Hydrometallurgical treatment of Haib copper ore

by W.J. Schlitt*

Synopsis

The Namibian Copper Joint Venture has been formed to undertake development of the Haib copper deposit, one of the largest undeveloped copper resources in Africa. The development programme includes extensive hydrometallurgical testwork that is being managed by Kvaerner Metals (formerly Davy International). Various parts of the programme have been contracted to four third-party laboratories.

The testwork involves sample selection and characterization, amenability tests, and a parametric study on the leaching variables. All portions of the programme are described and the results are presented. The ores selected for testing are quartz feldspar porphyries. The amenability tests include 'mapping' studies to assess optimum leach conditions, tests at high oxidation potentials, oxidation with various bacterial strains, and heap leach modelling. The parametric programme includes such variables as ore grade, leach temperature, crush size, pH, and ferric iron content of the lixiviant.

Testing was done on both whole ore and concentrate. Results show that bacterial leaching is effective and that ore grade and temperature are key variables. Crush size and pH have less impact on copper extraction. For mid-grade ore, about 0.25 per cent copper, extraction is projected to reach 50 per cent after two years when heap leaching tertiary crushed ore at 60°C and pH 1.5.

The heap leach operation would fit well with the proposed roast-leach operation used to treat higher grade ores. The latter could supply the low cost heat and acid needed in the heap leach.

Introduction

Background

The Haib is one of the largest undeveloped copper resources in Africa. The deposit is located in extreme southern Namibia, about 9 km from the Orange River (boundary with South Africa). The location is in rugged terrain just east of the main (sealed) highway that connects Cape Town, South Africa and Windhoek, Namibia.

The existence of the Haib has been known for some time and there is evidence of limited mining activities on outcrops and high grade, surficial material. The main exploration activity to date was conducted by Rio Tinto Exploration (Rio Tinto). Between 1973 and 1975, Rio Tinto drilled over 40 km of bore holes and assayed more than 250,000 core samples. Rio Tinto's conclusion was that the Haib is a disseminated hypogene ore body with good shows of molybdenum in the zones with higher copper values. The gross geologic resource contains upwards of 1.5 billion tonnes at 0.15 per cent copper.

Ongoing interpretation of drill hole information indicates that selective mining of the three higher grade zones would yield a single pit containing 604 million tonnes of ore at an *in situ* grade of 0.31 per cent copper and a waste:ore strip ratio of 0.72:1. There are higher grade blocks of mineralization that are generally fault-bounded. The owners are presently reviewing the mining schedule with reference to the higher grade blocks, and will be able to produce higher grade feed, greater than 0.4 per cent copper, in the first five years of operation. Chalcopyrite is the main copper mineral, except in the poorly defined zones comprising the leach cap and transition ore. These may contribute some 50 million tonnes of material at an uncertain grade.

In 1976, Rio Tinto opted not to proceed with development, focusing their attention on Palabora copper and Rössing uranium, instead. Since then, several groups have attempted to obtain the rights to continue with exploration and further development of the resource. However, none of these were successful until Great Fitzroy Mines NL (GFM) (now Copper Mines and Metals Ltd) was able to purchase the property.

In addition to drilling programmes undertaken by Rio Tinto and others, several leach testwork programmes have been conducted on Haib ores. The earliest was done by the National Institute for Metallurgy in 1968, so it actually predated the Rio Tinto exploration effort. These tests involved acid leaching of high grade oxidized copper ores

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(~ 2 per cent Cu) from the Haib river valley. Nine samples were submitted, all of which contained copper oxides and silicates, mainly as chrysocolla and brochantite. Recovery from minus 6 mm material in static tests typically exceeded 80 per cent in three days. Acid consumption was variable amongst the samples, but averaged about 20 kilograms per tonne of ore (kg/t).

In 1993, Van Eck and Lurie (Pty) Ltd ran some bucket leach tests on low-grade Haib ore from an old adit. The head sample assayed 0.184 per cent Cu. After 3 days of acid leaching, copper recoveries ran 8 to 12 per cent. Also in 1993, the University of the Witwatersrand conducted static bed leach tests on ore grading 0.384 per cent Cu. Acidic bacterial leach tests showed that 93.6 per cent of the copper was extracted from 0.25 x 0.50 mm material in 83 days. After 126 days, 70.7 per cent of the copper was extracted from 2.83 x 5.60 mm ore and 40.0 per cent was extracted from 10 x 19 mm ore in 97 days. Although these results were very encouraging, later evaluation suggested that the test material was from the transition ore zone, which has a high acid-soluble component. None of this earlier testwork has apparently been published in the open literature.

Overall development programme

The Namibian Copper Joint Venture (NCJV), now comprised of Copper Mines and Metals Limited (CMML) (20 per cent) and Namibian Copper Mines Inc. (80 per cent), has been conducting an extensive programme to assess the development of the Haib Copper Project. Initially, Lycopodium Pty Ltd of Belmont, Western Australia performed several conceptual evaluations and cost studies on various processing options, including heap leaching, flotation concentration, and roasting and leaching followed by solvent extraction-electrowinning (RLE). Subsequently, NCJV retained Minproc Engineers Limited and Davy International (now Kvaerner Metals, Davy Nonferrous Division) from Perth, Western Australia as a joint venture team (MDJV) to perform further technical studies on the post-mine processing of the Haib ore. These included the study of various processing options, followed by flowsheet development, engineering and cost assessment for the selected treatment route.

As part of its overall responsibility, the MDJV also developed and managed an extensive metallurgical testwork programme to characterize the ore from the Haib deposit. The work focused on ore comminution and flotation, sulfide roasting, calcine leaching, and heap leaching. Results from this activity were used to define the design criteria for flowsheet development and engineering design.

Hydrometallurgical programme

The testwork programme was handled in two parts. Those activities related to concentrate production and treatment, including roasting and calcine leaching, were directed and reported on from Minproc's office in Perth. The whole ore leaching activities were similarly handled from the Davy (Kvaerner) office in San Ramon, California. As originally set forth, the hydrometallurgical programme was to have three phases—(1) Amenability testing, (2) Heap leach testing: rock size, and (3) Heap leach testing: ore type variability. Ultimately, the third phase was postponed and the programme was divided into amenability and parametric phases, preceded by a separate sampling activity at the Haib deposit. In addition to the testing of whole ore, a limited number of tests were conducted on samples of flotation concentrate prepared from the main ore composite.

Davy provided overall management of the hydrometallurgy programme and an integrated interpretation of the results. However, the testwork was performed by third party laboratories selected on the basis of competitive bidding. Participating laboratories included BacTech (Australia) Ltd of Belmont, Western Australia (BacTech); METCON Research, Inc. of Tucson, Arizona, USA (METCON); MINTEK of Randburg, RSA (MINTEK); and the University of the Witwatersrand, in cooperation with Technikon Witwatersrand, Johannesburg, RSA (Wits). The specific testwork activities and the responsible parties are listed below.

Sampling

- Selection of ore types to be tested—NCJV and MDJV
- Collection of ore samples—NCJV
- General preparation of samples—MINTEK

Amenability testing

- Oxidation of Haib chalcopyrite mineralization with thermophylic bacteria—BacTech
- Heap leach modelling on low-grade Haib ore—Wits
- Column leaching tests on low-grade Haib ore using different lixiviants—METCON
- Screening tests on Haib concentrate—MINTEK
- Mapping studies to assess optimum leach conditions for Haib ore—MINTEK
- Sterile and bacterial column leach tests on low grade Haib ore—MINTEK
- Dissolution of Haib ore mineralization at elevated redox potentials—MINTEK

Parametric testing

- Effects of temperature, ore grade, and crush size—METCON
- Effect of acid level (pH)—MINTEK

Testwork programme

Sampling

Geological interpretation has defined two major rock types in the Haib deposit—quartz feldspar porphyry (QFP) and feldspar porphyry (FP). These may represent different types of alteration on a primary plutonic intrusive. In addition to these two rock types there are several important variants. One of the most prevalent is potassium-altered QFP (QFP– K alt). Others include a highly altered (foliated) FP, a high mafic QFP, an altered fine grained QFP (with loss of texture), and a sericitic altered QFP.

When NCJV drove a new adit from the surface into one of the high grade areas, three ore types were encountered. These were QFP, QFP–K alt, and near-surface transition ore (>0.3 per cent Cu, with 10 to 50 per cent acid soluble Cu). The result prompted a decision to conduct the amenability studies on a composite sample of QFP–K alt ore from the adit. This became known as the AD5 composite. A portion of this was subjected to comminution and flotation so that both a bulk sulfide concentrate and a cleaner concentrate were also available for testing. Other ores needed for leach testing were

selected from the large amount of Rio Tinto drill core still stored at the site. Although the core was quite old, it appeared largely unoxidized, probably due to the extremely low humidity in the Namibian desert. Head assays indicated very low acid soluble copper levels, confirming the lack of oxidation.

Amenability testwork

As indicated above, the amenability programme involved six different major activities at four different laboratories. Therefore, it is not possible to provide detailed descriptions of all the laboratory procedures that were utilized. As an alternative, summaries of the methodologies employed are provided below. These are intended to provide a clear understanding of the types of tests performed and the procedures involved. As all work was done under contract, the vendors can no doubt be contacted directly for more details on the testwork they can perform.

BacTech

Tests were conducted to determine whether or not thermophylic bacteria are capable of effectively leaching (oxidizing) the Haib chalcopyrite mineralization. Testing was performed on both the bulk sulphide concentrate (3.5 per cent Cu) and the cleaner concentrate (11.2 per cent Cu). Each sample was halved, with one portion ground to a particle size of P_{100} = 45 μm and the other given an undisclosed pretreatment (possibly ultra-fine grinding). Splits were agitation leached in 1.5 L of inoculum at 10 per cent solids and 45°C. Tests continued as long as 786 hours, with daily solution assays for copper and iron. Final leach residues were washed and assayed to confirm extractions. Washed residues were leached for 48 hours with 0.5 g/L sodium cyanide at pH 10.5 to determine gold recovery.

Wits

Column tests were run on the AD5 composite ore to determine the leaching characteristics of this material and then develop a mathematical model of the heap leaching process. In these tests the reaction rates of both the sulphide minerals and gangue constituents were monitored as a function of ore particle size (1.4 x 2, 2.8 x 4, 6 x 8, and 14 x 19 mm, plus bulk ore).

Several batch tests were run initally to identify optimum conditions for the long-term column tests. These included a sulphuric acid leach at pH 2.0, a similar leach with nitric acid added to determine the effect of strong oxidizing conditions, and a hydrochloric acid leach to determine calcium solubility without gypsum formation.

Following evaluation of the results, six columns were loaded with ore of various sizes and given an acid preleach to condition the material and remove any acid-soluble copper. Then the columns were leached in locked cycle with recirculated bacterial leach solution in aerated columns at 30°C and pH 2.0. Parameters being monitored were temperature, redox potential (Emf in mV), acidity (pH), and soluble copper, iron, aluminium, magnesium, and potassium levels. On a few occasions, baterial acitivity fell off and the leach solution was replenished with a fresh batch from an external bacterial growth system. At the end of the tests the residues were rinsed, drained and unloaded. The material was sized to determine the degree of acid-induced rock decrepitation, then assayed to confirm extractions and examined mineralogically.

