

# The intensive cyanidation of gold-plant gravity concentrates

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## SYNOPSIS

The development of a cyanidation route for the treatment of gold-plant gravity concentrates is described. This work was carried out as a natural consequence of earlier improvements to the recovery of gold in the gravity sections of existing gold plants. Concern about the environmental aspects of the conventional amalgamation process was further motivation in the development of an alternative procedure.

It was found that the partial pressure of oxygen in solution was the most critical reaction-rate determining factor. The rate of gold dissolution was shown to be controlled by the reaction rate below approximately 25°C and by diffusion above that temperature. Thus, the efficiency of pulp agitation and oxygen partial pressure permitted a high degree of control over the rate of gold dissolution. The use of oxygen rather than air significantly increased the dissolution rate of both gold and silver. The solubilities of sodium, calcium, and potassium aurocyanide complexes under simulated reaction conditions were studied, and it was established that the values for the sodium and calcium complexes were limited, requiring the use of lower pulp densities and higher reaction temperatures for the treatment of very high-grade concentrates. The presence of excessive amounts of tramp iron in such concentrates of up to 25 per cent by mass was found to result in poor gold dissolution owing to the cementation of gold under certain reaction conditions. The maintenance of high oxygen partial pressures, together with high cyanide concentrations, high pH levels, and the presence of calcium in the system, all aided in passivating the tramp iron. Copper, gold, and iron were shown to be the main cyanide-consuming elements.

Optimum gold dissolution was obtained in a closed bench-scale reactor incorporating a flotation-machine type of agitator for effective pulp agitation and oxygenation, the oxygen being recirculated. The reactor was designed to operate under a very small positive pressure. After reaction times of 2 to 6 hours at temperatures in the region of 30°C, both the gold and the silver dissolution averaged 99.5 per cent. In the case of very high-grade concentrates, it was shown that gold could be floated in the same reactor by drawing off the flotation concentrate through a port in the reactor. A flotation recovery of 87.4 per cent of the gold in 3.9 per cent by mass was obtained.

The recovery of osmiridium from cyanide-leach residues and the electrowinning of gold from the pregnant liquors were also considered as processing aspects in the treatment of gold-plant belt concentrates. Final table tailings with a gold concentration of 10 to 12 g/t were obtained.

The results of the investigation strongly suggested that an alternative cyanidation route for the treatment of gold-plant concentrates was technically feasible, provided the system included effective agitation and oxygenation. In the case of lower-grade concentrates, cyanidation followed by the electrowinning of gold appears to be the simplest route, while, for high-grade concentrates, flotation and smelting of the concentrate followed by cyanidation of the flotation tailings are suggested. The use of oxygen rather than air for sparging the pulp would depend largely on the grade of material to be treated. A preliminary comparison of the costs related to the amalgamation and cyanidation processes for the recovery of gold from such concentrates indicated very similar capital and operating costs. It was concluded that other considerations, e.g. toxicity, safety, and lower labour requirements, were more important than such small cost differences. Furthermore, the 'opening' of the gold-plant gravity circuit, which would be made possible by a cyanidation route, would further reduce the operating costs because of the subsequent decrease in the tonnage to be treated.

## SAMEVATTING

Die ontwikkeling van 'n sianiedproses, vir die behandeling van goudaanlegswaarkonsentraat, word beskryf. Die werk was 'n natuurlike voortvloeiing van vroeëre verbeteringe van die herwinning van goud in gravitasie-afdelings van bestaande goudaanleggings. Daarbenewens was besorgdheid oor omgewingsbesoedeling deur die konvensionele amalgameeringsproses verdere motivering vir die ontwikkeling van 'n alternatiewe prosedure.

Dit was bevind dat die parsiele druk van suurstof in oplossing die mees kritiese reaksietempobepalende faktor was. Onder ongeveer 25°C was die tempo waarteen goud opgelos het reaksietempobehoor, terwyl dit diffusiebehoor was bokant hierdie temperatuur. Die doeltreffendheid van pulpberoering en suurstof parsiele druk het dus 'n hoë graad van beheer oor die oplossingstempo van goud moontlik gemaak. Die gebruik van suurstof, eerder as lug, het die oplossingstempo van beide goud en silwer aansienlik verhoog. Die oplosbaarheid van natrium- kalsium- en kaliumaurosianied-komplekse was onder gesimuleerde reaksieomstandighede bestudeer en dit was vasgestel dat die oplosbaarheidswaardes vir natrium- en kalsiumkomplekse beperk was en dat die gebruik van laer pulp digthede en 'n hoër reaksie temperatuur nodig was vir die behandeling van baie hoëgraad-konsentraate. Dit was bevind dat die teenwoordigheid van oormatige hoeveelhede dwaalyster (tot 25 persent volgens massa) 'n swak oplossing van goud tot gevolg het, omdat dit onder sekere omstandighede die sementasie van goud veroorsaak het. Die handhawing van hoë suurstof parsiele druk, tesame met hoë sianiedkonsentrasies, hoë pH-waardes en die teenwoordigheid van kalsium in die sisteem het alles bygedra tot die passivering van dwaalyster. Koper, goud en yster was aangedui as die hoof sianied-verbruikende elemente.

Optimum oplossing van goud was verkry in 'n geslote bankreaktor met 'n flottasie masjien-tipe roerder vir effektiewe pulpberoering en verbinding met suurstof. Suurstof was deur die reaktor hersirkuleer. Die reaktor was ontwerp om onder 'n baie lae positiewe druk te werk. Na reaksietye van 2-6 uur teen temperature in die omgewing van 30°C, het gemiddeld 99,5 persent van beide die goud en silwer opgelos. In die geval van baie hoëgraad-konsentraate was dit aangetoon dat flottasie van goud in dieselfde reaktor gedoen kan word deur die geflopteerde konsentraat deur 'n opening in die reaktor af te trek. Die konsentraat wat 3,9 persent van die massa was, het 87,4 persent van die goud bevat.

Die herwinning van osmiridium uit sianiedloogoorblyfsels en die eletrolitiese herwinning van goud uit dragtige loeloplossing was ook na gekyk as prosesaspekte in die behandeling van goudaanlegbandkonsentraate. Finale tafeluitskotte van 10 tot 12 g/t goud was verkry.

Die resultate van die ondersoek dui sterk daarop dat indien effektiewe beroering en oksigenering in die sisteem ingesluit kan word, 'n alternatiewe sianiedproses vir die behandeling van goudaanlegkonsentraate tegnies moontlik is. In die geval van lae- en hoëgraad-konsentraate blyk sianidering gevolg deur elektrolitiese herwinning van goud die eenvoudigste proses te wees terwyl in die geval van hoëgraad-konsentraate, flottasie en smelting van die konsentraat, gevolg deur sianidering van die flottasie-uitskotte aanbeveel word. Die gebruik om suurstof, eerder as lug, deur die pulp te blaas sal grootliks afhang van die graad van die materiaal wat behandel moet word. 'n Voorlopige vergelykende studie van die kostes verbonde aan die amalgameering en sianideringsprosesse vir die herwinning van goud uit sulke konsentraate het groot ooreenkomstigheid in die kapitaal en lopende kostes aangedui. Die gevolgtrekking was gemaak dat faktore soos giftigheid, veiligheid en arbeid benodig, belangriker is as die klein kosteverskille. Verder sal die onderbreking van 'n goudaanleggravitasiebaan, moontlik gemaak deur die daarstelling van 'n sianied-proses, die lopende koste verder verlaag deur die vermindering van die massa wat behandel moet word.

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## Introduction

When mining operations first commenced on the Witwatersrand, the method of extracting gold from the ore was by stamp-milling and amalgamation<sup>1</sup>. With the introduction of cyanidation in 1890, the amalgamation of stamp-mill pulp was still retained so that as much gold as possible could be recovered at an early stage. With the later development of the closed gravity circuit, classified concentrates were again treated by amalgamation, and this practice is in use at the present time. The process involves the batch treatment of high-grade riffle-belt concentrates with mercury in amalgamation barrels. The barrel is rotated for a period ranging between 14 and 18 hours in order to liberate and polish the gold particles. Suitable quantities of lime, caustic soda, cyanide, and detergent are usually added to assist in this cleaning and polishing of the free gold. An adequate amount of mercury is then added to the barrel, and the rotation is continued for several hours during which time the mercury and gold form a soft amalgam. After amalgamation, the barrel is emptied over a single amalgam plate and the soft amalgam is then scraped by hand from the plate surface. Alternatively, the barrel contents are discharged through a hydrocyclone, and the amalgam is recovered from the spigot discharge. After being cleaned, the amalgam is filtered and retorted. The retorting of the pressed amalgam is carried out in an electric furnace at approximately 600°C, the condensed mercury being collected in buckets and recycled, while the sponge gold is smelted and cast into bars. Mercury losses during retorting should not normally exceed 0,02 g per ton of ore milled. The recovery of gold from gravity concentrates using amalgamation is about 96 to 97 per cent, although the overall recovery may be even higher because the amalgam tailings are returned to the head of the gravity circuit. The process may involve a considerable health hazard related to mercury ingestion, and is at the same time very labour-intensive.

As a result of earlier improvements to the recovery of gold in the gravity sections of existing gold plants<sup>2</sup> and the consequent need to treat larger tonnages of concentrates, an alternative processing route for gold-plant gravity concentrates was indicated. Concern about the pollution and health aspects of the amalgamation process was further motivation in the development of an alternative procedure. The advantages sought from such a route included low-risk working conditions, together with the elimination of the large recycling load of concentrates in the gold-plant gravity circuit and perhaps a less labour-intensive operation. A gold recovery equal to or better than the amalgamation route was also a prime requirement of such an alternative process. Earlier work carried out in these laboratories on the recovery of gold from both belt-concentrator tailings and amalgamation tailings had indicated that the cyanidation of high-grade gravity concentrates merited further investigation. In view of the varied and relatively long history of the gold-mining industry in South Africa, it may have been expected that the cyanidation of high-grade concentrates was well documented in the literature. However, a literature survey carried out in early 1974 failed to indicate any relevant information in this respect. The

only useful information was derived from numerous publications dealing with investigations on the dissolution of pure gold, but the extent to which such data could be applied to gold concentrates had yet to be established. Concurrently with the literature survey, work started on the on-site sampling of gold-plant belt concentrates and preliminary cyanidation tests.

The preliminary cyanidation testwork, which was carried out in stirred beakers, was most significant in that a very wide range of gold extractions, indicating very poor reproducibility, was obtained. This poor reproducibility suggested that some hitherto unforeseen experimental variable (or variables) was playing a critical role in the gold dissolution. Further investigations directed towards a better understanding of such reaction parameters were thus most necessary.

## Composition of Belt Concentrates

Weekly composite samples of belt concentrates from Welkom Gold Mine, designated W1 to W6, were washed in acetone and dried at 110°C prior to blending and splitting. The detailed analysis of four of these samples is presented in Table I, from which it can be seen that the gold content varied from 6 to 22 kg/t, reflecting the extreme difficulty in obtaining representative samples of such high-grade materials.

The average  $U_3O_8$  content can be seen to be approximately 23 kg/t, while the average platinum content of 44 g/t indicates a value for the total platinum-group metals (PGM) according to Cousins<sup>3</sup> of approximately 277 g/t. The presence of such high concentrations of uranium and PGM was indicative of the recycling of a high percentage of material in the gold-plant gravity circuit. The relatively large amounts of metallic tramp iron present (19 per cent) should be noted, because such material, if not effectively passivated, will precipitate the gold from the cyanide solution. The presence of copper (0,46 per cent), nickel (0,13 per cent), cobalt (0,10 per cent), and arsenic (0,29 per cent) should also be noted as possible major cyanide-consuming elements. The samples studied had a very similar composition with regard to their pyrite (35 per cent) and silica (33 per cent) contents.

Table II presents the screen analysis of samples W1 to W4, showing that, on average, 7,2 per cent by mass of the material is coarser than 295  $\mu\text{m}$  while containing only 4,4 per cent of the total gold. On average, 6,1 per cent of the material is below 43  $\mu\text{m}$ , but it contains as much as 26,1 per cent of the total gold content. Furthermore, on average 85 per cent of the particulate gold present is smaller than 147  $\mu\text{m}$ .

Because of the presence of considerable concentrations of PGM in such concentrates, the analyses for residual gold, silver, and platinum were carried out by acid digestion of the samples followed by atomic-absorption analysis.

