The 'lock-up' of gold in run-of-mine mills at Randfontein Estates*

by A.B. PHILLIPS† and D.B. PATERSON‡

SYNOPSIS

In view of decreasing head grades, the relatively high gold price, and recent developments in metal evaluation, an exercise was initiated to determine the extent of gold concentration, the mode of occurrence, and the areas of concentration within the run-of-mine milling circuits of Randfontein Estates Gold Mining Company (Witwatersrand) Limited.

The total gold content of three of the thirteen run-of-mine mills in the Group was measured as a typical lock-up of approximately 200 kg per mill. The majority of this gold was concentrated either behind the grid-type body liners or within the pockets of the body liners.

This paper is an account of the experimental procedure and the results obtained, and highlights possible means of limiting gold lock-up in future.

SAMEVATTING

'n Steeds verlagende goud erts graad, die relatief hoë goud prys en onlangse vooruitgang in metaal waardering, het 'n ondersoek genoodsaak om die mate en wyse van goud konsentrasie, asook die lokalisering daarvan te bepaal binne die semi-outogene vermaling sisteme van Randfontein Estates Gold Mining Company (Witwatersrand) Limited.

Die totale goud inhoud in drie van die dertien semi-outogene meulens is bepaal en 'n tipiese konsentrasie van ongeveer 200 kilogram per meul getoon. Die goud het hoofsaaklik agter die rooster tipe meulvoerings asook binne die voeringholtes gekonsentreer.

Hierdie referaat behandel die eksperimentele prosedure, toon die resultate verkry en beklemtoon die toekomstige moontlikhede om goud opbouing te beperk.

INTRODUCTION

The Randfontein Estates Gold Mining Company (Witwatersrand) Limited operates three metallurgical complexes for the recovery of gold and uranium from the Witwatersrand gold reefs in the town of Randfontein, 40 km west of Johannesburg. Mining operations in the area started in March 1889, and ceased temporarily in 1965 when the viable ore reserves had been depleted. Exploratory drilling in the 1960s showed viable deposits of gold and uranium in the Elsburg Reefs[1], south of the Witpoortjie Horst Block, which formed the southern limit of the previously exploited areas.

Three shaft complexes known as the Cooke Section were subsequently developed to exploit this area. The Cooke Recovery Plant was commissioned in 1978 to extract gold and uranium from these reefs based on conventional forward-leaching practices, the comminution of the ore being carried out in (4,27 m dia. by 7,32 m) run-of-mine mills. The Millsite Gold Plant, originally commissioned in 1911, has been modernized with the recent installation of two (4,27 m dia. by 10 m) run-of-mine mills. Five of these mills have also been installed at the new Doornkop carbon-in-pulp gold-recovery plant.

Discrepancies have for many years existed between the measured head grades at the shaft and those at the plant,

being based on underground chip sampling and thickener-underflow measurements respectively. These measurements have invariably indicated that the head grade at the plant is lower than that at the shaft.

In an attempt to improve metal-evaluation standards from the face to the final product, additional sampling points were introduced at the shaft head and at the plant receiving bins in 1984 (Fig. 1). These samples offered a means of control over reef dilution by waste rock at the shafts, and gave an indication of the actual grade received at each of the plants. The method used for the sampling of these streams was manual stop-belt sampling[2]. Although scepticism exists regarding the accuracy of this method, it remains the only practical means of sampling large tonnages of run-of-mine ore.

The introduction of these control measures indicated a continual variance between the grade of the ore received at the plant receiving bins and the head grade at the plant as measured on the thickener-underflow pulp streams. This variance was particularly evident at the Millsite Plant, where a new run-of-mine milling installation was being commissioned. This suggested that, unless one or other measurement was biased, large quantities of gold were being locked-up in the Number 1 run-of-mine mill.

The phenomenon of gold lock-up in milling circuits is well known. As early as 1930, a paper entitled 'The Absorption of Gold in Tube Mills' was presented by White[3]. While various estimates of the quantity of gold absorbed have been suggested, no physical measurement of total lock-up had been carried out in the mines of the Johannesburg Consolidated Investment Company (JCI) Gold Division.
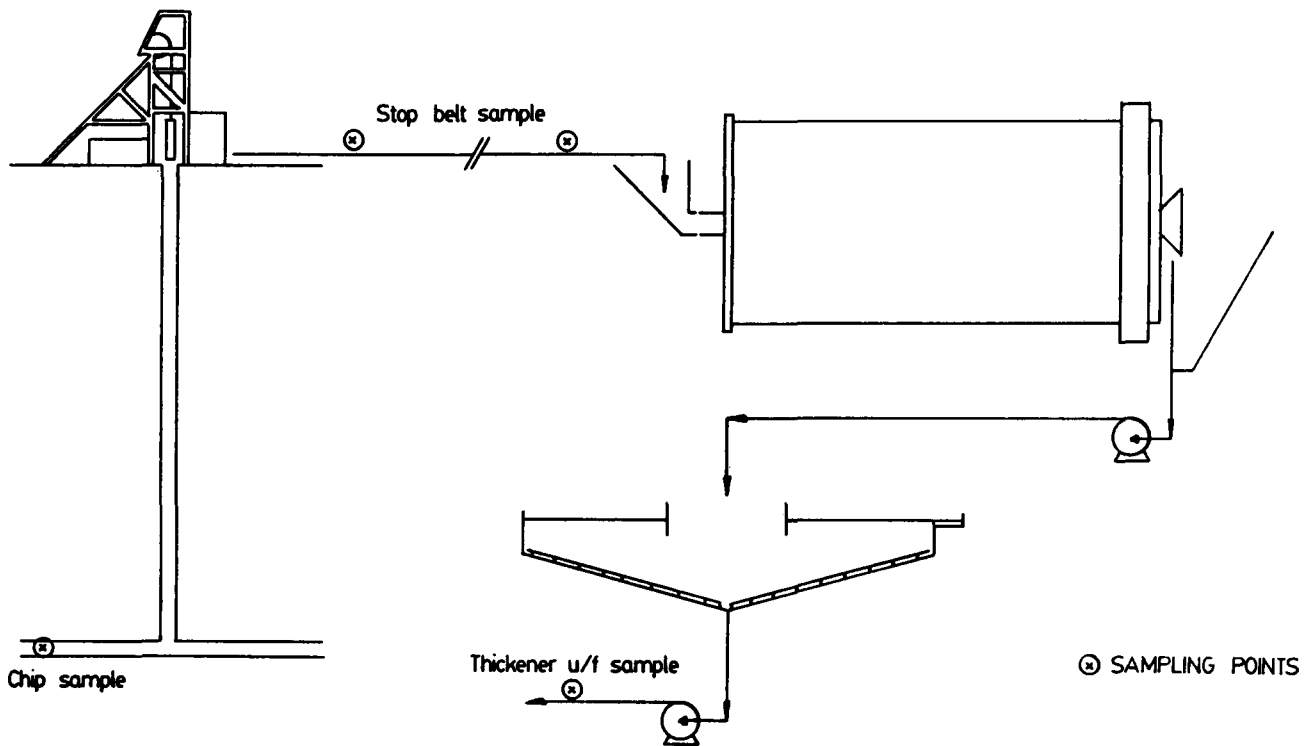
Fig. 1—Schematic diagram showing sampling points

An exercise was initiated in September 1986 to strip the liners from the Number 1 run-of-mine mill at the Millsite Plant, and to evaluate and recover the gold from its contents. This paper gives an account of the results and subsequent exercises, and suggests ways of limiting the quantity of gold lock-up in run-of-mine mills.