METCON

Column tests were run on the AD5 composite ore to determine how this material responds to a preleach acidulation step and to different lixiviant systems. Following sample preparation, a series of four 5-day bottle roll tests were performed on minus 1.65 mm material. These were designed to provide initial data on the leachability of the ore, the effects of iron concentration and chloride ion in the lixiviant, and acid consumption requirements. All tests utilized 0.5 kg of ore (33 per cent solids) at ambient conditions, with the lixiviant compositions shown in Table I.

Following completion of the bottle roll tests, four long-term column leaches were initiated. Each had an ore charge of 115 kg contained in 200 mm diameter by 2.2 m high columns. Leaching was done in locked cycle at a flow rate of 0.10 L/min/m2. Column effluents were monitored daily to determine copper and iron extraction, acid consumption, acidity (pH), and Emf. Futher details are shown in Table II.

Initially, all columns were fed with lixiviant at pH 2.0. However, based on tests at MINTEK (which follow later), a feed pH of 1.5 was found to be more effective. This change was made to all columns on test day 120. The two acid cured columns were taken off-line on test day 168. On test day 184 the temperature of the remaining columns was increased from about 30 to 45°C in order to gain some insight to the effect of temperature on leach kinetics. CL-04 (chloride-bearing) was taken off-line after 282 days and CL-02 (sulphate-only) was taken off-line after 316 days of leaching. Following shutdown, all columns were rinsed and drained, then the residues were rescreened to determine the degree of rock decrepitation and splits were assayed and examined mineralogically.

Table I

Lixiviant compositions planned for METCON bottle roll tests

Test No.	H_2SO_4, g/L	HCl, g/L	$FeCl_3$, g/L	$Fe_2(SO_4)_3$, g/L
1	10	10	3	-
2	10	10	10	-
3	20	-	-	3
4	20	-	-	10

Table II

Test parameters for METCON column tests

Column No.	Acidulation (acid cure)	Lixiviant
CL-01	10 kg/t H_2SO_4, w/ 2-wk cure	5 g/L H_2SO_4 + 3 g/L $Fe_2(SO_4)_3$
CL-02	None	As above
CL-03	5 kg/t @ H_2SO_4 & HCl, w/ 2-wk cure	2.5 g/L @ HCl & H_2SO_4, 3 g/L $FeCl_3$
CL-04	None	As above

Mintek

MINTEK conducted a comprehensive four-part amenability programme intended to identify the parameters required to effectively leach low-grade Haib ore. Each part is described below.

Tests on concentrate

An initial series of five tests was run on Haib concentrate. These were conducted to characterize the leachability of the Haib sulphide minerals without the added complexities associated with possible lixiviant-gangue mineral interactions. Three of the tests were conducted under sterile conditions, wherein the ferric iron needed to oxidize the sulphides was supplied externally. The remaining tests were bacterial leaches in which the ferric iron was generated *in situ* via bacterially catalyzed oxidation of the iron. All testing was done in small stirred reactors under ambient conditions.

Testing was done on the same AD5 cleaner concentrate used by BacTech in their programme. The sterile tests each utilized 20 g of concentrate in 125 mL of lixiviant in baffled glass reactors with magnetic stirrers. The redox potential was maintained at 430 mV by making small potassium permanganate additions. This provided a constant ferrous:ferric ratio at a total iron content of 3.45 g/L. Sulphuric acid additions were used to maintain a pH of 0.5. After three days of agitation, the tests were stopped and the residue was recovered by filtration. Then the material was weighed and assayed to confirm extraction.

The bacterial tests were larger scale, requiring 1.765 kg of concentrate in 10 kg of leach liquor. Initially the reactor was heated and agitated, with acid added to maintain a pH level of 1.5. This removed any acid soluble copper and iron and conditioned the ore. One of MINTEK's proprietary mixed-strain inoculants was added after seven days and the tests were run for another 30 days. Various parameters were monitored, including copper, iron and sulphate levels, pH, Emf, and temperature. Inlet and outlet gas flows were monitored for oxygen and CO_2.

Mapping studies

In this portion of the programme 12 runs were made to monitor the Haib chalcopyrite leach rate over a wide range of temperatures and redox potentials. The objective was to find the combination of parameters that provided the optimum leach rate. A secondary objective was to determine if there was a 'window of leachability' where rapid leaching occurred at low Emf values.

These tests used 10 g of the AD5 cleaner concentrate in mechanically stirred reactors containing 6.5 L of leach solution and operating at pH 0.5. The high liquid:solid ratio assured that solution composition would not change significantly as the concentrate was oxidized. Four different Emf levels (430, 500, 570, and 680 mV) were tested at each of three temperatures (45, 60, and 70°C). Test duration was either 50 or 72 hours, depending on conditions.

The redox potential or Emf was controlled by adjusting the ferric:ferrous ratio in the leach solution. The correct ratio for each run was calculated from the Nernst equation

$$E=E_w+(RT/nF)\ln[Fe^{3+}/Fe^{2+}] \quad [1]$$

where $E_w=E^o-E_{ref}$ was corrected for temperature. The total iron concentration was 30 g/L in all tests. This was large enough that the Emf value would not vary by more than 30 mV, even if all concentrate was oxidized.

To start the tests, the leach solution was first brought to the desired conditions. Then concentrate was added and stirring began. Chalcopyrite dissolution was tracked by periodic measurement of the copper concentration in the solution. After evaluating the results, three runs were repeated, with magnetic stirring being substituted for the mechanical agitation.

Column tests on whole ore

Eight long-term column tests were conducted using the AD5 ore composite to provide information on the effects that selected operating parameters have on copper recovery. In four of the columns the active lixiviant (ferric iron) was bacterially generated *in situ*. These were run at feed pH levels of 1.0, 1.2, 1.5, and 2.0. The latter was reduced to pH 1.7 after 237 days in an effort to find the optimum pH level.

In the remaining columns, the required ferric levels were maintained by external reoxidation of the ferrous iron in one of MINTEK's proprietary bacterial ferric iron generators. Otherwise, these tests were run much like the bacterial columns. They were intended to provide preliminary information on the effects that temperature, crush size and redox potential have on leach performance. The duration of the tests varied from column to column. A matrix of all test conditions is shown in Table XIX.

As with the METCON programme, bottle roll tests were conducted first in order to measure the initial acid demand by the gangue constituents. These were run for 24 hours at a liquid:solid ratio of 2:1 and a pH of 1.5. The amount of acid required for curing was determined by wetting four 1-kg samples of minus 19 mm ore and adding enough acid to give pH levels of 1.0, 1.2, 1.5, and 2.0. After standing for 16 to 18 hours, enough water was added to give a liquid:solid ratio 2:1 and the slurry was bottle rolled for two hours. Iron and copper extractions were recorded.

Typical bacterial column curing and start-up began with the loading of 38 kg ore charges into columns 150 mm diameter by 1.5 m high. Each charge was prepared by recombining the proper amount of ore from each size fraction. The charges were agglomerated with the acid cure dosage determined above. Loaded columns were allowed to stand for 72 hours, then leached in open circuit until the acid soluble copper was extracted (about 4 days). At that point the tests were changed to locked cycle and the columns inoculated with bacteria. These columns were aerated at 25 mL/min and maintained at 30°C.

The leach solutions exiting the columns were collected in sealed reservoirs, and sampled for copper, iron, acid, pH and Emf. Then the solutions were recycled to the top of the columns. Later in the programme, aluminium, magnesium, and calcium contents were also monitored to study the rate and mechanism of acid attack on the host rock. Periodically copper was stripped from the column drainage by solvent extraction and acid was added to maintain the desired pH levels.

The bacterial tests were started in April 1996. The sterile tests were initiated about 40 days later. The test at 60°C was started after the test at pH 1.0 was terminated. The initial solution application rate in all tests was 0.12 $m^3/h/m^2$, a level much higher than planned. This was lowered to the correct value of 0.12 L/min/m^2 in late July 1996. Leach extraction actually improved slightly when the flow rate was lowered. However, the overall results and conclusions did not appear to be compromised by the flow rate effects.

Tests at elevated redox potentials

This involved a limited series of tests to determine which mechanism controls the leach rate of copper in the Haib chalcopyrite ore–

- Chemical reaction at the chalcopyrite mineral surface, or
- Diffusion of reactants/products through the gangue rock matrix between the disseminated copper mineralization and the surrounding leach solution.

The testwork involved two batch tests on the AD5 cleaner concentrate and two continuous column tests on crushed AD5 ore. All were run at 30°C and utilized an acidic chloride lixiviant sparged with chlorine gas to maintain an Emf of 1100 mV, or more. Two lixiviants were tested in the batch tests. One was 1 M (molar) hydrochloric acid. The other was a 100:1 (v/v) mixture of 1 M sodium chloride and 1 M hydrochloric acid.

In the concentrate tests, 1300 mL of lixiviant were added to a baffled reactor, the temperature was adjusted to 30°C, stirring began and the reactor was sealed and vented. Then 13 g of concentrate were added. Solution samples were taken after 15, 30, 60, 90 and 120 minutes. These were filtered, the pH and Emf were checked, and the samples were analyzed for copper, total iron, sulphate, and total acid. At the end of each test the residual solids were recovered, washed, dried, and assayed to confirm extractions.

The continuous tests were run in polypropylene columns 290 mm in diameter and 750 mm high. Both tests used the 100:1 lixiviant and were identical, except that one column charge had a 19 mm top size and the other had a 12 mm top size. The chloride lixiviant was sparged with chlorine gas in an external reactor and delivered to the columns at a flowrate of 2 L/min/m^2. Leaching was conducted for eight hours per day, with a total leach cycle of 36 to 40 hours. After reviewing the results, leaching of the minus 19 mm column resumed and continued until the total time under leach reached 97 hours. Analytical procedures were the same as those used for the stirred reactor tests.

Parametric test work

Only two laboratories, METCON and MINTEK, participated in the parametric portion of the programme. METCON's column leach programme was greatly expanded to include a systematic assessment of the effects of temperature, crush size and initial ferric content of the lixiviant on copper extraction and acid consumption. Two additional QFP-K alt ores were included to provide grade vs extraction information. The low and high grade materials assayed 0.17 per cent and 0.80 per cent copper, respectively, vs the AD5 (mid-grade) assay of 0.27 per cent copper. The parameters studied were as follows:

- Temperature—30, 45, and 60°C
- Top size—9.5, 12.7, 19.0, and 25.4 (mid-grade only) mm
- Initial ferric level in lixiviant—0.5, 2.0, and 5.0 g/L Fe^{3+}.

The columns at the two higher temperatures were maintained in insulated, thermostatically-controlled cabinets in which the temperature varied by less than ±2°C. Otherwise, except for using slightly smaller columns, the METCON parametric programme was run in the same manner as described for their column amenability programme.

The MINTEK portion of the parametric study was designed to examine the effect of lixiviant acid strength on leach kinetics and acid consumption. The acid strength was indicated by pH, with testing done at pH levels of 1.0, 1.2, 1.5 and 2.0 (subsequently reduced to 1.7). These tests were conducted concurrently with the amenability studies, as described previously.

Presentation of results

General sampling programme

Except for the high and low grade ores used in the METCON parametric studies, all work was done on the AD5 ore composite or flotation concentrates produced from this ore. Rather complete chemical and mineralogical studies were performed on this material, with the results shown in Tables III (crush size), IV (chemical composition), and V (mineralogy). Similar information is provided for the AD5 cleaner concentrate in Figure 1 (size distribution), Table VI (chemical composition), and Table VII (other characteristics). Limited assay information on the bulk sulphide concentrate (used by BacTech only) is provided as a footnote to Table VI.

