



# Processing of Konkola copper concentrates and Chingola refractory ore in a fully integrated hydrometallurgical pilot plant circuit

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## Synopsis

The Konkola Deep Mining Project (KDMP) seeks to exploit the deep level potential of the Konkola Deeps orebody, producing high grade copper concentrates for treatment at the Nkana Smelter and Refinery.

As an alternative to smelting, Konkola Copper Mines (KCM) has examined a processing route involving pressure leaching of the Konkola concentrates at Nchanga, with the leach liquor being treated by solvent extraction and electrowinning to produce high purity copper cathode. The solvent extraction circuit would be operated to achieve a high 'delta' copper, thereby providing a strong acid raffinate which can be used in the existing Tailings Leach Plant (TLP) to treat Nchanga oxide copper materials.

The main focus of the hydrometallurgical option is to capitalize upon the synergy that exists between the treatment of sulphide copper concentrates (acid producing) and oxide copper ores (acid consuming).

The paper discusses the key features of a comprehensive piloting campaign carried out at Anglo American Research Laboratories, which involved the co-treatment of Konkola copper concentrates and Chingola refractory ore in a fully integrated pilot plant circuit.

## Background

On the 31st March 2000 Konkola Copper Mines (KCM) signed a vesting agreement to purchase the Konkola, Nchanga and Nampundwe mines and metallurgical facilities from Zambia Consolidated Copper Mines Limited (ZCCM).

Konkola and Nchanga are mines situated on the Zambian copperbelt. A map of Zambia showing the location of the main copperbelt towns and that of the Nampundwe mine near Lusaka, is presented in Figure 1. The copperbelt is a north-west trending zone 150 km long by 50 km wide, extending from the Luanshya mine in the south to the Konkola mine near the town of Chililabombwe in the north. Together with the copperbelt of the Democratic Republic of Congo (DRC) situated across the border immediately to the north, the Zambian copperbelt constitutes one of the world's great mining regions, charac-

terized by the size and quality of its copper and cobalt deposits. The Nampundwe mine situated south-west of Lusaka is of strategic importance to the metallurgical operations on the copperbelt, supplying a high sulphur pyrite concentrate to the cobalt roast-leach-electrowinning (RLE) plants, and to the copper smelters to make up for a sulphur deficit in the smelter feed.

The Konkola mine commenced production in 1957. The mine is one of the wettest in the world, with a total of 300 000 cubic metres of water being pumped daily from the underground mine. Current mine production from Konkola's Kililabombwe orebody amounts to approximately 2 million ton per year, and this ore is treated through a conventional crushing, milling and flotation plant. The ore mineralogy permits the production of a high grade concentrate containing 38–40% copper, and this concentrate is blended with lower copper, higher iron concentrates from other mines on the copperbelt, and treated either at the Nkana smelter, Kitwe, or at the Mufulira smelter owned and operated by Mopani Copper Mines (MCM).

KCM's long-term future lies with the Konkola Deep Mining Project (KDMP) which seeks to exploit the deep level potential of the Kililabombwe orebody. KDMP's mineable ore resource is estimated at 139 million tons at a grade of 3.81 per cent total copper. KCM intends to exploit these reserves by sinking a new shaft and building a new concentrator. Commissioning is planned for the last quarter of 2005. Mine production will increase to about 6 million ton ore per year, with copper production rising from the current 50 000 ton per year to approximately 220 000 ton per

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Figure 1—Zambian copperbelt

year. Reserves are sufficient for a life of mine in excess of 30 years.

This paper describes the testwork and engineering studies undertaken to examine the potential for treating KDMP's copper sulphide concentrates by a process of oxidative pressure leaching at Nchanga, rather than smelting at Nkana. The concept of hydrometallurgical processing at Nchanga includes the co-treatment of a partially refractory low grade oxide ore reclaimed from stockpiles at Nchanga. The synergies that result from co-treatment of sulphide and oxide copper materials are discussed, as are the potential benefits offered by integration of the new pressure leach facility with the existing Tailings Leach Plant (TLP) at Nchanga. Finally, the relative environmental impact of the smelting and hydrometallurgical processes are compared, and the potential for recovery of cobalt from hydrometallurgical leach liquors is considered.

### The Konkola deep mining project

The period from 2001 to 2010 will see major changes in the nature of KCM's mineral assets, as existing resources are depleted and are replaced by new feedstock sources. These changes will impact significantly upon the downstream requirements for metallurgical processing.

Most significantly the KDMP project will commence in late 2005, ramping up quickly to a production of about 210 000 ton of copper contained in concentrates. In contrast, the Nchanga Open Pit (NOP) will cease to operate by the year 2004 and the Nchanga Underground Mine will close in the year 2010.

Closure of the NOP in 2004 will bring to an end the processing of oxide copper mineralization at Nchanga, and as a result flotation tailings will no longer be routed to the TLP for treatment. The Nchanga Tailings Leach Plant came into operation in 1972 to process current and reclaimed oxide copper flotation tailings<sup>1</sup>. It was the largest copper solvent extraction plant of its time and continues today, in an expanded form, to operate successfully at Nchanga.

There exists at Nchanga an extensive low-grade copper resource known as the Chingola Refractory Ore (CRO) which comprises a mix of conventional oxide copper mineralization and a refractory cupriferous mica ore. Reserves of CRO have been estimated at approximately 140 million tons containing 0.87 per cent total copper. The CRO was stockpiled separately from waste during the mining of the NOP, and testwork has shown it to be an attractive feedstock for treatment in the TLP once tailings resources are depleted. The mineralogical characteristics of the cupriferous micas present in the CRO

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have been well described previously, as has the metallurgical response of the micas to acid leaching<sup>2</sup>.

