal prices of US$1.1/lb and US$10/lb for copper and cobalt respectively, the project returns a NPV of US$649million @ 6% discount, after financing of capital costs;
- **Kalukundi JV**—At the Emerging Central African Copper & Gold Conference of 30 March 2006, Rubicon Minerals Corp presented the results of its investigations at the Kalukundi Mine located east of Kolwezi. From Measured and Indicated Resources of 12.2Mt grading 2.4%Cu and 0.6%Co and preliminary economic/cost assumptions, Rubicon assigned a value of US$209/t of metal;
- **Kolwezi Tailings**—At the Emerging Central African Copper & Gold Conference of 30 March 2006, Adastra Minerals Corp presented the results of a definitive feasibility study on the retreatment of tailings material located near Kolwezi. The study reported that there were some 113Mt of Proved and Probable Mineral Reserves grading 1.49%Cu and 0.32%Co. Using metal prices of US$1.1/lb and US$12/lb for copper and cobalt respectively, the project estimated a return of US$100/t of metal;
- **Tenke Fungurume**—At the Emerging Central African Copper & Gold Conference of 30 March 2006, Tenke Mining Corp indicated that it expected a cash operating cost of about US$0.27/lb Cu with a 4:1 revenue to cost ratio from the feasibility study which is due for completion by mid-2006. Tenke has reported Measured and Indicated Mineral Resources of 233Mt grading 2.9%Cu and 0.3%Co. The project return has been estimated as US$127/t of metal;
- **Ruashi**—During 2005, Metorex Limited commenced the construction of the Phase I flotation plant aimed to treat the 3.2Mt of stockpiled material at Ruashi, grading 1.76%Cu and 0.49%Co. Phase II of the project will involve open pit mining of the 32Mt in the Ruashi orebodies and treatment through an expanded concentrator and SX/EW plant. Using metal prices of US$0.91/lb and US$10/lb for copper and cobalt respectively, the two phases of the project were shown to yield a return of US$193million @ 12% discount.

These comparable transaction values have been applied to the total equivalent copper content of the Kananga and Tilwezembe projects (Table 13.8) and the resulting values are shown in Table 13.9.

Table 13.9: Comparable Transaction values for Kananga and Tilwezembe

Comparable Project	Project value per tonne Cu-equiv	Value Kananga	Value Tilwezembe
	(US$/t Cu-eq)	(US$m)	
Kamoto JV (Katanga Mining)	206	38.1	70.4
Kalukundi JV (Rubicon Minerals)	209	38.0	70.2
Kolwezi Tailings (Adastra Minerals/First Quantum)	100	18.2	33.6
Tenke Fungurume (Tenke Mining/Phelps Dodge)	127	23.1	42.7
Ruashi (Metorex)	139	25.3	46.8

Note that the values in Table 13.9 relate to the gross in-situ Cu-equivalent tonnage and not the quantum attributable to GEC.

13.6.4 Future exploration expenditure.

Based on the planned exploration expenditure as set out in Table 3.1, exploration-based valuations for the Kananga and Tilwezembe projects are set out in Table 13.10.

Table 13.10: Exploration-based values for Kananga and Tilwezembe

Project	Exploration-based Value
	(US$m)
Kananga	5.3
Tilwezembe	3.3

13.6.5 Summary Values

In summary, the alternative values attached to the Kananga and Tilwezembe projects are set out in Tables 13.11 and 13.12 respectively.

Table 13.11: Alternative Values for Kananga

Alternative		Value Range
Metal in ground	(US$m)	37.9 to 85.3
KOV DCF value as US$/t Cu-eq	(US$m)	27.6
Comparable transaction as US$/t Cu-eq	(US$m)	18.2 to 38.0
Exploration budget (2006/8)	(US$m)	5.3

Table 13.12: Alternative Values for Tilwezembe

Alternative		Value Range
Metal in ground	(US$m)	56.3 to 112.6
KOV DCF value as US$/t Cu-eq	(US$m)	51.1
Comparable transaction as US$/t Cu-eq	(US$m)	33.6 to 70.4
Exploration budget (2006/8)	(US$m)	3.3

SRK has assigned the average of the numbers in Tables 13.11 and 13.12 as the values for Kananga and Tilwezembe, US$37m and US$55m respectively. Note that these values relate to the gross in-situ Cu-equivalent tonnage and not the quantum attributable to GEC.