Crush size distributions and partial chemical compositions for the three QFP-K alt ores used by METCON are presented in Table VIII; mineralogical characteristics are provided in Table IX. These include data on the split from the AD5 ore composite that was shipped to METCON.

The forms of copper given in the tables were determined using the sequential method of analysis given by Parkinson and Bhappu[1]. This involves a determination of the acid soluble copper (Cu_{AS}) on a fresh assay split. Then the residue from this step is assayed for cyanide soluble copper (Cu_{CN}). Finally, the latter residue is assayed for residual copper. The sum of the three assays should equal the total copper (Cu_{Tot}) assay within normal analytical accuracy. Generally speaking, the sum of the acid and cyanide values provides a measure of the readily leachable copper. The residual copper typically represents refractory chalcopyrite.

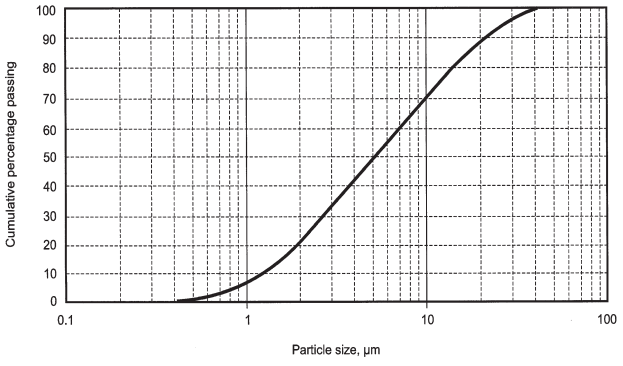

Figure 1—Particle size distribution, Haib cleaner concentrate

Table III

Crush size distribution on bulk AD5 ore composite

Size range, mm	Weight, kg	Size distribution, wt %
19 x 12	671	35.77
12 x 6	764	40.72
6 x 3.5[1]	156	8.32
3.5 x 2.85[1]	52	2.77
< 2.8	233	12.42
Total	**1876**	**100.00**

1. Size fractions combined for purposes of reconstituting column charges.

Table IV

Partial chemical composition of the AD5 ore composite[1]

Cu_{Tot}, %	0.28	Cu_{AS},[2] %	0.03	Cu_{CN},[2] %	0.03
S_{Tot}, %	0.39	S^{2-}, %	0.34	$(SO_4)^{2-}$, %	< 0.5
Fe, %	2.46	Si, %	28.5	Al,[3] %	8.5
K,[3] %	4.39	Ca,[3] %	2.8	Na,[3] %	1.10
Mg,[3] %	1.0	C_{Tot}, %	0.09	$(CO_3)^{2-}$, %	0.35[4]
Mo, ppm	41.3	As, ppm	79.8	Ag, ppm	< 2.5

1. Recalculated head based on assays for each size fraction and weighted on basis of size distribution.
2. AS is acid-solube copper; CN is cyanide-soluble Cu.
3. Semi-quantitative analysis.
4. Calculated value.

Table V

Mineralogy of the AD5 ore composite

Host Rock—Quartz porphyry containing quartz and feldspar. Microcline phenocrysts common and most feldspar extensively altered to sericite.
Rock Constituents—Major (> 20%) minerals are quartz, anorthite and microcline; minor (1 to 20%) are chlorite and sericite; trace (< 1%) are sulphides, magnetite, apatite, and leucoxene.
Sulphide Mineralization–76% chalcopyrite, 18% pyrite, 6% other (molybdenite and galena). Pyrite: chalcopyrite ratio ~ 1:4. Median sulphide grain size ~ 60μm.
Mode of Chalcopyrite Occurrence–Disseminated throughout rock matrix including layered silicates, feldspars and occlusions in quartz. At minus 2.8 mm particle size, 55% of sulphides still locked in silicates, only 8% fully liberated.
Other—Chlorite predicted to be major acid consuming constituent.

Table VI

Partial chemical composition of the AD5 cleaner concentrate[1]

Cu,	11.4	S_{Tot}, %	15.5	S^{2-}, %	15.3
$(SO_4)^{2-}$, %	<0.2	S°, %	< 0.2	Fe, %	15.9
Si, %	15.7	Al, %	4.85	Ca, %	1.16
C_{Tot},%	0.38	$C_{Organic}$, %	0.3	$(CO_3)^{2-}$, %	0.40
Mg, %	0.84	Au, g/t	9.5	Ag, g/t	28.5
Mo, ppm	645	Pb, ppm	595	Zn, ppm	200

1. Partial analysis of bulk sulphide (rougher) concentrate—3.53% Cu, 10.10% Fe, 5.03 % S_{Tot}, 3.81% S^{2-} 1.53 g/t Au

BacTech programme

The work done by BacTech showed that with their pretreatment step more than 98 per cent of the contained copper could be released by leaching with proprietary thermophylic bacteria. However, acid consumption was quite high, averaging nearly 300 kg/t. Without the pretreatment, copper release averaged only about 50 per cent, but with one-tenth the acid consumption. For these samples the slopes of the curves of extraction vs time had become nearly horizontal when the tests were terminated. However, ample chalcopyrite remained available for reaction. This suggests that some type of passivation may have occurred with the untreated concentrate.

Gold recovery from the leach residue was better in the standard trials than the ones with pretreatment. Extractions were approximately 95 per cent against 83 per cent. Final metallurgical balances for both concentrates are presented in Table X. Typical leach curves are shown in Figure 2.

Wits programme

All aspects of the work done by Wits have been discussed in considerable detail by Tomicki[2], as this was the basis for his Masters dissertation. To summarize, the preleach batch tests showed the following:

- In general, metal extraction decreased by an order of magnitude as the particle size increased from 0.053 to

Table VII

Characteristics of AD5 cleaner concentrate

Particle Size—P_{10} = 1.3 μm, P_{50} = 5.2 μm, P_{90} = 21.3 μm.
Bulk Density—3.066 kg/m[3].
Mineralogical Constituents—Major (> 20%) are quartz and feldspar, minor (1 to 20%) are chlorite, mica, chalcopyrite, and pyrite.
Mineralogical Modal Analysis, mass basis—Gangue minerals, 64%; chalcopyrite, 29%; pyrite, 4.6%; iron oxides, 1.4%.

Table VIII

As-received size distributions and partial head grades for the three QFP-K alt ores used by METCON in their parametric testwork programme

	Ore grade		
Parameter	**Low-grade**	**Mid-grade (AD5)**	**High-grade**
Size fraction, mm		**Size distribution, wt %**	
> 18.9	0	27.4	0
13.3 x 18.9	44.9	20.0	45.2
9.4 x 13.3	16.4	9.1	16.8
6.7 x 9.4	13.6	10.7	13.4
1.65 x 6.7	15.4	16.7	14.8
< 1.65	9.7	16.1	9.8
Total	**100.0**	**100.0**	**100.0**
Assay component		**Head grade %**	
Cu_{Tot}	0.17	0.27	0.80
Cu_{AS}	0.007	0.022	0.032
Cu_{CN}[1]	0.020	0.035	0.082
Fe	2.78	2.43	3.82
S_{Tot}	0.30	0.37	2.31

1. Cu_{AS} is acid soluble copper; Cu_{CN} is cyanide soluble copper.

2.8 mm. In terms of mass extracted, Fe > Al > Mg >> Cu. At the finest size, about 1000 mg/kg of magnesium, aluminium, and iron were extracted, vs 200 mg/kg for copper.

- ➤ Extractions increased by a factor of five to ten in the presence of nitric acid. Maximum iron extraction was about 11,000 mg/kg, vs about 4000 mg/kg for magnesium, aluminium, and copper.
- ➤ About 30 per cent of the calcium dissolved in the hydrochloric acid leach at an initial pH of 0.2.

The results of the column leach tests are summarized in Table XI. Here the leach time includes the initial acidic preleach (12 to 13 days) and periodic sulphuric acid washes.

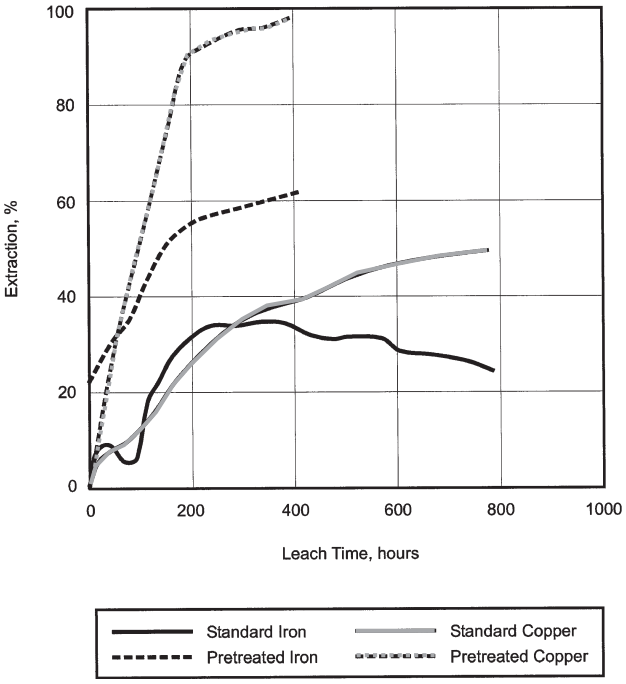

Figure 2—Typical leach curves for cleaner concentrate in the BacTech programme with thermophylic bacteria at 45°C

Table IX

Mineralogical characteristics of the three QFP-K alt ores used by METCON in their parametric testwork programme

High-grade—Host rock dominated by quartz, plagioclase, muscovite, and epidote, with minor microcline and apatite. Chalcopyrite is the dominant reflective phase, with grains typically 0.10 mm in size. No other copper minerals seen. Finer grained chalcopyrite found predominantly along cleavage lines of micas or chlorites; coarser grained, interstitial to quartz and feldspars. Pyrite less abundant, with little association between sulphides.
Mid-grade (AD5 Split)—Host rock similar to above but with more biotite and microcline. Obvious replacement of feldspars. Both chalcopyrite and pyrite seen, with pyrite:chalcopyrite ratio closer to 1:1. No other sulphides noted. Chalcopyrite distributed as above. Pyrite accounts for relatively minor portion of total iron in sample. Same is true for high-grade.
Low-grade—Gangue is dominated by quartz and muscovite/sericite, with less plagioclase and microcline; biotite seldom seen. Some calcite as feldspar replacement. Lower sulphide content than higher grade ores. Sulphide grain size only about 0.05 mm. Smaller grains predominantly lodged on silicate grain boundaries.

Table X

Final metallurgical balances in BacTech testwork programme

Conditions	Test duration h	Cu release %	Fe release %	S Oxidation %	Acid consumed kg/t
Bulk, Pretr'd	330	94.7	13.9	99.0	184
Bulk, Std	786	59.4	35.0	63.1	37
Cleaner, Pretr'd	402	98.4	60.3	93.8	405
Cleaner, Std	786	48.3	41.3	51.5	25

Table XI

Copper extraction as a function of ore particle size—Wits programme

Column no.	Ore size, mm	Test duration[1], d	Overall Cu recovery[2], % of Cu in ore	Cu in precip- % of Cu recovered
1	1.4 x 2	321	60.65	25.76
2 (low pH)	2.8 x 4	324	37.33	4.20
3	2.8 x 4	327	26.75	4.09
4	6 x 8	332	21.05	4.80
5	14 x 19	328	13.58	1.13
6 (bulk ore)	19 (top size)	268	13.92	n/a

1. Time includes acid preleach step and periodic acid washes to dissolve precipitates.
2. Final figure based on all analytical data; includes Cu recovered from jarosite.