## Cyanidation of Gravity Concentrates

### Consumption of Cyanide

Because of the excessive cyanide consumptions observed in the treatment of gravity concentrates, the

TABLE I  
THE CHEMICAL COMPOSITION OF FOUR COMPOSITE SAMPLES OF GOLD-PLANT BELT CONCENTRATES

Sample	Semi-quantitative spectrographic analysis (g/t)																							
	Zr	Mn	Ba	Pb	Zn	Sb	Sn	Li	Be	Mo	Nb	Cu	Ni	La	Ag	Co	As	Bi	V	Y	Sr	Na	Ca	Ti
W1	4000	4000	150	5000	80	<50	80	<100	<0,1	5	<100	1500	600	<100	100	400	400	3	50	100	<50	150	200	200
W2	4000	3000	100	8000	100	<50	70	<100	<0,1	8	<100	3000	1000	<100	400	800	1000	6	100	200	<50	150	200	200
W3	6000	4000	100	3000	60	<50	50	<100	<0,1	8	<100	1500	700	<100	300	400	400	3	70	100	<50	150	200	200
W4	1500	3500	200	6000	250	60	100	<100	<0,1	5	<100	900	3000	100	250	400	5000	8	15	400	<10	100	2000	200

TABLE II  
PARTICLE-SIZE AND GOLD-DISTRIBUTION ANALYSES OF FOUR COMPOSITE SAMPLES OF GOLD-PLANT BELT CONCENTRATES

Sample	Semi-quantitative spectrographic analysis (g/t)										X-ray fluores. (kg/t)										Chemical analysis (%)										Fire assay (g/t)			
	Al	Ge	Fe	Si	Cr	Mg	B	W	U <sub>3</sub> O <sub>8</sub>	ThO <sub>2</sub>	Rb	SrO <sub>2</sub>	Fe	Fe°	Co	Cu	Ni	S	As	As	Ni	S	As	Au	Ag	Pt	Pd							
W1	>3000	<10	>3000	>3000	2000	2500	<10	—	—	—	—	46,9	41,8	21,9	0,08	0,48	0,10	13,0	0,24	0,24	0,10	13,0	6300	671	40	2								
W2	>3000	<10	>3000	>3000	3000	2000	<10	—	1,4	0,59	—	31,3	39,3	15,4	0,15	0,65	0,17	22,5	0,43	0,43	0,17	22,5	22000	2500	39	4								
W3	>3000	<10	>3000	>3000	2500	1000	<10	—	2,3	1,21	—	34,4	33,0	13,4	0,08	0,28	0,10	18,4	0,19	0,19	0,10	18,4	12200	1200	44	2								
W4	>3000	10	>3000	>3000	2000	500	<10	1000	1,2	0,82	—	22,3	46,4	25,0	0,08	0,43	0,13	21,8	0,31	0,31	0,13	21,8	19400	2200	54	—								

Size fraction (μm)	Sample W1			Sample W2			Sample W3			Sample W4		
	Mass distr. (%)	TPM* (kg/t)	% of total TPM	Mass distr. (%)	TPM* (kg/t)	% of total TPM	Mass distr. (%)	TPM* (kg/t)	% of total TPM	Mass distr. (%)	TPM* (kg/t)	% of total TPM
+205	5,8	2,4	1,9	5,9	11,8	3,0	5,9	7,3	3,2	11,5	15,6	9,3
+208-295	10,5	2,3	3,2	7,1	10,7	3,2	7,1	7,3	3,9	9,1	11,8	5,5
+147-208	22,3	2,7	8,0	14,0	8,4	5,1	15,3	5,7	6,6	16,2	11,0	9,2
+104-147	22,2	4,5	13,2	16,9	5,4	4,0	20,2	5,7	8,6	24,4	12,0	15,1
+74-104	21,6	7,5	23,7	15,7	11,2	21,5	25,6	11,2	17,8	17,8	18,0	16,5
+53-74	8,5	12,9	14,6	23,7	28,7	19,4	13,6	17,5	17,9	11,8	28,6	17,4
+43-53	4,9	23,3	14,9	8,2	39,9	14,2	5,7	23,3	10,0	4,2	38,7	8,4
-43	4,2	40,4	22,6	8,6	93,9	35,0	6,6	57,1	28,3	5,0	72,2	18,6

\*Total precious metals

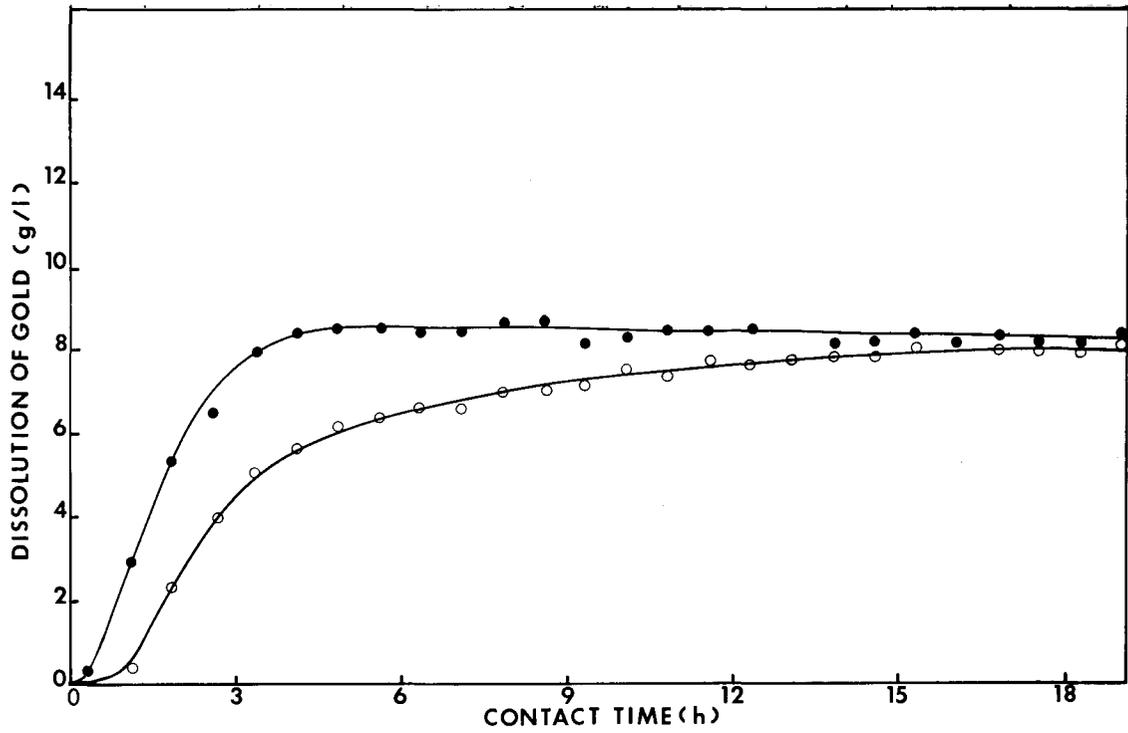


Fig. 1—Dissolution of gold from sample W1  
 ● Oxygen flow-rate 180 cm<sup>3</sup>/min ○ Air flow-rate 180 cm<sup>3</sup>/min NaCN addition 25 kg/t pH value 12,4 to 12,7 Liquid:solid 1:1

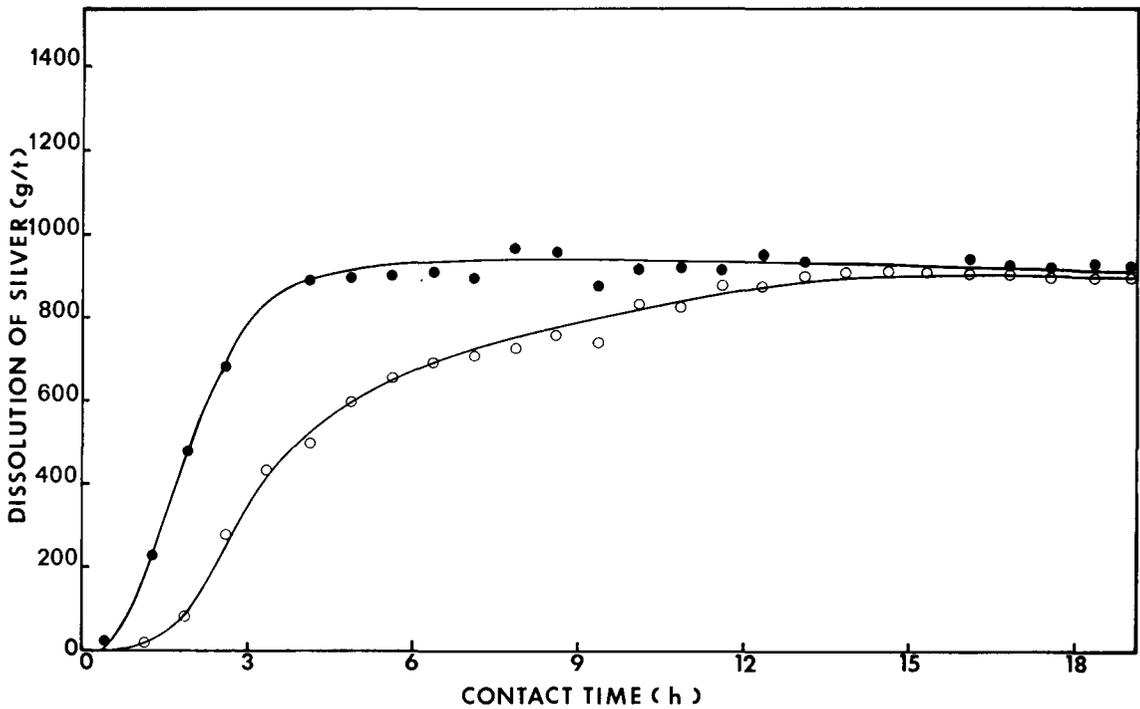


Fig. 2—Dissolution of silver from sample W1  
 ● Oxygen flow-rate 180 cm<sup>3</sup>/min ○ Air flow-rate 180 cm<sup>3</sup>/min NaCN addition 25 kg/t pH value 12,4 to 12,7 Liquid:solid 1:1

TABLE III

CYANIDATION OF SAMPLE W1 GOLD-PLANT BELT CONCENTRATE AT  $\approx 25^\circ\text{C}$ 

Test no.	Residue		Extraction		Sodium cyanide		Gas flow-rate		Conditions	Final $\frac{\text{Fe}}{\text{Cu}}$ in soln (g/t)	Initial pH	Final pH
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Added (kg/t)	Consumed (kg/t)	Oxygen (cm <sup>3</sup> /min)	Air (cm <sup>3</sup> /min)				
1 M	22,6	4,6			25,0	20,0	180	—	50 g/t Pb* added	650	12,4	12,7
1 NM	17,7	4,8	99,6	99,4						5000		
1 R	29,0	5,0										
2 M	227	14,1			25,0	19,5	—	180	2,5 kg/t CaO added	—	—	—
2 NM	589	23,2	95,0	99,3								
2 R	320	10,0										
3 M	105	12,0			25,0	19,5	—	180	2,5 kg/t CaO added	360	12,5	12,7
3 NM	285	21,2	97,5	98,1						4900		
3 R	140	20,3										
4 M	33,5	4,2			25,0	23,1	180	—	50 g/t Pb* added	1560	12,4	12,6
4 NM	21,1	6,2	99,7	99,0						4900		
4 R	38,2	6,2										
5 M	492	703			25,0	24,9	180	—	H <sub>2</sub> SO <sub>4</sub> added	1450	11,4	10,3
5 NM	32,5	28,0	93,1	20						4100		
5 R	339	750										
6 M	370	84,1			25,0	25,1	20	—	Reduced flow-rate (50 g/t Pb*)	2300	12,3	12,6
6 NM	1123	167	87	80					Increased cyanide addition (50 g/t Pb*)	3600		
6 R	729	148										
7 M	32,3	4,6			35,0	31,2	20	—		2500	12,3	12,9
7 NM	19,1	7,1	99,7	98,2						4900		
7 R	24,4	5,4										
8 M	392	43,1			35,0	25,0	20	—	Air pretreat for 12 h (50 g/t Pb*)	570	12,1	12,4
8 NM	654	82,1	92,1	82						4100		
8 R	554	64,2										
9 M	33,0	2,6			25,0	24,3	180	—	No Pb added	1700	12,0	12,5
9 NM	39,2	2,3	99,5	99,5						4700		
9 R	33,1	2,1										

M Magnetic fraction residue

NM Non-magnetic fraction residue

R Composite residue

\* Lead added as Pb(NO<sub>3</sub>)<sub>2</sub>

dissolution of gold and silver together with iron, copper, cobalt, nickel, and arsenic was investigated as a function of time so that the major cyanicides could be defined. The use of oxygen rather than air to aid the gold dissolution was also briefly considered.

