

# Recovery of gold from ashed woodchips\*

by M.W. JOHNS† and I.D. MATTHEWS‡

## SYNOPSIS

This paper describes the process adopted for the recovery of gold from a dump originating from clean-up operations at Durban Roodepoort Deep Gold Mine. The gold-bearing dump material was refractory in nature, not being amenable to cyanidation because of the high level of carbonaceous material present.

The laboratory tests on the material involved particle-size and chemical analyses, a mineralogical examination of the material, preliminary tests, and optimization of the process for the treatment of the dump material. The preliminary tests included tests on roasting, flotation, pre-leaching, adsorbent-in-leach, and blinding. Of these options, carbon-in-leach showed the greatest potential. The rate of gold dissolution, the rate of gold adsorption by the carbonaceous material, and the ratio of carbon to pulp were then investigated. It was determined that the ratio of carbon to pulp necessary to yield a reasonable gold recovery resulted in a carbon loading that was not high enough for economic elution and regeneration. A process incorporating washing of the carbon with cold acid between adsorption cycles was tested, and reasonable gold recoveries and carbon loadings were obtained.

Durban Roodepoort Deep Gold Mine piloted the conceptualized process and determined that it was economically feasible. Two Browns tanks on site were modified, and an acid-wash tank was installed. The capital outlay for the plant was recovered within some 3 months.

## SAMEVATTING

Hierdie referaat beskryf die proses wat aanvaar is vir die herwinning van goud uit 'n mynhoop wat ontstaan het as gevolg van opruimingsbedrywighede by die Durban Roodepoort Deep-goudmyn. Die goudhoudende hoopmateriaal was ekstraksievas aangesien dit nie vir sianidisering vatbaar is nie vanweë die hoë vlak van koolstofhoudende materiaal daarin aanwesig.

Die laboratoriumtoetse op die materiaal het 'n partikelgrootte- en chemiese ontleding, 'n mineralogiese ondersoek van die materiaal, voorlopige toetse, en die optimering van die proses vir die behandeling van die hoopmateriaal behels. Die voorlopige toetse het toetse in verband met roosterling, flottasie, voorafloing, adsorbeeringsmiddels in die loog, en verstopping ingesluit. Van hierdie opsies was koolstof-in-loog die belowendste. Daarna is die tempo van goudopsorping, die tempo van goudadsorpsie deur die koolstofhoudende materiaal, en die verhouding van koolstof tot pulp ondersoek. Daar is vasgestel dat die verhouding van koolstof tot pulp wat nodig is om 'n redelike goudherwinning te gee, gelei het tot 'n koolstoflading wat nie hoog genoeg vir ekonomiese eluering en regenerasie is nie. 'n Proses wat die was van die koolstof met koue suur tussen adsorpsiesiklusse behels, is getoets, en redelike goudherwinnings en koolstofladings is verkry.

Die Durban Roodepoort Deep-goudmyn het die proses wat uitgedink is, aangevoer en vasgestel dat dit ekonomies uitvoerbaar is. Twee Brownstenks op die perseel is gewysig en daar is 'n suurwastenk geïnstalleer. Die kapitaaluitleg vir die aanleg is binne ongeveer 3 maande verhaal.

## Introduction

Before being dumped, all underground wood at Rand Mines' Durban Roodepoort Deep Gold Mine, be it logs picked off the belt or chips screened from the pulp in the milling circuit, is processed for the recovery of the gold it contains. The wood is burnt and the ash, after being milled, is treated by gravity concentration to recover the gold. Although this operation is economically favourable, a dump of some 18 kt has accumulated in which the gold content is some 7 to 13 g·t<sup>-1</sup>. This constitutes about 170 kg of gold. The material on the dump is refractory in nature and is not amenable to direct cyanidation because of the carbonaceous material that it contains.

Declining ore grades and rising costs made it necessary for Durban Deep to investigate every potential source of revenue, and Mintek undertook laboratory tests on a suitable method for extracting the gold. The object of the work was to find a process route (preferably using existing equipment) requiring a low capital outlay and moderate labour. The process would need to be relative-

ly simple and so provide a speedy return on expenditure, and would need to operate on an on/off basis so that the general production would not be affected.

## The Nature of the Material

### Size Distribution

A representative sample of the dump material was screened, and the different size fractions were assayed for gold and analysed for their carbon and sulphide contents. The results are listed in Table I.

The fineness of grind of the material was 62 per cent passing 75 μm. The fraction larger than 500 μm had a gold grade of 17 g·t<sup>-1</sup> and contained 3,7 per cent of the gold, and the fraction smaller than 150 μm contained 88 per cent of the gold. The total carbon content of all the fractions was high, but was highest in the fractions between 500 and 38 μm. Most of the sulphides were contained in the fraction smaller than 75 μm.

### Mineralogical Investigation

Mineralogical investigation of the sample revealed that the dominant mineral phase was quartz. Also present were lesser amounts (less than 5 per cent) of hematite, pyrite, tramp iron, elemental copper, wood fibre, rutile, pyrrhotite, chalcopyrite, and cobalt sulphides. The wood

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† Mintek, Private Bag X3015, Randburg, 2125 Transvaal.

‡ Durban Roodepoort Deep Gold Mine, Roodiepmy, Roodepoort, 1728 Transvaal.

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TABLE I  
ANALYSES OF DUMP MATERIAL

Size fraction m	Mass %	Chemical composition				Distribution %	
		Au g·t <sup>-1</sup>	S <sup>-2</sup> %	Total carbon %	Organic carbon %	Gold	Organic carbon
+ 600	2,9	17,4	<0,1	7,9	6,5	5,4	4,6
- 600 + 300	1,0	18,4	<0,1	—	—	1,9	—
- 300 + 150	5,9	6,1	<0,1	—	6,5	3,8	9,5
- 150 + 75	28,0	6,5	0,38	3,7	3,3	19,2	22,7
- 75 + 38	19,3	10,8	0,68	3,7	3,2	22,0	8,5
- 38	42,9	10,5	0,12	6,1	5,2	47,7	54,8
Calculated head		9,5	0,29	4,6	4,1		

fibre had been carbonized to varying degrees. Although the particle size of the organic carbon was usually smaller than 100  $\mu\text{m}$ , numerous large grains (up to 400  $\mu\text{m}$ ) were also present. The copper particles were surrounded by iron oxide, which varied in thickness from particle to particle. Some of the gold was occluded in the pyrite, as well as in occasional porous hematite grains resulting from the roasting of the sulphides. The texture of some of the iron oxides and iron silicates appeared to have formed during a melting-crystallization phase.

The refractoriness of the material was considered to be due to the 'lock-up' of the gold in silicates, pyrite, and iron oxide; to the presence of 'preg-robbing' carbonaceous material; and to chemical interference resulting from the dissolution of elemental copper and pyrrhotite.