The latter were found necessary to recover copper that was leached from the ore, then precipitated in the jarosite-type compounds that formed within the columns. The quantity of precipitate decreased as the particle size increased, with anywhere from 1.13 to 25.76 per cent of the leached copper being recovered from the jarosite. This copper loss proved to be unique to the Wits programme; neither MINTEK nor METCON could find any significant amount of copper in precipitated iron salts removed from their leach columns.

A detailed discussion of the modelling work done by the Wits team is beyond the scope of this paper. For this, the interested reader is referred to Tomicki[2]. In doing the modelling work, efforts to apply equations describing either purely chemical or diffusional control did not provide good fits with the Haib data. Therefore, another leaching model was selected. This was originally derived by Letowski[3] and takes into account the effects that faster decomposition of secondary (gangue) ore components has on the slower leaching refractory chalcopyrite. The basic form of the Letowski equation is as follows:

$$1-2/3(\alpha/\beta)-[1-(\alpha/\beta)]^{2/3}+2/3(\alpha/\beta)[(1-(\alpha/\beta)^{1/3}-1]In[1-\beta/(A-1)]=kt \quad [2]$$

Here α is the fraction of copper leached; β is the fraction of secondary components decomposed; A is a parameter dependent solely on the ore grade, g_{Cu}, with $A=(1-\sqrt{g_{Cu}})^{-1/3}$; k is the rate constant for the leaching process; and t is the leach time. For the AD5 ore split used at Wits, g_{Cu} was 0.24 per cent, with A=1.163.

Table XII

Values of leaching rate constant k and projection of leach extraction as a function of time and particle size

Column no.	Particle size, mm	Rate constant (k)	Fraction of Cu extraction (α)		
			1 Year	2 Year	3 Year
1	1.4 x 2.0	7.269 x 10^{-4}	0.41	0.69	0.79
2	2.8 x 4.0	6.208 x 10^{-4}	0.34	0.62	0.75
3	2.8 x 4.0	5.106 x 10^{-4}	0.26	0.55	0.70
4	6.0 x 8.0	3.739 x 10^{-4}	0.16	0.33	0.51
5	14 x 19	5.952 x 10^{-4}	0.15	0.31	0.50

The first three terms on the left-hand side of Equation [2] are related to transport through a porous bed or porous fraction of the ore fragment to the unreacted copper sulphide surface. The remaining left-hand terms are related to the development of the copper sulphide surface accessible for leaching. According to the model, the leaching rate of the sulphide minerals [α=f(t)] is determined by the leaching rate of the secondary components [β=f(t)].

When solving Equation [2], α was determined as the slope of the leach curves where per cent extraction was plotted against time under leach. The term β was determined from the initial and final mass ratios for iron, aluminium and magnesium. Once values of α and β were available, they were used to estimate the value of k for each column or the equivalent particle size. These values of k, together with extrapolated values of β, were then used to predict long-term copper extraction for each range of particle sizes tested. The resulting three-year projections are shown in Table XII. The modelling can also be extended to estimate values of α for other particle sizes that were not tested.

As a check, the model was used to predict column extractions at the end of the test period. The disagreement was greatest for the finest particle size, about 11 per cent. The others were within 5 per cent. In all cases of disagreement, the model understated the extraction, suggesting that it provided a conservative estimate of copper recovery.

METCON programme

Amenability studies

The results of the preliminary bottle roll tests are presented in Table XIII. As shown, the presence of chloride ion was marginally effective in boosting copper extraction. However, the iron level had essentially no impact on extraction. Acid consumption was also slightly higher in the presence of chloride. The average acid consumption was about 25 kg/t for all four columns.

The leach curves for the four columns are shown in Figure 3. The time line includes a 15-day cure cycle before leaching began on the two acid-cured columns. Additional comparative data are included in Tables XIV and XV. The former shows results for all four columns after 175 days, at which point the acid-cured columns were fully rinsed and drained. The latter table compares the results of the two uncured columns after 282 days of leaching, when the mixed sulphate-chloride column was taken off-line. The extended leach cycle included 100 days during which the column temperature had been increased to 45°C.

Table XIII

Copper extraction and acid consumption in the METCON bottle roll tests

Test no.	Actual lixiviant, g/L				Cu extraction, %		Total acid consumption, kg/t
	H_2SO_4	HCl	$Fe_2(SO_4)_3$	$FeCl_3$	24 h	120 h[1]	
1	12.1	9.0	–	3.0	3.5	3.9	28.0
2	12.0	8.9	–	10.0	3.4	4.2	25.5
3	21.2	–	3.0	–	2.6	3.3	25.1
4	21.5	–	10.0	–	2.4	3.1	21.3

1. Includes copper in wash/rinse water.

Table XIV

Copper extraction and acid consumption after 175 days of leaching in the METCON amenability tests

Test no.	Lixiviant	Acid cure dosage, kg/t	Cu extraction, %	Acid consumption	
				kg/t ore	kg/kg Cu
CL-01	H_2SO_4 + $Fe_2(SO_4)_3$	10.58	10.66	22.5	86.2
CL-02	As above	–	10.45	18.4	63.3
CL-03	H_2SO_4 + HCl + FeCl3	10.47	7.58	15.9	84.5
CL-04	As above	–	5.69	6.4	41.3

Table XV

Copper extraction and acid consumption after 282 days of leaching in the METCON amenability tests

Test no.	Lixiviant	Cu extraction, %	Acid consumption	
			kg/t ore	kg/kg Cu
CL-02	H_2SO_4 + $Fe_2(SO_4)_3$	21.04	34.5	59.1
CL-04	H_2SO_4 + HCl + $FeCl_3$	16.80	9.0	19.7

The column leach results show that for both the sulphate-based and mixed sulphate-chloride lixiviants, the acid cure accelerated the initial release of copper. This was a short-lived phenomenon, however. After about ten days, the leach rate curves for the respective cured and uncured columns stopped diverging and began tracking each other and gradually converging. Thus, the cure did not affect the longer term release rate for copper. The initial addition of the cure acid did increase long term acid consumption, both on a kg per tonne of ore basis and a kg per kg of recovered copper basis.

Based on the parallel programme under way at MINTEK, the pH of the feed solution to the columns was reduced to 1.5 after 120 days of operation (105 days of leaching for the cured columns). In addition to increasing the level of acid consumption by the gangue rock, the iron levels in solution rose progressively from 2.0 to 5.5 g/L in the sulphate-only columns and from 0.5 to 3.0 g/L in the mixed chloride-

sulphate columns. However, as shown in Figure 3, there were no significant changes in the copper leach rates that accompanied the decrease in pH.

An unexpected result was the slower release of copper in the presence of chloride. Just the opposite effect had been noted in the bottle roll tests, but Figure 3 and Tables XIV and XV clearly show slower copper recovery with the mixed lixiviant. This was accompanied by lower acid consumption, as well. Further discussion of these results is provided in the section headed 'Leaching characteristics of the Haib ores'.

An interesting observation was that bacterial activity developed in all four columns. Since the columns were not intentionally inoculated and were isolated from each other, the bacterial activity appears to have arisen from micro-organisms originally present in the ore itself. Based on most probable number (MPN) cell counts, bacterial activity developed within one month in the sulphate-only system. At that point the MPN value was 10^5, increasing to nearly 10^7 after another month. However, no bacterial activity was detected in the chloride-bearing columns for the first 60 to 75 days of leaching. Once established, bacterial activity was never as great, with MPN values running between 10^3 and 10^4. These observations correlate nicely with Emf values. In the sulphate-only columns, readings rose from about 300 to 650 mV within the first 30 days. For the chloride-bearing columns, the Emf values remained below 500 mV until leaching had been under way for 60 to 70 days. See Figure 4.

The increase in temperature clearly boosted the leach rate, but did not favour one lixiviant over the other. The observed temperature effect established this parameter as one to be studied in more detail in the follow-on parametric programme.

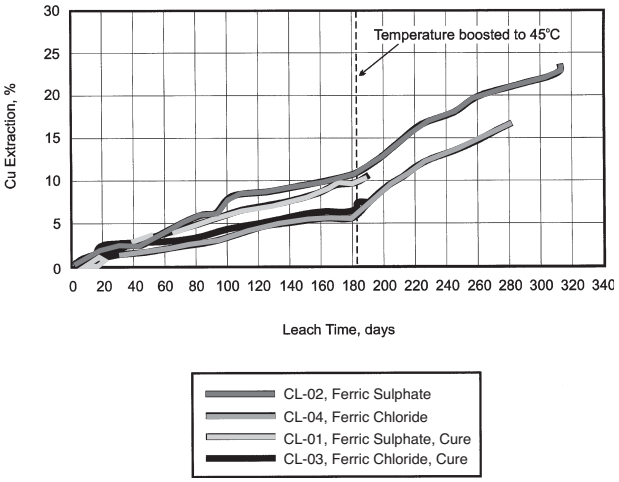

Figure 3—Leach curves obtained in the METCON amenability programme

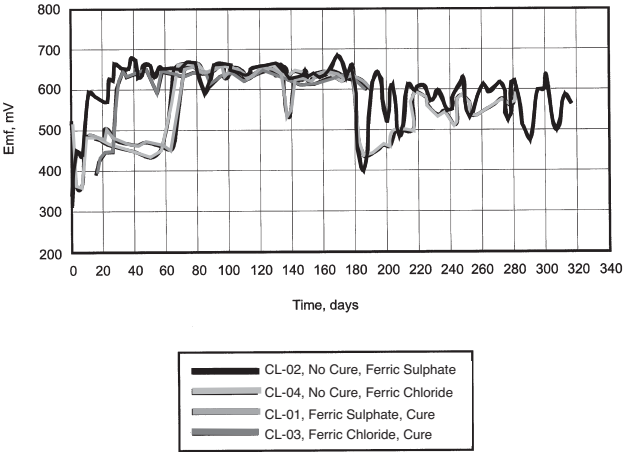

Figure 4— Variation in the Emf values of the pregnant leach solution as a function of time in the METCON amenability programme

Parametric studies

The variables studied in the METCON parametric programme were ore grade, leach temperature, crush size, and ferric ion content of the feed solution. The ferric effects were tested only on the mid-grade and high-grade ores. As shown in Figure 5, the ferric level in the feed (0.5, 2.0 or 5.0 g/L) had no discernible impact on copper recovery. Therefore, this portion of the parametric programme was terminated after about 110 days. In effect, other parameters such as the nature of the ore and the temperature controlled the iron levels in solution.

Temperature proved to be the most significant variable in the METCON programme. Increasing the temperature from 30 to 45 to 60°C doubled or even tripled the rate of copper extraction, depending on the ore grade. This is shown in Figure 6 for the high-grade ore and in Table XVI, which is a summary of the results from the parametric programme. Higher temperatures undoubtedly boosted both the diffusion and chemical reaction rates. The tendency toward passivation of the chalcopyrite, as observed in the BacTech work and at low temperature by MINTEK (see the next section), was apparently reduced at higher temperatures.