13.7 Valuation for GEC

Assigning a value to GEC is not simply a case of taking a 75% share of the DCP valuation. In terms of the JVA, GEC funds Gécamines' share of the capital requirement and records this as a loan. The loan together with interest at an agreed rate, is then repaid from future cashflows before any amounts are paid to Gécamines.

This process is correctly incorporated into the FM and is the basis on which a portion of the KOV value has been assigned to GEC (Table 14.1).

14 SUMMARY VALUATION AND CONCLUDING REMARKS

14.1 Summary Valuation

The summary valuations ascribed to GEC and DCP set out in Table 14.1 are based on a sum of the parts approach, as described in Section 14 of this CPR.

Table 14.1: Summary Valuation for DEC and GEC

		Value Attributable to	
Property	Units	DCP	GEC
KOV (DCF @ 10%)	(US$m)	1,447.9	1,085.9
Kananga	(US$m)	37.0	27.7
Tilwezembe	(US$m)	55.0	41.4
Total Asset Valuation	**(US$m)**	**1,539.9**	**1,155.0**
Adjustments:			
Estimated net (debt)/cash at 1 July 2006	(US$m)	26	26
Equity Value	**(US$m)**	**1,513.9**	**1,129.0**

14.2 Concluding Remarks

SRK has conducted a comprehensive review and assessment of all material issues likely to influence the future operations of GEC and DCP as these relate to the KOV Project. The Kananga and Tilwezembe properties have been treated as exploration projects for valuation purposes. In terms of the JVA with Gecamines, DCP has the right to use or access to the Processing Facilities. As these facilities do not relate directly to the DFS for KOV, they have been examined on the basis of material contracts and excluded from the valuation process.

Glossary of terms, abbreviations and units

Glossary

aeolian	erosion, transport, and deposition of material due to the action of wind at or near the earth's surface
arenaceous	term describing sedimentary rocks with a modal grain size in the sand fraction
argillaceous	term describing sedimentary rocks with a modal grain size in the silt fraction
assaying	the chemical analysis of mineral samples to determine the metal content
basement complex	the widespread association of igneous and metamorphic rocks which are covered uncomfortably by unmetamorphosed sediments
basal conglomerate	a conglomerate formed at the earliest portion of a stratigraphical unit
basinal	a basin like depression that may be erosional or structural in origin.
biotite/titic	trioctahedral
bornite	a copper ore, Cu_5FeS_4, found in copper veins
chalcocite	a copper ore, Cu_2S, found mainly in the enriched zones of sulphide deposits
chalcopyrite	the maincopper ore, $CuFeS_2$, found mainly in veins
coefficient of variation	the coefficient of variation is a dimensionless number that allows comparison of the variation of populations that have significantly different mean values
concentrate	a metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
crushing	initial process of reducing ore particle size to render it more amenable for further processing
cut-off grade	the grade of mineralised rock which determines as to whether or not it is economic to recover its gold content by further concentration
decline	a surface or sub-surface excavation in the form of a tunnel which is developed from the uppermost point downwards
detrital	a term applied to any particles of minerals, or, more rarely, rocks, which have derived from pre-existing rock by processes of weathering and erosion
dilution	waste which is unavoidably mined with ore
dip	angle of inclination of a geological feature/rock from the horizontal
dolomites	rocks with more than 15% magnesium content
drill-hole	method of sampling rock that has not been exposed
effective date	date on which all assumptions in the report were finalised and the date from which the valuation was done
fault	the surface of a fracture along which movement has occurred
feldspar/s	the most important single group of rock forming silicate minerals

