KAMOTO COPPER COMPANY

EXECUTIVE SUMMARY REPORT

for

KAMOTO REDEVELOPMENT PROJECT

HATCH™

,

ISO 9001

Table of Contents

Table of Contents – Tables

Table of Contents – Figures

Appendices

Financial Models
A.1 KCC Base Case 50% Third Party Debt
A.1.1 Amortized Debt Base Case
A.1.1.1 150,000 tonnes copper per year scenario

1. INTRODUCTION

The Kamoto Copper Company feasibility study has been commissioned by Kinross Forrest Limited (KFL) the owner of a 75% interest in the Kamoto Joint venture. The other 25% of the Kamoto Joint venture is owned by La Générale des Carrières et des Mines (Gécamines). The feasibility study was commissioned to develop a comprehensive plan for the rehabilitation and redevelopment of the Kamoto mine and related infrastructure located near Kolwezi in the Democratic Republic of the Congo (DRC). Katanga Mining Limited (KML) holds an option to purchase one-hundred percent of Kinross Forrest Limited in the Joint Venture.

This Report was prepared for KFL by a team of companies. HATCH was responsible for developing the metallurgical /plant engineering studies including mechanical/electrical engineering; surface infrastructure and financial modelling studies. McIntosh RSV LLC, in association with Caracle Creek International Consulting Inc. were responsible for the resource and reserve studies, including mine planning, and SRK Consulting Engineers and Scientists developed the environmental, tailings and groundwater studies.

The feasibility study premise was based in part on a Pre-Feasibility study that was completed by Hatch in 2003. The feasibility study has confirmed the general concepts of a phased redevelopment, restoration of economically viable operations within a very short time and low capital costs relative to the restored production capacity.

The objectives of the feasibility study were defined as follows:

- Complete a mineral resource and reserve evaluation in compliance with international standards;
- Complete a mine design and mine plan to support the mineral reserve estimate;
- Develop and define the production ramp up for the plant facilities in line with the mine plan;
- Complete an Environmental Impact Assessment;
- Define in detail the Scope of Work for the plant areas, underground and open pit mines and infrastructure, necessary to achieve the ramp up plan;
- Carry out sufficient engineering to enable the project capital and operating costs to be defined;
- Update and Refine the financial model developed in the Pre-Feasibility Study based on the results of the Feasibility Study.

Because of the extensive use of used equipment and the inherent issues surrounding refurbishment and replacement issues in the existing plants a slightly lower estimation accuracy is introduced when compared to a typical feasibility study. However, significant effort has gone into the study to identify and account for critical pieces of equipment in order to restore and maintain reliable operations.

2. STUDY METHODOLOGY

The re-establishment of operations is based on a phased approach over a four year period. This was based on an assessment of the condition of the plant sections, the capacity constraints of the facilities and the condition of the mines. From this, logical and cost effective incremental throughput steps were established.

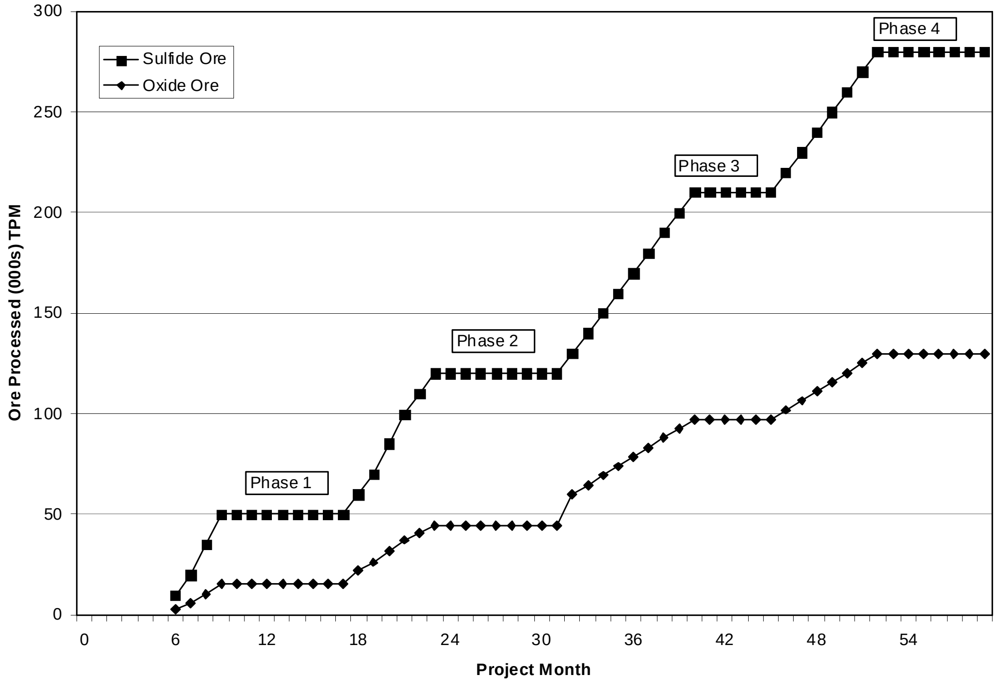

Figure 1 - Production Phases

The production rates of phase 4 were maintained for the 20-year analysis period.

2.1 Study Results

The study demonstrates that economically viable operations can be restored within a relatively short time frame. Capital costs, on a comparative basis for the restored production capacity, are very low. On an annual basis the operation becomes cash neutral in year four and cash positive in year eight.

Operating and capital costs, per pound of copper, during the project are summarized as follows:

	Phases 1-3	Phase 4 Onward	Life of Project
Site Operating cost	$1.10	$0.71	$0.75
Cobalt Credit			($0.52)
Total After Cobalt Credit			$0.22
Transportation and Marketing Expenses			$0.16
Royalty & Lease			$0.05
Initial Capital			$0.09
Sustaining Capital			$0.05
Total Production Costs			$0.58

Note: Columns may not add due to rounding

Table 1 - Operating and Capital Costs (per pound of Copper)

The amortised debt discounted cash flow evaluation of the KCC redevelopment project shows an IRR of 23.8% and a NPV 612 million USD using a 6% discount rate and an 8.5% debt rate (Appendix A.1.1).

The financial base case carries the following assumptions:

- Execution capital cost USD 426.7 million:
- Sustaining capital costs USD 231.3 million:
- Evaluation Period (LOM) 20 years:
- Copper revenue USD 1.10/lb:
- Cobalt revenue USD 10/lb:
- Total production of copper throughout LOM 2.17 millions tonnes (4 778 million lb);
- Total production of cobalt throughout LOM 0.113 millions tonnes (250 million lb).

The project is most sensitive to a change in copper recovery and operating costs.

Appendix A.1 shows the KCC financial base case based on 50% third party debt.

3. RESOURCES

The Property's mineral reserves and resources as of May 16, 2006 are as follows:

Kamoto Mineral Reserve Estimate, May 16, 2006

Classification	Ore Tonnes (000s)	Copper Grade %	Cu Tonnes (000s)	Cobalt Grade %	Co Tonnes (000s)
Proven Mineral Reserves	75,583	3.15%	2,382	0.32%	241
Probable Mineral Reserves	17,017	3.19%	542	0.27%	47
Proven + Probable Reserves	92,600	3.16%	2,924	0.31%	288

Notes: Mineral reserves are separate from mineral resources.

Table 2 - Mineral Reserve Summary

Kamoto Mineral Resource Estimate, May 16, 2006

Classification	Ore Tonnes (000s)	Copper Grade %	Cu Tonnes (000s)	Cobalt Grade %	Co Tonnes (000s)
Measured Mineral Resources	51,174	3.44%	1,760	0.41%	211
Indicated Mineral Resources	17,728	3.54%	628	0.33%	59
Measured + Indicated Resources	68,902	3.47%	2,388	0.39%	270
Inferred Mineral Resources - Kamoto	11,826	5.28%	624	0.15%	18
Inferred Mineral Resources – Open Pits	17,493	3.41%	596	0.32%	56
Total Inferred Mineral Resources	29,319	4.16%	1,220	0.25%	74

Notes: Mineral resources are exclusive to mineral reserves.

Table 3 - Mineral Resource Summary

4. OVERVIEW

Corporate mining activity in Katanga began in 1906 with the formation of Union Miniere du Haut Katanga (UMHK). In 1967, following national independence the operations of UMHK were nationalized and incorporated as La Générale des Carrières et des Mines (Gécamines). At its peak, Gécamines produced about 7 percent of global copper mine production and 62 percent of global cobalt production. In 1986, Gécamines produced 476,000 tonnes of copper and 14,500 tonnes of cobalt, 63,900 tonnes of zinc, 34.3 tons of silver plus cadmium and other minor metals. The majority of this production came from the Kolwezi district. By 1995, production had fallen to 32,500 tons of copper 3,950 tons of cobalt, 4,500 tons of zinc. The decline in metal production has continued to the point that primary production in the Kolwezi area has now virtually stopped with much of the current production coming from site cleanup activities. This has been due to a number of factors including:

- The political isolation of what was then Zaire in 1991
- The loss of financial credit lines
- The lack of sustaining capital and maintenance improvements
- Social and political environment within the country during this period
- The collapse of the central portion of the Kamoto underground mine

The Kamoto underground mine is accessed by twin declines, two primary shafts and three secondary shafts. Primary access is through the declines and ore handling is through the main shaft where crushed ore is transferred directly onto a conveyor to the Kamoto concentrator. Exploration and development in the Kamoto underground area began in 1959. Underground production, which began in 1969, used a variety of large-scale techniques including cut and fill, room and pillar and sub-level caving. Production steadily increased to reach the rate of 3,000,000 tonnes per year by mid-1970. Production reached a peak in 1989 when the mine produced 3.29 million tonnes of ore. From the mines start-up in 1969 through 2005, the mine has produced a total of 59.3 million tonnes of ore at an average grade of 4.21% Copper and 0.37% Cobalt. In 1990, a major collapse in the central portion of the deposit resulted in the loss of approximately 15 million tonnes of resource. Since that time, due to the issues noted above, production from Kamoto steadily decreased to the point that production has essentially stopped.

The DIMA open pit group consists of the pits Dikuluwe, Mashamba West and Mashamba East. These pits primarily provided oxide ore to the Kamoto Concentrator (DIMA sections). The DIMA pit group operated from 1975 through 1998 during which time a total of 57.7 million tonnes of ore grading 4.96% Cu and 0.16% Co was mined. By 1998, due to the lack of funds and increasing costs, these pits were allowed to flood. No significant production has come from Musonoie-T17.

Dikuluwe – Began operations in 1975 and ended in 1993. The pit produced a total of 26 million tonnes of ore at an average grade of 5.47% Copper and 0.10% Cobalt.

Mashamba West – Mining operations began in 1978 and ended in 1995. The pit produced a total of 21.8 million tonnes of ore at an average grade of 4.35% Copper and 0.14% Cobalt.

Mashamba East – Operated from 1985 through 1988, the pit produced a total of 9.8 million tonnes of ore at an average grade of 4.96% Copper and 0.35% Cobalt.

The Kamoto concentrator consists of four sections, Kamoto 1 and 2 built in 1968 and 1972 respectively and DIMA 1 and 2 built in 1981 and 1982. The Kamoto 1 and DIMA circuits were designed to process mixed ore types and Kamoto 2 was designed for sulphide ore. From 1969 through 2000, the Kamoto Concentrator processed over 126 million tonnes of ore at an average grade of 4.33% Copper and 0.28% Cobalt. In its current configuration, the Kamoto concentrator is capable of processing 7.5 million tons of ore per annum. This throughput was exceeded from 1983 through 1987 with the peak production year being 1985 when production exceeded 7.6 million tons of ore.