For a given set of conditions, raising the temperature accelerated the gangue-acid reaction and increased acid consumption when expressed on the basis of kg of acid per tonne of ore. However, temperature had an even greater effect on the copper leach kinetics. Thus, when expressed as kg of acid per kg of copper leached, acid consumption dropped as the temperature rose. These observations held for all three ore grades.

Table XVI

Summary of results from the METCON parametric study on -12.7 mm ore

Material	Copper recovery, %					
	30°C		45°C		60°C	
	142-day	267-day	142-day	267-day	142-day	267-day
High-grade	10.2	12.7	20.5	23.9	37.2	45.4
Mid-grade	9.0	11.6	13.9	19.4	23.4	33.4
Low-grade	6.3	9.9	9.7	12.3	15.5	22.6
Material	**Acid consumption[1]**					
	30°C		45°C		60°C	
	kg/t ore	kg/kg Cu	kg/t ore	kg/kg Cu	kg/t ore	kg/kg Cu
High-grade	42.9	41.7	43.4	22.5	60.2	16.4
Mid-grade	63.3	195.8	65.1	113.4	74.1	79.6
Low-grade	58.1	386.5	58.1	342.6	69.6	202.5

1. After 267 days of leaching.

Crush size had an impact on extraction, with the extraction rate increasing as the average particle size decreased. However, the effect was greatest on the low-and mid-grade ores. The crush size was relatively unimportant for the high grade material. These effects are shown in Figures 7 and 8. The explanation may be that the low- and mid-grade ores had relatively fine, disseminated mineralization so that finer crushing helped expose more copper to the lixiviant. By contrast the high-grade ore had relatively coarse mineralization that occurred in more accessible locations such as fracture fills or grain boundaries. Thus, the lixiviant could still penetrate to the copper sulphides as the particle size increased.

The QFP-K alt ores did exhibit a strong grade-recovery relationship. As indicated in Table XVI, for any given set of conditions, the extraction rate increased as the grade increased. There was also a synergistic effect with temperature, as the spread between the ore grades increased

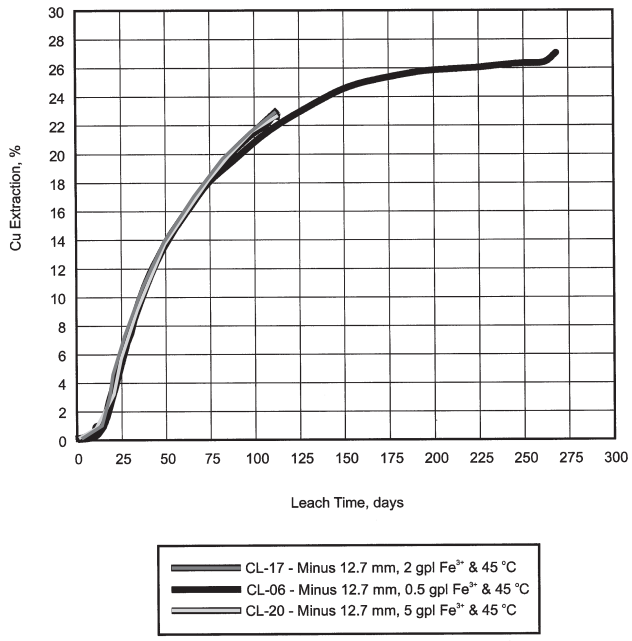

Figure 5—Leach curves for the high-grade ore showing the lack of impact that ferric concentration has on the extraction of copper in the METCON parametric programme

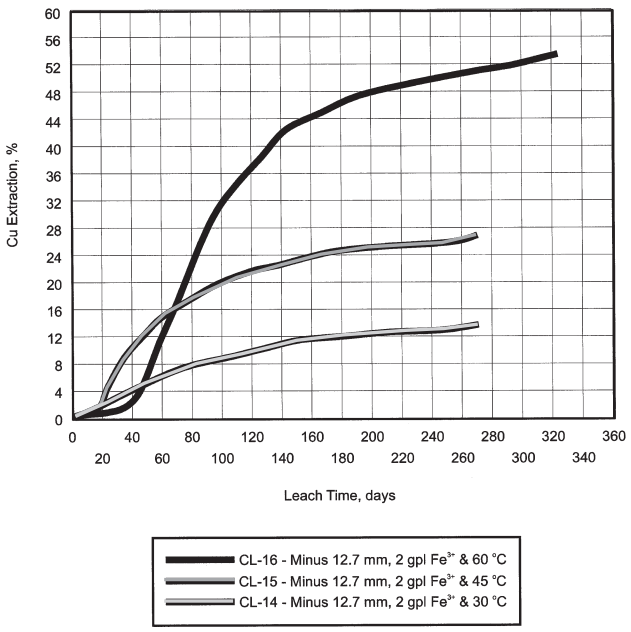

Figure 6—Impact of temperature on the copper leach rate for the high-grade ore in the METCON parametric programme

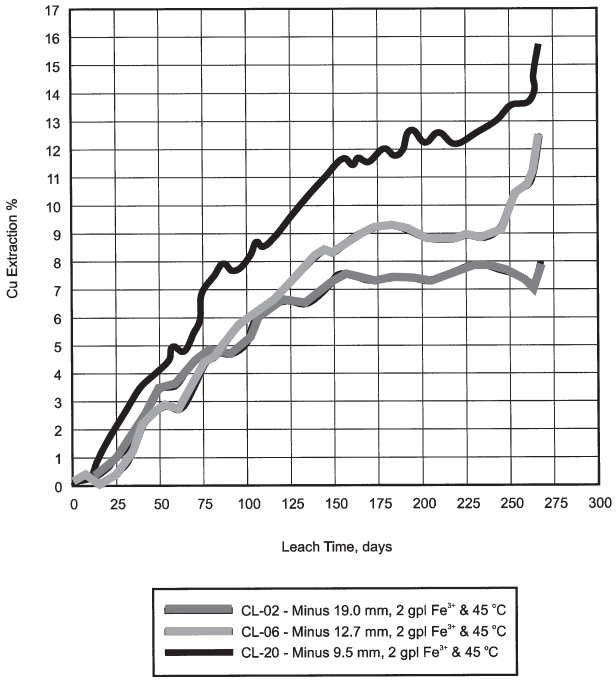

Figure 7—Impact of crush size on copper extraction for low-grade ore in the METCON parametric programme

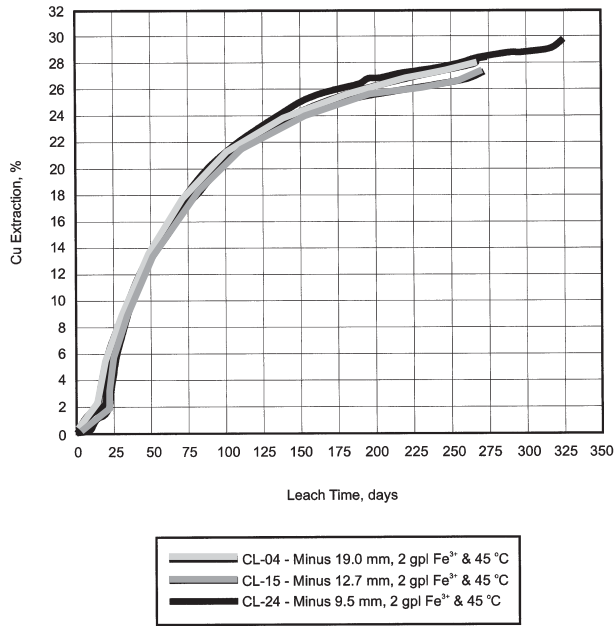

Figure 8— Impact of crush size on copper extraction for high-grade ore in the METCON parametric programme

Table XVII

Summary of results from MINTEK's amenability tests on Haib concentrate

Test description	Test duration, d	Extraction, %[1]		
		Cu	Fe	S^{2-}
Sterile, unmilled, 35°C	3	8.1	12.7	2.7
Sterile, unmilled, 70°C	3	94.5	73.6	72.8
Sterile, remilled, 70°C	3	99.1	82.0	80.1
Bacterial, unmilled, 35°C	37	22.5	42.2	45.5
Bacterial, unmilled, 35°C	37	24.8	41.5	47.1

1. Includes copper and iron removed in acid preleach treatment

Table XVIII

Summary of results from MINTEK's mapping study on Haib concentrate

Test no.	Test duration, h	Temperature, °C	Emf, mv	Cu extraction, %
1	72	45	430	5.2
2	72	45	500	5.1
3	72	45	570	7.0
4	72	45	680	8.0
5	72	60	430	8.3
6	72	60	500	11.7
7	50	60	570	11.3
8	50	60	680	16.5
9	50	70	430	10.6
10	50	70	500	15.0
11	50	70	570	20.4
12	50	70	680	24.9

as the temperature increased. Since it was the percentage extraction that increased, the implication is that the copper mineralization became more accessible as the grade increased.

Ore grade also appeared to impact bacterial activity. Just as it did in the amenability studies, bacterial activity developed naturally in each column within the first 30 days of leaching. For the low- and mid-grade ores, MPNs were typically 10^4, with a few readings as high as 10^6 or 10^7. For the high-grade ore, which is higher in both iron and sulphide sulphur, the MPNs were typically 10^5, with some values above 10^9. These high values were essentially independent of temperature, and occurred at both 30 and 60°C.

The maximum extraction achieved in the parametric programme involved minus 9.5 mm material leached for 357 days at 60°C. At this point, the extraction from the high-grade ore was 46.2 per cent; from the mid-grade, 42.3 per cent; and from the low-grade, 38.3 per cent.

MINTEK programme

Study on concentrate

Results of the concentrate leach tests are summarized in Table XVII. The low recovery in the sterile test at 35°C (~ 8 per cent) demonstrated the inherently refractory nature of the Haib chalcopyrite. The much higher recoveries at 70°C (94.5-99.1 per cent) showed that complete leaching is possible, given aggressive enough conditions. Based on subsequent studies, magnetic stirring in these tests was found to contribute to the high extraction by providing enough attrition to remove any film forming on the surfaces of the chalcopyrite grains.

Bacterial activity was easily established following inoculation of the reactors. This was evidenced by a rapid increase in the Emf levels to about 700 mV and a pickup in oxygen uptake to 2 kg/m^3/day. Based on the downward drift in pH during the test, pyrite was clearly being oxidized according to the reaction

$$FeS_2 + 3.5O_2 + H_2O \rightarrow FeSO_4 + H_2SO_4. \quad [3]$$

The fall in pH suggests that acid was being generated faster than it was being consumed. Based on examination of the residues and the system mass balance, pyrite oxidized in preference to chalcopyrite. Iron in the chalcopyrite also oxidized in preference to the copper. Bacterial leaching of copper did occur, as evidenced by extraction of some 20 per cent of the copper. However, after 34 days at 35°C, the reaction essentially stopped. This was attributed to passivation of the chalcopyrite, as ample material remained available for leaching.

Mapping study

The mapping study involved measurement of copper extraction over a range of temperatures and Emf levels. The results are summarized in Table XVIII and show that extraction improved continuously as the temperature and Emf levels were raised. Thus, the best extraction (24.9 per cent) was achieved in 50 hours at 680 mV and 70°C. This result was not entirely expected, as some chalcopyrite samples show good extraction over a narrow range of lower Emf values, a so-called 'window of leachability'.

Column studies

The MINTEK column studies included both amenability tests and parametric tests (copper extraction as a function of pH). These were run concurrently, with the results summarized in Table XIX. Leach curves (plots of copper extraction vs leach time) are presented in Figure 9.