filtration	process of separating usually valuable solid material from a liquid
flat	flatter dipping areas
flotation	the process by which the surface chemistry of the desired mineral particles is chemically modified such that they preferentially attach themselves to bubbles and float to the pulp surface in specially designed machines. the gangue or waste minerals are chemically depressed and do not float, thus allowing the valuable minerals to be concentrated and separated from the undesired material.
geochronological	the measurementof time intervals on a geological scale
grade	the measure of concentration of copper or cobalt within mineralised rock
hangingwall	the overlying side of an orebody or slope
haulage	a horizontal underground excavation which is used to transport mined ore
hydrogeology	a science that deals with sub-surface water and with related geologic aspects of surface water
Indicated Mineral Resource	that part of a mineral resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. it is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. the locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed
Inferred Mineral Resource	that part of a mineral resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. it is inferred from geological evidence and assumed but not verified geological and/or grade continuity. it is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability
intrusives	a body of igneous rock which has forced itself onto pre-existing rocks, either along some definite structural feature or by defamation or cross-cutting of the invaded rocks
lithology/ical	geological description pertaining to different rock types
Measured Mineral Resource	that part of a mineral resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. it is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. the locations are spaced closely enough to confirm geological and grade continuity
Metasediments/tory	metamorphosed sedimentary rock
mica/micaceous	layer-lattice minerals of the three-layer type, and may be divided into the dioctahedral muscovite group and the trioctahedral phlogopite-biotite group

milling	a general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product
Mineral Reserve	the economically mineable material derived from a measured and/or indicated mineral resource. it is inclusive of diluting materials and allows for losses that may occur when the material is mined. appropriate assessments, which may include feasibility studies, have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. these assessments demonstrate at the time of reporting that extraction is reasonably justified. mineral reserves are sub-divided in order of increasing confidence into probable mineral reserves and proved mineral reserve
Mineral Resource	a concentration or occurrence of material of economic interest in or on the earth's crust in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. the location, quantity, grade, continuity and other geological characteristics of a mineral resource are known, estimated from specific geological evidence and knowledge, or interpreted from a well constrained and portrayed geological model. mineral resources are sub-divided in order of increasing confidence, in respect of geoscientific evidence, into inferred, indicated and measured categories
mineral rights	a right or any share therein acquired, in terms of the minerals act to any right to dig or mine
mining assets	the material properties and significant exploration properties as defined in Section 1.1
mining authorisation	any authorisation issued in terms of the minerals act
on-going capital	capital estimates of a routine nature which are necessary for sustaining operations
ore reserve	see mineral reserve
orogeny	an orogeny is a period of mountain building leading to the intensely deformed belts which constitute mountain ranges.
Probable Mineral Reserve	the economically mineable material derived from a measured and/or indicated mineral resource. It is estimated with a lower level of confidence than a proved mineral reserve. It is inclusive of diluting materials and allows for losses that may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, and including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction is reasonably justified.
project capital	capital expenditure which is associated with specific projects of a non-routine nature

proterozoic	era of geological time between 2.5 x 109 and 570 x 106 years ago
Proved Mineral Reserve	the economically mineable material derived from a measured mineral resource. it is estimated with a high level of confidence. it is inclusive of diluting materials and allows for losses that may occur when the material is mined. appropriate assessments, which may include feasibility studies, have been carried out, including consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. these assessments demonstrate at the time of reporting that extraction is reasonably justified
pyrititic (pyrite)	the most widespread sulphide mineral, FeS_2
remnant	ore blocks left behind as a result of the underground mining method
SAMREC code	south african code for reporting of mineral resources and mineral reserves
schist/s	a regionally metamorphasised rock chracterised by a parallel arrangement of the bulk of the constituent minerals
sedimentary	pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks
shaft	an opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste
smelting	a high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
SRK Group	srk global limited
stratigraphy	study of stratified rocks in terms of time and space
steep	steeply dipping areas
syncline/clinal/clinorium	a basin shaped fold
tailings	finely ground waste rock from which valuable minerals or metals have been extracted
total cash costs	all total cash costs are based on public quoted nominal production costs, include retrenchment costs, rehabilitation costs, corporate costs, by-product credits for silver, sundry revenues, and exclude amortisation costs and inventory changes
total expenditure	all expenditures including those of a operating and capital nature
trust fund	a fund required by law to be set up, to which annual contributions are paid so that the remaining environmental liability of the operation is covered
volcanics	one of three groups into which rocks have been divided. The vocanic assemblage includes all extrusive rocks and associated intrusive ones
volcanics/volcanoclatics	one of three groups into which rocks have been divided. The vocanic assemblage includes all extrusive rocks and associated intrusive ones