MILLING PRACTICE

Randfontein Estates operates thirteen run-of-mine mills, each operating in closed circuit with a single-stage hydrocyclone, as shown in Fig. 2.

Ore is fed to the mills by means of vibratory feeders, which discharge ore onto the conveyor belts carrying mill feed. The mill product is discharged into a sump, from which the diluted pulp is pumped to a single-stage hydrocyclone. The cyclone underflow reports back to the mill, while the overflow is discharged via an automatic cross-stream sampler onto a chip screen. The oversize from the chip screen is returned to the mill, and the undersize reports to the thickening circuit.

Three basic instrumentation loops control the mill operations:

(1) the discharge sump is maintained at a constant level by the addition of water from the mill circuit,

(2) the cyclone volume flow is measured by a flowmeter, the output of which controls the variable-speed cyclone-feed pump, and

(3) a load cell situated below the mill-inlet trunnion bearing measures the mass in the mill and controls the feed rate of the ore by altering the vibration amplitude of the vibratory feeder.

Typical operating parameters are given in Table I, the individual mill specifications being listed in Table II.

TABLE I
TYPICAL MILL OPERATING PARAMETERS

Operation	Magnitude
Flowrate of cyclone feed volume	700 to 800 m³/h
Relative density of cyclone feed pulp	1,35 kg/l
Relative density of cyclone overflow pulp	1,05 to 1,15
Relative density of mill-discharge pulp	1,75 to 1,90
Feed rate of run-of-mine ore	55 to 75 t/h
Ratio of circulating load	3:1

Number 1 Run-of-Mine Mill at Millsite

Based on previous random sampling of the gold concentrates within a run-of-mine mill, it was generally accepted that most of the gold would occur between the backing plates and the mill shell (Fig. 3). In order to confirm this and to obtain the maximum amount of information, the following conditions were determined for the exercise.

(1) The mill was not flushed before the liners were removed (as is normal practice prior to shutdowns) but was operated at typical control conditions (i.e. maximum power draw) until shutdown.

(2) The mill contents were sampled in such a manner that the following could be determined:

   (a) the total gold content;

   (b) the distribution of gold behind the liners, in the liner pockets, in the feed and discharge-end liners, and in the mill charge;

   (c) the distribution of gold behind the liners down the length of the mill;

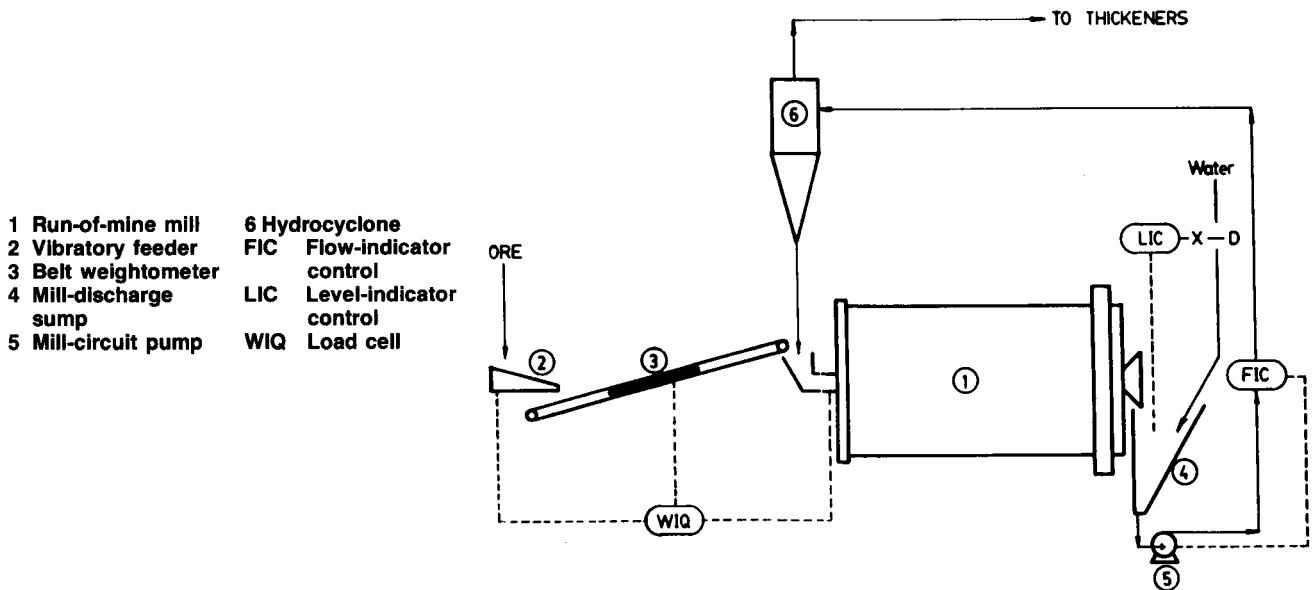   (d) the distribution of gold in the mill load as a function of particle size; and

Fig. 2—A typical run-of-mine milling circuit

1 Run-of-mine mill
2 Vibratory feeder
3 Belt weightometer
4 Mill-discharge sump
5 Mill-circuit pump

6 Hydrocyclone
FIC Flow-indicator control
LIC Level-indicator control
WIQ Load cell

TABLE II

MILL SPECIFICATIONS

	Cooke Plant	Millsite Plant	Doornkop Plant
Number of units	6	2	5
Supplier	3 ×Vecor; 3 ×Kobe	Vecor	Vecor
Size, m	4,27 dia. × 7,32	4,27 dia. × 10	4,27 dia. × 10
Type of mill	Trunnion discharge	Trunnion discharge	Trunnion discharge
Design of body liner	Solid liners in 1 mill		Solid liners in 1 mill
	Grid liners in 5 mills	Grid liners	Grid liners in 4 mills
Monthly rated capacity, t	50 000	40 000	40 000
Installed power, kW	1760	1760	2280
Power utilisation, %	75	90–100	50–60
Feed rate, t/h	75 to 80	70	70
Critical speed, %	90	90	90
Mill speed, r/min	18,75	18,75	18,75
Addition rate of steel balls, kg/t	0,5	0,2	0–0,2
Diameter of steel balls, mm	100	100	100
Diameter of cyclone, mm	1100	1100	1100
Supplier of cyclone	Spargo	Spargo	Spargo
Cyclone-feed pumps	2-stage—1 fixed speed —1 variable speed	1-stage—variable speed	1-stage—variable speed
Screen for wood chips, m	1,52 × 3,6 vibratory	1,76 × 4,56 vibratory	12 m² linear screen
Screen aperture wood chips, mm	3	1	0,75
Type of ore storage	Open stockpile	1 silo/mill	1 silo/mill
Capacity of ore storage, t	≈ 30 kt live	≈ 3 kt live/silo	≈ 3 kt live/silo
Mill product, %: >74 μm	63	75–80	75–80
<150 μm	9	1–3	1–3
Process control system	Outokumpu–Proscon	Yewpack–Hewlett Packard	Yokogawa–Centum

(e) the particle-size distribution of the steel-ball charge within the mill.

The lock-up evaluation exercise was scheduled to coincide with planned maintenance on the ore-storage silo, which was expected to take four weeks. This allowed ample time for accurate and efficient sampling and screening.