#### Chemical Analysis

The chemical analysis of the material is given in Table II, which shows the carbon content to be high.

TABLE II  
CHEMICAL ANALYSIS

Constituent	Concentration*
SiO <sub>2</sub>	62,6
S <sup>-2</sup>	0,28
Au	14,1 g·t <sup>-1</sup>
Fe	6,5
Cu	0,11
C	5,1
CO <sub>3</sub>	2,8
Al	>1
Si	>1
B	0,1 to 1
Ca	0,1 to 1
Mg	0,1 to 1
Mn	0,1 to 1
Na	0,1 to 1
Pb	0,1 to 1
Ti	0,1 to 1
Zn	0,1 to 1

\* All values are given as percentages except where stated otherwise

#### Preliminary Tests

A number of options are available for the recovery of

gold from refractory gold-bearing material. The processes tested in the present investigation were cyanidation, roasting, flotation, pre-leaching, adsorbent-in-leach, and blinding. A preliminary series of 'scouting' tests was undertaken on which process would extract the gold efficiently and economically.

#### Direct Cyanidation

In the first test, the pulped dump material was contacted with lime and cyanide for 24 hours. Although direct cyanidation was not expected to yield a high gold recovery, the test would at least show what recovery could be expected from the conventional route. It was found that only 6 per cent of the gold dissolved from the ore, the cyanide consumption being 1,0 kg·t<sup>-1</sup> and the lime consumption 5,7 kg·t<sup>-1</sup>.

#### Roasting

Two samples of the material were roasted at 600°C for 1 hour in a rotary kiln, after which one of the roasted samples was milled for 10 minutes in a stainless-steel rod mill. Both samples were then pulped, and lime and cyanide were added. The pulp was leached for 21 hours at ambient temperature.

The roasting followed by cyanidation gave a gold dissolution of 43 per cent, while the roasting followed by grinding and cyanidation resulted in a gold dissolution of 8 per cent. The lime consumption was 3,4 kg·t<sup>-1</sup> and the cyanide consumption 0,3 kg·t<sup>-1</sup>.

It was hoped that roasting would render the gold that was locked up in pyrite accessible to the cyanide, and would ash the carbonaceous material, which would otherwise adsorb the aurocyanide ion from the solution phase ('preg-robbing'). However, although roasting and cyanidation resulted in a gold dissolution of 43 per cent as against the 6 per cent obtained with direct cyanidation, the gold dissolution was only 8 per cent when the sample was milled after roasting. These results indicate that roasting burns away the surface carbonaceous material but not the occluded carbon, which is then liberated during milling. Therefore, either roasting at a higher temperature or roasting after milling would be indicated but, since Durban Roodepoort Deep does not have a roaster, and since the capital and operating costs associated with roasting are high, it was decided that this was not a viable process option.

#### Adsorbent-in-leach

If material is amenable to direct cyanidation but carbonaceous material is present, the gold will be leached and will enter the solution phase as the aurocyanide ion, which will then be adsorbed onto the carbonaceous material. Hence, a gold assay of the material before and after cyanidation will indicate a low degree of gold dissolution when this might not be the case. If an adsorbent (activated carbon or resin) is added with the cyanide, the adsorbent will compete with the carbonaceous material for the dissolved gold. However, if sufficient adsorbent is added, most of the dissolved gold will be adsorbed onto the adsorbent, and the 'preg-robbing' nature of the carbonaceous material will be reduced to a minimum.

Two tests were undertaken to show the validity of this concept. The material was pulped, and lime and cyanide

were added. In the first test, activated carbon ( $40 \text{ g} \cdot \text{l}^{-1}$ ) was added whereas, in the second test, a strong-base resin ( $80 \text{ ml} \cdot \text{l}^{-1}$ ) was added. In both cases, contact was allowed to continue for 24 hours, after which the coarse adsorbent was screened from the pulp and assayed for gold.

In the test using activated carbon, 74 per cent of the gold was adsorbed and, in that using resin, 63 per cent of the gold was adsorbed. This indicates that the 'preg-robbing' ability of the carbonaceous material is one of the major causes of the refractoriness. This finding was not surprising, since both the mineralogical examination and the chemical analysis had indicated this possibility. The carbon-in-leach process route showed promise, and was investigated further. This work is presented later in the paper.

#### Pre-cyanidation Leaching

The presence of a coating of iron oxide on some of the gold particles meant that these particles would not be amenable to cyanidation, and pre-leaching was tested as a way of destroying this coating.

For the four acid pre-leachings (Tests B to E in Table III), the material was slurried with dilute acid to lower the pH value to below 1. After 30 minutes, the pH value had risen to 5, and more acid was added to maintain the pH value at between 1 and 2. The acid pre-leachings of 6 and 24 hours' duration were carried out at ambient temperature. For the two pre-leaches using chlorine gas (Tests F and G), the temperature was  $85^\circ\text{C}$ , and the chlorine gas was added by being bubbled through the pulp.

After each of the six pre-leaches, the pulp was filtered and the residue was washed to a pH value of 5. The residues were then repulped, and lime, cyanide, and activated carbon were added and contacted for 24 hours. Since another sample had to be obtained from the dump before this set of tests, the carbon-in-leach test (Test A) was repeated. The results of these seven tests are listed in Table III.

The gold recovery in the carbon-in-leach test was only 33,5 per cent, compared with 74 per cent for a previous sample. It was presumed that the second sample was not as representative of the dump material as the previous one, and this was confirmed during later testwork.

As Table III shows, pre-leaching with acid increased the gold recovery but, since the sample was not representative, it was not possible to quantify the exact extent of this increase. Because of the high acid consumption and the need for extra units at the plant if acid pre-leaching were to be introduced for the dump material, it was concluded that this would not be a viable process route.

TABLE III  
RESULTS OF ACID AND CHLORINE PRE-LEACH FOLLOWED BY  
CARBON-IN-LEACH

Test	A	B	C	D	E	F	G
<i>Pre-leach</i>		$\text{H}_2\text{SO}_4$		$\text{HCl}$		$\text{Cl}_2$ gas	
Type							
Duration, h		6	24	6	24	2	5
Acid consumption, $\text{kg} \cdot \text{t}^{-1}$		182	228	116	116	—	—
<i>Carbon-in-leach</i>							
Cyanide consumption, $\text{kg} \cdot \text{t}^{-1}$	1,4	1,6	1,6	1,4	1,4	1,6	1,6
Lime consumption, $\text{kg} \cdot \text{t}^{-1}$	7,8	13,9	13,9	13,9	13,9	14,0	13,9
Gold recovery, %	33,5	64,6	75,7	74,5	74,7	21,2	23,2

#### Flotation

Flotation of the sulphides to produce a concentrate was tested next: frother was added, the carbon was floated, the slurry was conditioned, and the sulphides were floated. The sulphide concentrates were cleaned in all the tests except Test 4.