Concentrate samples of 2 kg each were leached with 2 litres of cyanide solution in a covered 5-litre beaker. The pulp was well stirred by a large paddle-type stirrer, while the pulp was sparged at a constant flow-rate of gas through a sintered-glass gas-distribution thimble. The reaction was carried out at ambient temperature, and, unless otherwise specified, the leaching times were confined to 20 hours. A micropump was employed to bleed the leach system continuously of filtered solution at a rate of 12,5 ml/h, and the solution was collected in a fraction collector for atomic-absorption analysis. After cyanidation the pulp was filtered, the filtrate being titrated for residual cyanide and the washed residue assayed for gold and silver. In certain instances, the leach residues were further separated into magnetic and non-magnetic fractions prior to analysis. Deionized water was used throughout the investigation.

Figs. 1 and 2 illustrate the dissolution of gold and silver respectively from sample W1 (gold 6,3 kg/t), either oxygen or air being used to oxygenate the pulp. It can be clearly seen that oxygen has a markedly beneficial effect on the kinetics of both gold and silver leaching. After 20 hours, an extraction of 99,6 per cent of the gold and 99,4 per cent of the silver using oxygen can be compared with an extraction of 97,5 per cent of the gold and 98 per cent of the silver using air (Table III).

In a further test run in which the pH value was maintained at approximately 10,3 by the use of sulphuric acid (Test No. 5, Table III), a gold extraction of 93,2 per cent and a silver extraction of only 20 per cent were obtained. Such poor dissolution, together with the very high gold and silver contents of the magnetic fraction of the residue and the high soluble iron but low copper content of the leach liquor, strongly suggested the cementation of gold, silver, and even copper onto the metallic tramp iron present. This conclusion is supported by earlier findings in which iron powder was added to the leach pulp prior to cyanidation at pH 10,5. The deleterious effect of the iron powder was most marked when air rather than oxygen was used to oxygenate the pulp, indicating that the oxidizing potential of the system plays a vital role in such a mechanism. The reduction in the oxygen flow-rate from 180 cm<sup>3</sup>/min to 20 cm<sup>3</sup>/min (Test No. 6, Table III) further substantiates the above suggested mechanism, where again a poor gold extraction (87 per cent) was obtained together with abnormally high amounts of soluble iron in solution. Such poor recoveries can, however, be improved by the use of higher cyanide additions (Test No. 7, Table III) but result in an excessive consumption of cyanide. Alternatively, prolonged pre-aeration of the pulp (Test No. 8, Table III), producing an excessive iron hydroxide precipitate, resulted in a marked decrease in soluble iron. The resulting poor gold recovery (92 per cent) is not easily explained, but excessive adsorption of gold and silver by such a precipitate (very high positive zeta

potential) is certainly suggested. The addition of lead in the form of lead nitrate prior to cyanidation appeared to have little or no effect on the gold dissolution under these specified conditions (compare Test Nos. 1 and 9, Table III). Likewise, small additions of lime had no noticeable effect on the gold dissolution (see later).

An effort was made to approximate the contributions to the total cyanide consumption made by gold, silver, copper, and iron, neglecting the small contribution made by cobalt, nickel, and arsenic as well as that lost by oxidation. The following co-ordination numbers were assumed for the corresponding cyanide complexes: 6 (iron), 3,6 (copper), 2 (gold), and 2 (silver). The solution concentration of the respective elements at the end of each 20 hour leach (Table III) were then used in the calculation of the cyanide consumption. In most instances, the calculated consumption agreed fairly well with the observed consumption. Thus, for the best gold extraction obtained (Test No. 7, Table III), it was shown that 63 per cent of the cyanide was consumed by copper, 19 per cent by gold, 15 per cent by iron, and 3 per cent by silver. These contributions can be compared with a very poor gold extraction (Test No. 6, Table III), where only 41 per cent of the cyanide was consumed by copper, 12 per cent by gold, 2 per cent by silver, and as much as 45 per cent by iron. This excessive cyanide consumption by iron is in many instances associated with poor gold and silver extractions.

The leaching of a second sample of belt concentrate designated W2 (gold 22 kg/t) using variable oxygen flow-rates and cyanide additions (Test Nos. 1-3, Table IV) yielded very poor gold and silver extractions. Such poor extractions, together with slower gold dissolution rates and the observation that there was sometimes a white crystalline precipitate in the pregnant liquor, suggested the saturation of the liquor with some gold complex. The subsequent analysis of such a dried precipitate indicated the formula NaAu(CN)<sub>2</sub>. It is interesting to note that, in those tests in which excessive additions of sodium cyanide were not made (Test No. 1, Table IV), the saturation of the pregnant liquor occurred at a gold concentration of approximately 13 g/l. However, when the addition of cyanide was excessive (Test No. 2, Table IV), only about 2 g/l gold was found in solution. This suggests that the solubility of the sodium aurocyanide complex decreases considerably with increasing cyanide concentration owing to a common ion effect. Such an effect may well be significant at the start of a reaction, when cyanide concentrations are relatively very high.

In a demonstration that the above solubility effect slows down and even halts the dissolution of gold and silver, leaches were conducted with equimolar additions of potassium cyanide under similar conditions. (From the available information it appears that potassium aurocyanide is approximately 6 to 7 times more soluble than sodium aurocyanide.) It was immediately evident (Test No. 5, Table IV) that, under the prescribed experimental conditions, i.e., a liquid:solid ratio of 1:1 and 25°C, the solubility of the gold cyanide complex plays a most significant role in the leaching of such rich concentrates. A gold recovery of 99,9 per cent, together with an iron

TABLE IV  
CYANIDATION OF SAMPLE W2 GOLD-PLANT BELT CONCENTRATE AT  $\approx 25^\circ\text{C}$

Test no.	Residue		Extraction		Cyanide		Oxygen flow-rate (cm <sup>3</sup> /min)	Conditions	Final Fe— Cu soln (g/t)	Initial pH	Final pH
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Added (kg/t)	Consumed (kg/t)					
1 M	3620	585			30,0 (NaCN)	30,0	20	Pretreat with O <sub>2</sub> for 12 h	2000	12,1	12,7
1 NM	10800	1022	63	64					3000		
1 R	8400	928									
2 M	4500	966			70,0 (NaCN)	42,2	180	NaCN added to excess	2300	12,0	12,6
2 NM	12900	303	53	60					4400		
2 R	10844	2375									
3 M*	3225	320			35,0 (NaCN)	30,1	500	Initially large O <sub>2</sub> flow reduced to 180 cm <sup>3</sup> /min after 0,75 h	1000	12,0	12,6
3 NM	8132	806	75	75			Reduced to 180		3900		
3 R	5700	600									
4 M*	2425	263			35,0 (NaCN)	26,3	180	25 l/t of H <sub>2</sub> O <sub>2</sub> added	940	12,0	12,6
4 NM	7200	697	73	75					5900		
4 R	6340	629									
5 M	20,3	5,2			47,0 (KCN)	42,5	180	Equimolar quantity of KCN to compare with 35 kg/t NaCN	1080	12,6	13,2
5 NM	16,4	3,1	99,9	99,9					4400		
5 R	15,1	2,0									
6 M	1000	110			35,0 (KCN)	34,9	180	25 l/t of H <sub>2</sub> O <sub>2</sub> added	240	12,2	12,9
6 NM	1300	150	92,7	92,5					4500		
6 R	1600	180									
7 M	350	35			35,0 (KCN)	34,0	180	KCN added in batches; 25 l/t of H <sub>2</sub> O <sub>2</sub> added	100	12,5	13,1
7 NM	1500	200	97,0	98,3					3300		
7 R	650	40									

\* Reaction time of 6 h.  
M Magnetic fraction residue  
NM Non-magnetic fraction residue  
R Composite residue

concentration of only 1080 g/t in the final leach liquor, should be noted. The dissolution of gold, silver, copper, and iron from sample W2 using potassium cyanide while sparging with oxygen is illustrated in Fig. 3. The addition of small quantities of hydrogen peroxide (Test Nos. 6-7, Table IV) can be seen to have little effect on the gold dissolution, but significantly decreases the iron dissolution.

### Solubility of Sodium and Potassium Aurocyanide

During the course of the present investigation on the dissolution of gold from high-grade belt concentrates, precipitates of aurocyanides frequently occurred. Such precipitates invariably coincided with poor gold extraction, which necessitated an investigation into the factors affecting the solubility of aurocyanide. Literature data on the subject revealed very little information, and only the solubility of potassium aurocyanide at 25°C could be derived (from the works of Basset and Corbet<sup>4</sup>).

Both potassium and sodium aurocyanide were prepared by evaporation of pregnant liquors, these being obtained by the leaching of gold concentrates with sodium or potassium cyanide respectively. The condensed solutions were chilled, and the precipitated aurocyanide salts were filtered off, washed with ice water, and dried at 110°C. Analysis of the salts showed that small quantities of alkalis and silver, and traces of iron, were still present. It was assumed that such small quantities of impurities would have no significant bearing on the solubility data.

A thermostatically controlled water-bath connected to a refrigeration unit was used for temperature equilibration. Small Erlenmeyer flasks containing water or leach

liquor were placed in the water-bath and saturated somewhat above the required temperature with the respective aurocyanide. The required temperature was allowed to stabilize for at least 6 hours while the flasks were shaken frequently. Then 10 ml quantities of saturated solution were pipetted into 100 ml weighed volumetric flasks, the pipette always being heated to above the test temperature to prevent crystallization of the aurocyanide during pipetting. After the sample had been weighed, the flasks were filled with distilled water and analysed for gold. The sodium and potassium concentrations were determined by atomic-absorption spectrophotometry. The error introduced by pipetting at temperatures other than the standard temperature of the pipette was found to be less than 1 per cent.

The effect of temperature on the solubilities of potassium and sodium aurocyanide in solutions of different cation concentrations is shown in Fig. 4, while Fig. 5 shows the dependence of the sodium aurocyanide solubility on the total concentration of sodium ions.

The two diagrams show that both the temperature change (Fig. 4) and the total cation concentration (Fig. 5) have a considerable effect on the solubility of the aurocyanide. Although the solubilities of both potassium and sodium aurocyanide increase with increasing temperature, the rate of increase is significantly greater for the potassium salt. The detrimental effect of the increase in cation concentration is probably related to both the salt effect and the common ion effect; there may even be a significant influence of anions such as cyanide or hydroxyl ion. Moreover, it has been observed that the aurocyanide concentration in saturated solutions is also affected by the total salt concentration. However, the influence of these parameters was not

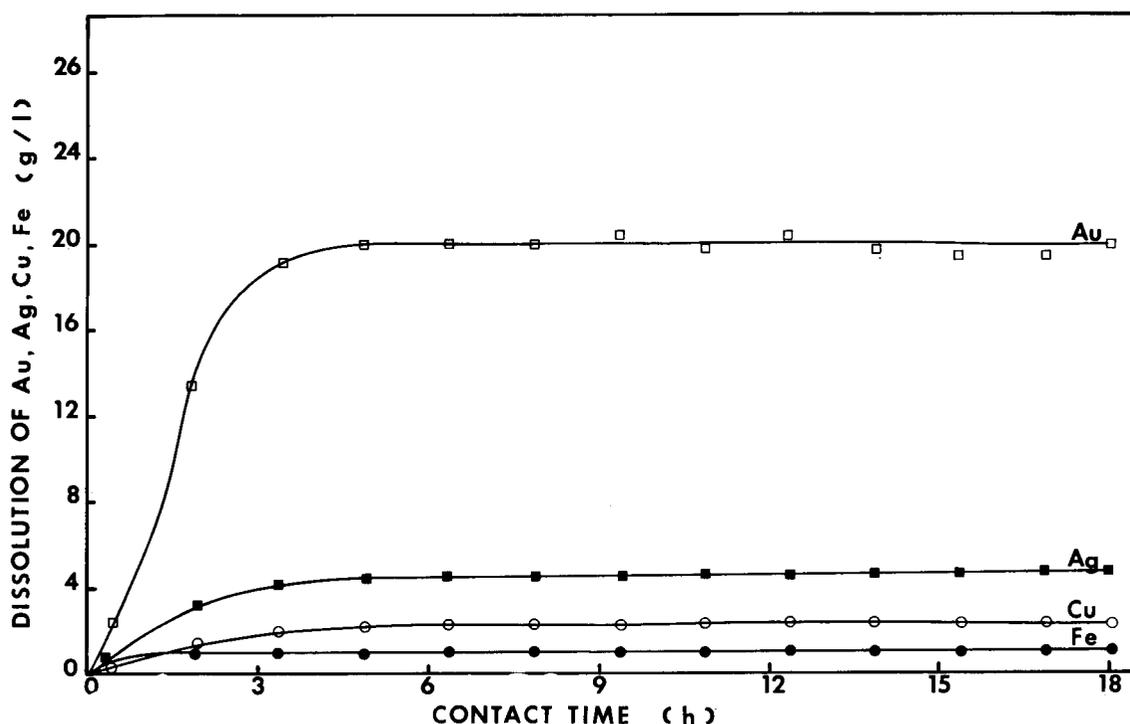
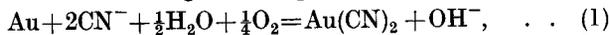


Fig. 3—Dissolution of gold, silver, copper, and iron from sample W2  
KCN addition 47 kg/t Oxygen flow-rate 180 cm<sup>3</sup>/min Liquid: solid 1:1 pH value 12.6 to 13.2

investigated because of the extent and complexity of the work involved.