The results at 30 and 60°C re-emphasize the importance of temperature, as observed in the METCON programme. At 60°C the copper extraction level reached 32.6 per cent. This was the only column test where recovery exceeded the level at which MINTEK observed passivation of the chalcopyrite under ambient conditions. Even with leach times approaching one year, extraction in the other columns was

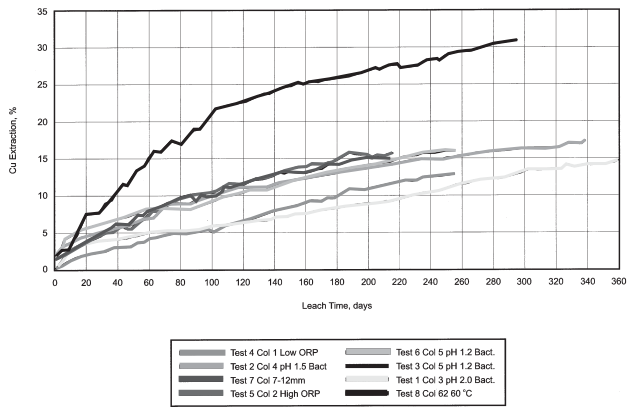

Figure 9—Summary leach curves for the MINTEK column tests

Table XIX

Summary of results from MINTEK's column leach tests

Parameter	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7	Test 8
pH level	2.0/1.7	1.5	1.0	1.5	1.5	1.2	1.5	1.5
Top size, mm	19	19	19	19	19	19	12	19
Temperature, °C	30	30	30	30	30	30	30	60
Emf, mV	High	High	High	Low	High	High	High	High
Other	Bacteria	Bacteria	Bacteria	Sterile	Sterile	Bacteria	Sterile	Sterile
Test days	358	338	46	254	216	253	209	294
Acid Consd, kg/t	28.1	56.7	68.7	55.0	52.2	91.9	80.5	88.1
Cu extn, %	15.0	19.0	7.1	12.8	16.7	18.5	19.2	32.6
Fe extn, %	16.6	31.3	23.8	30.3	16.6	33.4	31.1	34.4
Mg extn, %	19.9	25.7	44.7	35.4	35.5	50.4	47.2	40.8

less than 20 per cent under all conditions at 30°C.

The pH of the feed solution to the columns had a lesser effect on copper extraction than temperature. Decreasing the level from pH 2.0 to 1.5 increased copper extraction by about 25 per cent, but doubled the level of acid consumption. Dropping the pH level further (to 1.2) did not improve copper recovery at all, but did nearly double acid consumption again. For the AD5 ore, leaching at a pH of about 1.7 would probably provide the optimum balance between copper recovery and acid consumption.

By the end of the tests, the ongoing rate of acid consumption had dropped to a low level. This suggests that gangue acid requirements were largely satisfied, with further acid consumption probably attributable to the reaction

$$2Fe^{2+}+2H^{1+}+O2 \rightarrow 2Fe^{3+}+H_2O \quad [4]$$

Acid consumption per tonne of ore did increase with temperature, matching the results of the extensive parametric tests performed at METCON.

The column results also confirmed that leaching at a low Emf level is detrimental to copper recovery from Haib ore. The higher Emf levels associated with bacterial leaching are preferable to lower values.

Final extraction levels for both magnesium and iron are reported in Table XIX. Magnesium extraction tracked total acid consumption closely and offers a good indication of gangue rock reactivity. Iron extraction is more difficult to interpret. In most columns the iron content of the leach solution had reached a steady state by the end of the tests. Thus, as further dissolution occurred, there was an offsetting precipitation of iron salts. Phase identification techniques indicated that the precipitate was potassium jarosite, thus providing a sink for both iron and sulphate. This is consistent with the high potassium level in the AD5 ore.

Table XX

Results of leach tests on AD5 concentrate in chlorine-sparged lixiviant at high Emf levels

Time, min	Test in 1 M HCl			Test in Naci-HCl mixture		
	pH	Cu extn %	Fe extn %	pH	Cu extn %	Fe extn %
0	0.34	0	0	1.40	0	0
1+5	0.44	75.2	59.1	0.72	65.2	48.0
30	0.28	97.6	80.6	0.62	96.7	75.7
60	0.12	98.6	84.4	0.47	101.6	84.9
90	0.14	101.6	87.6	0.45	99.5	85.1
120	0.10	101.6	89.2	0.40	104.1	89.9
Final		99.5	91.6		100.1	89.7

Study at elevated Emf levels

The initial batch tests were conducted on AD5 cleaner concentrate and were intended to demonstrate that chlorine is an effective oxidizing agent for the Haib chalcopyrite. Results for the two lixiviants, 1 M HCl and 1 M acidified NaCl, are compared in Table XX.

In both tests the chlorine-sparged lixiviants produced very high Emf levels, with values between 1060 and 1170 mV. This was sufficient to oxidize more than 95 per cent of the chalcopyrite within 30 minutes and essentially 100 per cent of it within one hour. Iron extraction paralleled the copper, but reached a plateau at slightly less than 90 per cent. This implies that about ten per cent of the iron was present as insoluble gangue species.

In both tests the acidity increased continuously as the runs continued, with the 1 M HCl producing somewhat more acid. This was indicated by the lower pH in this test. Due to the aggressive conditions, about half the sample was consumed. The mass of the solids decreased from 13 g to 6.5 g at the end of both tests. In all the chlorine-sparged tests, sulphide was oxidized and acid generated according to the reaction.

$$MeS_2+7Cl_2+8H_2O \rightarrow Me^{2+}+2(SO_4)^{2-}+16H^{1+}+14Cl^{1-} \quad [5]$$

Here MeS_2 is a divalent base metal sulphide, either pyrite (FeS_2) or chalcopyrite ($CuFeS_2$). The final form of sulphur in this reaction could be either elemental sulphur (S^0) or sulphate $(SO_4)^{2-}$. The latter is favoured by high Emf levels. With experimental values of 1100 mV, or higher, complete oxidation to sulphate would be almost certain. This is consistent with the sulphate content of the final leach liquor.

Since the pH fell continuously in all tests, the chlorine dissolution reaction generated more acid than was needed to meet the gangue acid requirements. By contrast, external acid additions were required to maintain a solution pH of 2, or less, when leaching with acidic ferric sulphate lixiviants.

Following completion of the tests on concentrate, column tests were conducted on the minus 12 and 19 mm AD5 ore samples. The initial series of tests ran for 40 and 36 hours, respectively. The leach curve for the minus 12 mm sample is shown in Figure 9 and 36-hour results from both columns

Table XXI

Results of the column leach tests on AD5 ore in chlorine-sparged lixiviant at high Emf levels

Ore top size, mm	Duration, h	pH level		Extraction, %	
		Initial	Final	Cu	Fe
-12	36	1.98	1.39	8.1	1.4
-19	36	1.63	1.08	7.0	1.1
-19	97	1.48	0.71	11.1	3.9

are compared in Table XXI. This table also includes the results obtained by continuing the test on minus 19 mm ore for a total of 97 hours.

Figure 10 shows that the most rapid increase in copper extraction and the most rapid decline in pH occurred during the first four hours of the test. This is probably a consequence of the oxidation of sulphide mineralization exposed on the surfaces of the ore fragments. During the remainder of the test copper extraction increased linearly with time, while the pH declined linearly. The test on the minus 19 mm ore displayed the same characteristics. Due to the very acidic conditions, there was significant gangue dissolution. The final 97-hour leach liquor contained the following: 3210 ppm Ca, 516 ppm Al, 405 ppm Mg, 225 ppm K, and 3800 ppm $(SO_4)^{2-}$.

Mineralogical studies were conducted on the whole leach residue after the 97-hour leach. The results showed that only the altered feldspar and associated sulphide minerals were significantly degraded. Exposed sulphide mineralization was also attacked. However, chalcopyrite inclusions in the quartz and layered silicates were largely unattacked. After 97 hours, the chlorinated lixiviant had penetrated about 150 μm into the bulk sample, exclusive of cracks and other openings.

Overall, the Haib concentrate leached at least two orders of magnitude faster in the chlorinated system than it did in any of the tests in acidified ferric sulphate. Likewise, the crushed ore in columns leached some 30 times faster with the chlorine-sparged lixiviant than with any of the acidified ferric sulphate solutions. However, copper extraction from the crushed ore was always far slower than from concentrate leached under the same conditions. Thus, when the leach results and mineralogical findings are considered, the extraction rate for leaching whole ore appears to be controlled by the accessibility of the chalcopyrite to the lixiviant. Dissolution of the gangue minerals to expose more mineralization contributes to the leachability of the chalcopyrite. While important, the kinetics of mineral dissolution are not rate controlling for the Haib ore.

Discussion and analysis

Leaching characteristics of the Haib ores

The leaching characteristics of the potassium-altered quartz feldspar porphyry ore from the Haib deposit have been well characterized by an extensive multi-laboratory testwork programme. This is particularly true of the mid-grade material, which would be a likely target of any commercial heap or dump leaching operations. As discussed in the following section, higher grade material would most probably be processed conventionally to produce a flotation concentrate.

Like most chalcopyrite ores, the AD5 composite must be classed as refractory. In part, this is due to the nature of the Haib chalcopyrite mineralization itself. There are also two practical considerations that would impact a commercial operation. One is the lack of any significant supergene or oxide mineralization, i.e. virtually all copper is present as $CuFeS_2$. The other forms are inherently more leachable and, if present, often contribute much of the copper that is leached rapidly.

The second consideration has to do with pyrite. Unlike most low-grade porphyry copper ores, the AD5 material is low in pyrite*. The volumetric ratio is 1:4, FeS_2:$CuFeS_2$. Normally the ratio is reversed and the author has seen instances where it exceeds 10:1.

Pyrite generally plays a beneficial role in heap leaching. As it oxidizes, pyrite provides *in situ* sources of both iron and acid, two components essential for leaching copper sulphides. When oxidized to the ferric state, iron provides the active lixiviant. The acid is required to maintain a pH regime where the ferric iron and copper remain soluble. When pyrite is deficient, the iron and acid must be supplied from other sources, including those external to the leach heap or dump. Based on the author's observations, pyrite also seems to enhance copper extraction when it is in intimate contact with chalcopyrite. This may be due to galvanic effects. However, the mineralogical reports indicate that such associations are rare in the AD5 material.

Mineralogical examination of the fresh ore did show that the chalcopyrite may be found in three distinct settings—disseminated in feldspar, the layered silicates, or quartz. Examination of the leached residues showed that solution readily penetrated the feldspar and leached the contained copper minerals. The other settings, especially the quartz, were harder to penetrate and less of the copper contained in these areas was leached.

Various tests on the Haib chalcopyrite (in the form of flotation concentrate) showed the mineral itself may be difficult to leach completely, unless conditions are right. MINTEK observed that the mineral became passivated after 20 to 25 per cent extraction when leached bacterially at 30°C. Similarly, BacTech found that even on reground concentrate, passivation occurred at 50 to 60 per cent extraction at 45°C. There was a lack of passivation at 60 to 70°C, with BacTech's pretreatment, or at high enough Emf levels.

In spite of potential problems with the leachability of the chalcopyrite, the modelling studies by Wits and the MINTEK studies with chlorine-sparged lixiviant showed that the leach rate of whole ore is controlled by accessibility of the leach solution to the copper sulphides. The Wits study is particularly important because it demonstrated that decomposition of gangue constituents enhances the extraction of copper from the chalcopyrite by making the latter more accessible to the lixiviant.