Method

The mill was shut down on 1st September, 1986, and the load was discharged through the mill door into a prepared paddock lined with filter cloth, care being taken to ensure that all the fines were contained in this paddock. The charge was then covered with plastic to prevent contamination by concentrates falling from the mill as the liners were removed. The steel balls and concentrates wedged into all the liner pockets were then loosened, bagged, labelled, and removed from the mill. The body liners were then removed systematically, the concentrates from each of the eight rows of liners down the length of the mill being recovered separately. The liners were loosened, and the concentrates locked between the liner
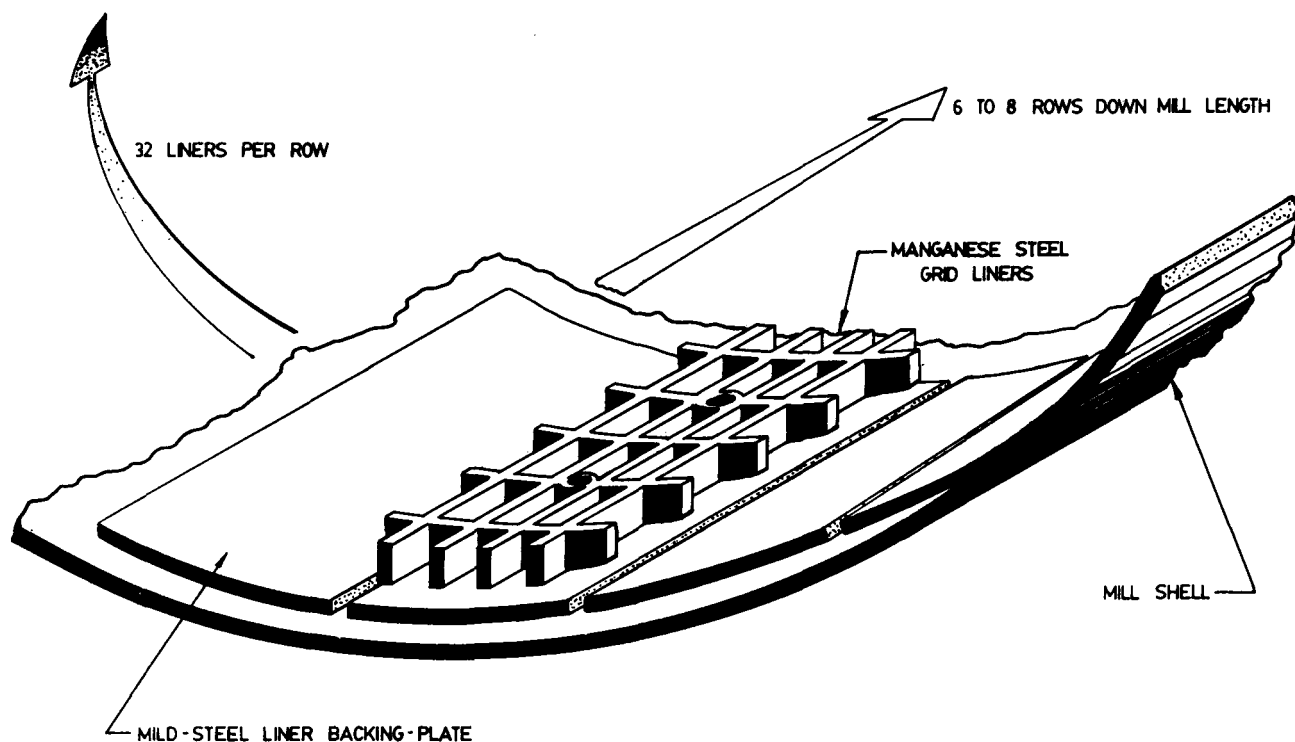
Labels on figure:
32 LINERS PER ROW
6 TO 8 ROWS DOWN MILL LENGTH
MANGANESE STEEL GRID LINERS
MILL SHELL
MILD-STEEL LINER BACKING-PLATE

Fig. 3—Schematic diagram showing the arrangement of the mill liners

and its backing plate, and between the backing plate and the mill shell, were removed, bagged, and labelled. All the concentrates adhering to the liner grid and the backing plate were removed by means of a wire brush. Once all the body liners had been taken out, the feed- and discharge-end liners were removed and the concentrates recovered in a similar manner. All the concentrates were stored in the gold smelthouse to prevent possible theft.

These concentrates were separated into fractions as listed below:

(i) concentrates from the grid-liner pockets,

(ii) concentrates locked between the liner and the backing plate, and between the backing plate and the mill shell, for each of the eight rows down the length of the mill,

(iii) concentrates from behind the discharge-end liners, and

(iv) concentrates from the feed-end liners.

These composite samples were dried and weighed and, after thorough mixing, they were levelled to a layer 0,3 m thick on a plastic sheet.

A grid of 0,1 m by 0,1 m spacing was core-sampled, and four 2 kg samples were prepared. Each sample was analysed to determine the average grade of the composite samples.

The bulk mill charge was wet-screened, using a trommel screen equipped with screen panels of 162, 82, 62, 25, and 6 mm. The minus-6 mm sand fraction was collected within a bund area, and water was drained from the material. The various screen products were then containerized, weighed, and transported to the JCI Minerals Processing Research Laboratory in Germiston for sampling and analysis, which were carried out as illustrated in Fig. 4.

Results

A total mass of 140,9 t of material was recovered from the mill, of which 15,4 t was steel balls and tramp iron. The gold content of the mill was 261 kg, of which 89,6 per cent was recovered from behind the mill liners and backing plates.

Details of the load mass and gold distribution are given in Table III, and the gold distribution is illustrated graphically in Fig. 5. The distribution of gold locked up behind the liner backing plates is given in Table IV, which is illustrated graphically in Fig. 6.

Number 1 Run-of-Mine Mill at Doornkop

In view of the massive scale of gold lock-up revealed at Millsite Plant, a second exercise was conducted at Doornkop Plant in January 1987. The purpose of this exercise was to quantify the gold locked up within the charge, within the liner pockets, and behind the liner backing plates. In addition, gold concentrates behind the backing plates would be recovered and sent to the by-products section of the Rand Refinery. In broad terms, the methods employed were similar to those used at Millsite, although the sampling was simplified in that no attempt was made to screen the charge into size fractions. Gold trapped between the mill liner and its backing plate was not included with the concentrates removed from behind the backing plates, but was included with the material from the liner pockets.