The Luilu metallurgical plant is located approximately 6 km north of the Kamoto Concentrator. It was originally constructed in 1960. In 1972 it was expanded to its present annual capacity of 175,000 tonnes of copper and 8,000 tonnes of cobalt. The site consists of three roasters, leaching circuit and electrolytic cells for copper and cobalt production. From 1984 through 1989, production at Luilu averaged 173,000 tonnes of copper and 5,900 tonnes of cobalt. The highest production year was 1986 with 177,500 tonnes of copper and 7,800 tonnes of cobalt.

By 1996, production had fallen to an estimated 27,000 tonnes of copper and 1,200 tonnes of cobalt and has continued to decline.

5. RESOURCE DEVELOPMENT

A substantial resource exists within the Kamoto mine that will be the initial target of a focused drilling program once operations restart. The first identified target in this resource development program will be the southern region of Kamoto. A ten hole program has been outlined to confirm and convert the high grade Inferred Resources in this area into Measured and Indicated categories. Other under-explored areas within the mine will also be targeted for additional exploration and development in the early years with the expectation that beneficial modifications to the current mine plan will be developed as more information is gained. Finally, outside of the current mine plan area, Kamoto resource potential is still open in most directions.

A district exploration program is also planned once operations are restarted. Recognizing that no systematic exploration of the project area has been carried out since the 1980's there are several highly prospective areas that will provide high quality exploration targets. The DIMA area currently holds over 20-years of reserves therefore exploration in this area will initially be focused on enhancing and evaluating the logical expansion of the planned pits.

6. KAMOTO MINE

6.1 Mining Method

The remaining resource areas to be mined consist largely of flat dipping areas, with some steeply dipping areas mainly concentrated on the western and southern edges of the ore body. Most of the near vertical or vertical ore body has been mined out, and most of the development access required is pre-existing to enable extraction of the remainder of the ore body with long hole drilling.

Three mining methods will be used:

- Flat dipping areas:
    1. Footwall benching of all areas pre-developed on the Room and Pillar mining method;
    2. Long Hole Retreat Stoping with top pillar drives.
- Steeply dipping areas:
    1. Long Hole Retreat Stoping with top pillar drives;
    2. Long Hole Retreat Stoping without top pillar drives.
- Near vertical or vertical areas:
    1. Cut and fill mining with secondary pillar extraction.

6.1.1 Room and Pillar Mining Method.

The application of the Room and Pillar method will be limited to areas of the ore body which have been developed according to this lay out, and where the footwall benching has not yet commenced. The room and pillar mining zones are as follows:

- Portions of zone 8 on the Ore Body Supérieur or Upper Ore Body (OBS).

- Portions of zone 5 on the OBS.
- Portions of zone 1 on the OBS.

The key mining activities for this method are the same as previously applied. The only change between the two methods is the application of backfill in the benched out area. Geotechnical analysis of the pillars revealed unfavourable safety factors in the slender pillar configuration after completion of the benching operation.

The backfill is required to:

- Provide confinement to slender pillars to assist in retaining their load bearing capability and preventing premature collapse;
- Reduce the volume of open stope in mined areas to minimize the probability and consequences of any pillar collapse and hanging wall caving that may occur.

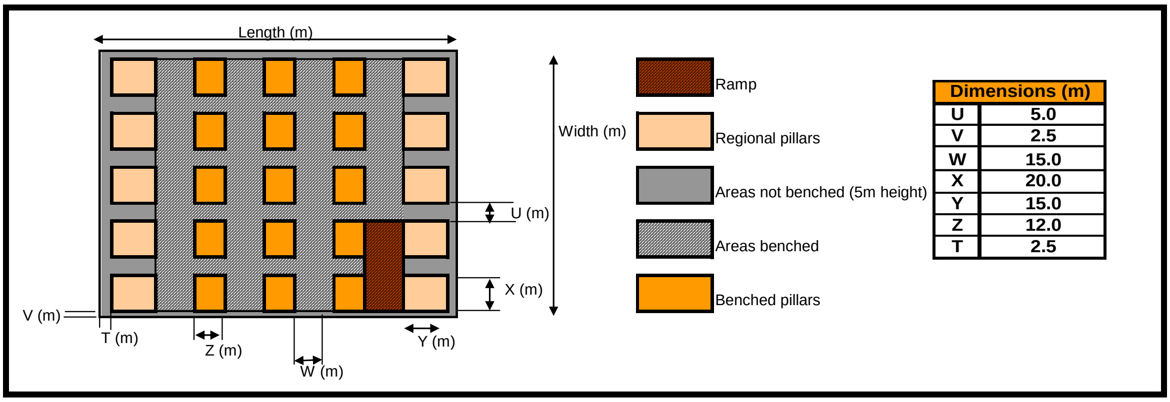

Figure 2 - Plan section – Room and Pillar Common Block

Figure 2 shows a plan view of a common block (smallest independent block which can be duplicated to form the mining lay out for the entire ore body). The key mining parameters are summarized as follows:

Undiluted Ore Flow			
	m^2	%	Tonnes
Total Area	16375		510,900
Geological loss	1638	10%	51,090
Initial extraction	8798	22%	114,368
Secondary extraction	7128	47%	238,680
Backfill required			86,063

Table 4 - Room and Pillar Extractable Tonnes

Room and Pillar Benching	Over break
Development over break (h/wall & s/wall)	5%
Benching	5%
Ore loss in stopes	3%

Table 5 - Input Parameters

6.1.2 *Long Hole Retreat Stoping*

The Long Hole Retreat Stoping method (LHRS) is the preferred mining method for the remainder of the Kamoto ore body. The mining method is easily adaptable to either flat dipping, steeply dipping or near vertical inclinations of the ore body, and is gaining popularity among other mines in the Copper belt region. The main advantages of the method are:

- High ratio long hole drilled metres compared to more costly short hole drilled metres, to extract a given tonnage profile per month.
- Increased utilization of ore body through improved total extraction rate.
- High extraction rates possible due to concentration of mining activities.

The main disadvantages of the method are:

- The need for backfill placement in mined out stopes and the associated binder cost.
- Stopes are no-entry areas, remote controlled load, haul dumpers (LHDs) required.
- Potential for excessive over break on hanging wall contact, thus resulting in increased dilution.

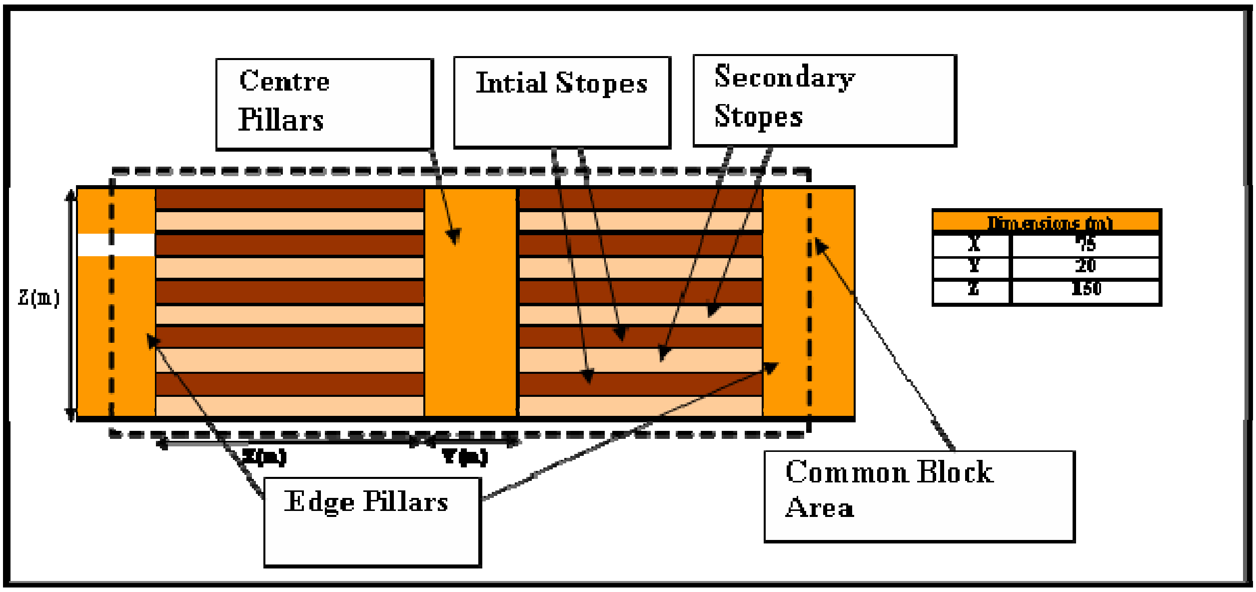

Figure 3 - Plan view – LHRS Common Block

Undiluted Ore Flow			
	m^2	%	Tonnes
Total Area	28,500		889,200
Geological loss	2,850	10%	88,920
Initial extraction	10,125	36%	315,900
Secondary extrac.	20,250	71%	631,800
Final extraction	22,950	81%	716,040
Area filled	20,250	71%	437,400
Cement required			21,870

Table 6 - Long Hole Retreat Stoping Extractable Tonnes

Long Hole Retreat Stoping	Meter	Over break
Sidewall over break (initial) - either side	0.5	7%
Sidewall over break (secondary) – 0.75m either side	0.75	11%
Hanging wall over break @ hanging wall value	1	8%
Development over break (h/wall & s/wall)		5%
Ore loss in stopes		3%
Mine Call Factor		100%
Geologic Loss		10%

Table 7 - Input Parameters

6.1.3 *Cut and fill mining with secondary pillar extraction.*

The key mining activities for this method is the same as historical mining methods. Only a relatively small portion of the remaining ore body has a vertical or near vertical inclination. The areas concerned are portions of zone 9 and division 5, which is largely pre-developed and mainly long hole drilling is required to extract the ore body.

6.1.4 *Kamoto Development Schedule*

Figure 4 shows the development requirements for the evaluation period. Development is a combination of 5-meter and 4-meter drifting and production and ventilation raises.