KCM's metallurgical treatment philosophy is influenced by the requirement to balance the sulphuric acid demand of oxide copper treatment at the TLP, against the acid generated by roasting and smelting of sulphide concentrates at Nkana. A shortage in the supply of acid from the acid plants at Nkana will affect treatment rates and copper production at the TLP; a surplus of acid could pose acid disposal problems if it is not possible to find a suitable outside purchaser for the acid. The philosophy which is to be used by KCM to ensure that an overall acid balance is achieved, involves the processing of the high acid-consuming CRO ore at a rate sufficient to consume the acid produced at Nkana. By this means, the acid is used productively to recover copper from the CRO ore prior to its eventual neutralization with limestone.

### Hydrometallurgical processing of KCM copper concentrates

#### Concept

The predominant minerals found in Konkola and Chingola sulphide concentrates are chalcocite ( $\text{Cu}_2\text{S}$ ), bornite ( $\text{Cu}_5\text{FeS}_4$ ) and to a lesser extent chalcopyrite ( $\text{CuFeS}_2$ ). This mineral assemblage distinguishes these concentrates from the predominantly chalcopyrite concentrates found at Nkana, Mufulira and Luanshya.

The characteristics of Konkola and Chingola concentrates—high copper (42% Cu), low sulphur (15% S), low iron (7% Fe) and high silica (10% Si)—render it necessary to blend the Konkola concentrates with other concentrates of higher iron and sulphur content, and to add limerock as a flux in order to produce a suitable feed for treatment in the Nkana smelter. These same characteristics, however, favour the processing of the concentrates by pressure leaching techniques, which rely on high temperature oxidation processes to solubilize copper (and by-product cobalt) whilst rejecting the iron predominantly as a non-hydrated haematite residue.

Total pressure oxidation of copper concentrates is a process option, which has been tested previously on a number of different concentrate types. The attractiveness of this route for treatment of copper-gold ores, in particular, has been recognised<sup>3</sup>. The process has still to be applied on a commercial scale, however, and this is largely due to the fact that advantageous circumstances are normally required to be present to enable the technique to be economically competitive with the more conventional smelting routes. The high copper, low sulphur concentrates at Konkola, coupled with the availability of oxide ores and an existing leach plant at Nchanga, present an attractive set of advantages for the KCM pressure leaching option. Against this however, is the fact that a smelter already exists at Nkana, and for the pressure leach option to be competitive it will be necessary to demonstrate advantages sufficient to compensate for the higher capital cost of the pressure leach option.

Pressure leaching of concentrates in a single stage oxidation autoclave provides an *in situ* means of transferring 'acid' from the sulphide to the oxide circuit, through the medium of the copper leached in the autoclave. Key to the

concept is the introduction of a new copper SX circuit operating at a high 'delta' copper which provides a raffinate of sufficient acid strength (70–80 g/l  $\text{H}_2\text{SO}_4$ ) to attack the refractory copper-bearing micas in the CRO ore at elevated temperatures.

It is proposed that sulphide concentrates from Konkola and Nchanga (both high and medium grade) are processed in a new pressure leach plant to be installed on the existing Nchanga TLP site, and that the acid resulting from this oxidation process is used to leach either flotation tailings or CRO ore in the existing TLP. Implementation of the concept will require the installation at Nchanga of a new solvent extraction plant (SX1) and two new tankhouse modules with a combined capacity of 178 000 ton copper per annum. These new facilities will be integrated with the existing facilities at the TLP in the manner shown in the flow diagram of Figure 2. The total copper production capacity of the integrated circuit would amount to 260 000 ton copper per annum, making it one of the largest plants of its type in the world.

The CRO ore will be mined from existing rock dumps and will be transported approximately 6 km to the existing Nchanga East Mill, where it will be crushed, milled and pumped to the pre-leach thickeners of the TLP. The quantity of CRO ore treated will be dictated by the acid availability from processing of the KDMP sulphide concentrates. This translates to a treatment rate of about 1 Mt per annum up until 2010, and thereafter once the treatment of stockpiled tailings has terminated, the treatment rate would be increased to about 6 Mt per annum. This is equivalent to a finished copper production from CRO ore of about 40 000 ton per annum.

In the hydrometallurgy option, KCM's total copper would be produced at the Nchanga facility (230 000 to 260 000 tonne copper/annum). There would be no need, in this instance, to be involved with either the Nkana facilities or the Nampundwe mine.

#### Sample descriptions

##### Konkola concentrate

Samples of the Konkola concentrate were delivered to the AARL in sealed drums. The benchscale testwork indicated that further milling of the material was not required and as such the sample was homogenized, sampled and repacked in sealed containers ready to be fed to the pressure leach circuit. The results of the chemical analysis and mineralogical analysis of the sample are shown in Tables I and II respectively.

The main copper sulphide minerals present in the test sample are bornite, followed by chalcocite, chalcopyrite and covellite. Cobalt occurs principally as the mineral carrollite in intimate association with the main copper sulphide minerals. The main gangue minerals are K-feldspar, quartz and mica. Dolomite and malachite are the dominant carbonate minerals present.

##### Chingola Refractory Ore (CRO)

Six composites of the Chingola Refractory Ore (CRO) material were delivered to the AARL as drill core samples. These individual composite samples were milled to attain a  $d_{75}$  of around 60 microns. After milling the composite samples were

