

The improved flotation of gold from the residues of Orange Free State ores

by P. A. J. CABASSI*, B. K. LOVEDAY*, B.Sc. Eng., Ph.D.,
F.S.A.I.M.M., P. H. RADCLIFFE†, B.A., M.A., (Met.),
M.S.A.I.M.M., and M. J. WILKINSON*, Hnc. (Met.)

SYNOPSIS

The recovery of gold, uranium, and pyrite from accumulated residues at mines in the Orange Free State is adversely affected by the presence of pyrophyllite, which is naturally floatable and must be prevented from floating with the desired constituents by the use of a mica depressant, and of thucholite (uraniferous kerogen), which contains significant quantities of uranium and gold but does not float readily in the presence of a mica depressant.

In laboratory tests on these residues, the use of a mixture of MIBC (methyl isobutyl carbinol) and paraffin in addition to the suite of reagents normally used was found to increase the gold recovery substantially, but the sulphur grade was lower. The recovery of thucholite also increased considerably, but the proportion of uranium associated with this mineral was small.

Plant trials, which are continuing, have been encouraging, indicating an increase in the gold recovery of 4,6 per cent.

SAMEVATTING

Die herwinning van goud, uraan en piriet uit die geakkumuleerde residu's by myne in die Oranje-Vrystaat word nadelig beïnvloed deur die aanwesigheid van pirofilliet wat natuurlik flotterbaar is en verhinder moet word om saam met die gewenste bestanddele te dryf deur die gebruik van 'n mikaonderdrukker en thucholiet (uraanhoudende kerogeen) wat beduidende hoeveelhede uraan en goud bevat, maar nie maklik in die aanwesigheid van 'n mikaonderdrukker dryf nie.

In laboratoriumtoetse op hierdie residu's het die gebruik van 'n mengsel van MIBC (metielisobutiëlkarbinol) en paraffien, benewens die reeks reagense wat gewoonlik gebruik word, die goudherwinning aansienlik verhoog, maar die swaefgraad was laer. Die herwinning van thucholiet het ook aansienlik toegeneem, maar die hoeveelheid uraan wat met hierdie mineraal geassosieer was, was klein.

Aanlegproewe, wat voortgaan, was bemoedigend en dui op 'n toename van 4,6 persent in die herwinning van goud.

Introduction

Gold and sulphides were first floated from Witwatersrand ore in 1935¹. The flotation of gold, uranium, and pyrite from accumulated residues² has been used fairly extensively in recent years to facilitate the economic processing of these residues. In most cases, conditioning under mildly acidic conditions was required to maximize the recovery, presumably by the cleaning of the oxidized, and coated surfaces of the pyrite and gold respectively. The collector used under these conditions (at a pH value of about 4) is mercaptobenzothiozole. The residues usually have a grading of 70 to 85 per cent passing 75 μm .

The ores in the Orange Free State contain about 10 per cent pyrophyllite, a mica that has a natural floatability. A depressant of low molecular mass is required to prevent the pyrophyllite from floating. Substituted guar is used at present at a dosage of about 50 g/t.

It has been assumed³ that the uraniferous kerogen in the ore (usually called thucholite) floats readily. This component of the ore usually contains significant quantities of uranium and gold (e.g. 20 to 300 g of gold and 3 to 10 kg of uranium per ton of thucholite), and may represent as much as 30 per cent of the gold in the feed to a flotation plant. Heavy-liquid separations on samples from Anglo American flotation plants in the Orange

Free State showed that only about 40 per cent of the thucholite was recovered by flotation. This low recovery could be related to the use of mica depressant, and Van Zyl⁴ was able to demonstrate the depression of thucholite in a laboratory cell using high concentrations of guar derivatives and carboxymethyl cellulose.

This paper describes laboratory tests aimed at the improved recovery of thucholite in flotation plants in the Orange Free State, and preliminary plant tests.

Laboratory Tests

The laboratory tests were initially confined to material treated in the flotation plants of Anglo American in the Orange Free State that produce concentrates for the Joint Metallurgical Scheme (JMS).

Fig. 1 shows a typical flotation response of sulphur (pyrite), gold, and uranium, clearly illustrating that some of the floatable gold is not associated with the sulphur. The correspondence of the curves for gold and uranium suggests a mineralogical association between these two elements.

The initial stages of the testwork were concerned with attempts to recover thucholite in a separate concentrate by the use of frothers or pairs of frothers only. The reasoning was based on the fact that the major constraint on the JMS flotation plants is the need to produce a pyrite concentrate at a grade suitable for roasting (ca 30 to 32 per cent sulphide sulphur), and it was considered that, if a first-stage thucholite or differential float could be developed by the use of frother alone, the subsequent pyrite float would suffer the least disturbance.

* Anglo American Research Laboratories, P.O. Box 106, Crown Mines, 2025.

† President Brand Gold Mining Company Limited, Shaft 2, Welkom, 9460.