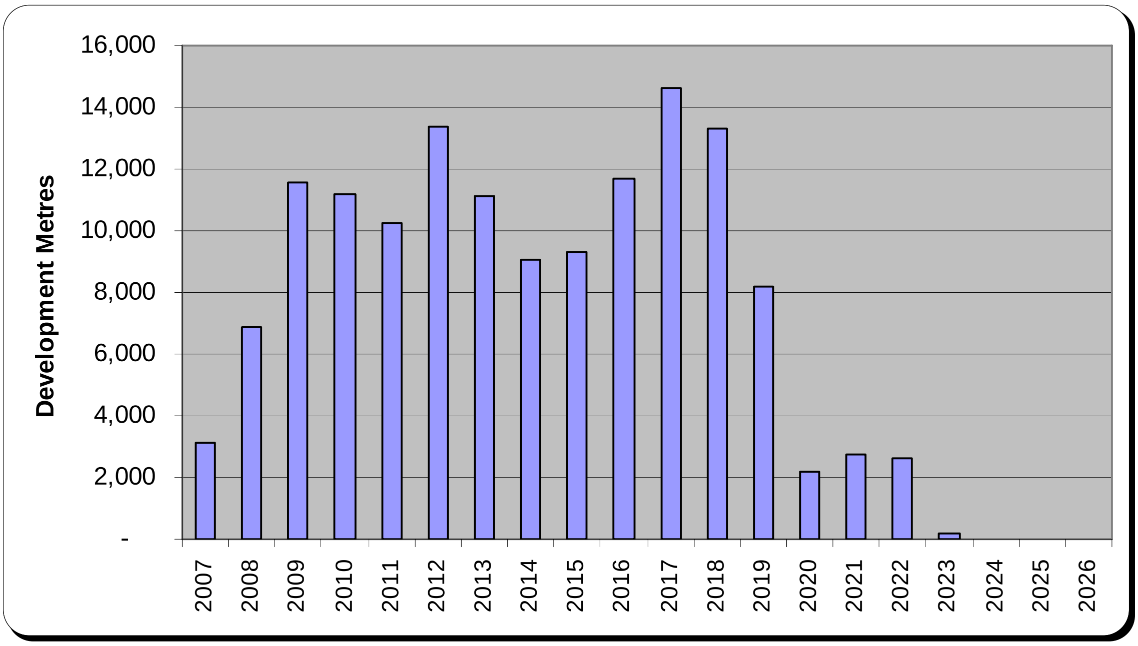

Figure 4 - Development Schedule

6.1.5 *Kamoto Production Profile*

Figure 5 shows the total ore production profile for the evaluation period. This is based on the current proven and probable reserve. As noted in the exploration section of this summary, it is expected that a focused exploration program will result in the expansion of reserves beyond that indicated by the current plan.

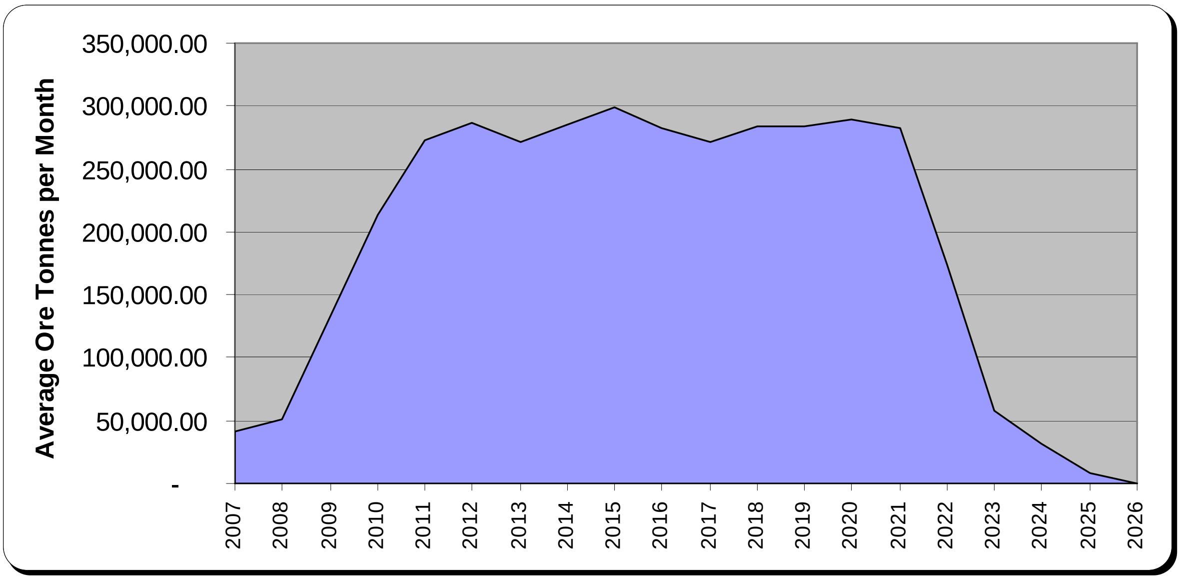

Figure 5 - Production Profile

7. OPEN PITS

7.1 Dewatering

Dewatering of the DIMA pits will require the use of barge mounted pumps and the drilling and commissioning of a new groundwater pumping wells.

It will require approximately three years to dewater Mashamba East. The Dikuluwe and Mashamba West pits will be drained over an extended period of approximately 12 years.

It is assumed that six wells will be required to bring the Mashamba East pit into production. The locations of the wells will need to be finalized at the commencement of mining in accordance with the planned pit configurations.

7.2 Operations

The open pits will be mined to provide oxide ore. Production will begin in the T17 pit and will continue there for approximately three years while Mashamba East pit is being dewatered and prepared for mining. Open pit mining will also be carried out in both Mashamba West and Dikuluwe in later years. Open pit mining will be done by a contractor.

7.3 Consolidated Open Pit Production Schedule

Figure 6 shows the consolidated production schedule for all the pits. It shows the increased production at end of mine life to compensate for the drop-off in underground tonnage

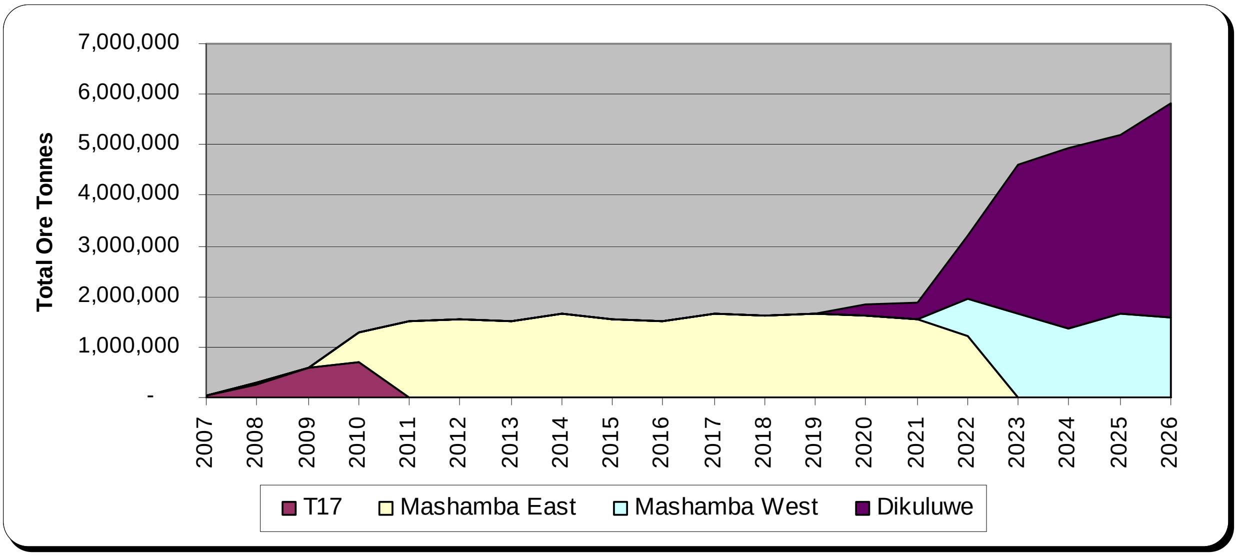

Figure 6 - Consolidated Open Pit Production Schedule

8. KAMOTO CONCENTRATOR

8.1 Summary

Gécamines have historically treated four types of ores at the Kamoto / DIMA concentrator using different process schemes.

Kamoto – DIMA process performance was reviewed over the life of the plant. Data was weighted from the actual data given for years 1990 and 1991 based on a Concentrator production report (for 1991) that was made available. The copper recovery for the concentration capacity averaged around 86% in the 1980's rising to around 90% during the 1990's when the plant was operating at lower throughputs.

Metallurgical research was undertaken as part of the Feasibility Study aimed at identifying areas where the performance of the existing operating plant could be improved. This was considered to be potentially achievable by optimizing the process flow sheet and applying new reagents. Due to the size of the sample composites, the preliminary nature of the testwork and project timing constraints there was not a lot of scope for the implementation of results of the preliminary test work within the process design. However, testwork outcomes have validated historical production performance data such as concentrate recoveries and grades, identified recovery limits and highlighted potential for operating cost reductions in the areas of reagent consumptions. Observations were made with regard to the potential that exists to improve recoveries in the area of cobalt. Further work is recommended on some of the ores.

Tests indicated that the potential exists to achieve more recovery through finer grinding taking into account traditional mineralogical constraints such as the tendency to over-slime chalcocite in the plant autogenous grinding circuit. Copper recoveries of up to 92% were achieved at rougher concentrate grades above 21%. Additional recovery was achieved as a result of a tailings regrind, with results consistent with historical plant modifications, indicating that this circuit should be further investigated if the required regrind capital and power costs were found to be low enough to be feasible. The cleaner and re-cleaner results indicated that the target Cu grades of 31% to 44% are easily achievable.

The target copper grade specifications were met for all composites at recoveries ranging between 70% to 90% Cu for the oxide ores Testwork has indicated that the copper and cobalt recoveries are sensitive to the over dosage of sulphidiser and there are indications that the dosage of this reagent can be reduced provided that the cheaper emulsion dosage was marginally increased from the low levels quoted in the tests, which were significantly lower than the consumptions quoted in production records. Gravity separation of a copper rich concentrate was shown to reduce the reagent consumption significantly on one specific mass fraction, but more work would be required to confirm that the copper and cobalt recoveries to the combined gravity/flotation concentrate would not be affected and that the balance of oxide remaining to be floated would not consume the same mass of reagent per unit volume of slurry generated. It must be borne in mind that test work sample head grades are 32% higher than forecast for sulphide and ranging from 56% to 123% higher than forecast in the testwork.

The rehabilitation project consists essentially of equipment replacement and rehabilitation aiming at improved maintenance, productivity and reduction of operation costs. The specification for the concentrator was based on a number of documents released from site personnel. Historical Equipment Lists, Engineers inspection lists, pump schedules and HATCH questionnaires answered by the client were consolidated to generate a number of phase specific project Mechanical Equipment Lists. The primary drivers of the overall mining schedule were sulphide mining production constraints and metallurgical oxide:sulphide balance constraints. Sulphide throughput has been prescribed with oxide feed levels subsequently determined by the neutral acid balancing of the refinery solution streams. ‘Mixed’ oxide / sulphide ore has been assumed to only be mined in Phase III based on the limited mining planning completed in the early part of this study. This will still require some validation. It is also assumed for the purposes of scope definition that dolomitic / mixed ore will be campaigned through the oxide circuit on a regular basis.

Oxide Mill circuit utilization will be very low in phase I and II (there will be a change to a larger 250 tph mill from a 100 tph mill in Phase III). Daily production will continue but on a single shift basis to facilitate optimum exploitation of the tailings for underground backfill purposes. Flotation residence time was not prescribed but was calculated from the operation of two 28’ Cascade Mills in parallel. Mineralogical investigations were undertaken to determine the necessity of the regrinding of concentrate and whether the practice should be re-introduced. Based on the prescribed design criteria the application of dewatering thickeners to the concentrate products was not deemed to be necessary. The decision to replace or refurbish the existing flotation capacity was made on the basis of a capital cost comparison

Some additional recovery benefit is expected to arise from the transition from smaller cells to larger cells in the later phase of the project. Significant maintenance cost benefits are also expected. An increase in Cu recovery had been measured to around 1% on similar plants that have been retrofitted with larger cells. Significant reductions in reagent consumptions have also been indicated when larger cells were introduced in previous projects.

Preliminary results would appear to indicate that the project could benefit from further detailed conclusive test work on representative (particularly T-17) composite material before concluding the level of recovery that can be expected from this ore. It is recommended that oxide tests should be considered in the future to further evaluate different emulsion component ratios and the effect of this on Rinkalore Booster to reduce NaSH, NaSiO2, Diesel and lime consumptions.