The increase in pH accompanying the increase in cation concentration (Fig. 5) is probably not due to a general dependence of aurocyanide solubility on pH, but rather to the manner of gold dissolution in cyanide solutions according to the equation



One mole of hydroxyl ion is produced for each 2 moles of cyanide ions consumed. Similarly, as mentioned before, hydroxyl ions are produced during the dissolution of iron and copper by cyanide.

It is perhaps surprising that saturation of the leach liquor should limit the gold dissolution during the leaching of concentrates. From available thermodynamic data, reaction (1) has a highly negative free energy ( $\Delta G^\circ_{298} = -24$  kcal/mol), even for saturated aurocyanide solutions. Thus, precipitation of aurocyanide during gold dissolution should act as a gold sink preventing attainment of equilibrium, rather than a dissolution inhibitor. Probably, the formation of a diffusion barrier

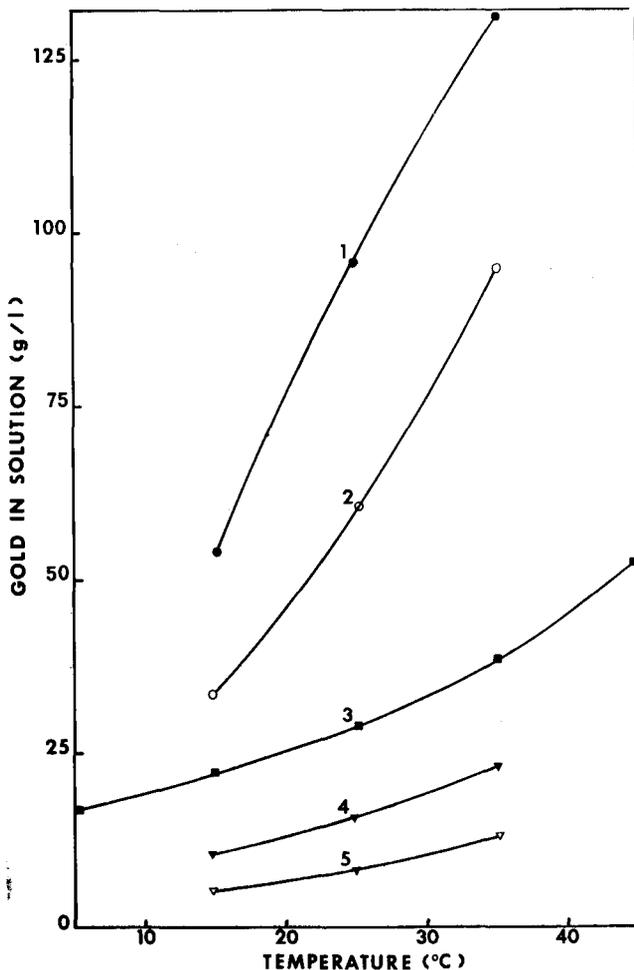


Fig. 4—Effect of temperature on the solubility of sodium and potassium aurocyanide

- $\text{K}[\text{Au}(\text{CN})_2]$ —K<sup>+</sup> increasing from 49 to 76 g/l
- $\text{K}[\text{Au}(\text{CN})_2]$ —K<sup>+</sup> increasing from 64 to 86 g/l
- $\text{Na}[\text{Au}(\text{CN})_2]$ —Na<sup>+</sup> increasing from 2,3 to 6,7 g/l
- ▼  $\text{Na}[\text{Au}(\text{CN})_2]$ —Na<sup>+</sup> increasing from 6,5 to 8,0 g/l
- ▽  $\text{Na}[\text{Au}(\text{CN})_2]$ —Na<sup>+</sup> increasing from 12,4 to 14,4 g/l

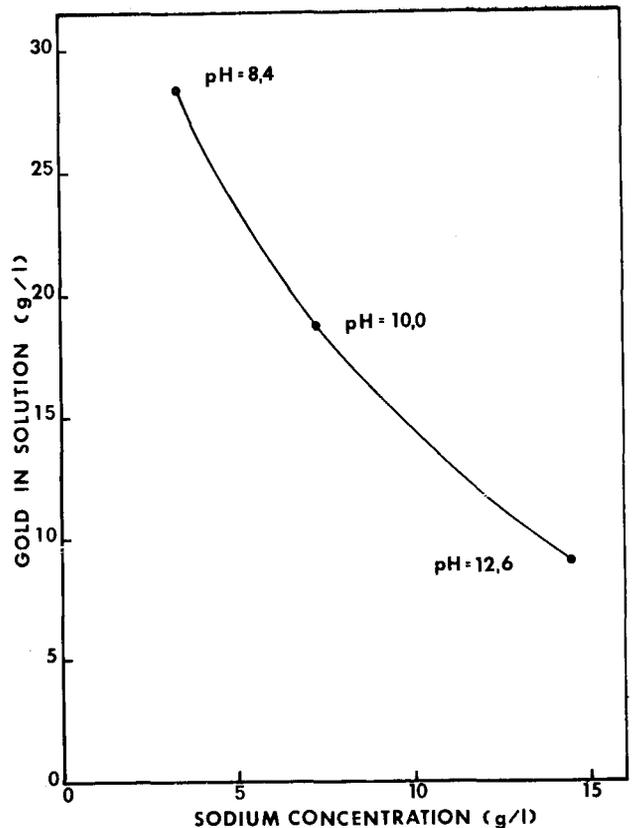


Fig. 5—Effect of sodium concentration on the solubility of aurocyanide at 25°C

by precipitated aurocyanide on the reaction surface (passivation) is responsible.

Because of the cessation of gold dissolution as soon as the precipitation of aurocyanide occurs, the solubility of the gold complex is of major importance in the cyanidation of concentrates. Consequently, with very rich concentrates (20 kg/t gold) the liquid:solid ratio becomes very critical. The liquid:solid ratio in turn has a pronounced effect on the size of leaching equipment required. For example, if a concentrate containing 20 kg/t is being leached with 25 kg/t sodium cyanide, the sodium concentration will be 11,7 g/l at a liquid:solid ratio of 1:1 and 5,9 g/l at a ratio of 2:1. If all the gold dissolved, a gold concentration of 20 g/l for the lower liquid:solid ratio and 10 g/l for the higher would be obtained. From curve 5 in Fig. 4 it is obvious that just over half the gold can dissolve at the low liquid:solid ratio even at 35°C, whereas a temperature in excess of 30°C is necessary to achieve complete gold dissolution at the higher ratio.

A liquid:solid ratio of approximately 3:1 would be required if leaching were carried out at room temperature. On the other hand, if potassium cyanide were used at room temperature, the liquid:solid ratio could be reduced to as little as 1:2. Although solubility data are generally represented as grams of solids in 100 grams of solution, the present representation of grams per litre was adopted for reasons of simplicity. Values can be converted to grams of solids per 100 grams of solution by the assumption of an increase in specific gravity from 1,02 to 1,03 g/cm<sup>3</sup> for sodium aurocyanide solutions and

1,11 to 1,19 g/cm<sup>3</sup> for potassium aurocyanide solutions in the temperature range 15 to 35°C.

It is clear from the results presented that high-grade gold concentrates pose certain practical limitations on the choice of leaching conditions when sodium cyanide is used as a leaching agent. Both higher reaction temperatures (30 to 40°C) and higher liquid:solid ratios (>2:1) are indicated if precipitation of the gold is to be avoided. While the use of potassium cyanide as an alternative leaching agent may well allow a far greater latitude in the choice of such reaction parameters, the additional cost and availability of potassium cyanide may militate against its use. The present programme was confined to the solubility of potassium and sodium aurocyanides, illustrating the important role played by solubility in the intensive cyanidation of gold concentrates. Such findings immediately suggest that the presence of calcium or magnesium may likewise play a very important role in the cyanidation of gold concentrates, and further investigation in this direction appears to be necessary (see later).

### The Effect of Oxygenation and Agitation

The earlier investigations had indicated that pulp aeration and agitation may play a very vital role in effecting efficient gold and silver dissolution from such high-grade material. It was thus necessary to investigate such variables using several means of mechanical agitation. The use of paddle agitation, a Super-agitator (Denver Equipment nomenclature), and a flotation machine was considered in the present investigation, which was confined to the treatment of sample W3 (gold 12,2 kg/t and silver 1200 g/t) at a liquid:solid ratio of 2:1 unless otherwise specified.

For paddle agitation, a series of experiments was designed so that variable flow-rates of oxygen and air, together with the efficiency of agitation at various reaction times, could be investigated. Cyanidation was carried out in a loosely covered 2-litre glass beaker maintained at 25°C in a water-bath in which the pulp was agitated by means of a paddle. Oxygen or air was added in controlled quantities through a sintered-glass gas distributor located near the bottom of the vessel. Two different impeller speeds and two different paddle sizes were investigated.

From Table V it can be seen that the kinetics of gold and silver dissolution are again faster with oxygen than with air, but that, at longer reaction times, the residual values obtained with air are similar to those obtained with oxygen (compare Test No. 7 with Test No. 15). Furthermore, the lower stirring speed (400 r/min) gave consistently poor gold extractions as compared with the higher speed (800 r/min). The effect of paddle size, although significant, is less obvious. On the other hand, the amount of oxygen added, either in the form of air or as pure oxygen, had no significant effect on gold extractions down to a critical level, which in the prescribed experimental conditions was approximately 10 cm<sup>3</sup>/min (see Test Nos. 11 and 12). Iron dissolution can be seen to be higher when the final cyanide concentration was low (Table V, Test Nos. 14, 14B, 16, 16B, 17, 19, 21, and 23), the lower sodium cyanide limit appearing

to be around 3 g/l. This appeared to indicate that cementation may occur at low concentrations of residual cyanide. Table V also shows that the cyanide consumption is not significantly affected by either air or oxygen. Longer reaction times increase the cyanide consumption marginally.

As the above results again emphasize the importance of efficient agitation and aeration, a similar exercise was carried out with a small Super-agitator type of system. The Super-agitator vessel was loosely covered, and air or oxygen was added at a controlled rate by means of a sintered-glass distributor located near the bottom of the vessel. The impeller speed was controlled at 1800 r/min, and the temperature was maintained at 25°C in all cases.

As Table VI shows, the kinetics of gold and silver dissolution are faster with oxygen than with air, but, given sufficient time, air produced residual values similar to that of oxygen. An increase in the leaching time from 6 to 16 hours lowered the residual values marginally when oxygen was used but dramatically when air was used. An increase in the leaching time from 16 to 24 hours using air decreased the residual values only marginally. Residual values tended towards a minimum, possibly determined by the amount of occluded gold present in the concentrate sample. With regard to the dissolution of iron and copper, it can be seen that the iron dissolution increases with time and is lower with oxygen than with air for shorter reaction times (6 hours). As before, when the final cyanide concentration was reduced, the iron dissolution was found to be higher, which again indicated that cementation occurs at lower cyanide concentrations, confirming previous observations. The copper dissolution, on the other hand, appeared to be quite unaffected by such changes in reaction variables.