Results

A total of 191 kg of gold was recovered from the mill, of which only 38 per cent was found behind the liner backing plates. The bulk of the gold (61,5 per cent) had been locked up in the pockets of the mill liners. Details
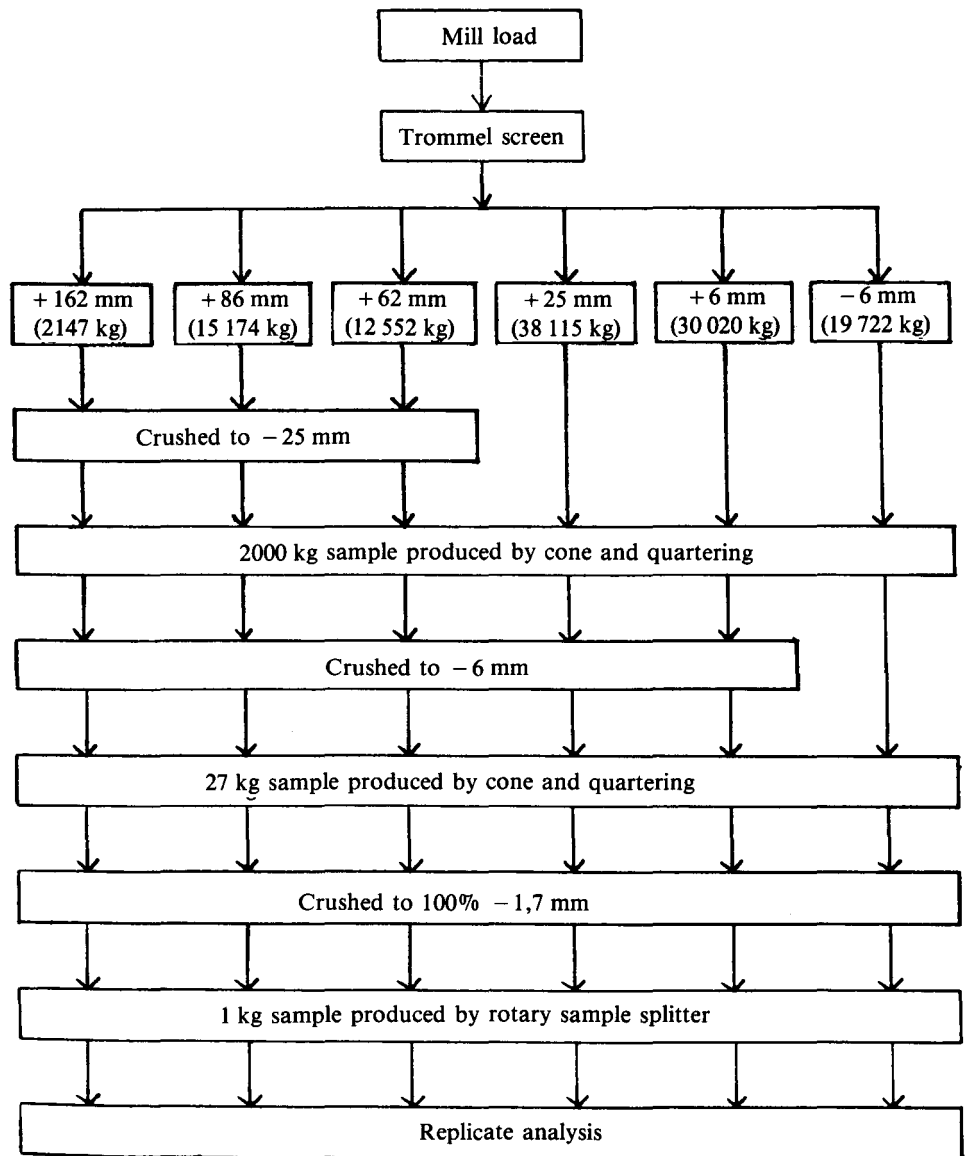
Fig. 4—Schematic flowsheet for the treatment of the load in the Number 1 run-of-mine mill at Millsite

Mill load → Trommel screen

+162 mm (2147 kg), +86 mm (15 174 kg), +62 mm (12 552 kg), +25 mm (38 115 kg), +6 mm (30 020 kg), −6 mm (19 722 kg)

Crushed to −25 mm

2000 kg sample produced by cone and quartering

Crushed to −6 mm

27 kg sample produced by cone and quartering

Crushed to 100% −1,7 mm

1 kg sample produced by rotary sample splitter

Replicate analysis

Behind backing plates and between liners and backing plates

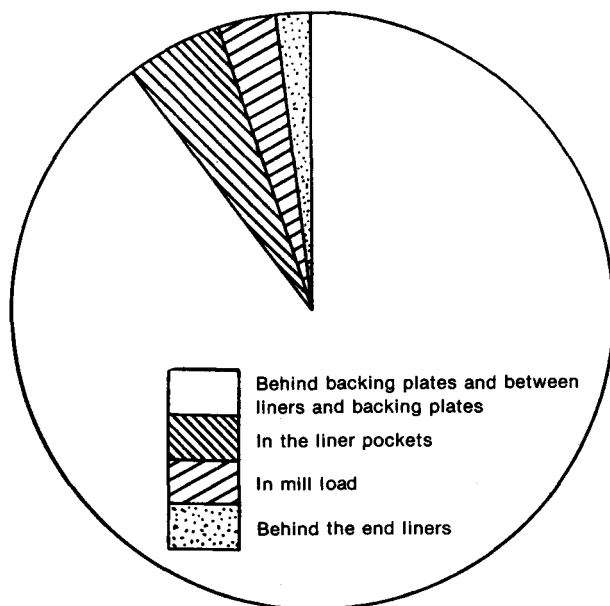In the liner pockets

In mill load

Behind the end liners

Fig. 5—The distribution of gold in the Number 1 run-of-mine mill at Millsite

of the gold distribution are given in Table V, and the results are illustrated graphically in Fig. 7.

Number 2 Run-of-Mine Mill at Doornkop

In an attempt to control gold lock-up, four of the eight rows of backing plates in the Doornkop Number 2 run-of-mine mill were rubberized on the side in contact with the mill shell, a 6 mm layer of 35 Shore A hardness rubber being used. The marked difference in gold distribution between the Millsite and the Doornkop Number 1 mills gave rise to fears of possible contamination of the load removed from the Doornkop mill, and it was decided to evaluate the load of Number 2 run-of-mine mill in February 1987. In this case, an additional objective was to evaluate the effectiveness of rubberized backing plates as a possible means of limiting gold lock-up.

Results

This exercise confirmed the gold distribution found in Number 1 run-of-mine mill at Doornkop. A total of 199 kg of gold was extracted from the mill, of which 39,8 per cent was removed from behind the liner backing plates

TABLE III
MASS OF CHARGE AND DISTRIBUTION OF GOLD IN THE NUMBER 1 RUN-OF-MINE MILL AT MILLSITE

Fraction	Subfraction	Mass kg	% of fraction	% of total mass	Gold grade g/t	Gold content kg	% of total gold content
Steel balls	+90 mm	3 260	21,1	2,3	–	–	–
	−90 mm +80 mm	3 640	23,6	2,6	–	–	–
	−80 mm +60 mm	4 540	29,4	3,2	–	–	–
	−60 mm +12 mm	2 000	13,0	1,4	–	–	–
	Tramp iron	2 000	13,0	1,4	–	–	–
Sub-total		15 440	100,0	10,91	Nil	Nil	Nil
Pebbles	+162 mm	2 150	1,8	1,5	2,76	0,01	0,004
	−162 mm +82 mm	15 170	12,9	10,7	2,18	0,03	0,01
	−82 mm +62 mm	12 550	10,7	8,9	4,21	0,05	0,02
	−62 mm +25 mm	38 120	32,4	26,9	3,61	0,14	0,05
	−25 mm +6 mm	30 020	25,5	21,2	3,79	0,11	0,04
Sand	−6 mm	19 720	16,8	13,9	4,23	0,08	0,033
Sub-total		117 730	100,0	83,19	73,76	0,44	0,204
Concentrates	Behind body liners	5 870	70,4	4,2	39 800	234,0	92,54
	Behind feed end liners	100	1,3	0,1	17 300	1,82	0,70
	Behind discharge end liners	210	2,6	0,2	13 200	2,81	1,11
	In liner pockets	2 150	25,8	1,5	6 400	13,78	5,45
Sub-total		8 340	100,0	5,89	30 200	252,41	99,83
Grand total		141 515		100,0	1844,8	252,85	100