In Test 1, the sulphide rougher concentrate was floated off and was cleaned after the addition of further reagents. In Test 2, the carbon was floated to visual extinction. In Test 3, all the reagents were added batchwise, and a different frother was used. In Test 4, the slurry was acidified to a pH value of 4 after the carbon had been floated off. The results, showing the grades and recoveries, appear in Table IV. Since the gold grade of the sulphide concentrate was highest in Test 1, that procedure would seem to be the optimum.

For flotation as the extraction route, there would be three possible streams:

- (1) sulphide cleaner concentration (recovery 49 per cent, gold grade  $212 \text{ g} \cdot \text{t}^{-1}$ ),
- (2) sulphide cleaner and rougher concentrate (recovery 61 per cent, gold grade  $61 \text{ g} \cdot \text{t}^{-1}$ ), and
- (3) sulphide cleaner, rougher, and carbon concentrate (recovery 72 per cent, gold grade  $42 \text{ g} \cdot \text{t}^{-1}$ ).

As the size of the dump does not warrant the installation of equipment to process the concentrate stream, the concentrate would have to be toll-refined, which would require a gold grade of  $500 \text{ g} \cdot \text{t}^{-1}$  unless the concentrate is high in sulphide. The grade criterion here excludes procedures (2) and (3). Since the sulphur grade of the concentrate from procedure (1) was only 6,4 per cent, it was

TABLE IV  
RESULTS OF FLOTATION TESTWORK

Fraction	Gold recovery, %				Gold grade, $\text{g} \cdot \text{t}^{-1}$				Mass recovery, %			
	1	2	3	4	1	2	3	4	1	2	3	4
Carbon conct	10,7	12,8	16,2	10,1	15,5	16,3	14,1	16,5	5,9	6,7	9,8	6,0
Sulphide ro tail	28,5	31,9	29,5	56,7	2,8	3,1	3,3	6,6	85,6	87,5	85,4	85,2
Sulphide cl tail	12,2	6,6	3,0	—	16,0	16,6	9,2	—	6,5	3,4	2,7	—
Sulphide conct	48,6	48,5	51,3	33,3	212,0	164,0	214,5	37,4	2,0	2,4	2,1	8,8

TABLE V  
CARBON-IN-LEACH TESTS ON DIFFERENT SIZE FRACTIONS

Item	Size fraction, mm				
	+0,5	-0,5 +0,25	-0,25 +0,075	-0,075 +0,038	-0,038
Mass distribution, %	0,6	6,9	34,4	23,2	34,9
Gold distribution, % by mass	1,0	8,5	32,1	31,8	26,5
Carbon concentration, %	8,75	6,0	3,9	3,6	6,4
<i>After CIL test</i>					
Gold in solution p.p.m.	0,023	0,023	0,012	0,016	0,008
Gold grade in ore, g·t <sup>-1</sup>	20,0	11,5	3,6	2,2	4,3
Gold loading on carbon, g·t <sup>-1</sup>	167	325	475	826	281
Gold adsorption, %	14,2	37,4	72,3	88,5	59,9
Gold distribution in residue, % by mass	2,9	19,1	29,9	12,2	35,8
Cyanide consumption, kg·t <sup>-1</sup>	3,3	1,8	1,4	1,0	1,6
Lime consumption, kg·t <sup>-1</sup>	15,9	13,7	12,9	12,7	15,8

concluded that flotation would not be a viable process route.

#### Blinding

The previous testwork had indicated that the main reason for the refractoriness of the material was the 'preg-robbing' carbonaceous material. Therefore, a possible method using the conventional cyanidation route would involve decreasing the adsorption ability of the carbonaceous material by blinding it with an organic material.

Two tests were undertaken in which a sample of the pulped material was contacted with paraffin in a flotation cell prior to cyanidation. As these tests yielded a gold dissolution of only 18 per cent, this simple route was not considered attractive.

#### Carbon-in-leach Tests

The preliminary tests had indicated that carbon-in-leach (CIL) was the most promising route by which the gold could be extracted, since not only did it give the highest recovery, but it is simple, cheap, and direct. For further quantification of the CIL route, tests were undertaken on the optimum size fraction, the rate of gold dissolution, the rate of gold adsorption onto the carbonaceous material and activated carbon, and the optimum carbon-to-pulp ratio.

#### Effect of Size Fraction

A sample was split into five size fractions and a CIL test was performed on each fraction. Lime was added to the pulp, which was aerated before the cyanide and carbon were added. After contacting for 24 hours, the gold contents of the carbon, solution, and ore were determined (Table V).

Although the fraction larger than 0,5 mm constituted only 0,6 per cent of the mass of the sample and contained only 1 per cent of the gold, it had the highest carbon content in the material and yielded the lowest gold recovery. The cyanide and lime consumption for this fraction was also the highest. In view of these aspects, as well as the problems that would be encountered in the handling of this fraction, it was decided that the fraction larger than 0,5 mm should be screened off and

dumped.

For all but the finest fraction, the gold adsorbed increased with increasing fineness. As can also be seen from Table V, the carbon content of these fractions followed a reverse trend, i.e. it decreased with increasing fineness. This observation provides further confirmation that the carbonaceous material is the prime cause of the refractoriness of the dump material. The gold grade of the residue, the lime consumption, and the cyanide consumption all followed the same trend as the carbon content of the fractions.

#### Rate of Gold Dissolution

To quantify the rate of gold dissolution from the ore, it was necessary to decrease to a minimum the adsorption of the gold by the carbonaceous material. This was done by the addition of excess activated carbon. The results of a dissolution test are shown in Fig. 1. Within 8 hours, 90 per cent of the soluble gold was leached, which indicates that a dissolution period of 24 hours would be sufficient for the leaching of the soluble gold. The rate expression used in a model to fit the observed results (as shown by the predicted line in Fig. 1) was based on second-order kinetics.