Table 8 incorporates the best estimates of the concentrator performance based on historical production data, nominal plant specification and scout test work.

Concentrator	**Phase 1**		**Phase 2**		**Phase 3**		**Phases 4**	
Key Data	% Cu	%Co	% Cu	%Co	% Cu	%Co	% Cu	%Co
Feed								
Sulphide Ore	3.34	0.26	4.10	0.27	3.39	0.28	3.18	0.32
Siliceous Oxide	2.73	0.31	2.88	0.30	2.69	0.62	3.04	0.34
Dolomitic Oxide			2.63	0.41	3.41	0.46	2.72	0.27
Concentrates								
Sulphide Concentrate	42.67	2.76	45.82	2.55	45.82	3.22	45.82	3.85

Siliceous Concentrate	22.82	1.80	22.82	1.60	22.82	3.60	22.82	1.75
Dolomitic Concentrate			16.13	0.76	16.13	0.66	16.13	0.48
Tailings								
Sulphide Tails	0.37	0.07	0.42	0.07	0.35	0.07	0.32	0.08
Siliceous Tails	0.74	0.16	0.71	0.15	0.66	0.32	0.75	0.18
Dolomitic Tails			0.89	0.37	1.20	0.43	0.92	0.24
Recoveries								
Sulphide Ore	89.7	75.5	90.5	76.6	90.5	76.6	90.5	76.6
Siliceous Oxide	75.5	53.1	77.9	53.1	77.9	53.1	77.9	53.1
Dolomitic Oxide			70.0	21.0	70.0	21.0	70.0	21.0
Concentrate Mass Fractions								
Sulphide Conc. (%)	7.0		8.1		6.7		6.3	
Siliceous Oxide Conc. (%)	9.0		9.8		9.2		10.4	
Dolomitic Oxide Conc. (%)			11.4		14.8		11.8	

Table 8 - Concentrator Performance

9. LUILU

9.1 Summary

The Luilu Plant is designed to recover copper and cobalt from sulphide, oxide and dolomitic concentrates produced at the Kamoto concentrator. The operation uses roasting, leaching, and precipitation circuits to produce copper and cobalt via electrowinning.

The principle of operation of the Luilu Plant is to use the acid generated by the roasting, leaching and electrowinning of copper from the sulphide concentrates to leach the oxide concentrates. Balancing the amounts of sulphide and oxide concentrates minimises the amounts of neutralizing agents or sulphuric acid needed to control the acidity of the process solutions and reduces the plant operating costs. Consequently, the proportion in which sulphide and oxide ores are mined, concentrated and presented to the refinery is a key process parameter.

Process flowsheets, process design criteria and mass balances were developed for the plant, based on available historical information and in-house knowledge. The major change compared to historical operation is the implementation of a process control system which constantly monitors the process conditions in the plant. It is expected that this change, if correctly implemented, will positively impact overall metal recovery and product quality. Therefore, it is strongly recommended that after the Luilu Plant has reached stable operation after phase 1 plant start-up, a detailed process review is performed to verify plant operation based on the developed process design criteria and mass balance and update the design if required.

The process environmental issues are limited to the roaster off gas system and tailings removal. The existing roasters are equipped with tail gas scrubbers, however it is unlikely that the sulphur removal efficiency of these scrubbers meets the applicable environmental legislative requirements. A dual-alkali off-gas scrubbing system is included for the newly installed roasters (during Phase 2 and 3). The off gas handling problem is therefore limited to Phase 1, when the existing roaster with tail gas scrubbers is in operation.

The current Luilu plant design assumes that all waste streams generated in the process are disposed off in the tailings dam with no water recycle to the plant. Incorrect handling and monitoring of the tailings disposal area could result in downstream handling problems.

Due to the need to balance the ratio between the oxide and the sulphide concentrate feeds, it is important that the roaster operation is reliable and the mine is able to produce the required ore ratio's. The reliability of the existing roaster poses the main process risk for the refinery in

Phase 1 due to its poor condition. A new roasters cannot be installed prior to Phase 2 due to the long lead time required.

9.2 Basis of Design

9.2.1 Process Objective

Table 9 identifies the key design parameters for the plant. Copper and cobalt will be recovered by electrowinning and sold in the form of copper cathode and cobalt broken cathodes (chips).

Upon completion of the simulations, the mass balance was frozen to allow engineering to commence. This mine plan was adjusted later in the project, resulting in altered production numbers for the operating cost estimate.

	Unit	**Design Value**				**Source**
		Phase 1	**Phase 2**	**Phase 3**	**Phase 4**	
Plant Operating Schedule	h/d	24	24	24	24	Kamoto
	d/wk	7	7	7	7	Kamoto
	d/y	365	365	365	365	Kamoto
Plant Availability	%	90	90	90	90	Kamoto
Sulphide Concentrate Feed	t/h	6.4	16.2	27.6	37.3	Kamoto
Oxide Concentrate Feed	t/h	2.3	7.1	15.2	20.5	Calculated
Dolomitic Concentrate Feed	t/h	0.6	1.9	4.2	6.4	Calculated
Overall Copper Recovery	%	90.5	90.4	90.3	90.2	Calculated
Overall Cobalt Recovery	%	58.6	58.6	57.1	56.4	Calculated

Table 9 - Key Plant Design Parameters

9.3 Control Philosophy

The control philosophy is designed to provide the safe and effective control of the process and equipment. It provides a simple integrated display of the process operating status and provides for safe interlocking of processes.

Total Recovery

The feasibility study utilized the following total recoveries for the economic analysis:

	Phase 1		**Phase 4**	
	Copper	**Cobalt**	**Copper**	**Cobalt**
Sulphide Ore	81.2%	47.1%	81.6%	46.0%
Oxide Ore	68.3%	33.2%	70.3%	31.9%
Dolomitic Ore	63.4%	13.1%	63.1%	12.6%

Table 10 - Total Recoveries

10. INFRASTRUCTURE

10.1 Power

A total of 303MVA is available to the Kolwezi area. Current consumption is in the region of 80MVA as measured at Substation West, the main transmission substation near Kolwezi.

Power transmission around the mine site is via 110kV lines while distribution is via both 15kV and 6.6kV power lines.

The maximum demand for the final phase of the KCC project, mining and process included, is 145MVA. Ramp ups are shown in the following figures. This is lower than the installed capacity of 240MVA. With the installed equipment a firm supply of 120MVA is available at the concentrator and Luilu.

Table 41 shows the installed power for the different phases. A diversity factor of 80% was used based on inputs from the various other disciplines and on equipment lists.

For the concentrator and Luilu the majority of equipment will be installed in the first two phases. This is because most auxiliaries need to run irrespective of production throughput. The figure also includes small power and lighting, ventilation and compressed air.

	Power Consumption			
	Phase 1	**Phase 2**	**Phase 3**	**Phase 4**
Kamoto DIMA Concentrator	41.6	44.9	48.6	52.1
Luilu	21.1	38.1	54.4	69.1
Underground Consumption	8.0	14.0	19.0	24.0
Total Power Consumption (MVA)	**70.7**	**97.0**	**122.0**	**145.2**

Table 11 - Power Consumption (MVA)

10.2 Water

Water required for diamond drilling and other exploration and mining activities for the Project comes from two sites. The Kamoto mine receives an estimated inflow of approximately 60,000 cubic meters per day. From this, a total of 4,000 cubic meters per day is pumped out as potable water and 19,000 cubic meters is pumped to the Kamoto concentrator for use in the metallurgical process. Historically, the Luilu metallurgical plant drew up to 620 cubic meters per hour from the Luilu River. Future operations are based on recycling process water which will reduce the fresh water demand to approximately 160 cubic meters per hour.

10.3 Tailings Sites

Seven candidate sites were identified for the impoundment of tailings from the Kamoto Concentrator and two additional sites were located for the disposal of tailings and solid waste from the Luilu Metallurgical Plant.

Field tests on both the Kamoto and Potopoto Tailings Dams have shown an average density of 1.4 t/m3 for the Kamoto Concentrator tailings at near surface and 0.950 t/m3 for the Luilu Tailings.

10.3.1 Kamoto Concentrator

After evaluating seven potential tailings sites, the existing Kamoto Tailings Dam was selected for use by the Kamoto Concentrator. By extending the dam footprint downstream the site can impound the entire study tonnage.

The preliminary design concepts involve the construction of two closure walls, the first across a tributary of the Luilu River. The second closure wall will be on the alignment of the current

Kamoto Tailings Dam embankment. The construction of the closure walls will not only allow additional storage capacity on the Kamoto Tailings Dam area but it would enable the area downstream of the existing wall to be integrated into a larger dam footprint.

The return water dam will be located downstream of the closure wall from where the penstock and under-drainage discharge can be returned back to the Plant.

The entire Kamoto Tailings Dam area will be isolated from the influence of storm events above the dam footprint area by the construction of storm water diversion structures.

The introduction of the diversion facilities will redirect current stream water crossing the dam. It will also to reduce the volume of water on the dam that has to be dealt with in the event of a storm until such time as the tailings dam rises by about another 5m in elevation at the eastern end of the dam.

10.3.2 Luilu

The new tailings impoundments at Luilu will consist of individual ponds with internal floor dimensions of 250m by 300m. The floor and perimeter walls will be constructed of compacted clay to a height of 3 metres above natural ground level. Each impoundment will have an inner basin area excavated to a depth of 3 metres. The ponds will be built with a dual synthetic liner with an internal leak detection and drain system installed. Each basin will hold approximately one year of tailings production. Once filled, the basin will be capped with a local material to shed water.

Water reuse from the tailings ponds will be maximized with the return water being pumped back to plant storage for reuse.

11. HUMAN AND SOCIAL ISSUES

11.1 Project Benefits

Katanga Mining Limited (KML) and Kamoto Operating Limited (KOL) will collectively design and drive social initiatives within the communities in the region, and specifically in the city of Kolwezi to ensure that the local and regional social infrastructure is benefiting from the project.

These sustainable development programs will be developed in an effort to increase the social services baseline in the community and will specifically address areas of general education, advanced technical training, general medical services, local social services, agricultural education programs and economic opportunities, and other micro-enterprises. These programs will be reviewed and managed by a collective interest group consisting of corporate / mine management and community stakeholders.

These programs will provide general guidance to assist in the local and regional economic rehabilitation resulting from the project's significant tax contribution, and the many tertiary economic opportunities that will develop.

During the course of the project the communities and the DRC will recognize the following economic benefits (000's USD):

DRC Royalty	$139,766
Tax on income	$759,037
Dividend tax	$146,623
Capital Equipment Duties	$15,657
Import Duties Consumables	$16,994

Payroll & Social Support	$413,787
Total	$1,491,864
Purchased Power Cost	$397,795

Table 12 - Economic Benefits

Part of the rehabilitation from a human capital perspective will be to reconstruct the workforce with experienced, capable workers, as well as skilled educated management and professional staff. These semi-skilled, skilled, management and professional people will most likely have gained their work experience with Gécamines.

11.2 Staffing Levels

The proposed staffing levels are based on the phased ramp up schedule.

The total workforce by phase is scheduled to be:

- Phase 1 : 1,466
- Phase 2 : 1,883
- Phase 3 : 2,154
- Phase 4 : 2,404

By utilizing contractors and other available skilled personnel, the company expatriate workforce will total 32.

12. ENVIRONMENTAL

The project has been planned to confine the entire project to the use of existing mining and processing infrastructure and footprint areas which have already been disturbed. In this way Kamoto intends to minimize cumulative impacts and set in motion a process whereby biophysical conditions in the area will gradually improve, along with a significant socio-economic improvement in the area.