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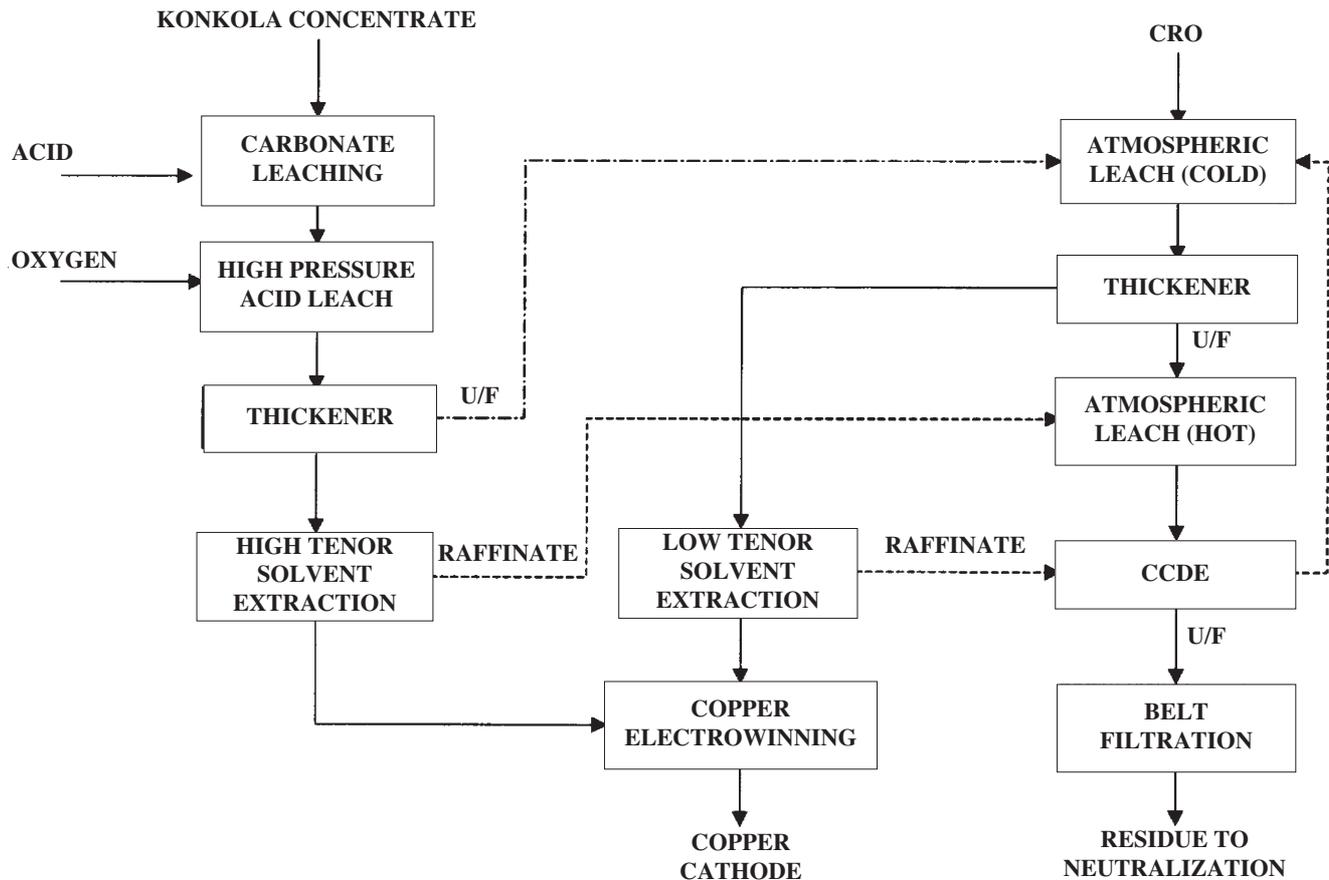


Figure 2—Block diagram of Konkola hydrometallurgical flowsheet

Table I  
Average head grade analyses—Konkola concentrates

Head analysis (%)												
Cu	Co	Fe	Al	Mg	Ca	Mn	Ni	Si	Zn	CO <sub>3</sub>	S	
41.44	0.40	6.51	3.01	0.88	0.35	0.02	<0.01	10.22	<0.01	2.0	~15	

Table II  
Konkola copper concentrate mineralogy

Minerals	Mass %
Bornite	22
Chalcocite	18
Chalcopyrite	11
Covellite	5
Pyrite	2
Unidentified sulphide	2
Cu oxides	1
Dolomite	1
K-feldspar	19
Quartz	8
Oxides	7
Unmatched	4
Total	100

homogenized and repacked into drums. Samples were taken from each drum and analysed. The average head grades are

shown in Table III. Mineralogical examination showed that copper was present in the CRO samples mainly as pseudo-malachite, malachite and cupiferous mica.

### Pilot plant description

Following a successful programme of batch scale laboratory testwork and a preliminary design and costing exercise, a decision was taken by KCM to proceed with a comprehensive programme of piloting at AARL. The pilot plant was configured to represent the flowsheet shown in Figure 2, and included the following process steps:

- A carbonate leach of Konkola concentrates ahead of pressure leaching
- A total pressure oxidation leach
- A high copper tenor solvent extraction circuit for treatment of pressure leach liquors
- A two-stage atmosphere leach of CRO, the second stage operating at elevated temperature
- A low copper tenor solvent extraction circuit (Existing TLP)
- A CCD and belt filter circuit (existing TLP)
- A copper electrowinning tankhouse.

The pilot plant circuit was designed to produce 1 kg per hour of LME Grade A copper cathode which corresponds to a treatment rate of approximately 4 kg Konkola concentrate per hour and 22 kg CRO per hour. The basis of control for the pilot plant was to adjust the rate of CRO addition to the circuit to exactly consume the acid generated in the pressure

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*Table III*  
**Average head grade analyses**

	Head Analysis (%)										
	Cu	Co	Fe	Al	Mg	Ca	Mn	Ni	Si	Zn	CO <sub>3</sub>
CRO 1	1.54	0.04	2.86	5.82	3.66	0.45	0.16	0.02	29.24	0.01	1.98
CRO 2	1.03	0.06	2.94	5.26	3.48	0.58	0.14	<0.01	29.72	0.02	2.48
CRO 3	0.79	0.05	3.13	5.98	3.16	0.81	0.12	<0.01	30.61	0.01	2.35
CRO 4	0.75	0.04	3.11	5.72	3.49	1.00	0.12	<0.01	30.46	<0.01	3.60
CRO 5	1.01	0.03	2.92	5.49	4.64	1.37	0.17	<0.01	28.41	<0.01	4.10
CRO 6	0.97	0.03	2.74	6.40	3.20	0.83	0.13	<0.01	29.71	0.21	2.46

leach section of the flowsheet. The rate of CRO addition therefore varied in accordance with the gangue acid consumption of the individual ore composites.