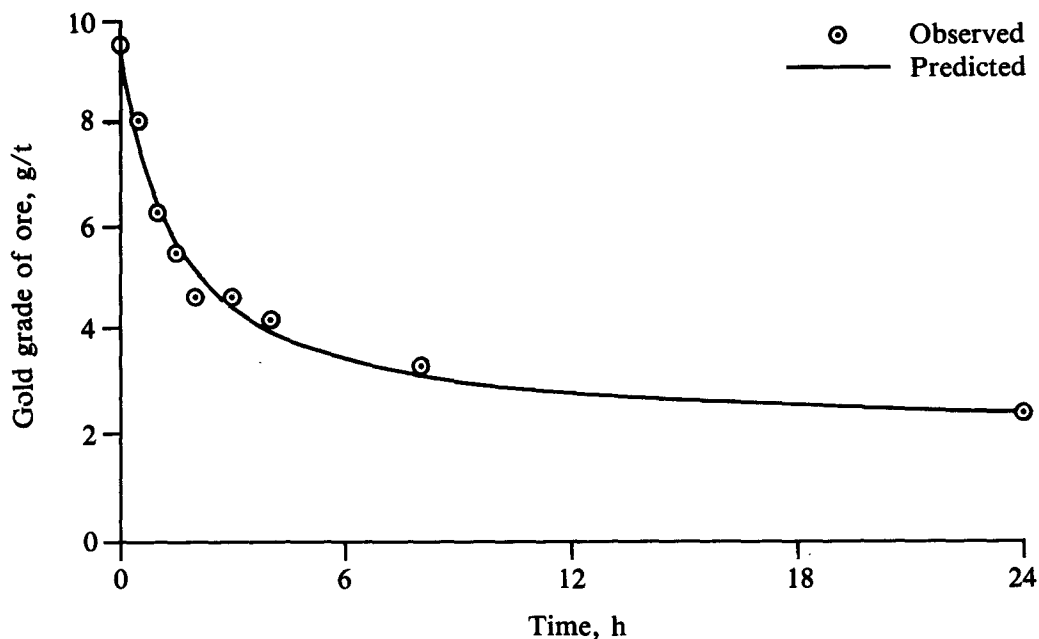
#### Rate of Gold Adsorption by Carbonaceous Material

To quantify the rate of gold adsorption by the carbonaceous material, it was necessary to decrease to a minimum the leaching of gold from the material. Since both cyanide and oxygen are required for the cyanidation of gold to take place, potassium aurocyanide (to ensure no free cyanide) was added to a pulped (relative density 1,45 kg·l<sup>-1</sup>) sample of the material to yield a gold concentration in solution of 15 p.p.m. Samples of the solution phase were taken at pre-determined times.

The results are plotted in Fig. 2, which shows that the carbonaceous material has the ability to adsorb most of the soluble gold. The rate expression used in a model to fit the observed results (as shown by the predicted line in Fig. 2) was based on film diffusion, and used the Freundlich isotherm to describe the equilibrium.

The tests on dissolution and carbonaceous adsorption provided proof that the gold can be cyanided and that

Fig. 1—Rate of gold dissolution from the material



○ Observed  
— Predicted

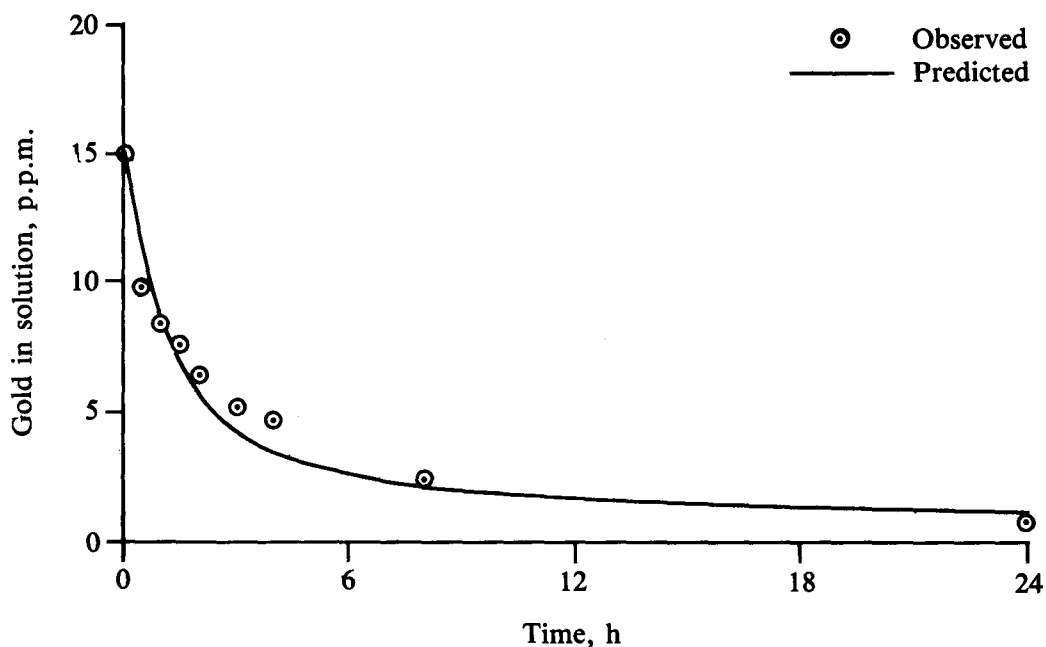


Fig. 2—Rate of gold adsorption onto carbonaceous material

○ Observed  
— Predicted

the carbonaceous material can adsorb the dissolved gold. It was therefore concluded that the carbonaceous material was the major cause of refractoriness in the dump material. The ability of activated carbon to adsorb gold is well known<sup>1</sup>, and hence no work was conducted on this aspect.

#### The Carbon-to-Solid Ratio

Once the reactions that take place in a CIL reactor (dissolution of gold from the ore, adsorption of gold by carbonaceous material, and adsorption of gold by activated carbon) had been ascertained, the combination of these three reactions, i.e. the CIL system, could be investigated.

Of the 9 CIL tests undertaken, 7 were on the effect of the carbon-to-solid ratio, and two on the effect of

water quality. In all the tests, lime was added to the pulp, which was aerated, and then the cyanide and carbon were added. The contact was continued for 24 hours, during which samples were taken at pre-determined times. All three phases were sampled and analysed for gold at the end of the contact (Table VI and Fig. 3).