TABLE IV
DISTRIBUTION OF GOLD CONCENTRATES BEHIND LINER BACKING PLATES BY ROW

Row number	Mass distribution			Gold assay		Gold distribution	
	Mass (kg)	% of total mass	Cumulative % of total mass	g/t	Total gold kg	% of total gold	Cumulative % of total gold
1 (Feed)	577,8	9,8	9,8	40 200	23,24	9,9	9,9
2	447,2	7,6	17,4	63 100	28,22	12,1	22,0
3	527,5	9,0	26,4	75 100	39,60	16,9	38,9
4	629,0	10,7	37,1	58 100	36,56	15,6	54,5
5	634,3	10,8	47,9	30 200	19,15	8,2	62,7
6	1087,9	18,5	66,4	25 400	27,66	11,8	74,5
7	825,5	14,1	80,5	34 500	28,47	12,2	86,7
8 (Discharge)	1143,9	19,5	100,0	27 200	31,11	13,3	100,0

TABLE V
MASS OF CHARGE AND DISTRIBUTION OF GOLD IN THE NUMBER 1 RUN-OF-MINE MILL AT DOORNKOP

Fraction	Dry mass kg	Mass distribution %	Gold grade g/t	Gold mass kg	Gold distribution %
'In mill' load	65 619	80,1	5	0,3	0,2
Concentrates in liner pockets	14 659	17,9	8 000	117,3	61,4
Concentrates behind liner backing plates	1 665	2,0	46 000	73,3	38,4
Total	81 886	100,0	2 340	190,9	100,0

Fig. 6—The distribution of gold behind the liners in the Number 1 run-of-mine mill at Millsite

— · — · — ·   Gold content, %
— — — —   Total mass, %
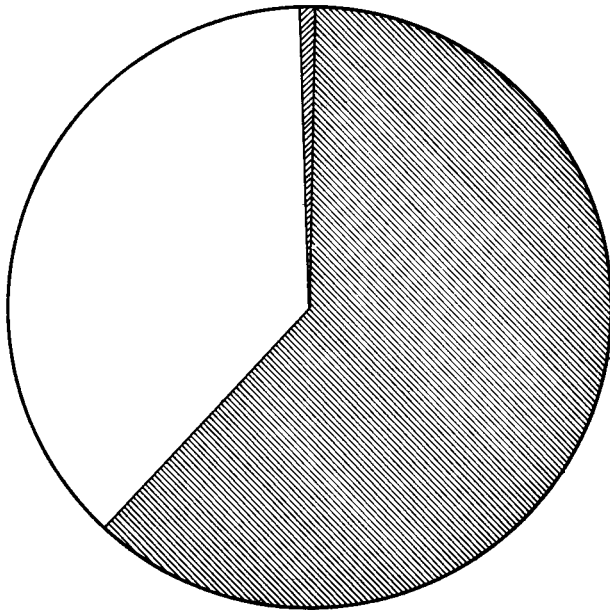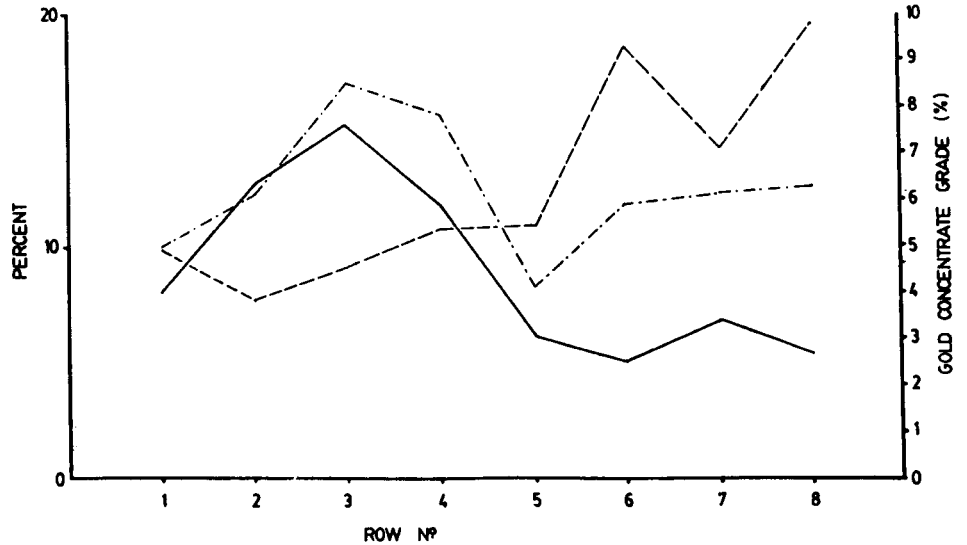———————   Grade of gold concentrate, %

Fig. 7—The distribution of gold in the Number 1 run-of-mine mill at Doornkop

☐ Behind liner backing plates

▨ In liner pockets

▨ In mill load

and 58,4 per cent was recovered from the liner pockets. Details of the gold distribution are given in Table VI, and the results are presented graphically in Fig. 8. Of particular interest was the fact that the use of rubberized backing plates reduced the mass of concentrates per row by 26 per cent but, as a result of the increased grade of the concentrate, the net gold lock-up per row of rubberized backing plates increased by 45 per cent.

MINERALOGY[4,5]

In order to understand the mechanism of gold concentration in run-of-mine mills, five samples of concentrates were submitted to the mineralogical section of the JCI Minerals Processing Research Laboratory for detailed analysis. These samples are listed below:

Fig. 8—The distribution of gold in the Number 2 run-of-mine mill at Doornkop

☐ Behind the normal backing plates

▨ Behind the rubber-lined backing plates

▨ In the liner pockets

░ Behind the end liners

(i) concentrate from behind the liners at Millsite Number 1 mill,
(ii) concentrate from behind the normal liner backing plates,
(iii) concentrate from behind rubberized backing plates,
(iv) concentrate from behind the end liners, and
(v) concentrate from pockets in the mill liners.

Samples (ii) to (v) were from Doornkop Number 2 run-of-mine mill.

Analysis by X-ray-diffraction, ore-microscope, and electron-microprobe techniques showed the Millsite sample to consist predominantly of aggregates of tramp iron, gold, and pyrite cemented in a matrix of the iron oxide hematite (Fe_2O_3).