The target operating conditions for the pilot plant are summarized in Table IV. The main features of the individual components of the flowsheet are briefly presented.

## Carbonate leach

Konkola concentrates are acid deficient under total pressure oxidation conditions, and a quantity of fresh acid must be added ahead of the pressure leach. This acid is used together with recycled raffinate to attack carbonates present in the Konkola concentrates in an atmospheric leach ahead of the autoclave. This prevents the release of excessive quantities of CO<sub>2</sub> gas inside the autoclave, thereby improving the efficiency of oxygen usage.

## High Pressure Acid Leach (HPAL)

The pilot autoclave is a horizontal six-compartment unit fabricated from grade 2 titanium with a total working volume of 25 litres. Each compartment is fitted with a high shear radial flow impeller and a sampling port. The autoclave is externally heated by individually controlled heating jackets on all six compartments. Raffinate is injected into compartments 2 and 3 to simulate the autoclave cooling which would be applied on a commercial scale. The autoclave is discharged through an automatically timed valve to a pressure let-down pot.

## CRO leach

The primary and secondary stages of the CRO leach consist of two and three agitated tanks, respectively. The first stage is an ambient temperature low density leach aimed at consuming the acid in the CCD wash solution ahead of solvent extraction. The second stage is conducted at a temperature of 65°C and a higher pulp density and acidity to achieve a high copper recovery from the refractory component of the CRO ore.

## Solvent extraction

The pilot plant incorporated two solvent extraction circuits each with 3 extraction and 2 strip stages.

The high copper tenor SX circuit operates with a high extractant strength and a high O/A ratio (4,9) to achieve a copper extraction of 50 g Cu/l. from a feed solution containing 60 g Cu/l.

The low copper tenor SX circuit operates with a lower extractant strength and an O/A ratio of 1.0 and aims to

produce a raffinate copper concentration of less than 0.5 g Cu/l.

## Electrowinning

The electrowinning circuit receives the combined advance electrolyte from the two SX circuits. The cell has 4 stainless steel cathode blanks of 0.78 m<sup>2</sup> area each and 5 lead/calcium/tin anodes. Cathodes are weighed every day to determine current efficiency and are pulled every four days. Guar gum is added to the electrolyte as a smoothing agent.

A contract to construct the pilot plant was awarded to Messrs LTA Autecon, MEIP in the third week in August 2000 and construction was completed by the end of September with cold and hot commissioning being completed by 24th October.

The control and instrumentation systems architecture is based on an integrated PLC/SCADA (Programmable Logic Controller/Supervisory Control And Data Acquisition) system.

A dedicated computer, interfaced with the PLC, is used to control the plant. The control system allows continuous monitoring of plant performance and targets. Based on the data generated, corrective action can be taken when a process stream is outside its target values.

Four shift crews were established, using mainly operators who had previously worked on other pilot plant campaigns at AARL. A foreman and a KCM metallurgist complemented each shift.

Prior to the recruitment of the operating teams, the shift metallurgists, under the direction of the production metallurgist and the project leader became fully conversant with the flowsheet, mass balance and equipment lists and proceeded to write the operating procedures for the plant.

*Table IV*  
**Pilot plant target conditions**

<b>Carbonate leach</b>		
Residence time	(h)	3
Temperature	(°C)	65
<b>Pressure leach</b>		
Residence time	(h)	1
Temperature	(°C)	200
Total pressure	(kPa)	2300
Oxygen pressure	(kPa)	700
<b>CRO leach—Stage 1</b>		
Residence time	(h)	2
Temperature	(°C)	30
<b>CRO leach—Stage 2</b>		
Residence time	(h)	6
Temperature	(°C)	65
<b>High tenor SX</b>		
Extractant concentration	(vol%)	32
Temperature	(°C)	40
Extraction stages	-	3
Strip stages	-	2
O/A ratio (extraction)	-	4.9
Copper extracted	(g/l)	50
<b>Low tenor SX</b>		
Extractant concentration	(vol%)	16
Temperature	(°C)	35
Extraction stages	-	3
Strip stages	-	2
Copper extracted	(g/l)	4 to 5
<b>Electrowinning</b>		
Number of cathodes	-	5
Cathode area	(m <sup>2</sup> )	0.78
Current density	(A/m <sup>2</sup> )	250
Temperature	(°C)	40

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Detailed operating, sampling and analytical procedures were drawn up for each section by the AARL management team while the plant was being built, and were used as a training tool prior to the plant commissioning.

## Results and discussion

The pilot plant operated continuously for a period of eight weeks. The overall statistics relating to materials processed and copper produced are as follows:

Total Duration	52 days
CRO ore fed	26.6 tons
Konkola concentrate fed	2.6 tons
Copper cathode produced	940 kg.

### High Pressure Acid Leach

The results achieved from the pressure leaching of Konkola concentrates over the full pilot plant campaign are summarized in Table V.

At the standard operating conditions of 1 hour retention time and 200°C, a retention time test looking at both the solution and solid retention times within the autoclave was conducted. The calculated retention time was 59 minutes for the solution and 56 minutes for the solids, very close to the retention time calculated based on flow and volumes.

During the campaign, a set of samples was taken from the pre-leach circuit and down the length of the autoclave to show the leaching profile and to demonstrate the kinetics of the system. The results are shown Figure 3. The solution analyses have not been corrected for the flashing of vapour that would have occurred during the sampling exercise. At 200°C, the concentrations inside the autoclave would be approximately 80 per cent of the values reported here.

Sample points 1 and 2 correspond to the feed and discharge solutions from the carbonate leach. The profile shows that under the influence of the fresh acid addition copper and iron are leached and the free acid rises to about 49 g acid/l.