Clearly temperature and ore grade are the two key parameters that control mineral accessibility. The test results suggest that temperature has a dual role in increasing

**The ratio for the other sulphide ore types and other areas of the deposit are not known. They may prove to be more typical of porphyry deposits.*

extraction. The MINTEK mapping studies showed that for any given Emf value, the leach rate of chalcopyrite itself increased with temperature. On the other hand, the METCON parametric programme showed that for any given ore grade, the leach rate of whole ore increased with temperature. The higher temperature should increase diffusion rates, making more of the disseminated mineralization accessible over a given time interval.

Increasing temperature provided another benefit that had not been expected. This has to do with acid consumption. As indicated by both the METCON and MINTEK column studies, increasing the temperature does increase the rate of acid consumption by the gangue minerals. Thus, acid consumption per tonne of ore increases moderately with increasing temperature. However, copper extraction increases faster. As a result, acid consumption per unit of copper recovered drops by roughly 50 per cent as the temperature rises from 30 to 60°C. See Table XVI.

The METCON testwork demonstrated a strong grade-recovery relationship for the QFP-K alt ore that also appears related to chalcopyrite accessibility. Based on mineralogical examination, the sulphide grain size increased and the degree of dissemination decreased as the ore grade increased. Particularly for the high-grade ore, much of the chalcopyrite was found on cleavage lines and interstitial locations between the quartz and feldspar. Such locations should be more accessible to the leach solution, so the increased level of extraction at higher ore grades is not surprising.

The less disseminated nature of the high-grade ore also makes it less sensitive to crush size than the mid- and low-grade ores. For these, finer crushing exposes more mineralization on the surface of the ore particles. In addition, the finer particles will have shorter diffusion paths to the remaining disseminated chalcopyrite and may have more micro-fractures and other pathways for solution access to the interior of the ore fragments.

There are several potential downsides to the finer crushing. One is that the incremental cost of the added crushing will exceed the value of the additional copper production from the finer ore. While an exhaustive cost study has not been made, results in Figure 7 show that extraction from the low-grade ore doubled when the top size decreased from 19 to 9.5 mm. Thus, tertiary crushing seems justified by the higher recovery.

Another potential problem with fine crushing is the possibility that heap permeability may become restricted. This could result from either excessive fines generated during crushing or from the subsequent acid-induced decrepitation of the ore fragments during an extended leach cycle. Since the Haib ore is quite competent as mined, fines generated during crushing do not appear to be a problem. This supposition seems to be confirmed by the multi-phase test programme. There were no percolation problems reported for any of the column studies, some of which lasted nearly a year. In addition, there was limited particle size degradation during leaching, especially at pH levels of 1.5 and 2.0. This is demonstrated by the data in Table XXII, which shows the change in particle size distribution during the MINTEK column tests at different pH levels.

The limited effect of the chloride ion on whole ore leaching is another phenomenon likely tied to mineral accessibility. Originally, chloride ion was added to the lixiviant due to the well known ability of ferric and cupric chloride to leach chalcopyrite more effectively than ferric sulphate. Indeed, chloride did provide some enhancement of the leach rate in the METCON bottle roll tests. However, here the ore charge was finely ground, so mineral accessibility was not a major factor.

In column tests on coarse whole ore, the presence of chlorides did not enhance the leach rate. In fact, it appeared to retard copper release. Since the diffusion rate, not the intrinsic reaction rate, controlled the kinetics of copper leaching, the presence of chloride provided no benefit. In fact, since acid consumption was lower in the chloride system, there was less decomposition of gangue constituents. Thus, following the Wits modelling, the acidic ferric sulphate leaches provided more second phase removal, giving better sulphide access and faster leaching.

Finally, the mapping study showed that leaching is most effective when conducted at the highest practical temperature and Emf level. This implies the use of a bacterial leach utilizing thermophylic strains. Such a system would provide the high temperature oxidation of ferric iron needed to maintain the maximum Emf level. Although three of the laboratories used external inoculation with their own in-house strains of bacteria, the work done by METCON showed that this would not be necessary in a commercial operation. Significant bacterial activity developed from indigenous strains under all conditions in the METCON tests. These covered temperatures from less than 30°C to as high as 60°C and included lixiviants that contained high chloride levels, as well as the typical acidic ferric sulphate solutions.

Options for commercial operation

In the overall Haib development programme, a substantial amount of time and effort was directed at defining the mill metallurgy of the ore. These results are not the subject of this paper, but showed that the various ore types responded very well to flotation, as long as the acid soluble copper content was less than ten per cent. The feasibility study showed that with an average head grade of 0.30 per cent copper and a

Table XXII

Change in particle size distribution while leaching at different pH levels

Rock size, mm	Head[1]	Size distribution, %			
		Test 1 pH = 2.0	Test 2 pH = 1.5	Test 6 pH = 1.2	Test 3[2] pH = 1.0
19 x 12	35.8	40.7	46.1	35.4	38.9
12 x 6	40.7	35.6	33.9	28.0	29.3
6 x 3	11.1	3.4	3.3	1.0	9.2
-3	12.4	20.3	16.7	35.6	22.6
Total	100.0	100.0	100.0	100.0	100.0

1. At present, neither the author nor MINTEK personnel have a explanation as to why the leach residues appear to have more 19 x 12 mm material than the head sample. Original log books indicate that the given distributions are correct. However, there seems to be a possibility that the weights were reversed on the two coarsest head fractions.

2. This was a short-term test lasting only 46 days. The others lasted 253 to 358 days.

mill cut-off of 0.22 per cent copper, flotation recovery averaged 90 per cent[4]. The most attractive route for subsequent processing of the concentrate was found to involve roasting, calcine leaching, and producing high quality copper cathode via solvent extraction (SX) and electrowinning (EW). A key factor, making this route the most attractive, was the added value of producing high quality cathode copper at the mine site. Production of low cost sulphuric acid from sulphur dioxide in the roaster off-gas was another benefit, as this is potentially saleable or could be used for leaching the calcine and ore heaps.

With this scenario, only sub-mill ore would be leached. Using the then-current mine plan* and a leach cut-off grade of 0.15 per cent copper, the leach material would run about 0.20 per cent copper and represent about 30 per cent of the ore mined. Thus, leach performance would follow results from the low-and mid-grade ores. Even though it was the best leaching material, the high grade ore would be reserved for mill feed.

Based on the results from the various testwork programmes, the optimum leach parameters from a technical point of view would start with fine (tertiary) crushing. This would produce ore sized to give a P_{100} of 15 mm and P_{80} of 9.5 mm. The pH of the feed solution to the heap (SX plant raffinate) would be 1.5 to 1.7. The Emf of the leach system would be about 650 mV, as dictated by good bacterial activity. A heap temperature of at least 60°C would be desired and acid consumption would run 30 to 60 kg/t (gross basis) as shown in Tables XVI and XIX.

Achieving the desired temperature and meeting the acid requirements are two key factors for engineering design. Typically, sulphide leach heaps operate above ambient temperature. This is due to the exothermic nature of the sulphide oxidation reactions, particularly pyrite. These reactions are driven by natural convective air flow through the heap[5]. For example, internal temperatures as high as 66°C have been measured on a million tonne test heap at the Bingham Canyon Mine in Utah, USA. Effluent air temperatures from this heap have typically run 30 to 35°C above the ambient air temperature[6].

A high pyrite:chalcopyrite ratio has contributed to the high temperatures noted at Bingham. The ratio varied from 1:1 to 4:1 for the ore types placed on the heap, with sulphide sulphur contents above 3 per cent for some of the material. By contrast, the Haib ore represented by the AD5 composite is low in sulphide sulphur, with a pyrite:chalcopyrite ratio of 1:4. Thus, *in situ* oxidation is not likely to be a big heat contributor and other sources will be needed in order to run the heap leach at 60°C.

Fortunately, there are at least two low-cost sources of heat available. One is waste heat from the roaster and acid plant. This can be transferred to the heaps by using the raffinate for roaster and gas cooling before returning this stream to the leach system. The second source of heat is the solar potential of the Namibian desert. With the long leach system pipe runs, there should be ample opportunity to use passive solar systems to heat the flows going to the heaps. Although detailed heat balances are beyond the scope of work undertaken so far, preliminary calculations indicate that a substantial amount of low cost heat would be available to the leach system.

The question of acid consumption is the second critical issue for a commercial leach operation. All of the laboratory programmes showed that the AD5 composite is a high acid consumer. The MINTEK parametric study on pH effects showed that long term acid consumption would run 30 to 60 kg/t. These are gross figures and do not take into account the various credits. Largest of these is the electrowinning credit of 1.54 units of acid per unit of copper plated.[7] This acid would come from the combined electrowinning operation via the raffinate streams from the RLE and heap leach facilities. Minor acid contributions would come from the electrolyte bleed and oxidation of any pyrite available in the heap leach ore.

The RLE portion of the electrowinning operation would be a major contributor of acid to the heap leach. This is due to the expected production split. Based on the Phase 2 Study,[4] the RLE route will produce over six times as much copper as the heap leach (85 800 tpa vs 13 775 tpa). Total EW acid output would be over 150 000 tpa.

In addition to these credits, there is a synergistic effect between consumption and the temperature. As noted above, acid consumption per unit of copper recovered falls dramatically when the leach temperature rises to 60°C. Acid not supplied from other sources would come from the by-product acid produced by treating the roaster off-gases. In reality, the cut-off grade for leach ore would be adjusted so that total acid demand by the ore balances with acid available from roasting and electrowinning.

Based on the information presented above, a preliminary flowsheet for a low-grade ore heap leach operation can be developed. Due to topographic considerations, the heap leach site will be located on a gently sloping area some kilometers from the more rugged area around the mine, mill, and roast-leach plant. For this reason, as well as major differences in

On-going revisions to the mine plan call for a more selective approach focusing on the various high-grade zones. This should raise the average mill-head grade and cut-off grade, and should also raise the leach grade marginally.

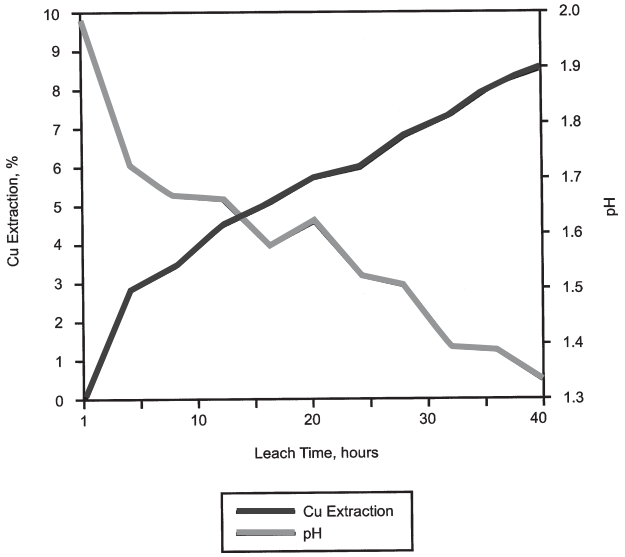

Figure 10—Plot of copper extraction and pH for minus 12 mm ore leached in chlorine-sparged lixiviant at high Emf

solution chemistry, the calcine leach (at 20 gpl Cu) and heap leach (at 1.6 gpl Cu) are expected to have separate satellite (SX) plants that exchange electrolyte with a common EW plant.