As required by the Mining Regulations, an Environmental Impact Statement (EIS) and an Environmental Management Plan of the project (EMPP) have been prepared by SRK Consulting. They are currently being translated and will be submitted immediately thereafter.

Final closure requirements and associated costs will be developed in consultation with Gécamines. Upon termination of the operating lease all properties and facilities will revert back to Gécamines. KCC will reclaim those facilities and operating sites that have been developed by KCC but which Gécamines does not wish to preserve.

13. COST ESTIMATE

13.1 Capital Costs

13.1.1 Capital Cost Estimate

The initial capital cost of rehabilitating the Kamoto assets have been estimated by the individual consultancy companies and compiled by Hatch. The ongoing capital cost for replacement of mining equipment was estimated using a zero-based model. The cost of replacing capital equipment in the process plants was estimated based on unit rates. The project calls for two distinct phases of capital infusion. The first phase relates to the four-year build to a sustainable production capacity. The second phase consists of ongoing capital replacement costs and lasts through year 16.

The capital costs for the initial production build up are summarized as follows:

Area	Total	Phase 1	Phase 2	Phase 3	Phase 4
KTO Mine	$ 80,377	$31,683	$20,158	$16,338	$12,158
Open Pits	$14,611	$13,161	$1,150	$150	$150
Kamoto Concentrator	$55,216	$23,492	$9,835	$14,703	$7,185
Luilu	$150,098	$38,772	$44,322	$50,368	$16,635
Infrastructure	$23,634	$18,018	$1,662	$3,201	$754
Indirect Costs	$54,318	$30,928	$8,421	$7,334	$7,635
Contingency	$48,572	$19,504	$10,972	$12,486	$5,611
Total	$426,786	$175,558	$96,522	$104,579	$50,128

Note: Columns may not add due to rounding
Costs in 000's USD

Table 13 - Capital Costs

Replacement and ongoing capital requirements for the life-of-mine analysis period are as follows:

Area	Total
U/G Mine	$103,017
Concentrator	$30,486
Hydro-Metallurgical	$79,310
General & Administration	$2,450
Dewatering	$16,000
Total	$231,263

Costs in 000's USD

Table 14 - Replacement and Ongoing Capital Costs

- The estimate is expressed in US Dollars (USD) with a base date of March 2006;
- The estimate consists of four distinct and separate phases in accordance with the requirements of the production ramp-up and is structured according to the Work Breakdown Structure (WBS);
- As far as possible, local costs and construction rates were used in the preparation of the cost estimate.

Capital Costs over the evaluation period life of the project are summarized as follows:

	Total('000s USD)	USD/t ore	USD/lb. Cu	USD/t Cu
Initial Capex - Phase 1	175,558	1.97	0.04	81
Initial Capex - Phase 2	96,522	1.08	0.02	45
Initial Capex – Phase 3	104,579	1.17	0.02	48
Initial Capex – Phase 4	50,128	0.56	0.01	23
Sub-Total Initial Capex	426,786	4.78	0.09	197
U/G Mine Replacement	103,017	1.15	0.02	48
Concentrator Replacement	30,486	0.34	0.01	14
Luilu Replacement	36,750	0.41	0.01	17
Dikuluwe dewatering	16,000	0.18	0.00	7
Additional tailings pond capacity	42,560	0.48	0.01	20
G & A Replacements	2,450	0.03	0.00	1
Sub-Total Replacement	231,263	2.59	0.05	107
TOTAL CAPITAL	658,049	7.37	0.14	304

Note: Columns may not add due to rounding

Table 15 - Capital Cost Summary

13.2 Operating Costs

The operating cost estimate is based on the phased mining and processing operations. Underground mining costs were developed by activity. Open pit costs are based on a rate per tonne on a contract-mining basis. Process costs were similarly developed based on first principles incorporating energy, reagent and manpower costs.

The usage of individual consumables was calculated in proportion to their major drivers: sulphide ore milled, oxide ore milled, copper production, cobalt production and operating hours.

The initial estimated operating costs are impacted by the plant throughput ramp-up and by the metals recovery ramp-up.

Operating costs were estimated for the Luilu plant for four levels of production corresponding to the four phases of the project. The Metsim mass balance was developed for these four discrete phases, and from this mass balance reagent and consumable consumption rates were generated per unit copper or cobalt production. The four sets of consumption rates relationships developed by the Metsim mass were applied to the relevant phases.

Staffing estimates were developed for each section of the operation. These were done using the Patterson band grading system. An additional dimension was added to cater for both local and expatriate labour.

The annual power consumption is based on the connected load of all operating equipment.

Maintenance costs are estimated based, where possible, on comparable operations, and also as a percentage of the capital costs required to establish a greenfields operation of similar capacity.

Transport costs were estimated based on the mass of consumables being sourced multiplied by a relevant quote obtained from a transport operator.

Import duties at three per cent for the duration of the project for all fuels, lubricants, reagents and consumables as per the DRC mining code.

Site operating costs by phase are as follows:

	Phase 1	Phase 2	Phase 3	Phase 4	Average
Tonnes (t) Copper Tonnes (t) Cobalt	27,070 964	68,468 2,729	109,693 6,228	1,961,812 103,705	
Underground Mining	$33,272	$41,044	$56,962	$738,412	
Open Pit Mining	$20,747	$70,591	$73,734	$847,944	
Kamoto DIMA Concentrator	$8,916	$13,927	$22,480	$428,070	
Luilu Plant	$23,289	$30,687	$49,130	$852,327	
General & Administration	$19,696	$15,527	$17,302	$207,410	
Total (‘000s USD)	$105,920	$171,775	$219,607	$3,074,163	
Cost per lb. (USD/lb. Cu)	$1.77	$1.14	$0.91	$0.71	$0.75
Cost per lb. Cu (with Co credit)	$1.42	$0.74	$0.34	$0.18	$0.22
Cost per tonne Cu	$3,913	$2,509	$2,002	$1,567	$1,648
Cost per tonne Cu (with Co Credit)	$3,128	$1,630	$750	$402	$492

Note: Columns may not add due to rounding

Table 16 – Site Operating Cost by Phase

Over the analyzed 20-year period, total production costs are as follows:

	Total (‘000s USD)	USD/t ore	USD/lb. Cu	USD/t Cu
Underground Mining	869,689	9.74	0.18	401
Open Pit Mining	1,013,017	11.34	0.21	467
Kamoto DIMA Concentrator	473,393	5.30	0.10	218
Luilu Plant	955,432	10.70	0.20	441
General & Administration	259,934	2.91	0.05	120
Site Operating Cost Sub Total	3,571,465	39.99	0.75	1,648
Cobalt Credit	(2,505,037)		(0.52)	(1,156)
Site Operating Cost Total After Cobalt Credit			**0.22**	**492**
Transport and Marketing Expenses	772,006		0.16	356
Royalty and Lease Obligations	255,251		0.05	118
Capital Costs	658,049		0.14	304
Total Production Costs	**2,751,734**		**0.58**	**1,270**

Note: Columns may not add due to rounding

Table 17 - Operating Cost Summary

14. ECONOMIC ANALYSIS

14.1 Introduction

The financial model used in the Pre Feasibility study has been updated and expanded in the feasibility study in consultation with KML.

The model has been developed in real terms i.e. no escalation in revenues or costs.

The model takes monthly capital, operating costs and revenue into account. These values are than annualised before the income statement from which the project returns are calculated.

The mine will be run by the Kamoto Operating Company, an independent company contracted to Kamoto Copper Company.

The financial model allows the returns to the different entities (KCC, KFL and GMC) to be calculated depending on the level of debt financing. The GMC valuation was based on Artlcle 6 of the agreement between GMC and KCC. According to Article 6 of the agreement between Gécamines and KCC, ownership of the assets would continue to reside with GMC with any equipment and facilities acquired outside of the leased assets being ceded to GMC at an agreed upon rate at termination of the agreement. Consequently, no attempt was made to establish a value for the assets and any liabilities that may accrue with ownership. Rather, the focus of financial modeling was on estimating the costs and revenues that would be produced for the specified production schedule.

The production schedule was driven by the capacity of the concentrator and hydro-metallurgical plants as well as the underground mine's sulphide ore production rate. The sulphide to oxide concentrate balance in the hydro-metallurgical plant then effectively created an oxide and dolomitic ore demand which the surface mine plan strove to achieve.

Revenue was estimated based on the grade of ore mined and the recovery achieved for the different ore types by the various plants. The shipping costs required to get the product to market were then subtracted to determine the net revenue.

Capital cost comprised of both the cost of rehabilitating the Kamoto assets in four phases as well as the ongoing sustaining capital cost for replacement of mining equipment and maintaining the plants.

Operating costs for the underground mine, concentrator, metallurgical plant and G&A were derived using a zero-based model and the mining plan. Operating costs for the open pit mine were based on contract mining rates.

14.2 Summary

The Kamoto Copper Company – Kamoto Redevelopment Project has been modeled with financial returns estimated for the following cases:

- The initial capital investment required to rehabilitate KCC funded by debt (8.5% interest rate) in four tranches, each amortized over 60 months. This is the base case (NPV based on a 6% discount rate). This evaluation does not attempt to finance any operational losses occurring in the first years. They are simply treated as negative cash flows in the first years of the project;
- KCC funded on a 100% equity basis (NPV based on a 15% discount rate);
- KCC funded on a 100% debt basis (8.5% interest rate), with principal repaid before dividends are declared to the partners (NPV based on a 6% discount rate).
- GCM 25% stake in KCC and a royalty with no equity contribution (NPV based on a 6% discount rate).

The financial base case carries the following assumptions:

- Execution capital cost USD 426.7 million;
- Sustaining capital costs USD 231.3 million;
- Evaluation Period (LOM) 20 years;
- Copper revenue USD 1.10/lb;
- Cobalt revenue USD 10/lb;
- Total production of copper throughout LOM 2.17 millions tonnes (4 778 million lb);
- Total production of cobalt throughout LOM 0.113 millions tonnes (250 million lb).

The amortised debt discounted cash flow evaluation of the KCC redevelopment project shows an IRR of 23.8% and a NPV 612 million USD using a 6% discount rate and an 8.5% debt rate.

Annual refined copper output peaks at 143,000 tonnes (315 million lbs), while a maximum of 10,000 tonnes (22 million lbs) cobalt is produced (not in the same year due to grade variations). Average annual production over the 20 year project life is 109,000 tonnes of copper (240 million lbs) and 5,680 tonnes of cobalt (12.5 million lbs).