If no activated carbon is added, the gold that is amenable to cyanidation flows from the ore into the solution, after which most of the gold flow is from the solution to the carbonaceous material. If activated carbon is added, the gold from the solution flows to both the carbonaceous material and the activated carbon. If sufficient activated carbon is added to the leach, the flow to the carbonaceous material is minimized, and the gold is in a recoverable form. The ratio of carbon to ore at which the 'preg-robbing' effect of the carbonaceous

TABLE VI  
RESULTS OF CIL TESTS AT DIFFERENT RATIOS OF CARBON TO SOLIDS  
Mass of ore 3 kg Volume of solution 3 litres Pulp density  $1,45 \text{ kg} \cdot \text{l}^{-1}$

Item	Test no.								
	1	2	3	4	5	6	7	8	9
Mass of carbon, g	60	90	120	180	240	300	360	120	120
Solution source	RWB	RWB	RWB	RWB	RWB	RWB	RWB	Mine	Mill
Head grade of ore, $\text{g} \cdot \text{t}^{-1}$	9,48	9,48	9,48	9,48	9,48	9,48	9,48	9,49	9,48
Ore grade at $t = 0,5 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	5,88	5,16	4,62	4,64	4,40	4,68	4,09	7,00	6,08
$t = 1,0 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	5,60	5,04	4,00	4,00	4,31	4,22	2,56	6,92	5,80
$t = 1,5 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	5,04	4,84	3,96	3,83	3,87	4,00	3,44	6,04	5,56
$t = 2,0 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	4,84	4,72	4,00	3,64	3,69	3,96	3,05	6,12	5,24
$t = 3,0 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	5,00	5,48	3,92	3,88	3,44	3,77	3,18	5,52	4,68
$t = 4,0 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	4,52	4,32	4,00	3,36	3,05	3,44	2,90	4,60	4,68
$t = 8,0 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	3,56	3,48	2,92	2,93	2,60	2,80	2,52	4,16	3,84
$t = 24 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	2,70	2,62	2,61	2,03	1,81	1,87	1,82	2,94	3,56
Carbon loading at $t = 24 \text{ h}$ , $\text{g} \cdot \text{t}^{-1}$	280	171	142	97	74	59	46	134	169
Solution concentration at $t = 24 \text{ h}$ , p.p.m.	0,15	0,023	0,22	0,19	0,17	0,17	0,17	0,23	0,12
Gold extraction*, %	68,0	66,3	69,3	74,4	77,1	76,4	75,7	65,5	65,8
Gold recovery†, %	66,2	66,0	66,8	72,4	75,0	74,3	73,5	62,8	64,7
Cyanide consumption, $\text{kg} \cdot \text{t}^{-1}$	1,11	1,21	1,86	1,31	1,34	1,54	2,01	1,32	1,26
Lime consumption, $\text{kg} \cdot \text{t}^{-1}$	13,2	13,3	12,5	12,3	12,1	11,6	11,5	13,2	13,2

RWB Rand Water Board, that is Witwatersrand tap water

\* Amount of gold leached from material

† Amount of gold loaded onto the carbon

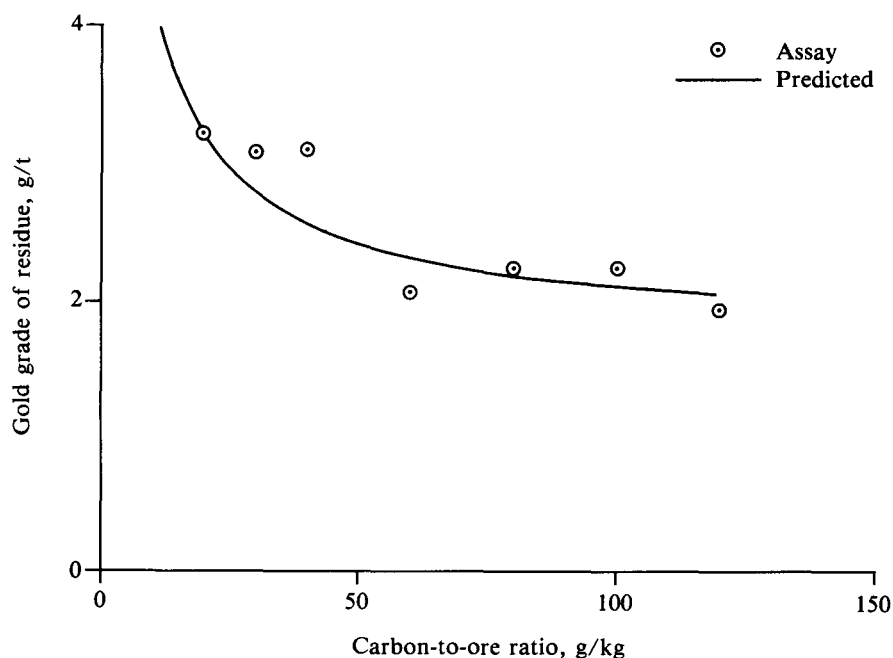


Fig. 3—Effect of the carbon-to-ore ratio on the gold grade of the residue

material is minimized is  $50 \text{ g} \cdot \text{kg}^{-1}$ . At that ratio, a carbon loading of some  $120 \text{ g} \cdot \text{t}^{-1}$  can be expected. In view of the costs of reagents, power, and labour, a gold loading as low as this would adversely affect the economics of the process owing to the high elution costs. The method by which higher loadings were obtained is outlined in the next section.

It was noted that, when tap water was used at a ratio of  $40 \text{ g} \cdot \text{kg}^{-1}$ , the gold in the residue was  $2,61 \text{ g} \cdot \text{t}^{-1}$ . With mine water it was  $2,94 \text{ g} \cdot \text{t}^{-1}$ , and with mill water  $3,57 \text{ g} \cdot \text{t}^{-1}$ . The increase of gold in the residue was prob-

ably due to the presence of calcium in the mine water ( $805 \text{ p.p.m.}$ ) and the mill water ( $975 \text{ p.p.m.}$ ).

#### Tests on the Recycling of Carbon

The tests showed that CIL, which is a simple method, would give a reasonable gold recovery. The optimum ratio of carbon to ore as determined from the batch tests was  $50 \text{ g} \cdot \text{kg}^{-1}$ . If the material were processed continuously through six stages, each with a nominal pulp residence time of 4 hours, the carbon concentration, calculated from the gold mass balance, would be  $218 \text{ g} \cdot \text{l}^{-1}$

for a carbon residence time of 1 day per stage. As this carbon concentration is high, it would be preferable to process the material on a batch basis, which would need a carbon concentration of  $37 \text{ g}\cdot\text{l}^{-1}$ . This would not be to the detriment of the process, which, according to the objectives, should be capable of operating on an on/off basis.

The gold loading on the carbon at the optimum ratio of carbon to ore was only  $120 \text{ g}\cdot\text{t}^{-1}$ . If carbon from one batch contact is used in the next batch contact, higher gold loadings should be obtained. However, since the loading of gold onto activated carbon is a reversible reaction and, since other elements will also accumulate on the carbon, the ability of the carbon to adsorb gold will decrease with each cycle.

A wash with cold hydrochloric acid was shown to remove most of the calcium on the loaded carbon, as well as lesser amounts of nickel, silica, magnesium, and aluminium. If it is true that the loading of gold onto carbon becomes irreversible after acid washing<sup>2</sup>, then reasonable recoveries and high gold loadings could be achieved if the carbon loaded from one batch of pulp were acid-washed and then re-used with subsequent batches.