TABLE VI

GOLD DISTRIBUTION IN THE LOAD OF THE NUMBER 2 RUN-OF-MINE MILL AT DOORNKOP

Fraction	Sub-fraction	Mass kg	Mass distribution %	Mean mass per row kg	Gold analysis, %			Gold mass kg	Gold distribution	Average mass per row kg Au	Average mass per row % of total Au
					Assay 1	Assay 2	Average assay 3				
Locked behind	Row 1	203,3	1,4		4,13	3,91	4,02	8,2	4,1		
non-rubberized	Row 2	235,9	1,6		3,66	4,46	4,06	9,6	4,8		
backing plates	Row 7 (1½ rows)	276,5	1,9		3,43	2,63	3,03	8,4	4,2		
	Row 8	130,5	0,9		6,08	6,13	6,11	8,0	4,0		
	Door liner	7,0	0,1		8,45	8,96	8,71	0,6	0,3		
Subtotal		853,2	5,9	189,6			4,07	34,8	17,4	7,7	3,9
Locked behind	Row 3	159,6	1,1		8,85	9,06	8,96	14,3	7,2		
rubberized	Row 4	147,0	1,0		7,57	8,38	8,00	11,7	5,9		
backing	Row 5	134,6	0,9		7,64	7,57	7,61	10,2	5,1		
plates	Row 6	122,2	0,8		7,10	6,90	7,00	8,6	4,3		
Subtotal		563,4	3,8	140,9			8,0	44,8	22,5	11,2	5,6
Inlet end	Behind liners	67,0	0,5		0,91	1,06	0,99	0,7	0,3		
Liners	Behind filler rings	24,6	0,2		4,77	4,33	4,55	1,1	0,6		
Subtotal		91,6	0,7				1,94	1,8	0,9		
Discharge-end liners	Behind liners	99,7	0,7		1,05	1,27	1,16	1,2	0,6		
	Behind spacer-box segments	46,2	0,3		1,23	1,69	1,46	0,7	0,3		
Subtotal		145,9	1,0				1,26	1,9	0,9		
Liner pockets		13 120,7	88,6		0,89		0,89	116,6	58,3		
Total		14 774,8	100,0				1,35	199,9	100,0		

Apart from the quartz and tramp iron, all the minerals present in the Millsite samples were constituents of the heavy-mineral assemblage of the run-of-mine ore. These minerals included sulphides such as chalcopyrite, arseno-pyrite, and pyrrhotite, as well as oxides such as chromite, magnetite, and uraninite. On average, the heavy minerals constitute only 3 per cent of the matrix of the mined ore, so that the apparent concentration of these heavy minerals in the mill-liner concentrates is significant.

Most of the gold detected in the sample consisted of liberated, slightly elongated grains, with an average length-to-breadth ratio of 3 to 1. These grains occurred together with pyrite and particles of heavily oxidized tramp iron in aggregates cemented by hematite. Less than 1 per cent of the gold occurred as inclusions within sulphide minerals, although a fairly common feature of the gold grains was the presence of small sulphide inclu-sions round the edges. It is possible that these had become embedded during milling.

The analyses of gold grains in the Doornkop samples for size and shape are given in Table VII.

In general, the mineralogical composition of these concentrates was similar to that at Millsite, but cementation of the minerals into aggregates had occurred only to a limited degree.

The concentrate trapped behind the rubberized backing plates was found to contain more magnetic material and considerably less coarse pebbles than the concentrate that was found behind the normal backing plates.

DISCUSSION

Measurements of the gold concentration in the Number 1 run-of-mine mill at the Millsite Plant and the Numbers 1 and 2 run-of-mine mills at the Doornkop Plant indicate

TABLE VII

SHAPES OF THE GOLD GRAINS, AND THEIR AVERAGE AND MAXIMUM SIZES, IN DOORNKOP CONCENTRATES

Sample	Total number of grains	Shape, %					Grain size, μm	
		Elongate	Angular	Rounded	Irregular	Total	Average	Maximum
Rubberized backing plates	3 334	46	8	20	26	100	75×35	515×325
Normal backing plates	2 349	42	9	22	27	100	70×25	675×225
End liners	801	39	10	24	27	100	40×25	270×185
Liner pockets	461	40	10	26	24	100	32×15	205×75

a substantial degree of concentration, which, at approximately 200 kg per mill, is greater than was originally expected.

The majority of gold present in the run-of-mine mill at Millsite was located between the body liner and the mill shell. The major area of concentration in the Doornkop mills was in the pockets of the grid-type body liners. This anomaly can be attributed to a difference in operating practices. At the Doornkop Plant, it is standard practice to remove the backing plate whenever body liners are replaced, thereby releasing any concentrates present between the backing plate and the mill shell. At the Millsite Plant, only the body liner was replaced, and the backing plate was not disturbed. A standard procedure has now been adopted, in which the backing plate is removed and the mill shell is cleaned by a high-pressure water spray before the body liners are replaced.

It is evident that the mechanism resulting in the concentration of gold in or behind the body liners is gravity concentration. Centrifugal forces acting on the particles within the charge increase the effective settling rate, resulting in a concentration of the heavier particles towards the periphery of the mill shell. Apart from quartz, all the minerals present in the samples of concentrates taken behind the body liners form constituents of the 'heavy mineral' assemblage of run-of-mine ore. Typically, 'heavy minerals' constitute only 3 per cent of the matrix of run-of-mine ore.

It is evident that any cracks or voids round the periphery of the mill will trap and concentrate gold and associated dense material. Evidence from the Doornkop Plant indicates that narrowing of the gap behind the mill liners results in preferential concentration of the gold in place of the other associated gangue material. This results in a decrease in bulk, but an increase in gold concentration, of the locked-up material.

The measurement of the gold concentration behind the individual rows of body liners at the Millsite Plant indicated a definite concentration pattern, with maximum lock-up occurring behind body-liner rows numbers 2, 3, and 4 (Fig. 6).

As the gold is finely disseminated in the run-of-mine ore, liberation may occur only after multiple passes through the mill. As a result of this, and the fact that the gold is concentrated in the cyclone underflow owing to its high density, the gold grade of the circulating load is typically about ten to fifteen times higher than that of new feed.

As the circulating load re-enters the mill-feed end, the heavy material settles and concentrates from the feed end to the discharge end. The lower relative density of the pulp at the feed end than that at the discharge end is also a contributing factor to the settlement of the gold in this area.

Rows numbers 2, 3, and 4 are the body liners with the highest wear rate within the mill (Fig. 9). This is the area where maximum impact occurs, and it is considered relevant that the maximum concentrates of gold are also found in this area. The presence of a uniform 10 to 15 mm thick layer of concentrate behind the body liners indicates that, during milling operations, the body liners/backing plates drift away from the mill shell. The rate of movement could be associated with the degree of impact that is taking place.
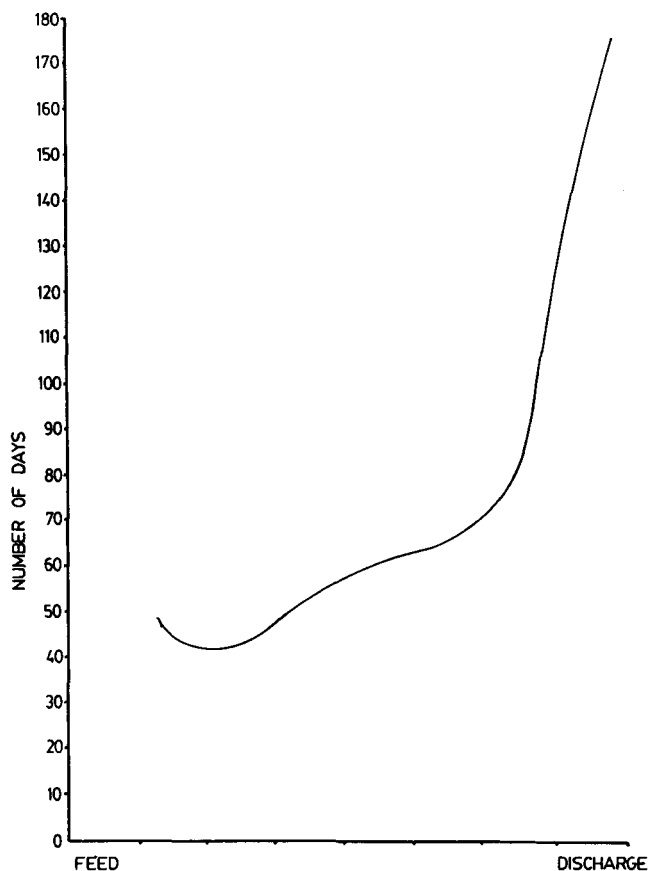

Fig. 9—The life of run-of-mine mill liners

Run-of-mine milling operations depend on the following variables:

● relative density of the pulp in the mill,
● feed rate,
● head grade of the run-of-mine ore,
● volumetric loading of the mill charge,
● addition rate of steel, and
● fragmentation of the run-of-mine ore.