As the above results again emphasized the importance of efficient agitation and oxygenation, a third mechanism, viz a conventional flotation machine, was also investigated. The leaching vessel was completely sealed, apart from a small vent hole. Air or oxygen was fed conventionally at a controlled rate through the agitator shaft of the flotation machine. The impeller speed was 1800 r/min and the reaction time 6 hours in all cases. The temperature was controlled by a water jacket. As shown in Table VII, the kinetics of gold and silver dissolution are faster with oxygen than with air, the effects being more marked with sodium cyanide than with potassium cyanide. An increase in the temperature from 25 to 35°C using oxygen also reduced the residual gold value from 32,6 to 18,6 g/t, and an increase in the cyanide addition from 30 to 70 kg/t reduced the residual gold value from 32,6 to 20,6 g/t — an effect that is probably purely kinetic. Potassium cyanide at a liquid:solid ratio of 1:2 gave lower residual values than sodium cyanide at a liquid:solid ratio of 2:1 for a given temperature and cyanide addition, both with air and with oxygen. The iron dissolution was again lower with oxygen than with air (possibly a cementation effect), and higher with potassium cyanide than with sodium cyanide (possibly owing to higher pH). An increase in the temperature from 25 to 35°C had no effect, but an

TABLE V  
CYANIDATION OF SAMPLE W3 CONCENTRATE USING A PADDLE IMPELLER

Test no.	Stirrer (r/min)	Impeller paddle size	Residue		Extraction		Sodium cyanide		Gas flow-rate		Reaction time (h)	Final Cu in solution (g/t)	Final Fe in solution (g/t)
			Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Added (kg/t)	Consumed (kg/t)	O <sub>2</sub> (cm <sup>3</sup> /min)	Air flow-rate (cm <sup>3</sup> /min)			
1	400	small	6993	TPM	—	—	—	—	25,0	15,6	20	1305	249
2	400	large	1497	TPM	—	—	—	—	25,0	17,1	20	1300	257
3	400	small	6024	TPM	—	—	—	—	25,0	14,7	10	1080	415
4	400	large	1997	TPM	—	—	—	—	25,0	16,5	10	1240	270
5	400	small	5197	TPM	—	—	—	—	25,0	15,5	100	1350	230
6	400	large	1848	TPM	—	—	—	—	25,0	16,1	100	1350	210
7	800	small	59,5	2,5	99,5	99,8	—	—	25,0	16,6	100	1360	190
8	800	large	79,0	3,0	99,4	99,7	—	—	25,0	17,0	100	1380	230
9	800	small	76,0	6,0	99,4	99,5	—	—	25,0	16,2	20	1380	190
10	800	large	81,0	3,5	99,4	99,7	—	—	25,0	16,6	20	1380	170
11	800	small	1025	TPM	—	—	—	—	25,0	18,0	10	1290	240
12	800	large	120	5,0	99,0	99,6	—	—	25,0	18,1	10	1410	190
13	800	large	82,5	7,0	99,4	99,5	—	—	30,0	18,0	—	1500	180
14	800	small	178	17,5	98,6	98,7	—	—	25,0	24,8	—	1420	790
15	800	large	60,0	9,0	99,5	99,3	—	—	30,0	18,6	100	1460	170
16	800	small	176	14,3	98,6	99,0	—	—	25,0	24,4	100	1390	820
17	400	large	5625	TPM	—	—	—	—	30,0	24,8	—	1245	1140
18	400	small	3404	TPM	—	—	—	—	30,0	15,7	500	1410	280
19	400	small	4132	TPM	—	—	—	—	30,0	27,1	100	1260	1435
20	400	large	84,0	10,0	99,4	99,3	—	—	30,0	18,4	100	1590	330
14B	800	small	114	23,7	99,0	98,2	—	—	30,0	24,9	500	1200	920
16B	800	small	149	26,2	98,9	98,0	—	—	30,0	25,5	100	1300	910
21	800	small	3279	TPM	—	—	—	—	25,0	25,0	—	9650	1230
22	800	large	284	85,3	97,5	92,7	—	—	30,0	18,0	500	1260	248
23	800	small	4685	TPM	—	—	—	—	25,0	25,0	—	8400	1330
24	800	large	585	120	95,1	90,3	—	—	30,0	16,8	100	1276	228

TABLE VI  
CYANIDATION OF SAMPLE W3 CONCENTRATE USING A SUPER-AGITATOR\*

Test no.	Residue		Extraction		Sodium cyanide		Gas flow-rate		Reaction time (h)	Final Cu in solution (g/t)	Final Fe in solution (g/t)
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Added (kg/t)	Consumed (kg/t)	Oxygen (cm <sup>3</sup> /min)	Air (cm <sup>3</sup> /min)			
1†	144	18,0	98,6	98,4	25,0	22,2	200	—	6	1060	920
2	25,4	7,5	99,8	99,4	30,0	17,7	200	—	6	1360	100
3	19,1	6,8	99,9	99,5	35,0	22,0	200	—	16	1390	580
4	203	20,0	98,5	98,5	35,0	17,8	—	1000	6	1230	241
5	27,5	6,3	99,8	99,4	35,0	18,7	—	1000	16	1426	312
6	24,5	5,4	99,8	99,3	35,0	19,3	—	1000	24	1370	345

\*Nomenclature of Denver Equipment

†Slow and inefficient filtration, and lower than normal residual cyanide

increase in the sodium cyanide addition from 30 to 70 kg/t again increased the iron dissolution considerably. The copper dissolution again appeared to be unaffected by changes in reaction conditions.

While the evidence presented for each type of agitator system is perhaps self evident, no definite conclusion can be drawn at this stage concerning the relative effectiveness of the flotation machine and the Super-agitator, the only conclusive results being that the flotation machine gave lower residual values when using air than did the Super-agitator for short reaction times, indicating better aeration characteristics. It can reasonably be expected that, when optimum conditions are used, the flotation-machine system will yield residual gold values of between 10 and 20 g/t after relatively short reaction times. While the present exercise indicated very clearly that the type of agitation is an important operational parameter in the cyanidation of gold-plant concentrates, further testwork on a larger scale would be required before the merits of the flotation-machine type of agitator can be evaluated.

### Effects of Temperature and Calcium

It had been earlier established that both the liquid:solid ratio and the temperature of the pulp have a very direct bearing on the solubility of the aurocyanide complex during gold dissolution. Furthermore, the effects of calcium on such a leaching system and its effect on gold solubility had yet to be established, and the significance of such reaction parameters had only been deduced by indirect means. Further investigation under actual leaching conditions was warranted.

The investigation was thus primarily concerned with an elucidation of the effects of temperature and calcium on the kinetics of gold dissolution. The investigation was confined to the treatment of sample W4 (gold 19,4 kg/t) using the Super-agitator and the flotation-machine type of agitator as previously described.

For cyanidation carried out in the flotation-machine type of agitator, the experimental conditions and results are summarized in Table VIII, which shows the effects of temperature on gold dissolution, together with the use of deionized water and of air and oxygen aeration. Likewise, Table IX shows the effect of temperature on gold dissolution in the presence of 0,28 g/l calcium added in the form of gold-plant mill return water. The above effects are also illustrated graphically in Fig. 6.

It can readily be seen that, in both instances (Tables VIII and IX), temperature has a very important kinetic effect on the dissolution of gold at temperatures below 20°C. Gold extractions can be seen to decline considerably below 20°C, but were little affected in the range 25 to 50°C when oxygen aeration was used. However, the differences in the rate of reaction between air and oxygen were markedly lower at the higher reaction temperatures, sparging with air yielding residual gold values of 114 g/t at 25°C and 30 g/t at 45°C after a contact time of 6 hours, which can be compared with 15 g/t at 25°C and 15 g/t at 45°C using oxygen. The use of mill return water (Table IX) appeared to have no detrimental effect on the gold dissolution.

When the experimental programme was repeated with

TABLE VII  
THE CYANIDATION OF SAMPLE W3 CONCENTRATE USING A FLOTATION MACHINE TYPE OF AGITATOR

Residue		Extraction		Cyanide		Gas flow-rate		Liquid solid ratio	Temp. (°C)	Final pH	Final Cu in solution (g/t)	Final Fe in solution (g/t)
Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Added (kg/t)*	Consumed (kg/t)*	Oxygen (cm <sup>3</sup> /min)	Air (cm <sup>3</sup> /min)					
56,6	22,0	99,5	98,3	25,0 (NaCN)	16,6	50	—	2/1	25	12,1	1340	160
32,6	6,6	99,7	99,5	30,0 "	17,0	100	—	2/1	25	12,30	1230	178
121	—	198,9	—	25,0 "	17,3	—	250	2/1	25	12,2	1270	300
44,0	12,6	99,6	98,9	30,0 "	18,4	—	500	2/1	25	12,2	1800	220
20,6	4,3	99,8	99,7	70,0 "	37,0	100	—	2/1	25	—	1290	1468
18,8	4,6	99,8	99,5	30,0 "	18,3	100	—	2/1	35	12,1	1235	165
19,5	10,6	99,8	99,0	25,0 (KCN)	20,8	250	—	0,5/1	25	13,1	4820	1420
14,6	4,4	99,9	99,6	30,0 (KCN)	23,2	250	—	0,5/1	25	13,1	4800	1875
56,8	17,4	99,5	97,7	30,0 (KCN)	22,6	—	1250	0,5/1	25	13,1	4680	2050

\*Expressed as equivalent NaCN

†Slow and inefficient filtration

a Super-agitator, it became evident that no conclusions could be drawn from the results owing to poor reproducibility. The difficulty in obtaining reproducible results was ascribed to the efficiency of agitation. To be consistently efficient, and especially to avoid 'dead' spaces where unleached material may collect, the Super-agitator system appears to require careful design, which could not be obtained with the device used in this testwork.

When the present results concerning the effect of temperature on gold dissolution are compared with the

findings of Julian and Smart<sup>5</sup>, who studied the dissolution of pure gold, similar trends are most evident. Their results, which are presented in Fig. 7, show that below 20°C the rate of gold dissolution increases much more rapidly with temperature than it does above approximately 30°C. In the temperature range 20 to 30°C, a transition region between the fast and the slow reaction is apparent. The present investigations on the temperature dependence of leaching rates of gold from concentrates and that of pure gold show that below about 20°C the leaching rates are most sensitive to temperature

TABLE VIII

EFFECT OF TEMPERATURE ON GOLD DISSOLUTION FROM BELT CONCENTRATES USING A FLOTATION-MACHINE TYPE OF AGITATOR

Temperature (°C)	Residue		Extraction		Residual NaCN (g/l)	Gas flow-rate		Reaction time (h)
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)		Oxygen (cm <sup>3</sup> /min)	Air (cm <sup>3</sup> /min)	
25	15,0	2,6	99,9	99,9	6,9	200		6
35	19,2	2,8	99,9	99,9	7,2	200		6
45	15,1	2,0	99,9	99,9	5,8	200		6
25	114	29,8	99,4	98,6	7,0		1000	6
35	70,0	26,0	99,6	98,8	6,2		1000	6
45	30,2	13,3	99,8	99,4	6,8		1000	6
15	446	54,4	97,6	98,9	8,5	200		2
20	170	31,6	99,1	99,4	7,8	200		2
25	128	21,1	99,3	99,6	8,1	200		2
30	26,1	10,1	99,9	99,8	8,0	200		2
35	81,4	23,0	99,6	99,5	7,7	200		2
40	58,3	21,1	99,7	99,6	7,6	200		2
45	79,2	21,2	99,6	99,6	8,0	200		2
50	110	24,5	99,4	99,5	7,8	200		2

Material: Sample W4 (19,4 kg/t Au)  
 Conditions: Liquid: solid ratio 2:1  
 NaCN addition 40 kg/t

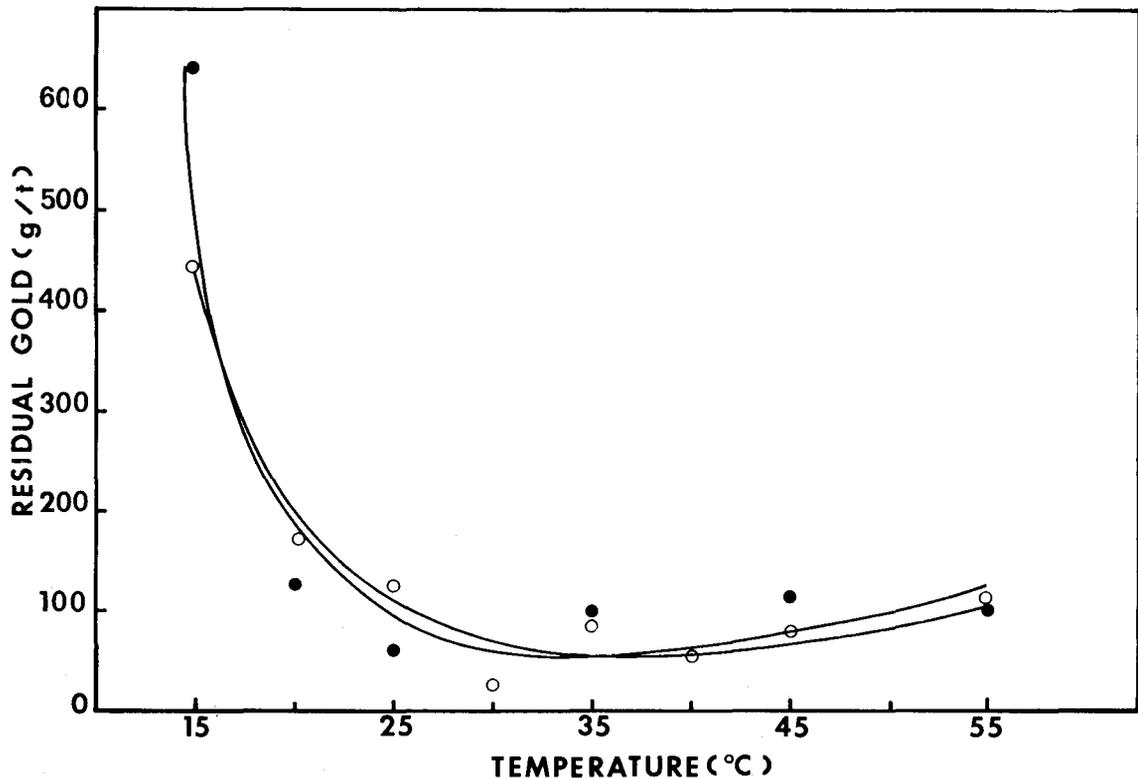


Fig. 6—Effect of temperature on the dissolution of gold from sample W5  
 ○ Using deionized water ● Using mill return water NaCN addition 40 kg/t Oxygen flow-rate 200 cm<sup>3</sup>/min  
 Liquid: solid 2:1 Contact time 2h

changes. On the other hand, above about 25°C there is no significant increase in the rate of gold dissolution with temperature.