© The South African Institute of Mining and Metallurgy, 1983.
SA ISSN 0038-223X/\$3.00 + 0.00.

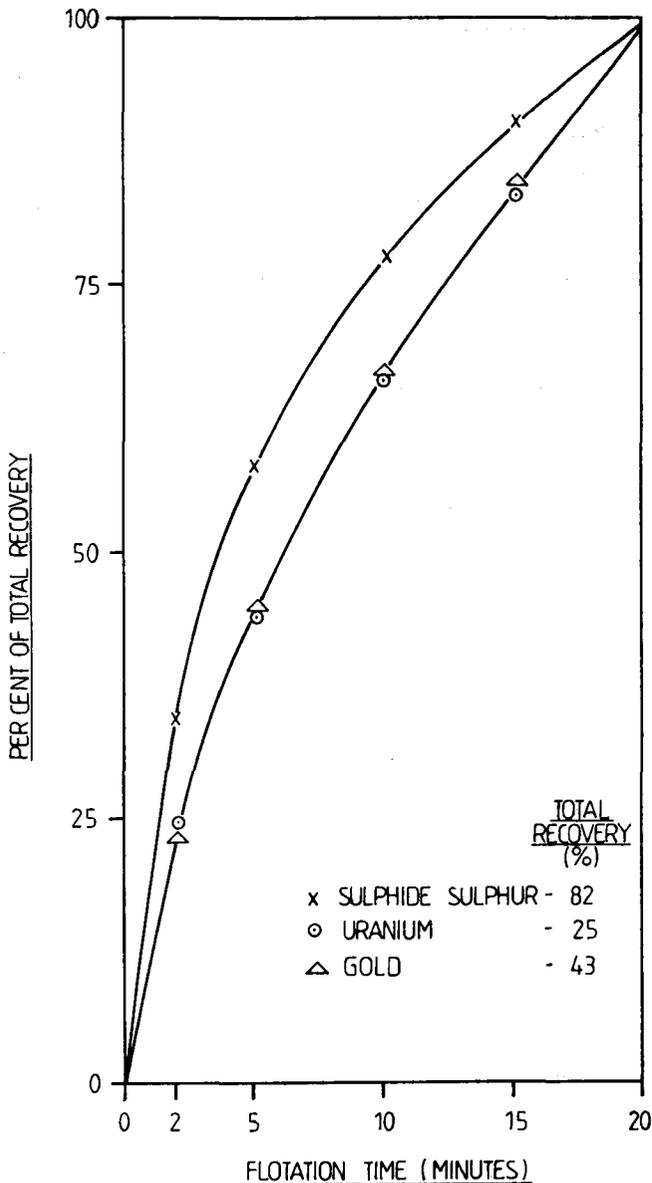


Fig. 1—Typical batch results for JMS feeds

A powerful depressant has to be added to ores from the OFS feeds to reduce the recovery of pyrophyllite to a minimum. Thus, although the term 'frother-only test-work' has been used to describe the early stage in the investigation, the reagent suite always, of necessity, includes pyrophyllite depressant. It was decided that the deportment (recovery) of thucholite should be monitored by means of the gold assay since wood pulp makes the analysis of carbon in the ore difficult and the final objective was, in fact, the recovery of more gold.

Among the frothers tested were the following:

- methyl isobutyl carbinol (MIBC)
- Dowfroth 200
- Montanol 3000
- tri-ethoxy butane (TEB)
- crecrylic acid.

These tests showed very little promise.

Attention was then diverted to the traditional way of activating naturally floatable minerals (molybdenite and coal) by the addition of paraffin and frother. This approach was viewed with trepidation by some, who had experienced very unstable froths when attempting to float pyrite from the current residues of a uranium plant (countercurrent decantation and solvent extraction). The initial tests confirmed the worries about paraffin, which produced sparse, brittle froths changing to loose, copious, and dirty froths with excessive pyrophyllite. However, the use of a mixture of paraffin and an oil-based frother solved this problem. A popular recipe for the flotation of coal is a mixture of 20 per cent MIBC and 80 per cent paraffin. This mixture has also been used for the flotation of molybdenite. The dosage rate of the mixture was 50 g/t in almost all the tests.

The results of this change in technique were encouraging, as shown in Table I, which compares the standard JMS test and a test using a mixture of MIBC and paraffin.

The points of note in connection with these tests are as follows.

- (i) Some 9 per cent extra gold was recovered in a mass of concentrate that was 0,6 per cent higher than in the standard float.
- (ii) There was a slight increase in the gold grade from 7,09 to 7,24 g/t, despite the higher mass.

An additional encouragement was that the froth was much improved, it stabilized more quickly, and its appearance was near to that in the standard flotation test.

At this time it was decided that the following two routes should be examined for the possible inclusion of the MIBC-paraffin suite of reagents used in the JMS flotation plants:

- (1) *Differential flotation.* The thucholite collector was added, and a concentrate was collected before the normal suite of reagents was added.
- (2) *Bulk flotation.* All the reagents were added at the start and conditioned together.