These must have some influence on the lock-up of gold and hamper an understanding of the mechanisms taking place.

METHODS OF CONTROLLING GOLD LOCK-UP

The results of the studies to date indicate that there are four broad areas at which methods to control lock-up could be aimed:

● reduction of gold circulating loads,
● reduction of gold lock-up in the liner pockets and the mill load,
● reduction of gold lock-up behind and between the liners, and
● continuous recovery of the locked-up gold.

Gold Re-circulating Loads

Studies to date within the JCI Group on metallurgical plants that operate a concentration stage for the removal of precious metals ahead of cycloning have shown that the lock-up of precious metals within the milling circuits is minimal. This appears to indicate that the inclusion of a gravity and/or a flotation circuit for the recovery of

free gold from the cyclone feed stream would be effective in reducing the gold recirculating load to the mills, which, in turn, results in a significant decline in the gold lock-up. Such circuits are expensive to construct and operate, and present severe security risks. As a result, alternative methods of control are being investigated.

Liner Pockets and Mill Load

Given the gravity-concentration properties of gold, it is unlikely that anything can be done to prevent liberated gold from concentrating within a mill at the liner–charge interface. Measures can be aimed at maximizing the subsequent size reduction of the gold particles and ensuring maximum release of this gold during flushing, at start-up and shutdowns for example. The grid liners of manganese steel currently used effectively shield this gold from further comminution and prevent efficient flushing. In view of the successful replacement of grid liners in the gold-plant run-of-mine mills at Vaal Reefs Number 9 Shaft with solid white-iron liner blocks, a set of these liners has been installed in the Number 5 run-of-mine mill at the Doornkop Plant. A further set of solid liners will shortly be installed at the Cooke Plant. This design of liner incorporates a tapered solid-brick type of liner locked into position with a combination key and lifter-bar arrangement. This should enhance the attrition of the gold concentrates at the liner–charge interface, and enable more of the locked-up gold to be flushed from the mill during start-up and shutdown procedures.

Initial indications from the Number 5 mill at Doornkop are that the total gold locked up within the mill has been significantly reduced by the use of these liners. It will be some months before the true effect of the liners can be accurately gauged. Current indications are that the total lock-up will be below 50 kg per mill.

Logic appears to indicate that routine flushing would be more efficient in a peripheral-discharge mill than in a mill fitted with screens and lifters. Mokken et al.[6] showed that the gold concentration in the mill discharge from a peripheral-discharge secondary mill after start-up was substantially lower than that in an identical mill fitted with lifters. This could indicate a lower initial degree of lock-up within the mill, and suggests an area for future investigation within the industry.

Behind and Between the Liners

The results of the testwork to date have shown that, for success to be achieved in this area, movement of the liners away from the mill shell must be eliminated, and all cracks, crevices, and pockets behind and between the liners must be sealed. A number of measures are currently being evaluated.

Cleaning behind Backing Plates during Relining

Provided that the replacement of high-wear liner grids is carried out systematically, a set of liner backing-plates should last through multiple sets of grid replacements. A comparison of the amount of gold locked up behind the liners in the Millsite and Doornkop mills and the mode of occurrence of this lock-up indicates that, unless the locked-up gold behind the backing plates is removed at each grid replacement, the layer of locked-up gold behind the liners will continue to increase in thickness, and

oxidation will result in the formation of cemented aggregates as was evident at Millsite. Current cleaning practice has effectively controlled locked-up gold behind the backing plate to below 80 kg per mill.

Rubberizing of Backing Plates or the Mill Shell

The results obtained from the Number 2 mill at Doornkop indicate that, unless additional measures are implemented to prevent drifting of the liner and its backing plate away from the shell, the use of rubberized backing plates or rubberizing of the shell will not be effective. The rubber layer prevents the coarse gangue from partially filling the voids, which in turn leads to a greater concentration of gold behind the backing plates. However, it is likely that the use of rubberized shells to seal voids, in conjunction with methods to minimize liner drift, will lead to the ultimate solution of the problem.

High-tensile Steel-body Liner Bolts

The liner bolts currently in use on all the run-of-mine mills are generally of a poor quality. No technical specification is stipulated, and the material used is EN8, which has a tensile strength of approximately 550 mPa. The bolts are manufactured from standard EN8 round bar, the head of the bar is deformed to the required profile, and the thread is machined with no subsequent controlled heat-treatment.

The concentrate present behind the body liners at Millsite occurred as a relatively uniform layer with a thickness of 10 to 15 mm. It was previously considered that the concentration behind the liners would occur in the voids and crevices present. The presence of this relatively thick layer implies that the liners are being allowed to drift away from the mill shell, thereby allowing concentrates to become embedded between the liner and the mill shell. The following factors could contribute towards this:

● deformation of the liner bolt,
● poor washer design, and
● poor tightening of the liner bolts during relining.

A technical specification has been drawn up that will be evaluated on one of the run-of-mine mills at the Cooke Plant. The material to be tested is EN26 that has been heat-treated to a hardness of approximately 300 B in a controlled atmosphere. In addition, a locking nut of high-tensile steel and a copper gasket will be utilized. The liner bolts will be tightened to 1000 mPa and re-tightened after 72 hours of operation.

Sealants

Preliminary trials have been conducted on the use of sealants to close the gap between the liner backing plates and so minimize the penetration of gold into the voids between the backing plates and the shell. Silicon rubber was evaluated at the Doornkop Plant but was found to be inadequate in resisting abrasion from the pulp. Nordback, a filler conventionally used in cone crushers, was tested on a limited scale at the Millsite Plant. Random sampling indicates that the lock-up of gold behind the backing plates has been reduced to below 20 kg per mill (25 per cent of that at Doornkop). This material is likely to withstand abrasion but is expensive, and its use would be labour-intensive and would result in a decrease in mill

availability.

Continuous Gold Recovery

Modifications to the milling circuit to allow for the addition of cyanide into the mills would probably result in a marked reduction in gold locked up within the mills, although there would be an increased lock-up of gold in the mill-circuit waters. Mineralogical examination of the concentrates from the mills indicates that this would have to be a continuous process, since the oxidized aggregates of gold that have been locked up are likely to be refractory to cyanidation. The potential of this method is currently being evaluated at the JCI Minerals Processing Research Laboratory. However, the use of this method could be considered only for the more modern plants such as Doornkop owing to the inherent risks of solution spillage from the mill-circuit waters.