The chemical behaviour of the gold in concentrates during leaching is similar to that of pure gold, and all the reactions that apply to pure gold appear to be valid for the gold present in concentrates. Numerous investigators have revealed that the dissolution of pure gold is usually controlled by diffusion. At low temperatures the rate control of gold dissolution can change from diffusion to reaction rate control. This change-over can be aptly

demonstrated from the data of Julian and Smart. If the data illustrated in Fig. 7 are plotted in the form of log (rate) against the reciprocal of absolute temperature, a curve similar in shape to that presented in Fig. 7 is obtained. However, the portions below about 15°C and from 25 to 70°C yield straight lines with different slopes. From these respective slopes, an activation energy of about 13 kcal/mol can be derived from the line through the low temperature points, and approximately 2 kcal/mol for the temperature range 25 to 70°C. For the type of heterogeneous reaction in the liquid phase that

TABLE IX

EFFECT OF MILL RETURN WATER ON GOLD DISSOLUTION FROM BELT CONCENTRATES AT VARIOUS TEMPERATURES USING A FLOTATION-MACHINE TYPE OF AGITATOR

Type of water	Temperature (°C)	Residue		Extraction		Residual (NaCN) (g/l)
		Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	
Deionized	15	446	54,4	97,6	98,9	8,5
Deionized	20	170	31,6	99,1	99,4	7,8
Deionized	25	128	21,0	99,3	99,6	8,1
Deionized	35	81,4	23,0	99,6	99,5	7,7
Deionized	45	79,2	21,2	99,6	99,6	8,0
Deionized	55	110	24,5	99,4	99,5	7,8
Mill Return	15	640	65,0	96,6	98,7	8,2
Mill Return	20	128	23,8	99,3	99,5	7,9
Mill Return	25	82,9	14,7	99,7	99,7	7,9
Mill Return	35	104	28,3	99,4	99,4	8,4
Mill Return	45	114	14,0	99,4	99,7	8,1
Mill Return	55	99,0	30,0	99,5	99,4	8,2

Material: Sample W4 (19,4 kg/t Au)  
 Conditions: Liquid: solid ratio 2:1  
 NaCN addition 40 kg/t  
 Oxygen flow-rate 200 cm<sup>3</sup>/min  
 Reaction time 2 h

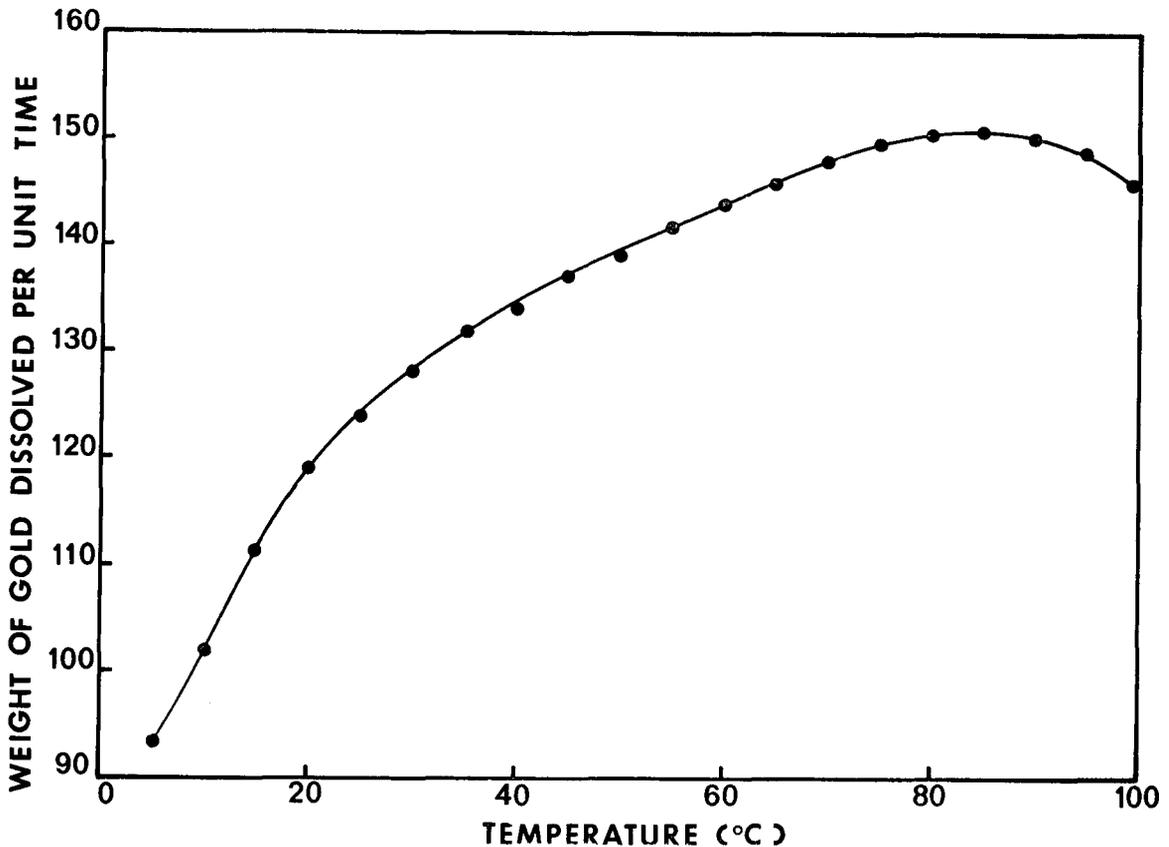


Fig. 7—Effect of temperature on the dissolution of pure gold (after Julian and Smart<sup>5</sup>)

is responsible for gold dissolution, the higher activation energy is characteristic for a reaction rate controlled process, while the lower one is typical for diffusion controlled systems. Evidence for a change from reaction rate control at low temperature to diffusion control at the higher temperature was also found in the present investigation, i.e., below 20°C the gold dissolution rate had a strong dependence on temperature, whilst above about 20°C no change of gold dissolution rates with increase in temperature could be determined. Further evidence regarding diffusion control above 20°C was that, under normal operating conditions, increased agitation and aeration and an increase in oxygen partial pressure resulted in increases in the gold dissolution rates.

### Other Reaction Variables

It became most apparent during the course of the present investigation that the reaction variables including pH, cyanide concentration, and time had very complex compounding effects upon one another, and as such were difficult to define. As the definition of these compounding effects necessitated a large number of tests, further study was confined to simple tests involving the rolling of bottles. Some 500 to 600 individual tests were conducted (the equipment being housed in a thermostatically controlled cabinet). For the sake of brevity, only the more pertinent features of this study are discussed here.

The effect of reaction time on the extraction of gold from various concentrates in the presence of air is shown in Fig. 8. It can be seen that, after an initial acceleration

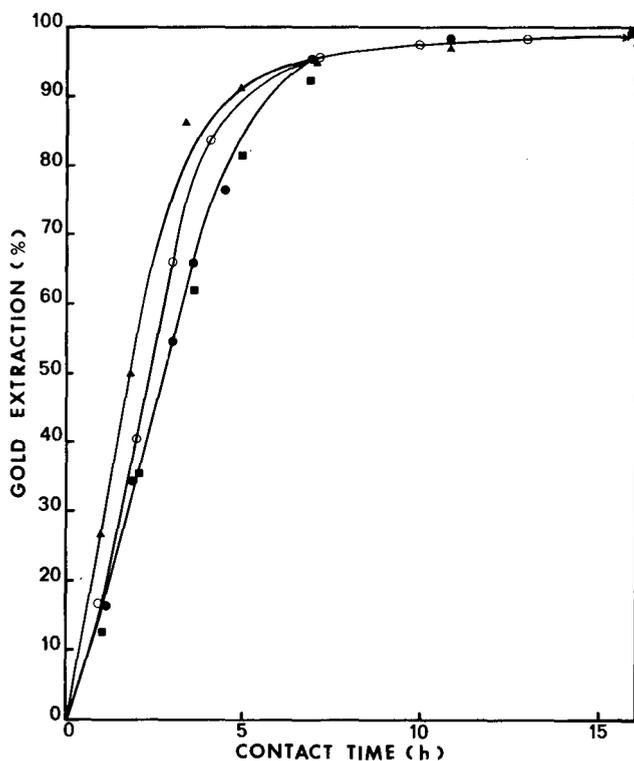


Fig. 8—Effect of contact time on the dissolution of gold from belt concentrate at 20°C  
 ○ Sample W1 (Au 6,3 kg/t)    ▲ Sample W3 (Au 12,2 kg/t)  
 ● Sample W2 (Au 22 kg/t)    ■ Sample W4 (Au 19,4 kg/t)

period, the rate of gold dissolution is almost constant for a considerable time (1 to 4 hours), and that this rate is roughly proportional to the original gold concentration, i.e., the higher the head grade, the faster the dissolution. In other words, the percentage extraction with time is roughly the same even for concentrates with very different gold contents. This cannot be strictly true for reasons that will be given later.

As expected, these reaction rates, which would yield a gold dissolution of 100 per cent in 4 to 5 hours irrespective of the head value, do not continue. After about 70 to 80 per cent of the gold has been dissolved, the rate of dissolution decreases significantly and becomes very slow after about 7 hours, when about 95 per cent of the gold has been dissolved. Approximately the same time is required for the dissolution of most of the remaining 5 per cent as for the dissolution of the first 95 per cent. Under similar experimental conditions but for extended leaching times (48 to 96 hours), the final residual gold values were approximately 10 to 20 g/t for all the concentrates so far investigated (W1 to W4). These residual gold values appear to be independent, within reasonable limits, of the magnitude of the reaction variables; that is, under all reasonable operating conditions, a final residual gold value, which is characteristic of the particular concentrate, can be achieved if leaching is continued for a long enough time. It also means that, at extended leaching times, all the confounding effects disappear. Thus, the effectiveness of any variation in the operating conditions can be assessed by the leaching time required to reach residual gold values of 10 to 20 g/t in the present instance or by how closely this residual value can be approached in a given time.

Generally, the dissolution of silver followed the same trend as that of gold, but the dissolution of silver appeared to be somewhat slower than the dissolution of gold and, during the progress of the reaction, the gold:silver residue ratio decreased from its original value of about 9:1 to roughly 5:1.

The leaching of sized fractions was undertaken for two reasons. First, it was suspected that the coarser gold might be responsible for the slower leaching rates observed after about 70 to 80 per cent of the gold had been dissolved. Second, it was hoped that this exercise might throw more light on the nature of the refractory or residual gold. In the past, speculations about the nature of this gold had ranged from its being occluded gold to its being gold in solid solution or even adsorbed gold. The observation that milling in cyanide or milling followed by cyanidation lowered the residual value considerably hinted strongly at occluded gold. Occluded gold particles were also observed during microprobe analyses of the leach residues.

A size analysis and gold distribution of the fractions used in the present investigation are presented in Table II, and the results of leaching tests are shown in Fig. 9. As expected, the +147 μm size fraction, which contained the lowest concentration of gold, had the slowest leaching rate, whilst the -74 μm fraction leached the fastest. In spite of the large difference in the head grade of gold between the coarsest and the finest fractions, the actual residual gold value after about 3 hours leaching

was already lower for the finest fraction. In addition to being the fraction with the slowest leaching rate, the +147  $\mu\text{m}$  material was also responsible for the greatest proportion of residual gold.