Abbreviations

ADTs	articulated dump trucks
AIDS	the Acquired Immune Deficiency Syndrome
AIM	Alternative Investment Market of the London Stock Exchange
BID	base information date
GEC	Global Enterprises Corporate Ltd.
CM	Chambishi Metals plc
CoG	cut-off grade
CoSX	Cobalt solvent extraction
CPI	Consumer Price Index
CP	Competent Person
CPR	Competent Persons' Report
CuSX	Copper solvent extraction
DCF	discounted cash flow
DRC	Democratic Republic of Congo
EIS	Environmental Impact Statement
EMP	Environmental Management Programme
EMPR	Environmental Management Programme Report
FM	Financial Models
GDP	Gross Domestic Product
HIV	Human immuno deficiency virus
HT	on-mine high tension
ISO	International Standards Organisation
IMM	International Mining and Metallurgy
LHD	Load haul dump
LME	London Metal exchange
LML	Large scale mining licences
LMPI	low manganese pig iron
LSE	London Stock Exchange
MSP	Mineral Separation Plant
NPV	Net present value
PPCF	cut and fill
RoM	Run-of-Mine
RoR	Rate of Rise
RSA	Republic of South Africa
SAMREC	South African Mineral Resource Committee
SANAS	South African National Accreditation System
SANS	South African National Standard
SHE	Safety, Health and Environment
SHEQ	Safety, Health, Environment and Quality
SLOS	sub-level open stoping
SRK	SRK Consulting (South Africa) (Proprietary) Limited.
SRK Group	SRK Global Limited
TBL	terminal benefits liability
TEC	Total Employees Costed
TE Models	Technical Economic Models
TEMs	Technical Economic Models
TEPs	Technical Economic Parameters
WACC	weighted average cost of capital
WHO	World Health Organisation

Units

cm	a centimetre
cmg/t	a centimetre gramme per tonne—a measure of metal accumulation
cfm	cubic feet per metre
g	grammes
ha	a Hectare
hrs	hours
k	one thousand units
kg	a kilogram
km	a kilometre
kt	a thousand metric tonnes
ktpm	a thousand metric tonnes per month
ktpa	a thousand metric tonnes per annum
m	a metre
m^2	a square metre—measure of area
m^3	a cubic metre
mm	a millimetre
ms^{-1}	metres per second
Mt	a million metric tonnes
Mtpa	a million metric tones per annum
MVA	a million volt-amperes
MW	a million watts
pa	per annum
Pa	a Pascal—a measure of pressure
s	a second
t	a metric tonne
tpa	metric tonne per annum
tpd	metric tonne per day
tpm	metric tonne per month
tm^{-3}	density measured as metric tonnes per cubic metre
US$m	a million United States Dollars
US$	United States Dollar
US$/lb	United States Dollars per pound
US$/t	United States Dollars per tonne
ZAR	South African Rand
ZARm	South African Rand million
ZAR/t	Rand per tonne
°	degrees
°C	degrees centigrade
ʻ	minutes
%	percentage

Chemical elements

Co	Cobalt
Cu	Copper
CuO	Copper Oxide
Fe	Iron
TCo	Total Cobalt
TCu	Total Copper

Part XI

Glossary of technical terms

The following definitions shall apply to the technical terms used herein:

"beneficiation"	the treatment of raw material by drying, flotation, gravity or magnetic separation
"cathode"	the cathode is the conductor through which electricity leaves the cell. For copper refining the cathode is where the refined copper is deposited
"Co"	cobalt
"concentrate"	material which has been processed to increase the percentage of the valuable mineral to facilitate transportation and downstream processing
"CAGR"	cumulative annual growth rate
"COSAC"	cobalt from slag and copper as a by-product
"Cu"	Copper
"electro-refining"	the process of refining impure copper metal by passing a current through a copper sulphate solution using impure copper anodes and copper stainless steel cathodes
"electro-winning"	the process of extracting copper from copper sulphate solutions by passing a current through the solution using a lead anode and either a copper or stainless steel cathode
"EPC"	engineering procurement contract
"grade"	proportion (by weight) of the valuable element within the mineralised rock
"hydro-metallurgical"	metallurgical processing in aqueous environments
"Indicated Mineral Resources"	that part of a mineral resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed
"Inferred Mineral Resources"	that part of a mineral resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability
"kt"	kilotonnes
"ktpa"	kilotonnes per annum
"lb"	imperial pound (mass) equivalent to 0.4536 kilogrammes
"leaching"	extracting a soluble metallic compound from an ore by selectively dissolving it in a suitable solvent

"lime"	calcium carbonate ($CaCo_3$)
"LME"	London Metal Exchange
"mine life"	the remaining life of the mine in years calculated by deducting the scheduled production rates (i.e., the rate at which material will be removed from the mine to the current defined reserves)
"mineral"	a natural, inorganic, homogeneous material that can be expressed by a chemical formula
"mineral resource"	a tonnage or volume of rock or mineralisation of intrinsic economic interest
"mineralisation"	the process by which minerals are introduced into a rock. More generally, a term applied to accumulations of potentially economic or related minerals in quantities ranging from anomalous to economically recoverable
"Mt"	million tonnes
"Mtpa"	million tonnes per annum
"MW"	megawatt
"nameplate capacity"	the initial capacity or throughput of fixed facilities such as a processing plant or material handling system as estimated by the design engineers
"open pit"	surface mining in which the ore is extracted from a pit or quarry
"ore"	a mineral or mineral aggregate containing precious or useful minerals in such quantities, grade and chemical combination to make extraction economic
"oxide"	a compound of oxygen with another element or group
"plant"	fixed or moveable equipment required in the process of winning or processing the ore
"Probable Reserves"	those measured and/or indicated mineral resources which are not yet "proved", but of which detailed technical and economic studies have demonstrated that extraction can be justified at the time of the determination and under specified economic conditions
"refining"	the final process of upgrading the metal quality
"reserves"	those parts of mineral resources for which sufficient information is available to enable detailed or conceptual mine planning and for which such planning has been undertaken. Reserves are classified as either proved or probable
"resources"	all of the potential minerals in a defined area based on points of observation and extrapolations from those points. Potential minerals are defined as minerals which have been or could be beneficiated to give a quality acceptable for commercial usage in the foreseeable future

"RoM"	a mineral in its raw, untreated state subsequent to extraction and prior to sizing and other beneficiation
"SAMREC Code"	the South African Code for Reporting of Mineral Resources and Mineral Reserves including the guidelines contained therein
"smelting"	thermal process whereby molten metal is liberated from a concentrate, with impurities separating into a lighter slag
"strike" or "strikes"	the direction or bearing of a bed or layer of rock measured along a horizontal line along the bed or layer
"t" or "tonne"	metric tonne equivalent to 2,204.62 pounds
"tailings"	the waste material produced from ore after economically recoverable metals or minerals have been extracted. Changes in metal prices and improvements in technology can sometimes make the tailings economic to process at a later date
"tpa"	tonnes per annum
"tpd"	tonnes per day