The fresh (make-up) water addition point for the combined leach operation will be the calcine leach area. Make-up water for the heap leach will be provided as raffinate transferred from the calcine leach SX plant. This will allow for maximum utilization of the high acid content in the calcine raffinate stream, as well as recovery of the residual copper. The latter would run 1 gpl Cu if there were 95% recovery in the calcine leach SX circuit. Finally, as noted above, the raffinate stream would serve as a medium for heat exchange between the roasters and the leach heaps.

The heap leach flowsheet is shown in Figure 11. It starts with 3-stage open circuit crushing, with both primary and fine ore stockpiles. The latter effectively de-couple the crushing operation from the upstream mine and the downstream acidulation and heap stacking operations. The acidulation step would provide a limited amount of acid. Based on test results, this acid would not have a significant effect on long term copper recovery. Instead, the intent would be to generate a sufficiently acidic initial heap outflow so that copper and iron would be carried in solution, rather than precipitated in the heap. The exact requirement would be based on further testwork.

The ore would discharge directly from acidulation drums onto a cross-country conveying system that feeds the radial stacker building the heaps. Once in place, the ore would be leached in at least two and perhaps three stages. Heated raffinate from the calcine leach-SX plant would be directed to the freshest ore. Outflow from this heap would report to one side of a divided PLS pond and produce the bulk of the pregnant solution for the heap SX operation. Raffinate from the heap SX plant would be acidified, if necessary, and advanced to the oldest heap areas via a passive solar heater. Outflow from this stage would drain to an intermediate leach solution (ILS) pond. This ILS would be advanced to a middle-aged heap area, again being acidified as appropriate and passing through a solar heater. Outflow from this heap area would flow to the opposite side of the PLS pond and be apportioned between recycling to the heaps and advancing to the SX plant. This type of heap recycling, sometimes referred to as solution stacking, is intended to maximize the copper tenor and minimize the volume of PLS being advanced to the SX plant. This minimizes the size and thus the cost of the SX facility.

Based on the mine plan considered during the Phase 2 Study[4], the heap leach operation would treat 14.5 Mtpa of ore grading 0.19 per cent copper. Using the final parametric test

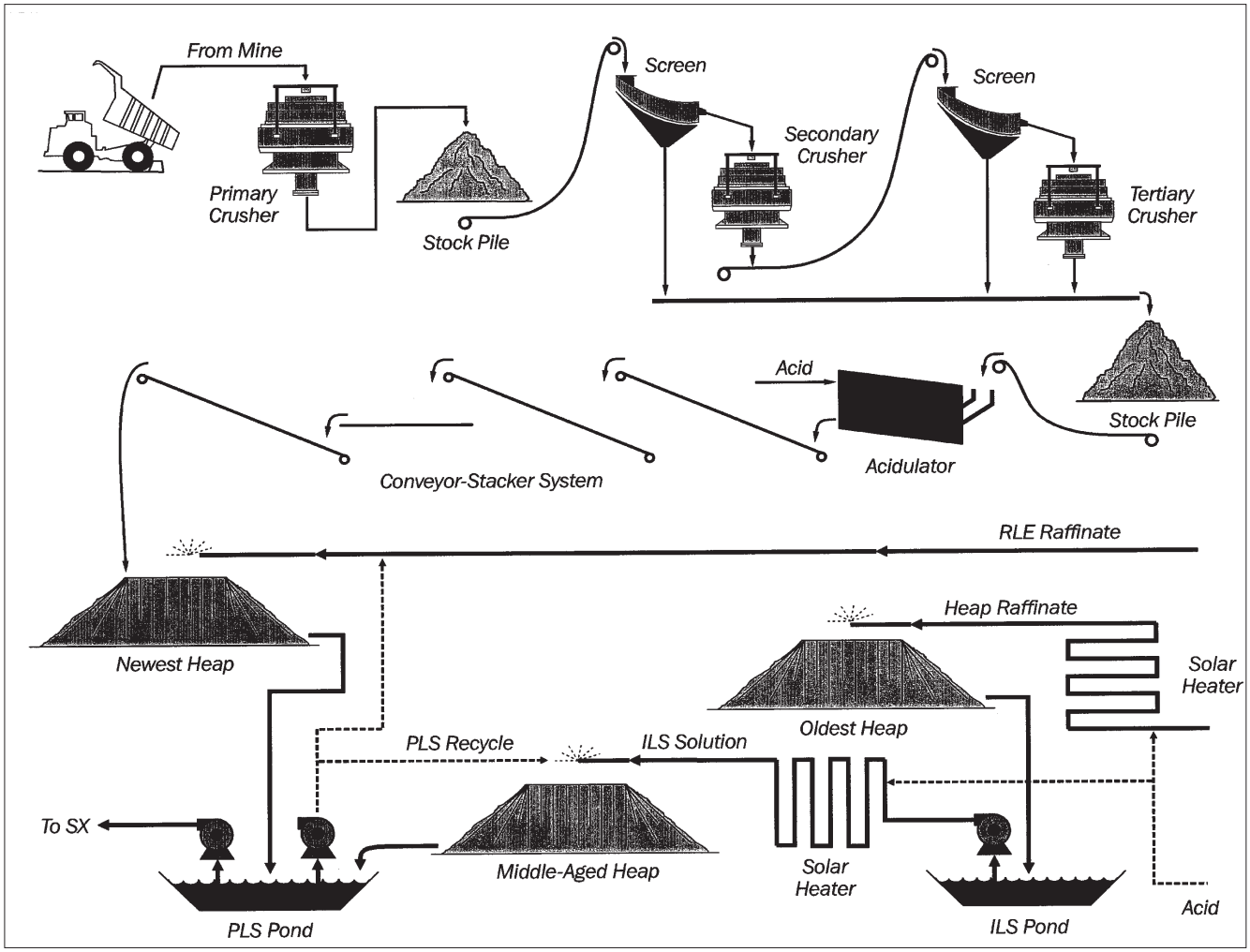

Figure 11—Schematic flowsheet for heap leach system showing solution recirculation

results, the ultimate level of copper recovery has been extrapolated to 50 per cent with a two to three year leach cycle. This assumes that a heap temperature of 60°C can be maintained. Under these conditions, steady-state cathode output would be 13 775 tpa. The Wits modelling work could not be used for the extrapolations since it is based on leaching at ambient conditions, not the 60°C level found to be optimum.

Plans for further work

Development of the Haib Copper Project clearly represents work in progress. Further hydrometallurgical testing is suspended at the moment, while attention is focused on better definition of the resource and a mine plan that takes advantage of the higher grade zones. The next phase of the hydrometallurgical work is expected to focus on two areas—(a) the oxide and transition ores, and (b) other sulphide ores.

The oxide and transition ores are defined as having greater than 50 per cent acid soluble copper and 10 to 50 per cent acid soluble copper, respectively. These represent surficial and near-surface material that will be mined to expose mill feed. Thus, there will be a need to treat these materials and develop an early cash flow from the operation. In addition, past testwork suggests that these ores will be more leachable than the primary chalcopyrite ore. The first priority will be to get a better definition of this part of the resource in terms of grade and tonnage. Then representative material will be selected and tested. This will involve monitoring copper recovery and acid consumption as functions of leach time, crush size, solution irrigation rates, and acid addition levels. A testwork programme would probably take about six months to complete.

Since oxide and supergene mineralization derive from remobilized copper, they should be more accessible (less disseminated) than primary mineralization. Thus, fine crushing may not be required.

Only the potassicly altered feldspar quartz porphyry ore type has been extensively tested. With this ore, both the feldspar and its sericitic alteration products are major acid consumers. Other sulphide ore types, particularly the quartz porphyries, may behave rather differently in terms of both copper recovery and acid consumption. Using optimum conditions from the present tests as starting points, representative samples of the other hypogene ore types will require testing to determine recovery levels and operating parameters. Such testwork could be started in parallel with the tests on the oxide and transition material. However, the programme is likely to run longer due to the refractory nature of the Haib chalcopyrite.

Conclusions

The overall conclusion is that Haib chalcopyrite ore, as represented by the potassium-altered quartz feldspar porphyry AD5 composite, is a refractory material. Long-term copper extraction from ore grading about 0.25 per cent copper should reach at least 50 per cent, but only after two to three years of leaching under optimum conditions. These include tertiary crushing and a bacterial leach at pH 1.5 to 1.7 and 60°C.

Other conclusions from the multi-laboratory testwork programme on whole ore leaching are as follows:

- The copper leach rate is controlled by the lixiviant's ability to access the disseminated chalcopyrite grains, and not by the rate of reaction at the sulphide grain surface.
- Mathematical modelling shows that dissolution of the gangue constituents is more rapid than the copper leach rate and, therefore, contributes to copper extraction by progressively making the sulphides more accessible to the lixiviant.
- Due to accessibility being the rate-controlling mechanism, the presence of chloride ion does not enhance the extraction of copper from chalcopyrite in a whole ore leach.
- Based on the three ore grades tested, there is a strong grade-recovery relationship, with extraction increasing as the grade increases, other parameters remaining constant.
- For a given set of conditions, copper extraction increases as the temperature and system redox potential increase, i.e. there is no 'window of leachability' at low Emf values.
- The optimum leach rate requires a temperature of at least 60°C.
- A conventional bacterial leach should be effective for the Haib ore. No bacterial inoculation seems necessary, as the indigenous bacteria were able to adapt to a wide range of temperatures (at least 30 to 60°C) and even to high chloride levels (>10 gpl). Bacterial activity increases with ore grade due to the increased amount of iron and sulphide sulphur present in the ore.
- During a typical bacterial leach, the pyrite is attacked in preference to the chalcopyrite and the iron is attacked in preference to the copper on the $CuFeS_2$ surface. Under certain conditions, especially lower temperatures, the chalcopyrite becomes passivated and leaching essentially stops, even though ample chalcopyrite remains available.
- Crush size is not as important as temperature, but for the lower grade ores, a finer crush size does correlate with higher leach recovery. For the high-grade ore, where the mineralization is not highly disseminated, crush size is not a significant variable. No percolation problems caused by ore fines were evident in any of the tests.
- The potassium-altered quartz feldspar porphyry ore is a high acid consumer.
- The optimum pH for a heap leach is 1.5 to 1.7, giving gross acid consumption levels of 30 to 60 kg/t. The copper extraction rate does not increase at lower pH levels, but acid consumption does.
- Acid consumption per tonne of ore increases as the temperature rises, but the copper leach rate increases even faster. Therefore, acid consumption per unit of copper leached falls by about 50 per cent as the temperature increases from 30 to 60°C.
- A low-grade heap leach on crushed ore fits well with a concentrate roast-leach operation on higher grade material. The roast-leach process produces low-cost

heat and acid that can be utilized to optimize recovery from the heaps.

More limited testing was done of Haib chalcopyrite concentrate produced from the AD5 ore composite. These tests showed that complete dissolution of the copper is possible given aggressive conditions such as high temperatures and high oxidation potentials. Gold recovery from the leach residue was effective (83 to 95 per cent) using a standard cyanide leach.

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Jaco–top Mintek bursar*

Jaco Blignault has been awarded a prize for being Mintek's top undergraduate bursar for 1998.

Jaco, who studied for a Chemical Engineering degree at Potchefstroom University for Higher Christian Education, achieved an average of over 80 per cent during his three years of study. ◆

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