Two independent parallel tests were performed in which 4 litres of pulp were contacted with carbon for  $21\frac{1}{2}$  hours. The carbon was then washed in cold hydrochloric acid for 2 hours before being water-washed and re-used with the next batch of pulp. Eight cycles were carried out. The results of the tests are listed in Table VII, and the gold recovery per cycle is plotted in Fig. 4 for both tests.

For eight cycles, the average gold recovery was 72,1 per cent, the average gold loading on the carbon being  $1020 \text{ g}\cdot\text{t}^{-1}$ . The degree of gold extraction decreased marginally as the carbon was recycled, probably for the following reasons.

- (1) After each cycle, a sample of the carbon was removed for assay purposes, causing the ratio of carbon to

TABLE VII  
RESULTS OF CARBON RECYCLING

Conditions

Lime added	$13,3 \text{ g}\cdot\text{kg}^{-1}$
Cyanide added	$2 \text{ g}\cdot\text{kg}^{-1}$
Carbon added	$50 \text{ g}\cdot\text{kg}^{-1}$
Acid wash	1 bed volume at a conen of $70 \text{ g}\cdot\text{l}^{-1}$

Cycle	Time h	Run A			Run B		
		Ore grade $\text{g}\cdot\text{t}^{-1}$	Gold loading $\text{g}\cdot\text{t}^{-1}$	Recovery %	Ore grade $\text{g}\cdot\text{t}^{-1}$	Gold loading $\text{g}\cdot\text{t}^{-1}$	Recovery %
1	0	8	0	—	9,88	0	—
	21,5	2,56	179	68,0	2,40	145	75,5
2	0	8,8	179	—	9,0	145	—
	21,5	2,22	267	74,8	2,30	264	74,4
3	0	8,8	267	—	10,6	264	—
	21,5	2,62	398	70,2	2,68	390	74,7
4	0	9,8	398	—	12,40	390	—
	21,5	2,7	483	72,4	2,86	513	76,9
5	0	8,41	483	—	9,0	513	—
	21,5	3,10	624	63,1	3,0	622	66,7
6	0	8,76	636	—	8,8	626	—
	21,5	3,10	759	64,6	1,62	788	81,5
7	0	8,88	759	—	8,2	788	—
	21,5	3,30	904	62,8	3,16	898	61,5
8	0	9,08	904	—	9	898	—
	21,5	2,74	1020	69,8	3,08	1020	65,7
		Average		68,2	Average		72,1

solids to decrease from  $50$  to  $41,5 \text{ g}\cdot\text{kg}^{-1}$  during the test. The model mentioned previously was used to predict the effect that this would have on the system, and it was found that the decrease in the carbon-to-solids ratio would result in a decrease in the gold recovery of 1,5 per cent.

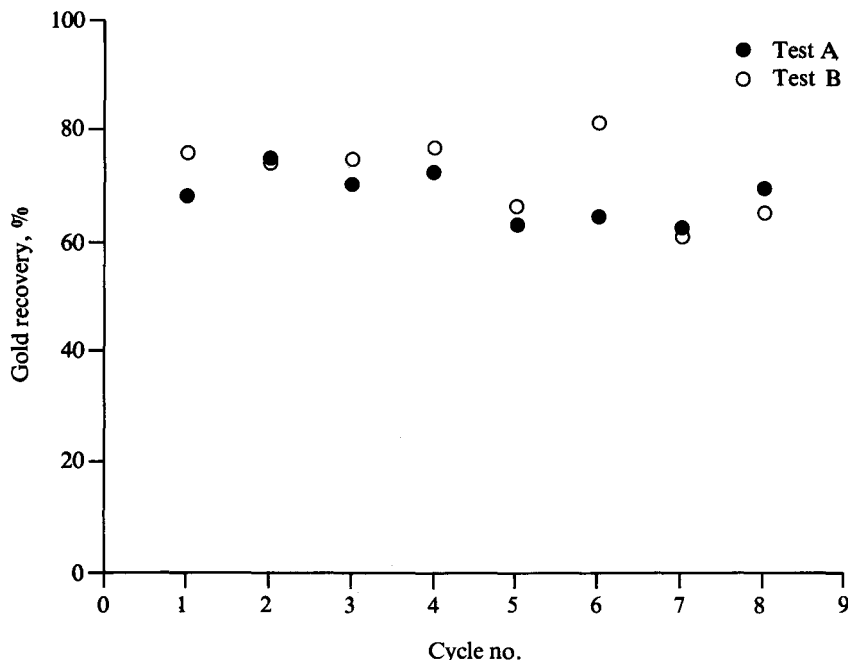


Fig. 4—Recovery of gold in the carbon-recycling tests

- (2) Although acid-washing removes calcium and is purported to render the gold loading on the carbon irreversible, other metals accumulate on the carbon, decreasing its activity.

An analysis of the loaded carbon after eight cycles (Table VIII) showed that base metals had been loaded but not to any significant degree. The silver loaded was more than the gold loaded.

TABLE VIII  
ANALYSIS OF THE LOADED CARBON ( $\text{g}\cdot\text{t}^{-1}$ )

Au	Ag	Zn	Ni	Co	Si	Fe	Ca	Cu	Pb
1020	1305	311	825	<50	2000	1780	617	776	214

The lime consumption was  $13,2\text{ g}\cdot\text{kg}^{-1}$  and the cyanide consumption  $1,35\text{ g}\cdot\text{kg}^{-1}$  of material treated, and the acid consumption per ton of carbon washed was 210 kg.

This testwork showed CIL to be a process that requires a small capital outlay, is low in labour requirements, can operate on an on/off basis, and is simple and potentially profitable.

### Design of the Plant

#### Design Considerations

In the design of a CIL plant to recover the gold in the dump, the capital and operating expenditures were to be minimal, the payback period was to be as short as possible, existing equipment was to be used where practicable, the treatment of the material was to start as soon as possible, and a batch-type process was to be used.

The resultant investigation into the implementation of a batch CIL process revealed that it would be possible to use existing redundant equipment, viz five  $350\text{ m}^3$  Browns tanks and two  $4,3\text{ m}^2$  vibrating screens. In addition, one of the existing cyanide storage tanks could be relocated, and the lime-slurry reticulation could be extended to serve the CIL process. The capital cost of dump reclamation (including the purchase of a front-end loader for use underground when the project had been completed) was approximately R290 000. The cost of an initial inventory of activated carbon, the installation of facilities for the storage of carbon and acid treatment, and all the ancillary equipment and residue-disposal facilities, was in the region of R380 000. The loaded carbon would be eluted and thermally regenerated at a plant operated by Rand Mines Milling & Mining Company, which would be capable of treating 2 kt of material per month. The payback period would be about 3 months, and the subsequent profit margin would be about R340 000 per month.

Early in 1987 it was decided to go ahead with the project, and work on the design commenced. So that capital expenditure could be kept to a minimum, the design was kept simple and practical and, where possible, metallurgical engineering and operating staff were used to construct and install the plant and equipment.