Sample points 3 to 8 are the samples taken from the six autoclave compartments, and sample 9 is the sample taken from the let-down pot. The results show that within the autoclave the iron concentration in solution reduces rapidly as oxidation and hydrolysis proceeds forming a mixture of haematite and various basic iron sulphates. Free acid and acid generated through hydrolysis of ferric ion in solution is

consumed by the leaching of copper and cobalt species. Analysis of residue samples taken from the various compartments shows that copper leaching is essentially complete at the end of 40 minutes whereas cobalt continues to leach slowly for the full 60 minutes residue time.

Mineralogical examinations of solid residues shows the order of leaching for the copper minerals to be Bornite>chalcocite>covellite>chalcopyrite.

Quantities of jarosite/alunite were found to be present in the leach residue and gave rise to scaling on some of the internals of the autoclave. Although it is believed that the scaling problem is accentuated at the small scale of operation, the final design has made allowance for installation of a third 'standby' autoclave to permit the possibility of a planned programme of autoclave de-scaling should this prove to be necessary.

Corrosion coupons were placed in the autoclave during the continuous run and were taken out on completion of the campaign. The coupons were placed in both the vapour and pulp phases of compartment 1 and compartment 6. The investigation showed conclusively that titanium grade 2 is the superior material of construction for autoclave internals.

### CRO leach

A total of six CRO composite materials were tested in the atmospheric leach representing different mineralogical zones within the stockpiled CRO dumps. For the duration of the composite 5 campaign, the pressure leach was deliberately stopped and the CRO ore was leached with fresh acid rather than raffinate. This was to determine whether there is any difference in CRO leach performance when using the two different lixiviant types.

The average results of the different runs for each of the composites are shown in Table VI.

Extractions were high for all composites, averaging between 75 and 80 per cent. The gangue acid consumption (GAC) values, with the exception of composite 5, varied between 36 and 58 kg/t. Composite 5 was significantly higher at 83 kg/t, and this is due to the high carbonate content of this CRO composite.

### Solid liquid separation

Extensive work was carried out to define the optimal flocculant and dosage rate for leach slurries. The work showed that Magnafloc M338 worked effectively for both the HPAL slurry and the CRO material. An average settling rate of between 0.25 and 0.30 m<sup>2</sup>/tpd for both the HPAL slurry and the CRO material was found.

The pilot plant used a horizontal belt filter to filter a combined feed of HPAL thickener and CRO residue. The process consisted of a single wash stage with a specified wash ratio of 0.4 m<sup>3</sup> wash/ton solids.

Vendors of filtration equipment were invited to carry out testwork to provide the necessary data for subsequent tendering. The results indicate that a filtration rate in the order of 300 to 320 kg/m<sup>2</sup>h is achievable for the combined CRO and HPAL thickener underflows when no flocculent addition is used.

### High tenor solvent extraction (SX1)

The high copper tenor solvent extraction circuit is relatively

Parameter	Mass balance value	Plant values		
		Minimum	Maximum	Average
1. Extractions				
Copper (%)	97	96.0	99.3	98.1
Cobalt (%)	-	89.4	99.1	96.5
2. Conc. acid addition (kg/t)	250	192.3	250	203.6
3. Discharge values				
Copper tenor (g/l)	82.6	63.5	109.6	89.0
Acid tenor (g/l)	5.0	0.40	15.7	6.5
4. Average temperature (°C)	200	195	204	198
5. Total average pressure (Kpa)	2200	2050	2225	2170
6. GAC (kg/t)				33