The effect of cyanide concentration can best be demonstrated by a graph of residual gold versus residual cyanide concentration. Such a graph is given in Fig. 10, which shows that, under the specified conditions, increasing concentrations of residual cyanide have a very significant effect on the rate of gold dissolution when the leaching is done in the presence of air, but only a very small effect on the gold dissolution rate when the leaching is carried out in the presence of oxygen. In both instances, the use of high cyanide additions invariably resulted in a higher consumption of cyanide, while, if the cyanide concentration dropped below approximately 0.5 g/l, the gold dissolution almost ceased. It was also found during this investigation that very little difference in gold dissolution rates were observed when 'flake' cyanide (50 per cent calcium cyanide) rather than sodium cyanide was used. In fact, a very beneficial effect was noted in that the solubility of the iron was markedly depressed by the use of calcium cyanide. Another advantage in the use of calcium cyanide rather than sodium cyanide is the increased solubility of calcium aurocyanide. Because analytical grades of both calcium cyanide and calcium aurocyanide could not be obtained at the time and were difficult to prepare, the solubility of calcium aurocyanide was estimated by an indirect method. A high-grade gold concentrate was leached with Aero Brand calcium cyanide (S.A. Cyanamid (Pty)

Limited) at a low liquid:solid ratio, and the maximum amount of gold that could be dissolved was considered to indicate the solubility of calcium aurocyanide under such leaching conditions. Thus, at a temperature of 20°C, a solution with a calcium concentration of about 25 g/l contained 18 g/l gold. This is more than double the quantity of gold that could be contained by 1 litre of sodium aurocyanide solution with a sodium concentration of about 15 g/l. However, the solubility of calcium aurocyanide is much lower than that of potassium aurocyanide. A solution containing approximately 75 g/l calcium as calcium aurocyanide could dissolve only about 7 g/l gold at approximately 20°C. Other cyanide reagents besides those discussed above have no practical significance and were therefore not investigated.

Leaching data on the effect of pulp pH on gold dissolution are presented in Table X, from which it is most evident that increasing pH has a significant detrimental effect on the rate of gold extraction under the specified conditions. The addition of a protective alkali, following the standard practice in plants with low-grade circuits, would therefore be detrimental only if applied to the leaching of gold concentrates, since the resultant pH during the leach is quite sufficient to prevent the evolution of cyanide. The opposite should rather be carried out; that is, acid should be added, but the danger of lowering the pH into a region where the evolution of hydrogen cyanide becomes significant would militate against such a practice except under strictly controlled

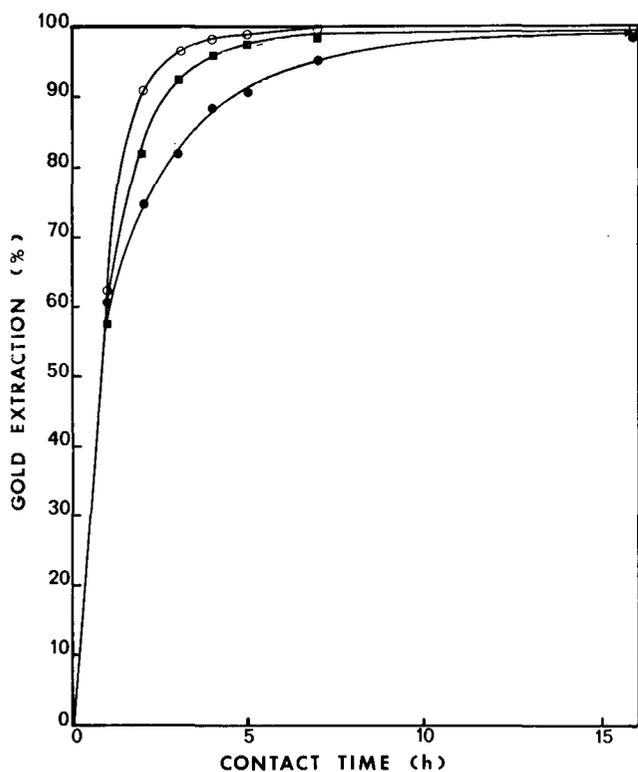


Fig. 9—Effect of particle size on the dissolution of gold from sample W4 at 30°C  
 ○ —74  $\mu\text{m}$  (Au 117 kg/t) ● +147  $\mu\text{m}$  (Au 45 kg/t)  
 ■ +74—147  $\mu\text{m}$  (Au 13.8 kg/t)  
 NaCN addition 40 kg/t Air aeration in rolling bottle  
 Liquid:solid 3:1

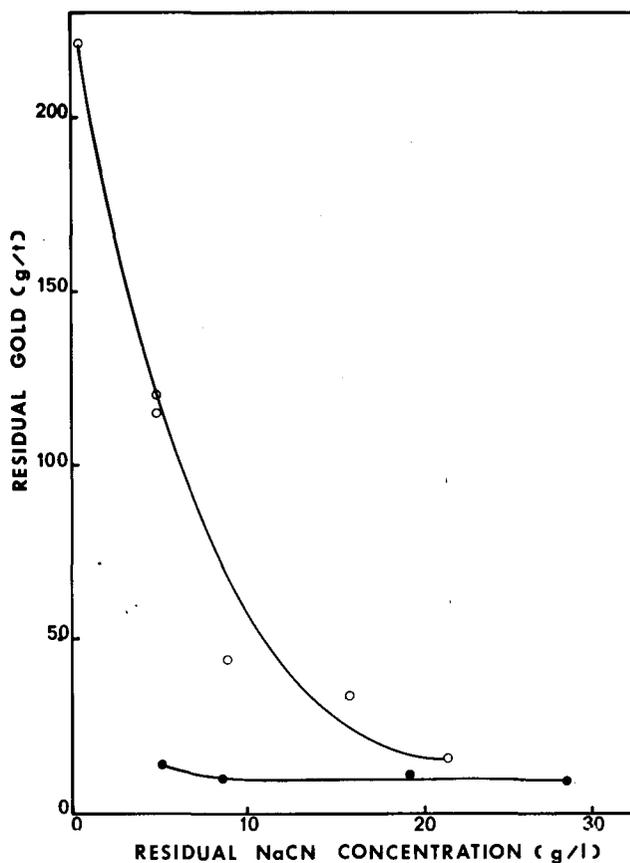
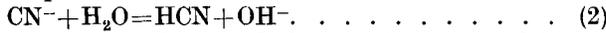


Fig. 10—Effect of residual cyanide concentration on the dissolution of gold from sample W3 at 20°C  
 ○ Aeration ● Oxygenation  
 Liquid:solid 2:1 Contact time 1h

conditions. Although the reasons for the high pH value observed during the leaching of concentrates are twofold, they are attributable to the same source. At the beginning of the reaction when cyanide is added to the pulp, some hydrolysis of the cyanide takes place according to the equation



The extent of the hydrolysis is demonstrated in the case of sodium cyanide and potassium cyanide in Table XI. It can be seen that, at the high cyanide addition required for the leaching of gold concentrates, a safe pH value is obtained even before the start of the reaction, and the addition of a protective alkali is no longer necessary. The rapidly ensuing gold dissolution reaction (1) produces more hydroxyl ions, which accounts for a rise in pH during the course of leaching. Thus, the higher the gold tenor of the concentrate, the higher the resultant pH will be and the slower the resultant leach rate.

Since increasing cyanide concentration also causes increasing cyanide consumption, especially when air-sparging is carried out, improvement of the gold dissolution rate by this method is not economically attractive owing to the current costs of reagents. The effects of pH become really detrimental only at a pH level of more than 12.5. Such high pH values can be obtained only from very rich concentrates and fairly low liquid:solid ratios. Even temperature, provided it is maintained above 25°C, does not appear to provide a ready means of controlling the reaction. This leaves oxygen partial pressure as the one variable of those currently investigated for effective rate control. Another factor that in practice is closely allied with oxygen partial pressure and that will permit a considerable amount of rate control is the type and rate of pulp agitation. Theoretical

considerations indicate that the rate and type of stirring may even permit the same control of the rate of gold dissolution as can be effected by oxygen partial pressures. To what extent use will be made of either or both of these factors in improving the rate of gold dissolution depends entirely on economic considerations.

### Cyanidation and Flotation in a Bench-scale Pilot Reactor

The efficiency of a flotation-machine type of agitator for the cyanidation of gold-plant belt concentrates having already been established, a bench-scale pilot reactor was constructed. It consisted basically of a stainless-steel leaching vessel and a Denver D12 flotation-machine agitator. The aim of the present investigation was to obtain reaction data on a larger scale for certain parameters that would affect the design of an envisaged full-scale plant. Because of gold solubility problems associated with the treatment of very rich high-grade concentrates, which would consequently seriously affect the capacity of such a plant, the feasibility of floating gold before cyanidation was also investigated.

### Equipment and Procedure

The leaching apparatus is shown in Fig. 11. The vessel was fitted initially with a heating element coupled to an automatic temperature controller. However, it was found that temperatures increased during the course of the reaction owing to the violent agitation, and the heating element was subsequently replaced by a cooling coil.

TABLE X

EFFECT OF pH ON GOLD DISSOLUTION FROM BELT CONCENTRATES

Contact time (h)	6	16	6	16
pH	Residual Au (g/t)		Au Recovery (%)	
11,5	180	55	99,1	99,7
12,3	340	95	98,2	99,5
12,7	3 100	240	84	98,8
12,9	6 400	440	67	97,7
13,0	9 000	750	54	96,1

Conditions: Cyanide addition 60 kg/t  
 Temperature 40°C  
 Material Sample W4  
 Liquid:solid ratio 2:1

TABLE XI

CHANGE OF pH WITH INCREASING CYANIDE CONCENTRATION AT 20°C

Cyanide concentration (g/l NaCN/KCN)	pH with NaCN	pH with KCN
1	10,7	10,7
2	10,9	10,9
3	11,0	11,0
5	11,1	11,1
7	11,1	11,1
10	11,2	11,2
15	11,3	11,3
25	11,3	11,4
35	11,3	11,5
50	11,3	11,5

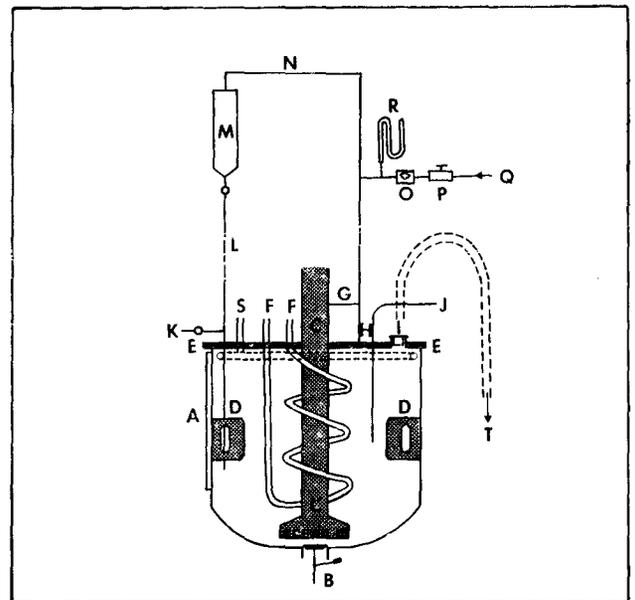


Fig. 11—The intensive cyanidation bench-scale reactor (10 kg capacity)

- A Perspex window, 5 cm by 20 cm
- B Discharge mechanism
- C Denver D12 flotation impeller
- D Baffles
- E Flanged cover plus gasket
- F Cooling coil
- G Air/oxygen feed
- H Oxygen recycle
- I Solids-feeding port/flotation port
- J Temperature control
- K Sampling/decantation tube
- L Cyanide-solution feed
- M Cyanide-solution storage
- N Pressure balancing line
- O Flowmeter
- P Pressure regulator
- Q Air/oxygen supply
- R Manometer
- S Washing ring
- T Flotation concentrate

When air was used, the system was open to the atmosphere, but, when oxygen was used, the vessel was sealed and oxygen was fed automatically through a pressure regulator, the pressure in the vessel being kept constant at approximately 7 to 14 kPa and oxygen being added only as it was consumed.

At the end of each leaching experiment, the solids were washed by decantation as follows: the solids were allowed to settle for 15 minutes for the first two decantations and for 10 minutes for subsequent decantations; the cycle of washing and decantation was continued until a neutral wash water was obtained, after which the residue in the vessel was discharged, filtered, and dried.

The on-site flotation of fourteen 1 kg samples of high-grade belt concentrates (gold 11 to 40 kg/t) by a well-established gold-flotation procedure yielded average gold recoveries of 88 per cent at rougher-concentrate grades averaging 48 per cent gold. The cyanidation of the resulting flotation tailings (gold 0,4 to 1,9 kg/t) was subsequently carried out in the bench reactor. The feasibility of floating gold direct from the bench reactor was also studied (see later).

### Cyanidation

When a high-grade gravity concentrate (sample W4 — gold 19,4 kg/t) was leached using oxygen sparging at a liquid:solid ratio of 2:1 and controlling the temperature in the region 30 to 35°C, gold extractions of 99,9 per cent were obtained after a contact time of 4 hours (Table XII). Similarly, for concentrate flotation tailings containing 432 g/t gold (Table XIII), the overall gold recoveries averaged 99,8 per cent when the flotation was taken into account. Once again, the use of oxygen would appear to be advantageous where high-grade concentrates are treated directly by cyanidation, but, with

lower-grade flotation tailings, air or a short period of sparging with oxygen followed by air would appear to be adequate. The lowest value for residual gold obtained in the present exercise (6,9 g/t) was obtained after sparging with oxygen for 1 hour followed by aeration for 16 hours. From a technical point of view, the present investigation indicated that a very simple operation yielding exceptionally high gold recoveries is technically feasible.