#### The Flowsheet

Fig. 5 shows the flowsheet of the process, incorporating reclamation, slurry pumping, screening, CIL, removal of

carbon, acid washing, and disposal of residue. The existing equipment and new installations are indicated, and the salient features of the process are as follows.

- Reclamation of the material involved the use of a loader and monitor guns to slurry the material into a penstock system.
- Material larger than 0,6 mm was removed by use of an existing 1,2 by 3,6 m vibrating screen.
- CIL was carried out in existing air-agitated Browns tanks.
- Carbon was removed and separated by use of a recessed impeller pump that delivers onto an existing 1,2 by 3,6 m vibrating screen.
- A storage tank was installed for concentrated hydrochloric acid, which fed a  $25\text{ m}^3$  acid-washing tank in closed circuit with a  $30\text{ m}^3$  carbon-storage tank.
- For carbon handling, use was made of  $1\text{ m}^3$  bulk bags and an electric hoist, by which carbon was added to the leaching tanks.

#### Description of the Operation

The dump was situated near the main plant in an area measuring 110 by 55 m and had a depth of 4 m. The most practical method by which the material could be reclaimed was by the use of a penstock system. A vertical excavation of about 4 m was connected into a drain feeding a sump just outside the main plant. A high-pressure centrifugal pump drew Rand Water Board (RWB) water from one of the old Browns tanks to supply hand-operated monitor guns. The loader placed material in heaps around the inlet to the penstock, and the material was monitored through a cylindrical screen of 25 mm-square mesh.

The slurry flowed via a drain to a vibrating screen (apertures 0,6 mm), which cut at approximately 0,5 mm. The underflow fed a sump, from which the slurry was pumped to the leaching tanks. The screen oversize was transported to the surface clean-up plant for further treatment by milling and gravity concentration.

Lime slurry from the existing circuit was discharged into the screen-underflow sump. However, when tested, it became evident that this was inadequate to maintain the correct protective alkalinity. A portable lime-storage silo (capacity 15 t) was located near the leaching tanks, bags of lime being filled manually and hoisted to the top of the leaching tanks.

Of the five existing Browns tanks, two could not be used because of corrosion. One tank had to be used for the storage of RWB water, leaving two tanks for leaching. The pulp was agitated by the injection of compressed air at about  $0,25\text{ m}^3$  of air per second. Each tank had a volume of  $350\text{ m}^3$  and was capable of leaching more than 250 t of solids per batch at a slurry density of  $1,46\text{ t}/\text{m}^3$ .

The required quantity of lime was added to a full charge of agitated pulp, and an aeration period of 2 to 4 hours was allowed before the cyanide and carbon were added. Liquid calcium cyanide was used for leaching, and was pumped from a  $15\text{ m}^3$  storage tank. Bulk bags of  $1\text{ m}^3$  were filled with acid-washed activated carbon, and hoisted to the top of the leaching tanks. A drawstring arrangement under the bags allowed the carbon to be discharged into the agitated pulp.



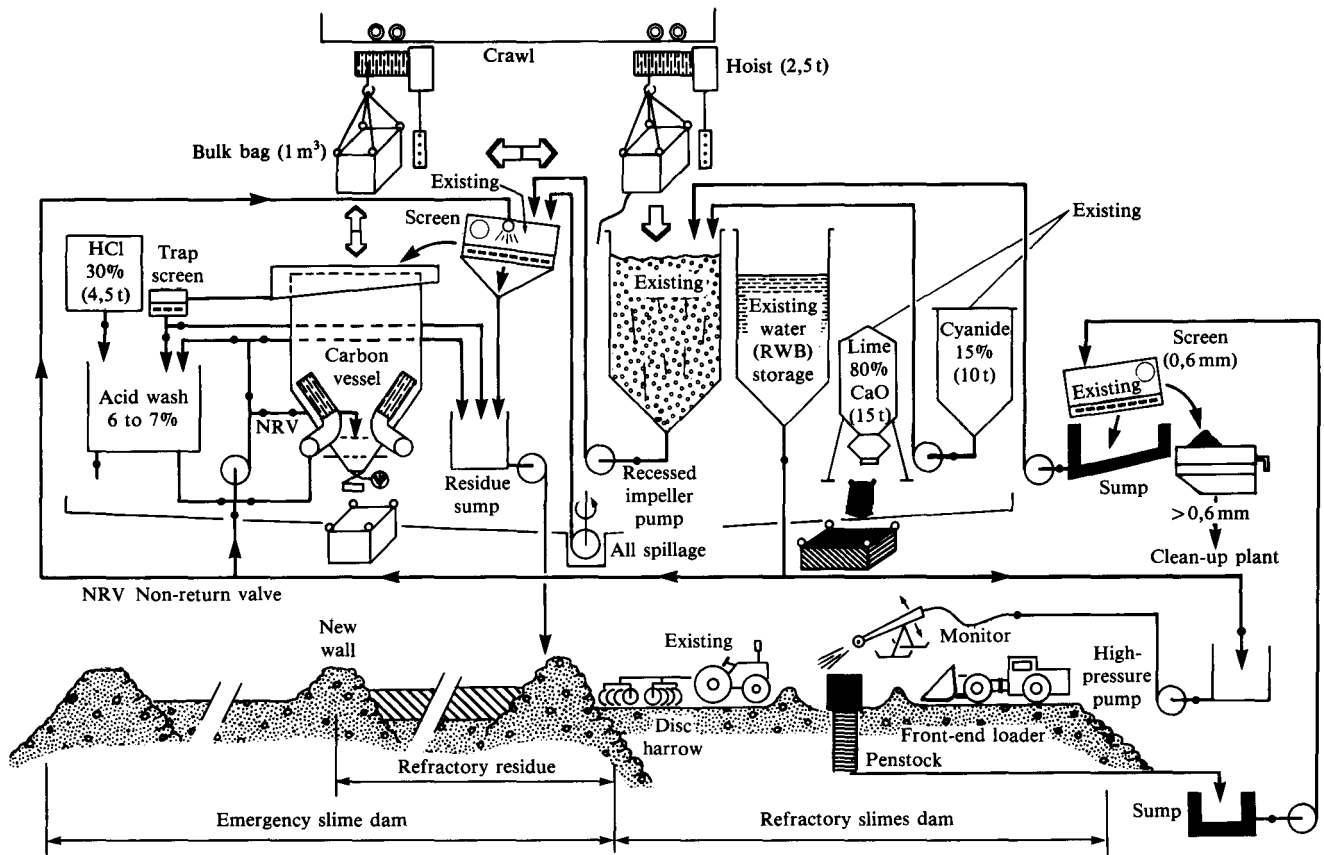


Fig. 5—The process flowsheet

Cyanide and carbon were added simultaneously, and the agitated pulp was leached for 22 hours. A carbon inventory of 10 t per batch was used, which is equivalent to a concentration of  $31 \text{ g} \cdot \text{l}^{-1}$ .