Autoclave Profile Solution Analysis

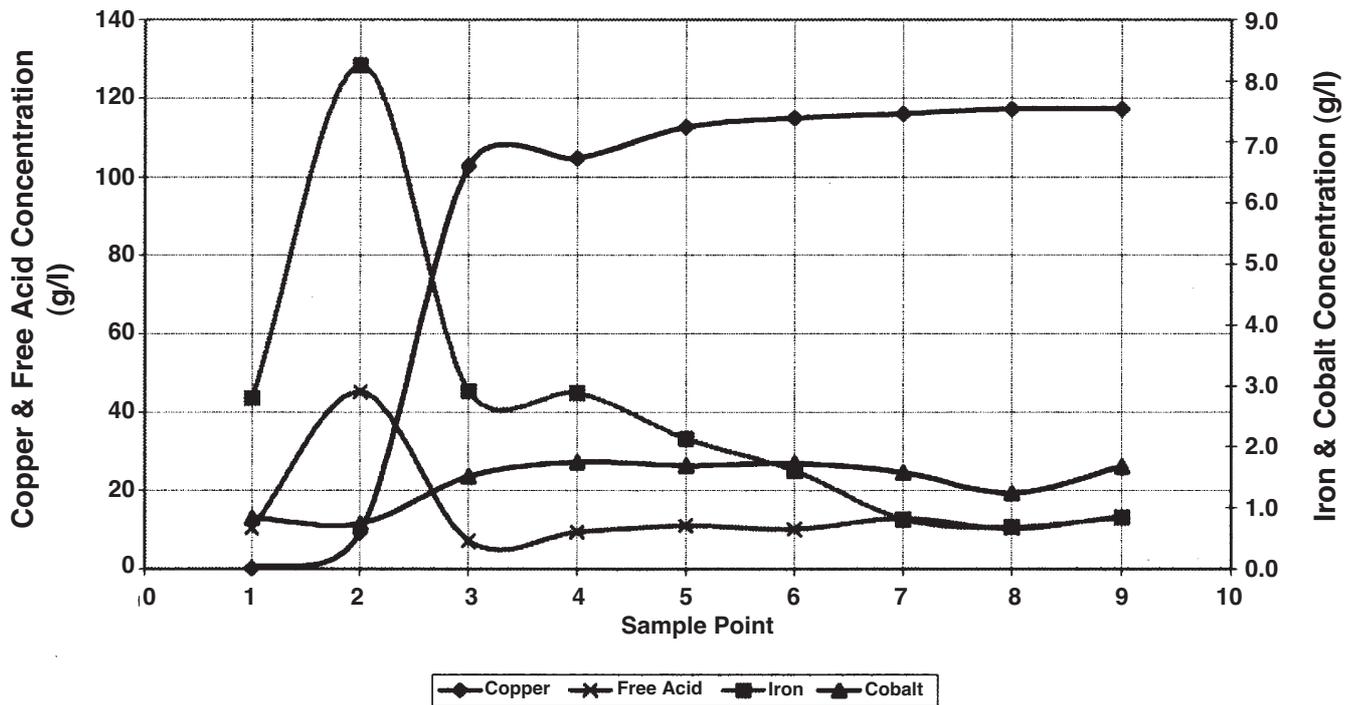


Figure 3—Leach profile solution tenors

Table VI  
CRO leach results

Parameter	Comp-1	Comp-2	Comp-3	Comp-4	Comp-5	Comp-6
Cu head grade (%)	1.54	1.03	0.79	0.75	1.01	0.97
PLS tenors -Cu (g/l)	9.8	7.2	6.0	5.0	3.2	5.2
-Acid (g/l)	2.9	2.9	3.0	2.5	2.7	2.5
Solids feed rate (kg/h)	22.9	19.5	23.9	19.7	16.5	22.5
Copper extraction (%)	80.7	75.5	74.1	77.3	77.0	80.3
GAC (kg/t)	36.7	38.4	41.1	58.7	83.4	46.8

novel due to the high delta copper on the PLS (50 g/l). This was achieved by a combination of high extractant concentration (32% v/v) and a high O/A ratio of 4.9.

Two types of extractant were tested; LIX 984N from Cognis, and M5640 from Avecia. In both cases an extractant concentration of 32% v/v was used, with Shellsol 2325 (25% aromatic content) as the diluent. The circuit was operated at approximately 40°C, which is the expected temperature that will exist in the commercial plant without additional heating. A summary of the operating performance for each of the extractants is given in Table VII.

The circuit performed exceptionally well meeting the required delta copper consistently with very few problems encountered. Some crud formation was observed (probably due to the solid content of the PLS that was at times higher than the target of <20 ppm). Most importantly, the pilot plant

demonstrated that it is possible to produce a raffinate containing approximately 11 g Cu/l and 75 g acid/l which is suitable as a lixiviant for the CRO ore in the TLP circuit.

Low tenor solvent extraction (SX2)

The low tenor solvent extraction circuit would be the existing solvent extraction circuit at the TLP but with the extraction concentration increased to permit the treatment of higher copper tenor solutions from the existing CCD circuit.

The two types of extractant used in SX1 were again tested. In this case however an extractant concentration of only 16% v/v used. A summary of the operating performance for each of the extractants is given in Table VIII.

The circuit performed well, meeting the required delta copper consistently. In the case of composite 1, the high head grades and high extractions seen in the leach meant higher than anticipated copper tenors in the PLS, and as such the raffinate copper tenors were also high. It was agreed not to make any changes to the circuit to compensate for this as the remaining CRO composites had lower copper heads and any change would have to be reversed. The circuit had reached stability under the desired process conditions by the end of campaign 3.

Extraction and stripping isotherms were again produced in the laboratory from samples taken from the pilot-plant circuit. The SX2 isotherm predicted the observed plant behaviour well.

Electrowinning

The electrowinning section of the plant ran relatively smoothly during the campaign. Some problems associated with high current densities, and subsequent gas evolution,

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Table VII

## SX1 performance

	LIX 984N			M5640	
	Comp-1	Comp-2	Comp-3	Comp-4	Comp-6
PLS - Cu (g/l)	59.2	59.8	61.4	60.9	60.3
- Acid (g/l)	2.4	5.6	5.2	4.8	4.4
Raffinate - Cu (g/l)	11.8	13.8	11.3	11.2	11.4
- Acid (g/l)	69.8	72.3	77.7	76.9	75.9
Stripped organic Cu (g/l)	5.3	4.8	4.1	4.9	5.7
Loaded organic Cu (g/l)	15.0	15.0	12.9	14.6	15.2

Table VIII

## SX2 performance

	LIX 984N			M5640		
	Comp-1	Comp-2	Comp-3	Comp-4	Comp-5	Comp-6
PLS -Cu (g/l)	9.3	7.3	6.1	4.9	3.3	5.2
- Acid (g/l)	2.5	2.7	2.9	2.4	2.7	2.6
Raffinate-Cu (g/l)	2.8	1.1	0.6	0.14	0.04	0.20
- Acid (g/l)	12.5	11.8	11.3	9.6	7.7	10.5
Stripped organic Cu (g/l)	2.8	2.4	2.2	3.0	2.6	3.2
Loaded organic Cu (g/l)	8.5	8.1	6.9	7.5	5.8	8.2

occurred early on in the campaign (composite 1), but these were overcome by increasing the submerged area of the cathodes (increasing the weir height) and controlling the maximum allowable current to the unit.

Due to the relatively short campaign length it was necessary initially to spike the electrolyte solution with iron (1700 mg/l) and cobalt (170 mg/l) to ensure correct Eh values and good plating characteristics. Further into the campaign chloride (20 mg/l) was also added (as HCl) to the electrolyte to improve the cathode surface quality. A summary of the performance of the electrowinning section during the campaign is shown in Table IX.

Cathode stripping was performed on four-day cycles with samples being taken of a selection of cathodes from each strip for chemical and physical analysis. The average chemical analysis obtained for the cathodes from each run is shown below in Table X.

The results indicate very high grade copper well within the LME Grade A specifications, with the exception of the oxygen content in the first two composites. This was considered to be due to the high current densities seen in the composite 1 and 2 run, which gave excessive gas evolution on the cathode, and an extended pulling time, which resulted in a significantly reduced cathode-anode spacing.