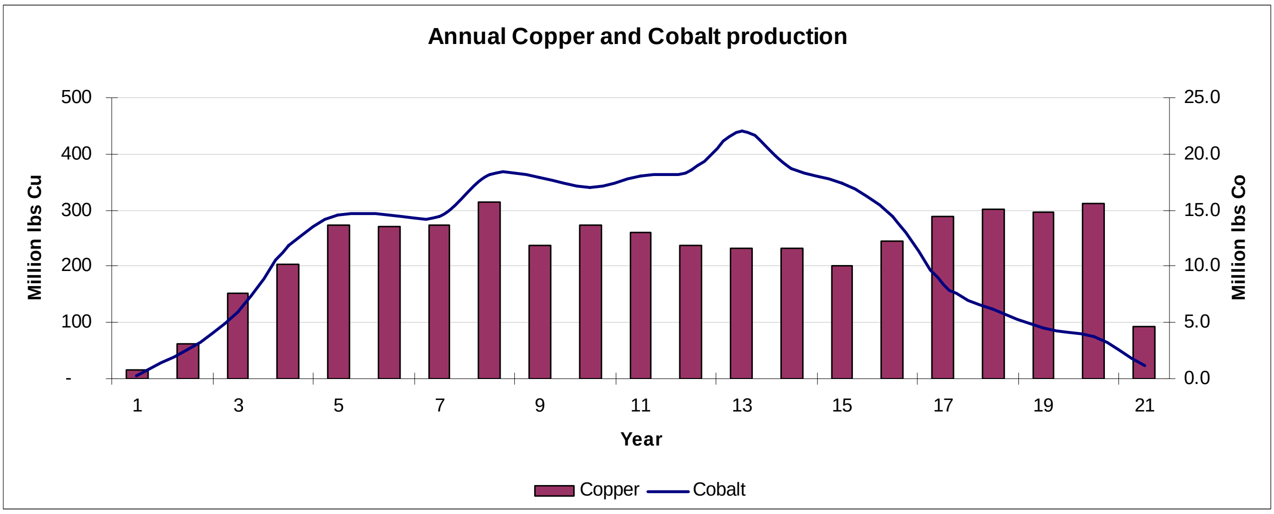

Figure 7 - LoM Metal Production

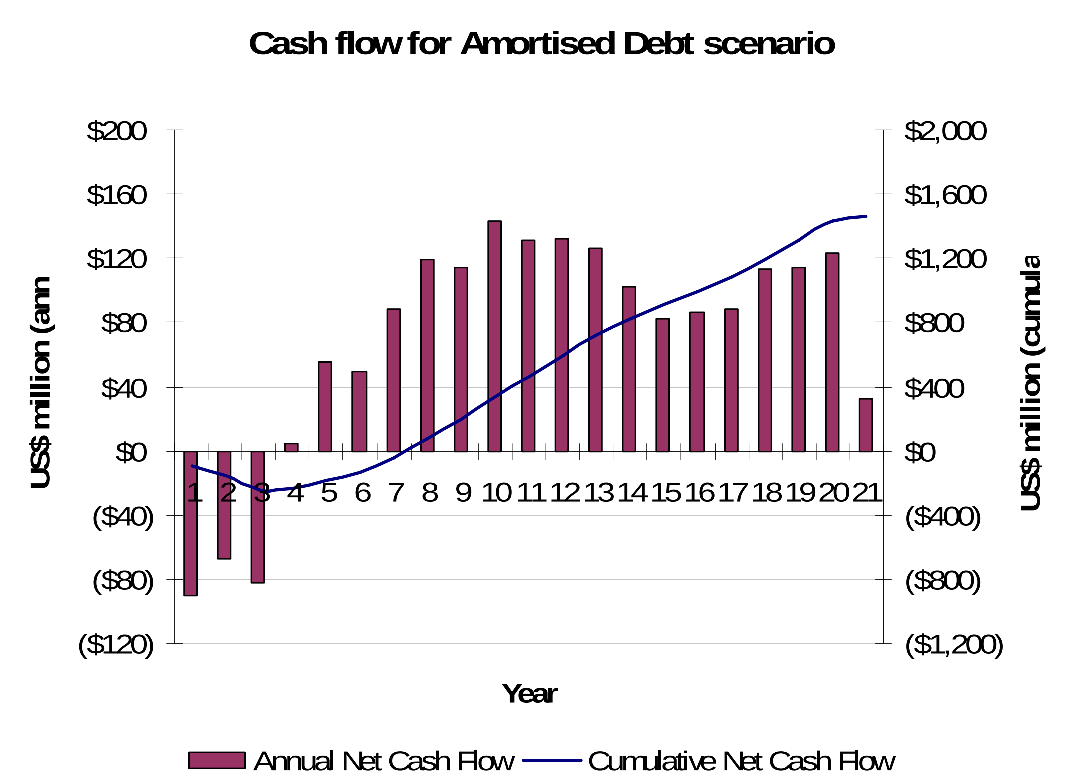

Figure 8 - LoM Cash Flow

Figure 8 illustrates the net cash flow received by KCC over the 20-year project analysis period under the base case (amortised debt scenario). The project cash flow can be divided into three main phases:

1. During the ramp up in years 1-3, KCC is cash negative due to the amortisation schedule and higher unit operating costs.

2. For years 4-7, KCC is repaying debt and generating an average of USD 61-million free cash annually.

3. From years 8 onward, the debt is retired and average free cash is driven by the grade profile of the mine.

14.3 Sensitivity Analysis

Sensitivity analysis considered the impact on the Base Case returns (USD 1.10 Cu and USD 10.00 Co) of variance in the following parameters:

- Metal Price;
- Process Recovery;
- Capital and Operating Costs.

For each, three cases were considered:

- The project returns to KCC;
- The project returns, assuming 100% equity finance / 100% debt finance;
- The returns to GCM.

14.3.1 Metal Prices

As seen in Table 18, the impact of a $0.10/lb increase in the copper price is roughly equivalent to a $2.00/lb increase in the cobalt price.

		KCC Amortized Debt		KCC 100% equity		Debt	GCM
Copper	**Cobalt**	**NPV$_6$**	**IRR**	**NPV$_{15}$**	**IRR**	**NPV$_6$**	**NPV$_6$**
$1.00	$9.00	$375	17.0%	$9	15.4%	$336	$155
	$10.00	$459	19.4%	$50	17.2%	$425	$177
	$11.00	$543	21.8%	$92	19.0%	$510	$198
	$12.00	$627	24.1%	$132	20.7%	$597	$220
$1.10	$9.00	$528	21.4%	$84	18.7%	$495	$194
	$10.00	$612	23.8%	$125	20.4%	$583	$216
	$11.00	$696	26.2%	$165	22.1%	$672	$238
	$12.00	$780	28.6%	$204	23.7%	$760	$260
$1.20	$9.00	$682	26.0%	$157	21.8%	$657	$234
	$10.00	$766	28.4%	$197	23.4%	$745	$256
	$11.00	$849	30.8%	$236	25.0%	$832	$278
	$12.00	$933	33.2%	$276	26.5%	$919	$300

Table 18 - Metal Price Sensitivity

14.3.2 Recovery

Table 19 indicates that returns are more sensitive to the recovery of copper than cobalt, with a 4% reduction in cobalt recovery being approximately equivalent to a 2% reduction in copper recovery:

		KCC Amortized Debt		KCC 100% equity		Debt	GCM
Copper	**Cobalt**	**NPV$_6$**	**IRR**	**NPV$_{15}$**	**IRR**	**NPV$_6$**	**NPV$_6$**
-4%	0%	$557	22.2%	$99	19.3%	$525	$202
-2%	0%	$585	23.0%	$112	19.9%	$553	$209
0%	0%	$612	23.8%	$125	20.4%	$583	$216
+2%	0%	$640	24.6%	$138	21.0%	$612	$223
+4%	0%	$667	25.4%	$151	21.5%	$641	$230
0%	-4%	$585	23.0%	$112	19.9%	$553	$209
0%	-2%	$599	23.4%	$118	20.2%	$568	$212
0%	0%	$612	23.8%	$125	20.4%	$583	$216
0%	+2%	$626	24.2%	$132	20.7%	$597	$220
0%	+4%	$640	24.6%	$138	21.0%	$612	$223

Table 19 - Recovery Sensitivity

14.3.3 Costs

Table 20 indicates that project returns are most sensitive to an increase or decrease in operating costs, and relatively insensitive to variation in the capital costs.

		KCC Amortized Debt		KCC 100% equity		Debt	GCM
Capex	**Opex**	**NPV_6**	**IRR**	**NPV_{15}**	**IRR**	**NPV_6**	**NPV_6**
+10%	0%	$574	21.8%	$95	18.9%	$539	$208
+5%	0%	$593	22.8%	$110	19.6%	$560	$212
0%	0%	$612	23.8%	$125	20.4%	$583	$216
-5%	0%	$631	25.0%	$140	21.3%	$605	$220
-10%	0%	$650	26.2%	$155	22.2%	$628	$224
0%	+10%	$484	19.5%	$56	17.4%	$448	$187
0%	+5%	$548	21.6%	$91	18.9%	$514	$201
0%	0%	$612	23.8%	$125	20.4%	$583	$216
0%	-5%	$677	26.2%	$158	21.9%	$652	$231
0%	-10%	$741	28.6%	$190	23.4%	$722	$245

Table 20 - Capital and Operating Cost Sensitivity

15. EXPANSION SCENARIO

15.1 Summary

Expansion of the Kamoto reserve base will be an immediate priority after operations resume. A substantial resource exists within the Kamoto mine that will be the initial target of a focused exploration program once operations restart. The first resource development target will be the southern region of Kamoto. Other under-explored areas within the mine will also be targeted for additional exploration and development in the early years with the expectation that beneficial modifications to the current mine plan will be developed as more information is gained. Finally, outside of the current mine plan area, Kamoto resource potential is still open in most directions.

The southern region of Kamoto contains substantial measured, indicated and inferred resources with an average weighted copper grade of 4.58%. A ten hole program has been outlined for this area with the goal of confirming and converting the high grade Inferred Resources in this area into Measured and Indicated categories. It is expected that with this drilling and detailed mine planning this area can be added into the mine schedule.

The scenario is based on these southern resources being converted to reserves and integrated into the mine plan starting in year nine. Planning accounts for the location and timing relative to the other mining activities as well as the base operating costs required for mining and processing. Some additional capital may be required if Kamoto production needs to expand significantly, however it is expected that ultimately the mining of this or any other higher grade area would ultimately displace or delay the mining of other lower grade areas.

The same criteria used in the financial base case are used in this scenario. The discounted cash flow evaluation of this scenario yields a full debt amortized financial base case IRR of 28.6% and a NPV 909 million USD using a 6% discount rate and an 8.5% debt rate (Appendix A.1.1).

Annual refined copper output peaks at 196,000 tonnes (433 million lbs), while a maximum of 11,200 tonnes (25 million lbs) cobalt is produced (not in the same year due to grade variations). Total copper production over the analysis period of 20-years is 2,572,000 tonnes while total cobalt production is 127,300 tonnes.

15.1.1 Operating Plan

Development and operation of the Kamoto mine would be identical to the base case up through year eight. By then the ventilation and development infrastructure will be sufficient to allow for the expansion of operations, if required. Open pit production may also have to be expanded to provide sufficient oxide ore to balance the sulphide production depending upon the final production profile from Kamoto.