CONCLUSION

The results obtained to date indicate that approximately 200 kg of gold is locked up in each of the run-of-mine mills at the Randfontein Estates Gold Mining Company. This results in a delayed cash flow and a loss of potential earnings. Several methods to minimise the lock-up of gold are being evaluated, and a plan of action has been formulated for future testwork. The incorporation of solid body liners and the use of liner bolts of high-tensile steel are considered to be the most effective measures at this stage.

ACKNOWLEDGEMENTS

REFERENCES

1. TUCKER, R.F. The sedimentology and mineralogy of the composite reef on Cooke section, Randfontein Estates Gold Mine, Witwatersrand, South Africa. M.Sc. thesis, University of the Witwatersrand, 1980. 289 pp.
2. BARTLETT, H.E., PIENAAR, A.H., and BRIGHT, D. Application of run-of-mine sampling of feed grade ores to the control and evaluation of surface and underground mining operations. GOLD 100. Proceedings of the International Conference on Gold. Johannesburg, The South African Institute of Mining and Metallurgy, 1986. vol. 1, pp. 63–72.
3. WHITE, H.A. The absorption of gold in tube mills. J. Chem. Metall. Min. Society S. Afr., vol. 31, no. 6. Dec. 1930. pp. 161–171.
4. DINHAM, P. Mineralogical examination of Millsite mill liner concentrates. Johannesburg, Johannesburg Consolidated Investment Co. Ltd, Minerals Processing Research Laboratory, unpublished report, 1987. 11 pp.
5. ROBERTSON, N. Mineralogical examination of gold concentrates from run-of-mine mill No. 2, Doornkop Plant, R.E.G.M. Johannesburg, Johannesburg Consolidated Investment Co. Ltd, Minerals Processing Research Laboratory, unpublished report, 1987. 6 pp.
6. MOKKEN, A.H., BLENDULF, G.K.I., and YOUNG, G.J.C. A study of the arrangements for pulp discharge on pebble mills, and their influence on mill performance. J.S. Afr. Inst. Min. Metall., vol. 75, no. 5. May 1975. pp. 257–289.

Deep-level mining and metallurgical equipment

The South African Institute of Mining and Metallurgy will be holding two colloquia on the above two topics. The Colloquia will be held in Johannesburg from 17th to 21st September, 1990, in conjunction with the Electra Mining Exhibition.

Technical Challenges in Deep-level Mining

The South African gold-mining industry is moving to progressively deeper working areas and prospects. Major planning work is being carried out for mining at significantly increased depths. The Colloquium is intended to include papers describing work done on these aspects by the local industry, as well as work on relatively deep operations worldwide. The problems involved will be identified, and progress made on providing suitable solutions to these problems will be discussed.

The following are some of the aspects that will be addressed:

Exploration, financing, access, mining methods, rock mechanics, ventilation, refrigeration, mechanization, backfilling, emergency provisions.

The Colloquium will be of particular interest to mining engineers and operators involved in deep mines and mines with similar problems.

Innovations in Metallurgical Equipment

Equipment for metallurgical processes is forever changing to cope with new ideas and demands. This Colloquium is aimed at recording the latest of these changes and bringing new equipment to the attention of practising metallurgists.

Papers covering recent developments in all spheres of extractive metallurgy from crushing to pyrometallurgy will be welcome.

The Colloquium will be of interest to anyone involved with the processes and maintenance of equipment associated with extractive metallurgy. A wide range of processing equipment will be on view at the exhibition to complement the presentations.

Enquiries

Pamela Binstead
SAIMM
P.O. Box 61019
Marshalltown 2107.

Tel. 832-2177. Telex 4-86431.

Analytical chemistry

The Analytical Division of the Royal Society of Chemistry is to hold SAC 89—An International Conference on Analytical Chemistry—at the University of Cambridge from 30th July to 5th August, 1989. This is the 9th in the series of triennial Conferences originated by the Society for Analytical Chemistry (hence SAC) in 1965.

Sponsorship by the Federation of European Chemical Societies (FECS event no. 122) and the International Union of Pure and Applied Chemistry (IUPAC) has been granted. IUPAC sponsorship implies that entry visas will be granted to all *bona fide* chemists provided application is made not less than three months in advance. If a visa is not granted one month before the meeting, the IUPAC Secretariat should be notified without delay by the applicant.

The scientific programme will be organized around plenary, invited, and contributed papers, which will take the form of both oral and poster presentations covering the whole field of analytical chemistry. As at previous conferences, special sessions on particular analytical themes will be organized by interested subject groups. Two regions of the Analytical Division and other organizations will also participate. The programme will include workshops, where research workers can demonstrate new apparatus and techniques. Update courses have been arranged to provide intensive all-day tutorial and practical demonstration sessions. These will be held on Wednesday, 2nd August, as an alternative to scientific or cultural visits.

The following plenary lectures will be given:

Professor J. Ruzicka (University of Washington, Seattle, USA): Enhancement of Instrumental Analysis by Flow Injection Techniques.

Professor W.H. Pirkle (University of Illinois at Urbana–Champaign, USA): Practical Applications of an Understanding of Chiral Recognition Requirements.

Professor Dr G. Kateman (Katholieke Universiteit, Nijmegen, The Netherlands): Evolutions in Chemometrics.

Professor A. Townshend (University of Hull, UK): Light and Life—Some Recent Applications of Chemiluminescence and Enzymes in Analytical Chemistry.

Many titles of papers have been submitted in response to the First Circular, but further contributions are invited. Papers should discuss original unpublished work. Titles, followed by names and addresses of contributors and abstracts (up to 200 words), must be set in double-spaced clear type, within the boundaries of 25 × 15 cm, on a single sheet of paper in a form suitable for publication in the Conference handbook. Three copies of each of these must be submitted before 31st October, 1988, to

Mr G.M. Telling
SAC 89 Scientific Programme Co-ordinator
c/o Analytical Division
Royal Society of Chemistry
Burlington House
London W1V 0BN.
UK.

Authors will be informed of the acceptance or rejection of submitted papers and whether presentation will be by lecture or poster by early February 1989. Manuscripts of accepted papers submitted by the end of August 1989 will be considered for publication in a special issue of *The Analyst*. Papers should include novel or relevant review material to satisfy the normal criteria for publication in *The Analyst*.

All presentations will be in English.

Provisional information concerning the scientific programme will be distributed with the registration forms early in 1989. Full details will be published in the March 1989 issue of *Analytical Proceedings*.

Dewatering

As a follow-up to the successful Dense Medium Operators' Conference in July 1987, the Southern Queensland Branch of The Australasian Institute of Mining and Metallurgy will host the Conference on Dewatering Technology and Practice in Brisbane from 9th to 11th October, 1989. The Conference is again in association with the Coal Preparation Society of Queensland. Technical contributions will have an emphasis on practical applications and operations, and will cover all aspects of solid–liquid separation.

An equipment exhibition will be held in association with the Conference.

The following topics will be discussed at the Conference:

Thickening (conventional and high rate)
Filtration (pressure, vacuum, and centrifuge)
Centrifuges

Cyclone dewatering
Dewatering screens
Drying
Flocculation
Pilot-scale operation
Plant design and operation
Instrumentation and control.

Information on all aspects of the Conference is available from

Dr Don McKee
Julius Kruttschnitt Mineral Research Centre
Isles Road
Indooroopilly, Queensland, 4068 Australia.
Telephone: (07) 378 1588
Telex: AA 40315
Facsimile: (07) 378 5775.