### Effluent treatment/cobalt recovery

Water balance and impurity build-up considerations dictate the need for a solution bleed from the primary circuit and this bleed is taken from the low copper tenor raffinate stream. Treatment of the CRO ore with its high cupriferous mica content places a particular importance on the neutralization of the bleed stream to ensure that impurities such as magnesium and aluminium, in particular, are removed prior

to discharge of the neutralized effluents to the residue dam.

Samples of raffinate were taken from the pilot plant once steady state conditions had been achieved, and were used in a laboratory programme of neutralization testwork. The basis of the testwork was to use milled limestone to achieve neutralisation to a pH of about 5.0 and thereafter to use milk of lime to raise the pH to a level sufficient to remove the remaining impurities.

The results of the testwork are shown in Table XI for a final pH of 10.5.

The results show that by raising the neutralization pH to 10.5 it is possible to reduce all impurities as well as the total sulphate content of the solution, to a level at which the neutralized slurry can be safely routed to a residue dam.

Limestone consumption to reach pH 5.0 was found to be 14 per cent over stoichiometric, and the lime consumption to achieve pH 10.5 was 22 per cent over stoichiometric.

Konkola concentrate contains a significant quantity of cobalt which is almost completely extracted during pressure leaching. CRO also contains cobalt which is partially extracted during leaching.

Table IX

## Electrowinning performance

	Comp-1	Comp-2	Comp-3	Comp-4	Comp-5	Comp-6
Advance electrolyte -Cu (g/l)	49.7	48.8	46.2	44.7	43.6	50.4
A -Acid (g/l)	155.9	164.6	168.7	168.7	178.4	160.2
Spent Electrolyte -Cu (g/l)	41.5	40.5	37.9	36.9	36.8	41.2
-Acid (g/l)	167.8	175.4	180.4	178.6	187.9	173.0
Temperature (°C)	41.6	41.5	40.9	41.0	37.5	41.2
Total copper plated (kg)	160.5	192.4	186.4	166.7	42.5	190.2
Average current density (A/m <sup>2</sup> )	288	235	230	206	321	244
Overall current efficiency (%)	94.8	93.8	91.7	92.1	88.1	93.7
Power consumption (kWh/t)	2004	2053	2037	2079	2006	2008

Table X

## Cathode chemical analysis (KCM analysis)

Impurities	LME A Grade Specs	CRO 1	CRO 2	CRO 3	CRO 4	CRO 6
O <sub>2</sub> (%)	0.02	0.043	0.039	0.014	0.011	0.020
As (ppm)	5	0.2	0.2	0.1	0.1	<0.1
Se (ppm)	2	0.1	<0.1	0.1	0.1	0.1
Te (ppm)	2	0.2	0.1	0.1	<0.1	<0.1
Bi (ppm)	1	<0.1	<0.1	0.1	0.1	0.1
Sb (ppm)	4	0.2	0.1	0.1	0.1	0.1
Pb (ppm)	5	1.0	0.9	0.7	0.3	1.7
Fe (ppm)	10	<2.0	<2.0	2.0	2.0	2.5
Ni (ppm)	10	0.4	0.4	<0.1	0.5	<0.1
Ag (ppm)	25	<2.0	<2.0	<2.0	<2.0	<2.0
Co (ppm)		0.2	0.2	0.9	1.0	<0.1
S (ppm)	15	<5.0	12	<5.0	<5.0	<5.0
Zn (ppm)		<0.1	<0.1	1.1	1.2	<0.1
Mn (ppm)		0.4	0.6	1.5	0.9	<0.1
Total	65 ppm (exc. O <sub>2</sub> )	<12	<19	<14	<14	<12

## Processing of Konkola copper concentrates and Chingola refractory ore

*Table XI*

**Neutralization of raffinate**

	pH	Solution assay (mg/l)						
		Cu	Co	Fe	Mg	Al	Mn	SO <sub>4</sub>
Raffinate	1.2	93	750	1980	7340	1256	1245	53000
With limestone	5.0	26	720	93	7696	20	1282	33000
With lime	10.5	0.1	0.1	0.5	62	0.5	0.5	1900

Dissolved cobalt reports to the raffinate bleed and for the KDMP project at full production the combined quantity of cobalt would amount to approximately 3000 ton per annum. The challenge which is faced in seeking to recover the cobalt economically from the raffinate stream is to achieve an effective separation of the cobalt from the impurities which also accumulate in the raffinate. Extensive laboratory testwork has demonstrated that it is possible to recover the cobalt from the raffinate using a combination of selective precipitation, solvent extraction and ion exchange techniques. The recovery of by-product cobalt would add significant value to the hydrometallurgical option for Konkola concentrates.

### Design and costing of the hydrometallurgical option

The pilot plant programme provided the necessary technical design information and confidence to proceed with an engineering pre-feasibility study for the hydrometallurgical option. This study was carried out by Anglo Technical Division (ATD), with specialist input from selected engineering companies including AGRA Simons (pressure leaching) and Kvaerner (solvent extraction/electrowinning).

The broad objective of the study was to produce a capital and operating cost estimate for a fully integrated hydrometallurgical complex at Nchanga capable of processing all of KCM's copper feed materials from mid-2005 until the exhaustion of the KDMP mineral assets in 2031. The project returns would then be compared to the pyrometallurgical treatment option incorporating modifications to the existing Nkana smelter and refinery.

The principal components of the process plant were as follows:

- ▶ New concentrate reception and offloading facility
- ▶ New concentrate handling and re-pulping facility including carbonate pre-leach
- ▶ New pressure leach autoclaves and let-down system
- ▶ New high-grade copper SX plant
- ▶ Two new copper tankhouses with combined capacity of 178 000 ton copper per annum
- ▶ Modifications and additions to the existing TLP plant to accommodate the treatment of CRO ore
- ▶ New reagent and utilities section—acid plant, oxygen plant, steam boilers, etc.

Some of the key features of the hydrometallurgical plant are shown in Table XII.