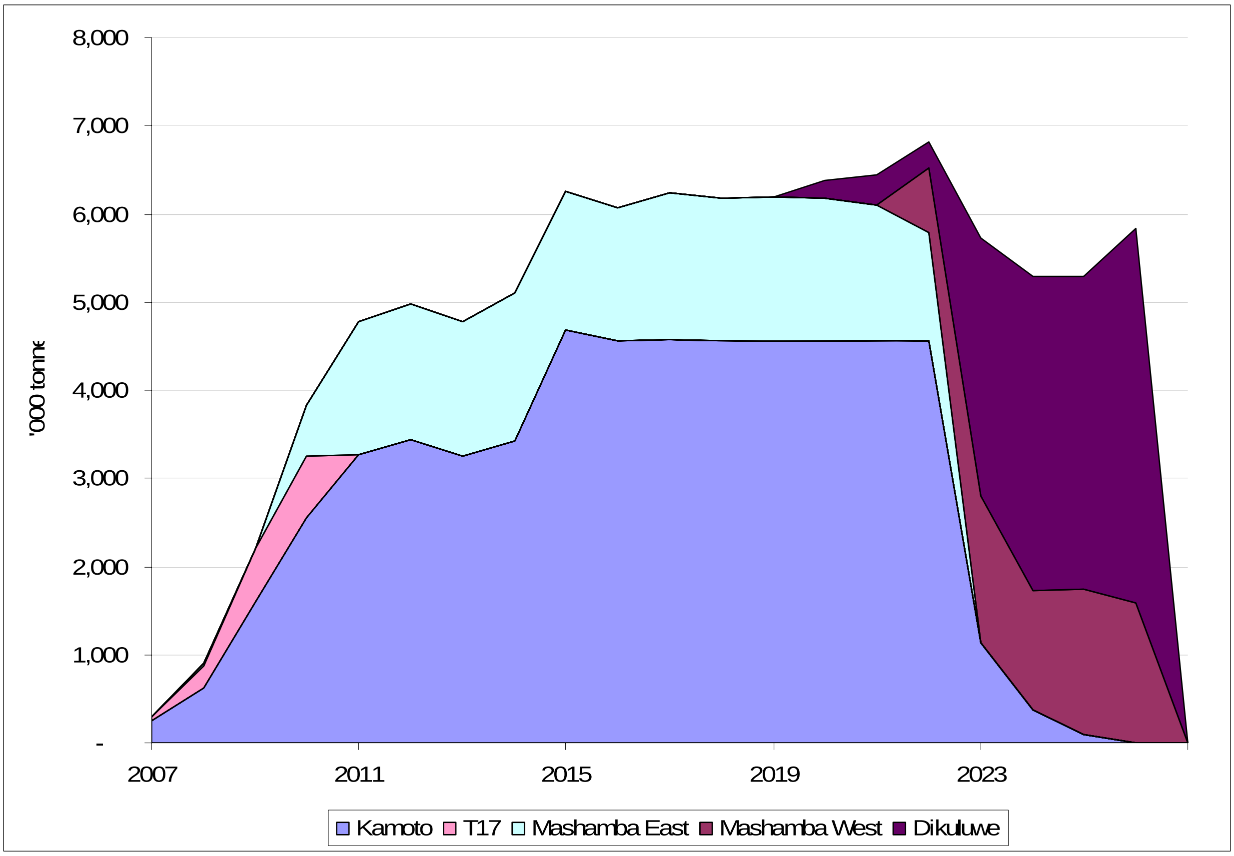

Figure 9 – Expansion Scenario Mine Consolidated Mine Production

15.2 Capital and Operating Costs

15.2.1 Capital Cost Estimate

Capital costs over the life of the project are summarized as follows:

	Total ('000s USD)	USD/t ore	USD/lb. Cu	USD/t Cu
Initial Capex - Phase 1	175,558	1.75	0.03	68
Initial Capex - Phase 2	96,522	0.96	0.02	38
Initial Capex – Phase 3	104,579	1.04	0.02	41
Initial Capex – Phase 4	50,128	0.50	0.01	19
Sub-Total Initial Capex	426,786	4.24	0.08	166
U/G Mine Replacement	103,017	1.02	0.02	40
Concentrator Replacement	30,486	0.30	0.01	12
Luilu Replacement	36,750	0.37	0.01	14
Dikuluwe dewatering	16,000	0.16	0.00	6
Additional tailings pond capacity	42,560	0.42	0.01	17
G&A Replacements	2,450	0.02	0.00	1
Sub-Total Replacement	231,263	2.30	0.04	90
TOTAL CAPITAL	658,049	6.54	0.12	256

Table 21 – Expansion Scenario Capital Cost

15.3 Operating Costs

The unit operating costs use the same methodology as the base case. They are based on the same unit rates for mining and processing by ore tonne and ore type.

Site operating costs by phase are as follows:

	Phase 1	Phase 2	Phase 3	Phase 4	Average
Tonnes (t) Copper	23,952	66,510	114,590	2,366,715	
Tonnes (t) Cobalt	890	2,757	6,725	116,934	
Underground Mining	29,106	37,542	55,362	834,876	
Open Pit Mining	20,747	70,591	73,734	841,872	
Kamoto DIMA Concentrator	8,415	13,696	22,752	456,897	
Luilu Plant	22,762	30,524	50,801	911,935	
General & Administration	19,596	15,579	17,392	210,052	
Total (000s USD)	100,626	167,931	220,041	3,255,632	
Cost per lb. (USD/lb. Cu)	1.91	1.15	0.87	0.62	0.66
Cost per lb. Cu (with Co credit)	1.53	0.73	0.28	0.13	0.17
Cost per tonne Cu	4,201	2,525	1,920	1,376	1,456
Cost per tonne Cu (with Co Credit)	3,382	1,611	626	286	365

Table 22 – Expansion Scenario Operating Cost by Phase

Over the analyzed 20-year life of the project, total production costs are as follows:

	Total (‘000s USD)	USD/t ore	USD/lb. Cu	USD/t Cu
Underground Mining	956,885	16.91	0.17	372
Open Pit Mining	1,006,945	22.89	0.18	392
Kamoto DIMA Concentrator	501,760	4.99	0.09	195
Luilu Plant	1,016,021	10.10	0.18	395
General & Administration	262,619	2.61	0.05	102
Site Operating Cost Sub Total	3,744,230	57.49	0.66	1,456
Cobalt Credit	(2,806,654)		(0.50)	(1,091)
Site Operating Cost Total After Cobalt Credit			**0.17**	**365**
Transport and Marketing Expenses	913,637		0.16	355
Royalty and Lease Obligations	293,934		0.05	114
Capital Costs	658,049		0.12	256
Total Production Costs	**2,803,195**		**0.49**	**1,090**

Table 23 – Expansion Scenario Operating Cost Summary

Under this scenario, the communities and the DRC would recognize the following economic benefits (000’s USD):

DRC Royalty	$162,596
Tax on income	$1,022,409
Dividend tax	$201,794
Capital Equipment Duties	$15,657
Import Duties Consumables	$18,620
Payroll & Social Support	$411,773

Total	$1,832,849
Purchased Power Cost	$405,530

Table 24 – Expansion Scenario Economic Benefits

16. ECONOMIC ANALYSIS

16.1 Summary

The alternative scenario has been modeled with financial returns estimated for the following cases:

- The initial capital investment required to rehabilitate KCC funded by debt (8.5% interest rate) in four tranches, each amortized over 60 months. This is the base case (NPV based on a 6% discount rate). This evaluation does not attempt to finance any operational losses occurring in the first years. They are simply treated as negative cash flows in the first years of the project;
- KCC funded on a 100% equity basis (NPV based on a 15% discount rate);
- KCC funded on a 100% debt basis (8.5% interest rate), with principal repaid before dividends are declared to the partners (NPV based on a 6% discount rate).
- GCM 25% stake in KCC and a royalty with no equity contribution (NPV based on a 6% discount rate).

This financial case uses the same financial assumptions carried in the Feasibility Study financial analysis.

		KCC Base case		KCC 100% equity		Debt	GCM
Copper	**Cobalt**	**NPV_6**	**IRR**	**NPV_{15}**	**IRR**	**NPV_6**	**NPV_6**
$1.10	$10.00	$909	28.6%	$241	24.2%	$885	$298

Table 25 – Expansion Scenario Economic Summary

Annual refined copper output peaks at 196,000 tonnes (433 million lbs), while a maximum of 11,200 tonnes (25 million lbs) cobalt is produced (not in the same year due to grade variations). Total copper production over the analysis period of 20-years is 2,572,000 tonnes while total cobalt production is 127,300 tonnes.

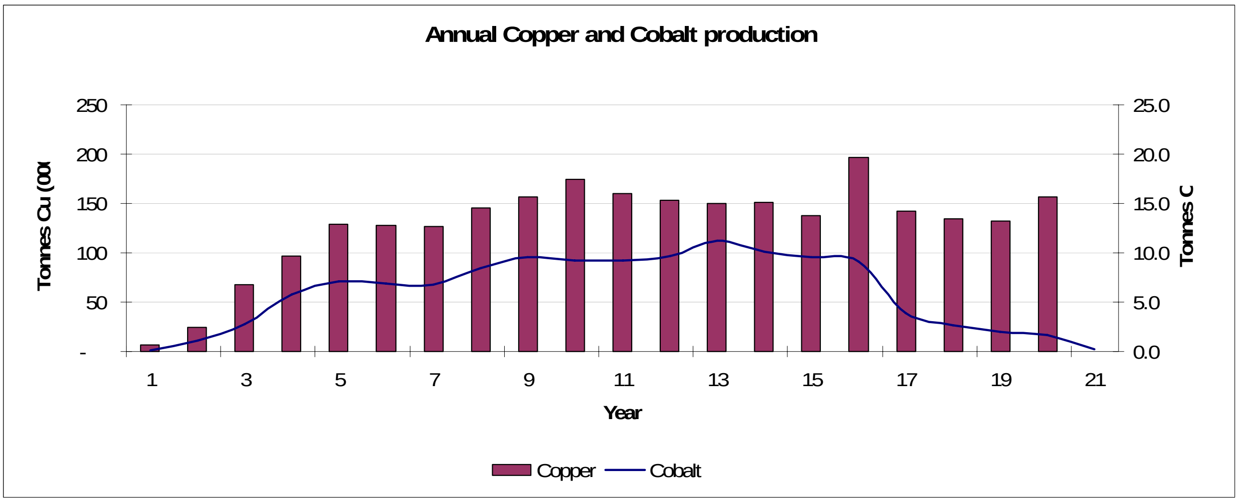

Figure 10 – Expansion Scenario Mine LoM Metal Production

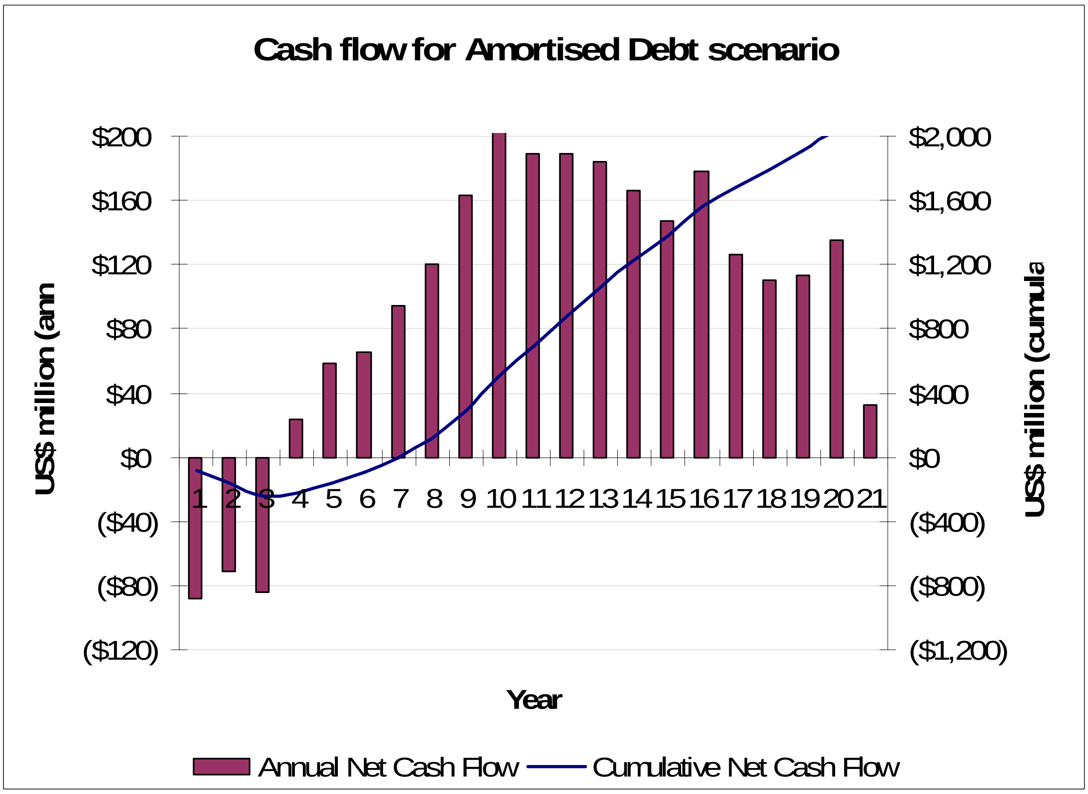

Figure 11 – Expansion Scenario LoM Cash Flow

Figure 8 illustrates the net cash flow received by KCC over the 20-year project life under the base case (amortised debt scenario). The project cash flow can be divided into three main phases:

1 During the ramp up in years 1-3, KCC is cash negative due to the amortisation schedule and higher unit operating costs.