### Direct Flotation from Reactor

Sample W6 of gold-plant belt concentrate (gold 2,5 kg/t) was submitted to flotation in the cyanidation reactor (Fig. 11), whereby the excess froth was piped off through a 25 mm plastic tube connected to the ore-feeding port in the lid of the vessel. After a flotation procedure involving 25 minutes, a concentrate grade of 7,3 per cent gold was obtained, representing 3,9 per cent of the original mass. The gold recovery was 87,4 per cent with tailings assaying 421 g/t gold. As such a flotation recovery compared very favourably with the more conventional procedure, it would appear to be recommended, especially for high-grade concentrates. Although the cyanide consumption was reduced only marginally, the cyanidation times can be expected to be considerably shorter. The possibility of floating gold direct from the cyanidation vessel thus appeared very attractive because of its simplicity and relative effectiveness.

### Recovery of Osmiridium from Leach Residues

Three composite samples of belt concentrates designated W1 to W3 were initially leached by intensive cyanidation, and the residues were tabled for the recovery of osmiridium. Approximately 10 kg of each

TABLE XII  
CYANIDATION OF BELT CONCENTRATES IN THE BENCH-SCALE REACTOR

Contact time (h)	Temperature rise (°C)	Residual Au (g/t)	Au extraction (%)	Sodium cyanide		Fines decanted (% of total)
				Added (kg/t)	Consumed (kg/t)	
3	16-53 No control	124	99,4	40,0	28,0	3,8
6	13-59 No control	270	98,6	40,0	34,4	6,7
3	18-56 No control	67,9	99,6	40,0	31,4	6,7
6	30-35 Controlled	939	95,3	40,0	40,0	5,5
4	30-35 Controlled	26,9	99,9	55,7	40,5	5,2

Material: Sample W4 (19,4 kg/t Au)  
Conditions: Liquid: solid ratio 2:1  
Oxygen aeration

TABLE XIII  
CYANIDATION OF BELT-CONCENTRATE FLOTATION TAILINGS IN THE BENCH-SCALE REACTOR

Contact time (h)	Temperature (°C)	Oxidizing agent	Impeller speed (r/min)	Residual Au (g/t)	Au extraction (%)	Sodium cyanide		Fines decanted (% of total)
						Added (kg/t)	Consumed (kg/t)	
4	20-25	Air	1450	9,1	97,8	30,0	21,9	7,5
4	30-35	Air	1450	38,9	90,8	37,5	25,7	5,8
4	40-45	Air	1450	6,5	98,5	30,0	18,1	7,6
3	30-35	Oxygen	1450	9,7	97,7	30,0	18,5	7,2
1	30-35	Oxygen	1450	9,6	97,7	35,0	25,2	7,4
1	30-35	Oxygen	1100	33,8	92,0	35,0	23,5	5,1
1	30-35	Oxygen	800	15,9	96,2	35,0	24,5	5,8
1+16*	30-35	Oxygen + Overnight air	1450	6,9	98,4	25,0+10*	24,3+8,1*	6,0

\*1 hour of oxygen followed by 16 hours of air.  
Material: Flotation tailings (432 g/t Au)  
Conditions: Liquid: solid ratio 1:1

leach residue composite was tabled on a small Wilfley shaking table, and concentrate and tailing fractions were taken for subsequent analysis. The results are presented in Table XIV, where it can be seen that an average platinum recovery (osmiridium) of 76 per cent was obtained in a mass distribution of 3,7 per cent. The concentrate fraction can also be seen to contain on average 24 per cent of the total gold and 49 per cent of the total uranium values. The results show remarkably good reproducibility with regard to the recovery of platinum and uranium. Such findings indicate that a very large percentage of the heavy-mineral phases are currently recirculating in the existing closed gold-plant gravity circuit. A mineralogical examination of various residue fractions indicated that the gold was almost exclusively occluded in the uraninite and pyrite phases. As the tabling of leach residues yielded tailings containing 10 to 12 g/t gold, roughly half of which was estimated to be occluded in the large recycling uraninite phase, the eventual opening of the gravity circuit should roughly halve the subsequent residual gold values.

The evidence presented, together with the findings on the cyanidation of gravity concentrates given earlier, shows that, in the case of composite belt concentrates, intensive cyanidation followed by a tabling operation for the recovery of osmiridium may eventually yield residues containing as little as 5 g/t gold. It is difficult to assess at present whether such residues merit further milling to reduce the gold values even further, or whether the simple calcining of such residues in a conventional acid plant would not be a more practical operation.

### Electrowinning of Gold from Pregnant Liquor

In view of the high-tenor gold solutions resulting from the cyanidation of gold-plant concentrates, as well as the relatively high residual cyanide concentrations of such solutions, the electrowinning of gold was considered to be a better alternative than zinc precipitation. As a result of investigation at these laboratories on the electrowinning of gold from relatively concentrated cyanide solutions, a cell consisting of an inner cathode compartment separated from an outer annular anode compartment by a semi-permeable cation-exchange

membrane was developed (Fig. 12). The use of a semi-permeable membrane allows for the regeneration of cyanide in the catholyte. Gold and silver were plated onto mild-steel wool, and the iron was subsequently removed from the product either by fusion and fluxing or by leaching with acid. It was shown that, with con-

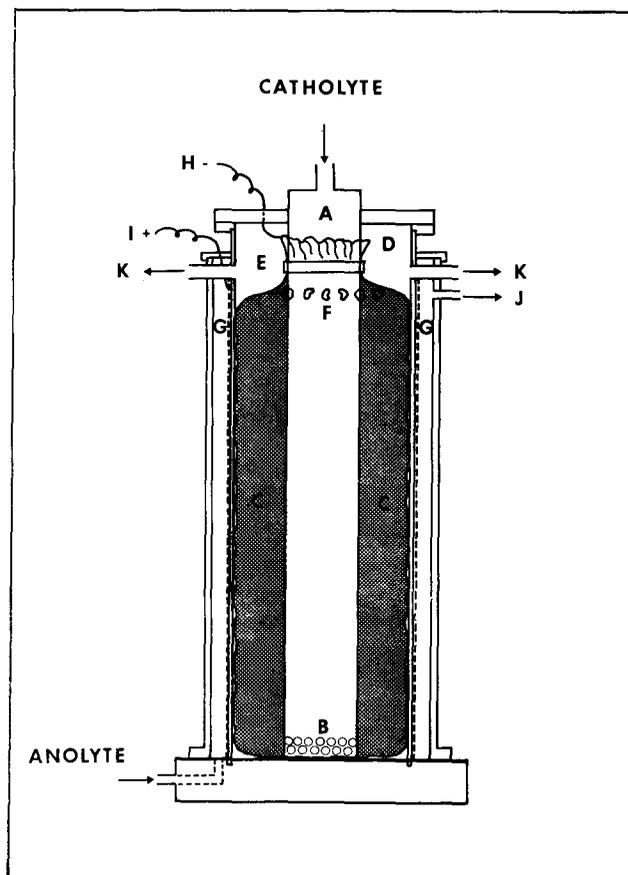


Fig. 12—The steel-wool gold-electrowinning cell  
**A** PVC inlet distribution tube **B** Catholyte distribution ports  
**C** Mild-steel wool **D** Sock made from filter cloth  
**E** Band for securing filter sock **F** Catholyte outlet holes  
**G** Annular perforated stainless-steel anode enclosing the cation-exchange membrane  
**H** Cathode connection **I** Anode connection  
**J** Recirculating NaOH/Na<sub>2</sub>SO<sub>4</sub> anolyte  
**K** Recirculating catholyte

TABLE XIV  
 RECOVERY OF OSMIRIDIUM FROM BELT-CONCENTRATE LEACH RESIDUES

Material	Mass distribution (%)	Concentrate grades			Distribution			
		Au (g/t)	Pt (kg/t)	U <sub>3</sub> O <sub>8</sub> (%)	Au (%)	Pt (%)	U <sub>3</sub> O <sub>8</sub> (%)	Pyrite (%)
<i>Sample W1</i>								
Concentrate	2,5	405	1868	345	28,0	77,0	47,0	1,0
Middling 1	16,6	26,1	26,4	—	12,2	7,4	—	24,5
Middling 2	74,5	24,8	10,9	—	52,1	13,6	—	67,0
Tails	6,4	41,8	11,4	—	7,7	1,0	—	7,5
<i>Sample W2</i>								
Concentrate	5,1	83	655	295	25,0	80,0	53,0	2,1
Middling	19,8	14,5	17,8	35	18,5	8,5	24,4	23,4
Tails	75,1	12,4	6,4	8,6	56,5	11,5	22,6	74,5
<i>Sample W3</i>								
Concentrate	3,6	76	625	221	20,0	70,0	47,0	1,8
Middling	22,3	14,3	16,5	21	22,0	10,2	27,0	28,2
Tails	74,1	11,1	93	6	58,0	20,8	26,0	70,0

centrated solutions exceeding 1 g/l gold, 100 per cent current efficiency was obtained, but this efficiency decreased at lower concentrations and was about 40 per cent at 150 g/t gold. The mild-steel wool was the type used for cleaning, and could be loaded with at least ten times its own mass of gold plus silver. Minor amounts (less than 0,1 per cent) of copper and zinc, and traces of nickel were deposited, but a gross amount of mercury originating from amalgamation (2,8 per cent) was coplated with the gold. For leach liquors approximating 2 g/l gold, the cost of electrolysis plus smelting to yield bullion was estimated to be approximately R1,22 per kilogram of gold recovered for a large scale of operation. The cost of power for electrolysis was approximately 0,3 cents per kilogram of gold.

### Discussion

The results presented in the investigation strongly suggest that an alternative cyanidation route for the treatment of gold-plant concentrates is technically feasible, provided effective agitation and oxygenation of the system are afforded. In the case of lower-grade concentrates, cyanidation followed by the electrowinning of gold would appear to be the simplest route, while, for high-grade concentrates, flotation and smelting of a concentrate followed by the cyanidation of flotation tailings are suggested. Again, the use of oxygen rather than of air for sparging the pulp will depend largely on the grade of the material to be treated.

A preliminary study of the costs of the amalgamation and cyanidation processes for the recovery of gold from gravity concentrates was also undertaken at these laboratories. For the sake of comparison, a plant treating approximately 2 t of concentrates per day was considered. The total capital costs (1977) were estimated to be as follows:

Amalgamation	R247 000
Cyanidation	R293 000
Cyanidation plus flotation from reactor	R294 000
Cyanidation plus separate flotation circuit	R328 000.

The following operating costs per ton of concentrate (1977) were estimated:

Amalgamation	R26,4
Cyanidation (air)	R24,0
Cyanidation (oxygen)	R26,6
Flotation plus cyanidation (air)	R24,4
Flotation plus cyanidation (oxygen)	R26,4.

Thus, while capital costs favour the amalgamation route, operating costs favour cyanidation, although differences in operating costs are relatively small. However, small additional costs would be incurred in the electrowinning of gold from solution in the cyanide routes, which would reduce still further the differences

in operating costs for the alternative processes. It was concluded that, in the selection of the best route for the treatment of concentrates, other considerations, e.g. toxicity control, safety, and lower labour requirements, are more important than the small cost differences. Furthermore, the cyanidation route and the flotation plus cyanidation route are expected to achieve a gold extraction of more than 99,5 per cent. This is somewhat higher than for amalgamation, which is claimed to extract 96 to 97 per cent of the gold, the balance being partially extracted elsewhere in the circuit. Amalgamation has the further disadvantage compared with the other routes of involving a fairly large and uncertain circulating load in the gold-plant gravity circuit; 'opening' this circuit would considerably reduce the capital and operating costs for the cyanidation route because of the decreased tonnage to be treated.

While many of the aspects related to the current investigation concerned the possible cementation of gold onto the large amount of tramp iron present in the material studied, the subsequent 'opening' of the gold-plant gravity circuit may well change the composition of such concentrates considerably. This aspect alone could modify the conclusions drawn from the present investigation. Only a full-scale plant operation with careful monitoring of the process variables and the composition of the material treated would provide all or some of the answers necessary for such an appraisal.

### Sequel

As a result of the investigation described here, a full-scale plant for the intensive cyanidation of gold-plant gravity concentrates was commissioned at the Welkom Gold Mining Company in March 1977. At the time of writing, average gold recoveries of more than 99 per cent were indicated, with gold in the leach residues running at 10 to 50 g/t.

### Acknowledgement

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