Discharging pulp was pumped via a recessed impeller pump over a vibrating screen identical to that used for the removal of coarse particles. The washed carbon discharged into a cylindro-conical rubber-lined storage tank of  $30 \text{ m}^3$  volume. Leached pulp from the screen underflow gravitated into a tank prior to the disposal of the residue.

Commercial hydrochloric acid was kept in a storage tank, RWB water from another storage tank being used to dilute the acid to 7 per cent. A 'multipurpose' hydro-seal pump circulated the acid solution up through the

carbon in the storage tank. The overflow flowed through a carbon-trap screen back into the acid-washing tank. Circulation continued until the strength had been reduced to between 2,8 and 3,0 per cent (4 hours). The solution was then drawn through 'candle' screens of 0,6 mm aperture in the storage tank. The carbon was then rinsed with RWB water until the pH of the solution was 5,0 to 5,5. The rinsing solution was discarded. A hand-operated butterfly valve allowed the carbon to be released into the bulk bags.

#### Operating Data

The plant was designed, installed, and commissioned during 1987, and production began in November of that year. The operating results are summarized in Table IX.

TABLE IX  
METALLURGICAL RESULTS

Item	Nov. 1987	Dec. 1987	Jan. 1988	Feb. 1988	Mar. 1988	Apr. 1988	May 1988	Jun. 1988
Amount treated, t	1172	1272	1343	1324	1584	1647	1672	1994
Pulp density, $\text{kg} \cdot \text{l}^{-1}$	1,26	1,27	1,28	1,26	1,32	1,33	1,34	1,33
Gold head grade, $\text{g} \cdot \text{t}^{-1}$	11,0	13,5	12,1	10,6	11,1	9,8	9,2	7,6
Gold in residue, $\text{g} \cdot \text{t}^{-1}$	2,9	3,3	3,2	3,9	2,9	3,2	3,8	3,0
Gold in filtrate, $\text{g} \cdot \text{t}^{-1}$	0,024	0,028	0,039	0,016	0,028	0,019	0,013	0,015
Gold recovery, %	74	76	71	63	74	68	59	60
Gold call, kg	9,5	13,0	11,5	8,9	13,0	11,0	9,3	9,1
Discharge CN, %	0,010	0,017	0,020	0,015	0,027	0,036	0,030	0,029
Discharge CaO, %	0,020	0,023	0,023	0,016	0,020	0,053	0,048	0,034
Gold loaded onto carbon, $\text{g} \cdot \text{t}^{-1}$	1250	1033	902	1529	1245	1026	1026	—
Gold on eluted carbon, $\text{g} \cdot \text{t}^{-1}$	65	49	357	43	84	126	75	—
Gold recovered, kg	8,7	14,2	5,8	16,3	15,9	13,2	11,4	12,8

The desired relative density of the pulp was  $1,46 \text{ kg} \cdot \text{l}^{-1}$ . However, owing to problems encountered on the dam, the actual relative density of the pulp was below this value, and the tonnage treated was initially well below the target level of 2 kt per month.

For the first 8 months of operation, the head value averaged  $10,6 \text{ g} \cdot \text{t}^{-1}$  and the residue value averaged  $3,3 \text{ g} \cdot \text{t}^{-1}$ , which indicates a gold recovery of 69 per cent. This figure compares very favourably with that obtained during the laboratory tests. The average loadings on the carbon were  $1144 \text{ g} \cdot \text{t}^{-1}$ . The consumption of cyanide was  $1,95 \text{ kg} \cdot \text{t}^{-1}$ , lime  $11 \text{ kg} \cdot \text{t}^{-1}$ , and hydrochloric acid  $215 \text{ kg} \cdot \text{t}^{-1}$ , all of which compare well with those determined in the laboratory.

#### Economic Considerations

The total capital outlay was R670 000, made up of R270 000 for the loader, R130 000 for the carbon, and R270 000 for the tank, pipes, pumps, etc. The operating

costs were R25 per ton, made up of R10 per ton for reagents, R5 per ton for labour, and R10 per ton for elution-regeneration. For this capital outlay, R3 040 000 was realized, which means that the payback period was 3 months. The operating revenue is R255 per ton, which converts to a profit of R420 000 per month.

#### Acknowledgement

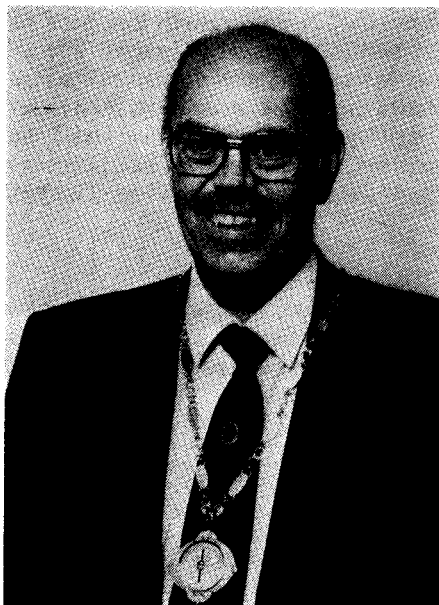
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## SA Corrosion Institute

Ivan Ogilvie has been elected President of the South African Corrosion Institute for a two-year term of office. Mr Ogilvie is Managing Director of the South African Copper Development Association, and has been associated with the copper-based non-ferrous industry for over twenty years.



## Import replacement\*

With the aid of the South African Bureau of Standards (SABS), another product that has been imported up to now at great cost will be replaced by a locally produced product. According to tests carried out by the SABS, the local product is an improvement on the imported product.

The product concerned is a special type of rivet that is used in great quantities during extensive servicing of aircraft by South African Airways.

Until recently these rivets had to be imported at exceedingly high cost from the original manufacturers of the aircraft. For this reason, it was decided to develop the same type of rivet for local manufacture.

During testing, the locally manufactured rivets were subjected to the most significant tensions that occur during flights. According to the test results, the locally developed rivets are superior to the imported product.

\* Released by the South African Bureau of Standards, Private Bag X191, Pretoria 0001.

## Call for papers

Readers are invited to submit papers for possible inclusion in this *Journal*. The papers, which should conform to the requirements outlined on page v, may deal with any aspect of mining, metallurgy, or mineral processing, but practical applications are preferred.

Please send your paper to Miss Caroline Jansen, the Production Secretary, at the following address:

The South African Institute of Mining and Metallurgy  
P.O. Box 61019  
Marshalltown  
2107 South Africa.