2 For years 4-7, KCC is repaying debt and generating an average of USD 60-million free cash annually.

3 From years 8 onward, the debt is retired and average free cash is driven by the grade profile of the mine.

Financial Models

A.1 KCC Base Case 50% Third Party Debt

			Year									
KFL & Third Party Debt	**Description**	**TOTALS**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Dividends	USD '000s	$1,403,440	$0	$0	$0	$0	$0	$0	$0	$23,642	$114,673	$142,277
KFL redemptions	USD '000s	$225,514	$0	$0	$0	$2,808	$58,031	$51,181	$55,386	$55,638	$2,365	$100
Third Party redemptions	USD '000s	$225,514	$0	$0	$0	$2,808	$58,031	$51,181	$55,386	$55,638	$2,365	$100
Subtotal inflows	USD '000s	$1,854,468	$0	$0	$0	$5,616	$116,062	$102,362	$110,772	$134,918	$119,402	$142,478
KFL investment	USD '000s	($193,044)	($107,261)	($50,193)	($35,590)	$0	$0	$0	$0	$0	$0	$0
Third Party investment	USD '000s	($193,044)	($107,261)	($50,193)	($35,590)	$0	$0	$0	$0	$0	$0	$0
Net cash flow	USD '000s	$1,468,379	($214,523)	($100,386)	($71,179)	$5,616	$116,062	$102,362	$110,772	$134,918	$119,402	$142,478
Discount Factor		6%	94.34%	89.00%	83.96%	79.21%	74.73%	70.50%	66.51%	62.74%	59.19%	55.84%
NPV	USD '000s	$582,551	($202,380)	($89,343)	($59,764)	$4,448	$86,728	$72,161	$73,670	$84,649	$70,674	$79,559
IRR		**18.86%**										

	Year										
KFL & Third Party Debt	11	12	13	14	15	16	17	18	19	20	21
Dividends	$130,285	$131,360	$125,617	$101,774	$81,971	$85,435	$87,211	$111,994	$113,160	$121,748	$32,293
KFL redemptions	$4	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Third Party redemptions	$4	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal inflows	$130,294	$131,361	$125,617	$101,774	$81,971	$85,435	$87,211	$111,994	$113,160	$121,748	$32,293
KFL investment	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Third Party investment	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net cash flow	$130,294	$131,361	$125,617	$101,774	$81,971	$85,435	$87,211	$111,994	$113,160	$121,748	$32,293
	52.68%	49.70%	46.88%	44.23%	41.73%	39.36%	37.14%	35.03%	33.05%	31.18%	29.42%
Discount Factor	$68,637	$65,282	$58,894	$45,015	$34,204	$33,631	$32,387	$39,237	$37,401	$37,962	$9,499
NPV											

A.1.1 Amortized Debt Base Case

KCC 100% Amortized Debt	Description	TOTALS	1	2	3	4	5	6	7	8	9	10
Profit After Tax	USD '000s	$1,514,375	($60,467)	($19,244)	($15,615)	($3,155)	$23,844	$67,398	$95,288	$120,523	$120,287	$141,478
Depreciation	USD '000s	$756,531	$0	$0	$0	$87,705	$126,704	$61,115	$49,881	$42,109	$32,604	$25,680
Interest	USD '000s	$98,584	$13,793	$18,776	$21,466	$18,539	$13,737	$7,388	$3,522	$1,164	$200	$0
Sub-Total Cash Generated By Operations	USD '000s	$2,369,490	($46,674)	($468)	$5,850	$103,089	$164,285	$135,901	$148,691	$163,796	$153,091	$167,158
Amortized Capital	USD '000s	($525,370)	($43,222)	($66,986)	($88,442)	($97,875)	($105,074)	($61,852)	($38,088)	($16,633)	($7,199)	$0
Replacement Capital	USD '000s	($231,263)	$0	$0	$0	($814)	($4,254)	($24,723)	($21,901)	($20,594)	($19,914)	($9,434)
Witholding Taxes	USD '000s	($146,623)	$0	$0	$0	$0	$0	$0	$0	($7,637)	($11,453)	($14,338)
Project Negative Cash Position	USD '000s	$0										
Net Cash Flow	USD '000s	$1,466,234	($89,896)	($67,454)	($82,591)	$4,399	$54,957	$49,325	$88,702	$118,931	$114,526	$143,385
Discount Factor		6%	94.34%	89.00%	83.96%	79.21%	74.73%	70.50%	66.51%	62.74%	59.19%	55.84%
NPV	USD '000s	**$612,395**	($84,807)	($60,034)	($69,345)	$3,485	$41,067	$34,772	$58,992	$74,619	$67,787	$80,065
IRR		**23.83%**										

KCC 100% Amortized Debt	11	12	13	14	15	16	17	18	19	20	21
Profit After Tax	$135,530	$136,212	$129,213	$99,514	$75,775	$77,875	$71,521	$94,648	$96,154	$101,714	$25,882
Depreciation	$35,493	$33,068	$33,670	$31,094	$23,394	$32,283	$33,220	$32,706	$32,455	$33,365	$9,987
Interest	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sub-Total Cash Generated By Operations	$171,023	$169,280	$162,883	$130,608	$99,169	$110,158	$104,740	$127,354	$128,608	$135,080	$35,868
Amortized Capital	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Replacement Capital	($26,733)	($23,904)	($23,842)	($17,797)	($8,260)	($15,271)	($7,740)	($3,040)	($3,040)	$0	$0
Witholding Taxes	($13,117)	($13,216)	($12,640)	($10,255)	($8,264)	($8,626)	($8,818)	($11,301)	($11,415)	($12,280)	($3,261)
Project Negative Cash Position											
Net Cash Flow	$131,173	$132,160	$126,401	$102,555	$82,645	$86,261	$88,182	$113,013	$114,153	$122,800	$32,607
Discount Factor	52.68%	49.70%	46.88%	44.23%	41.73%	39.36%	37.14%	35.03%	33.05%	31.18%	29.42%
NPV	$69,100	$65,680	$59,262	$45,360	$34,485	$33,956	$32,748	$39,593	$37,729	$38,290	$9,592

Project amortization cash flow based on the following schedules:

	Start Month	Term (months)	USD '000
Tranche 1	1	60	175,558
Tranche 2	13	60	96,522
Tranche 3	27	60	104,579
Tranche 4	44	60	50,128

	Tranche				
	1	2	3	4	Total
2007	43,222	0	0	0	43,222
2008	43,222	23,763	0	0	66,986
2009	43,222	23,763	21,456	0	88,442
2010	43,222	23,763	25,747	5,142	97,875
2011	43,222	23,763	25,747	12,341	105,074
2012	0	23,763	25,747	12,341	61,852
2013	0	0	25,747	12,341	38,088
2014	0	0	4,291	12,341	16,633
2015	0	0	0	7,199	7,199
2016	0	0	0	0	0
Total	216,111	118,817	128,736	61,707	525,370

A.1.1.1 150,000 tonnes copper per year scenario

KCC 100% Amortised Debt	Description	TOTALS	1	2	3	4	5	6	7	8	9	10
Profit After Tax	US$ '000s	$2,179,361	($58,775)	($22,508)	($17,374)	($3,451)	$48,739	$85,319	$103,540	$128,836	$170,192	$224,081
Depreciation	US$ '000s	$760,157	$0	$0	$0	$106,684	$105,016	$59,526	$47,596	$39,133	$36,178	$29,126
Interest	US$ '000s	$98,584	$13,793	$18,776	$21,466	$18,539	$13,737	$7,388	$3,522	$1,164	$200	$0
Sub-Total Cash Generated By Operations	US$ '000s	$3,038,102	($44,982)	($3,732)	$4,091	$121,772	$167,492	$152,233	$154,658	$169,133	$206,571	$253,207
Amortised Capital	US$ '000s	($525,370)	($43,222)	($66,986)	($88,442)	($97,875)	($105,074)	($61,852)	($38,088)	($16,633)	($7,199)	$0
Replacement Capital	US$ '000s	($231,263)	$0	$0	$0	($814)	($4,254)	($24,723)	($21,901)	($20,594)	($19,914)	($9,434)
Witholding Taxes	US$ '000s	($207,406)	$0	$0	$0	$0	$0	$0	$0	($11,837)	($16,314)	($22,161)
Project Negative Cash Position	US$ '000s	$0										
Net Cash Flow	US$ '000s	$2,074,063	($88,204)	($70,717)	($84,350)	$23,083	$58,164	$65,658	$94,669	$120,069	$163,143	$221,611
Discount Factor		**6%**	94.34%	89.00%	83.96%	79.21%	74.73%	70.50%	66.51%	62.74%	59.19%	55.84%
NPV	US$ '000s	**$909,212**	($83,211)	($62,938)	($70,822)	$18,284	$43,463	$46,286	$62,960	$75,333	$96,564	$123,747
IRR		**28.63%**										

KCC 100% Amortised Debt	11	12	13	14	15	16	17	18	19	20	21
Profit After Tax	$196,804	$195,742	$189,159	$165,400	$141,353	$168,261	$115,514	$96,865	$99,573	$116,916	$35,174
Depreciation	$37,961	$36,568	$37,242	$35,075	$28,310	$43,298	$30,998	$27,582	$27,716	$32,043	$104
Interest	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sub-Total Cash Generated By Operations	$234,766	$232,310	$226,401	$200,475	$169,664	$211,559	$146,512	$124,447	$127,289	$148,959	$35,278
Amortised Capital	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Replacement Capital	($26,733)	($23,904)	($23,842)	($17,797)	($8,260)	($15,271)	($7,740)	($3,040)	($3,040)	$0	$0
Witholding Taxes	($18,912)	($18,946)	($18,414)	($16,607)	($14,673)	($17,844)	($12,616)	($11,037)	($11,295)	($13,542)	($3,207)
Project Negative Cash Position											
Net Cash Flow	$189,121	$189,460	$184,144	$166,071	$146,731	$178,443	$126,157	$110,370	$112,953	$135,417	$32,071
Discount Factor	52.68%	49.70%	46.88%	44.23%	41.73%	39.36%	37.14%	35.03%	33.05%	31.18%	29.42%
NPV	$99,626	$94,156	$86,334	$73,453	$61,226	$70,244	$46,850	$38,667	$37,333	$42,224	$9,434