A capital cost estimate was prepared for the proposed hydrometallurgical plant which covered the integration of the new circuit components (pressure leach, high tenor SX, tankhouses) with the existing components at the TLP (atmospheric leach, CCD, belt filters, low tenor SX, tankhouse). The total capital requirements over the period 2004 to 2010 was estimated at US\$ 264 million, equivalent

to approximately US\$ 1100 per annual ton of copper produced. Typical operating costs for the processing of Konkola concentrates and CRO to final cathode (excluding mining, concentrator management overhead and realization cost) are given in Table XIII.

The operating cost summary shown in Table XIII assumes that the acid generated as a consequence of the treatment of Konkola concentrates by pressure leaching (approximately 420 kg acid per ton of concentrate or 600 ton acid per day) is passed to the CRO treatment circuit at nil cost. Had this acid been accorded a value of US\$ 80/ton, which is considered a realistic long-term price for acid in Zambia, the net by-product credit to Konkola concentrates would equate to about US 3.9c/lb Cu and the cost penalty to CRO treatment would be about US 19c/lb Cu. These figures illustrate the importance of securing a low cost acid source for economic treatment of the CRO.

### The hydrometallurgy option

A critique of the hydrometallurgy option in the light of a detailed comparison with the option of smelting concentrates at Nkana, shows both advantages and disadvantages for this process route.

The advantages are briefly as follows:

*Table XII*

**Design parameters—hydrometallurgical plant**

<b>Pressure plant leach</b>	
Design tonnage	566 000 tpa
Autoclave size	4.2 m diameter x 22 m length (190 m <sup>3</sup> working volume)
Number of autoclaves	2+1 (stand-by)
Pressure let-down	2 stage 'flash'
<b>CRO Leach</b>	
Design tonnage	6.5 mtpa
<b>Solvent Extraction</b>	
High tenor PLS flow	480 m <sup>3</sup> /h
PLS concentration	60 gCu/l
Raffinate concentration	10 gCu/l
Number of SX trains	2
O/A ratio	4.9
Low tenor PLS flow	3000 m <sup>3</sup> /h
PLS concentration	5 gCu/l
Raffinate concentration	0.5 gCu/l
Number of SX trains	4 (existing TLP)
O/A ratio	1.1
<b>Electrowinning</b>	
Existing tankhouse	88 000 tpa Cu
New tankhouse	178 000 tpa Cu
<b>Services</b>	
Acid plant	350 tpd
Oxygen plant	675 tpd
Steam boilers	60 tpd

*Table XIII*

**Operating cost summary**

Item	Units	Konkola concentrates	CRO
Copper production	tpa	198 000	40 000
Milling	c/lb	0	8.5
Leaching	c/lb	4.1	17.2
SX / EW	c/lb	6.4	8.0
Neutralization	c/lb	0.5	12.8
Plant Overhead	c/lb	0.7	0.7
Total on-site Costs	c/lb	11.7	47.2

## Processing of Konkola copper concentrates and Chingola refractory ore

- Production and management centralized at Nchanga
- Lower operating costs
- Higher copper recovery
- Platform of modern technology and new equipment established for the future
- Environmental benefits
- Productivity improvements
- Potential for economic recovery of by-product cobalt. The disadvantages are as follows:
  - Higher capital cost
  - Additional risk associated with introduction of new technology.

### Conclusions

Processing of Konkola concentrates and CRO in a fully integrated hydrometallurgical circuit based around the existing Nchanga TLP, has been shown to constitute a sound technical option for KCM. This approach is believed to offer managerial, technical, environmental and logistical benefits for KCM, but the hydrometallurgy option comes at a significantly higher investment cost. Opportunities exist to add value to the hydrometallurgy option and these are presently under investigation. These opportunities include the possibility of replacing the high copper tenor SX circuit with direct electrowinning (capital and operating cost savings), the

opportunity for economic recovery of by-product cobalt (enhanced profitability) and the possibility of achieving a higher copper recovery at the Konkola concentrator due to the lower constraint on concentrate copper grade for the pressure leach option (increased production).

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## Seventh annual course on the design of slurry pipeline systems\*

To be held in Cape Town from 12 to 15 March, 2002

Paterson and Cook Consulting Engineers will present the seventh annual course on the design of slurry pipeline systems in March 2002 at the Breakwater Lodge in Cape Town's Victoria and Alfred Waterfront.

The course was established in 1996 to provide guidance for the design of slurry pipeline systems for the mining and process industries. Slurry pipeline transportation is not adequately covered in most undergraduate engineering courses. Consequently, engineers and metallurgists are generally ill-equipped when faced with the task of designing a slurry transportation system or establishing why a system does not perform its required duty. The course aims to address this problem by giving delegates a sound understanding of slurry flow mechanisms and an appreciation of the design requirements for a successful slurry pipeline transportation system.

The course has proven to be extremely popular and is always fully booked. Delegates from Australia, Botswana, Brazil, Canada, Namibia, the Netherlands, the USA and Zimbabwe have attended the course. A fully instrumented transparent test loop is used during the lectures to demonstrate various aspects of slurry pipeline flow. Visual observations of flow phenomena provide an insight into slurry behaviour and clarify theoretical concepts. Design examples are used to give delegates an opportunity to apply

the principles presented during the lectures. The 2002 course has been extended to four days to cater for additional material that has been incorporated into the course.

The course covers the following:

- flow behaviour and modelling of different types of slurries
- centrifugal and positive displacement pumps
- pump and pipeline wear
- instrumentation and laboratory techniques
- valves for slurry service
- hydraulic design and engineering considerations.

Where possible the course is tailored to suit the delegate's requirements. For example additional lectures on the design of backfill distribution systems and high concentration slurry flow may be presented depending on the delegate profile.

The course will be held from 12 to 15 March 2002 at the Graduate School of Business, University of Cape Town, Breakwater Lodge, Victoria and Alfred Waterfront, Cape Town. ◆

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