The *standard flotation*, against which the two routes were assessed, was the same as the bulk flotation but excluded the thucholite collector, i.e. the standard laboratory technique developed over the past few years for the testing of JMS flotation feedstock. The reagent suite was as follows:

Senkol 50 (sodium mercaptobenzothiozole)	120 g/t
Acrol J2P 350 (substituted guar—pyrophyllite depressant)	80 g/t
Dowfroth 200	5 g/t
Copper sulphate	50 g/t

It may be of interest that all the testwork was conducted on 10 kg portions of the various samples, by the use of Denver D7 Sub A flotation mechanisms. These are 1 ft pilot-plant machines used individually as batch flotation machines. The investigation up to this stage had been conducted on samples of deposited residue, representing the feed to both the President Steyn and the President Brand flotation plants, repulped with monitor water

TABLE I
ROUGHER FLOTATION—OVERALL RECOVERIES

Procedure	Rougher concentrate				
	Mass %	Gold		Sulphur	
		Au g/t	Recovery %	Sulphide sulphur %	Recovery %
Standard JMS float of 15 min duration	5,0	7,09	52,3	12,0	77,8
Thucholite flotation (MIBC-paraffin float followed by JMS float, total flotation time 15 min)	5,6	7,24	61,4	11,4	82,9

and flotation-plant dilution water from President Steyn. After a further small series of tests had confirmed the results shown in Table I, the investigation was extended to cover the feeds to President Brand using repulping (monitor) and dilution water from that plant.

Both the differential and the bulk flotation routes were examined for a time, although it became apparent that bulk flotation would be the preferred route for a number of reasons, among them being the random recovery of pyrite into the so-called differential gold concentrate caused by varying amounts of reagents recycled with the plant waters, and the relative ease with which the bulk flotation technique might be tested on a plant. Finally, the inclusion of the thucholite in the roaster feed provided a means of liberating the gold.

Table II gives the weighted averages of a series of six directly comparable tests conducted on samples from President Steyn using President Steyn plant waters. The major finding was the 7,2 per cent increase in gold recovery (from 28,8 to 36,0 per cent) in the first 3 minutes of flotation with an increase in concentrate mass of 0,2 per cent (from 1,3 to 1,5 per cent). The grade of sulphide sulphur dropped from 34,5 to 29,5 per cent in this first concentrate. The overall recovery of gold was increased by 6,7 per cent with an increase in concentrate mass of 0,5 per cent (from 4,8 to 5,3 per cent).

Fig. 2 shows the gold recovery plotted as a function of sulphur grade, since the latter is all important for roaster operation. The grade of the first concentrate is an important control parameter on the Anglo American plants in the OFS, as this concentrate is not cleaned and must be close to final grade. The results shown in Fig. 2 indicate that it may be necessary for more of the rougher concentrate to be cleaned. However, the overall relationship between grade and recovery was significantly better.

The recovery of uranium did not change in this series of tests, and, as the thucholite was thought to be a major carrier of floatable uranium, one could conclude that the recovery of thucholite was unaffected.

Samples from a similar set of directly comparable flotation tests were deslimed at about 10 μm and were then separated by the use of heavy liquids. The density fractions were examined under a microscope and assayed. The fractions can be described as follows:

- (a) *Slimes*
Predominantly quartz with major amounts of pyrophyllite.
- (b) *2,6 r.d. Float Fraction*
Thucholite was observed, with accessory amounts of porous silicates in the concentrates and major amounts of silicates in the head and tailings samples.
- (c) *2,85 r.d. Float Fraction*
Quartz was the predominant constituent, with minor to major amounts of pyrophyllite. Some thucholite was observed.
- (d) *2,85 r.d. Sink Fraction*
Pyrite was predominant, with trace amounts of rutile, uraninite, chromite, and pyrophyllite.

The 2,85 r.d. sink fraction was sometimes separated by means of the Superpanner. A microscopic examination of the Superpanner concentrate occasionally revealed liberated particles of gold.

The distribution of gold in the head and tailings samples is listed in Table III, which shows the contribution of the various mineral associations to the overall recovery of gold. The uranium was concentrated in the pyrite and thucholite fractions (1000 to 2000 g/t), but, because of the small mass of the latter, the thucholite contained only about 6 per cent of the uranium in the head sample. This explains why no significant change in the recovery of uranium was found. Fig. 3 shows the progressive recovery of gold from the density fractions as the batch test proceeded. The substantial improvement in the recovery of gold from the thucholite fraction is readily apparent (i.e. the recovery of thucholite was improved).

With these encouraging results, the technique was applied to the then current feed to the President Brand flotation plant, which is very similar to the material at President Steyn. Not surprisingly, very similar results were obtained.

The investigation continued with tests on variations in the dosage of reagents, etc. The ratio of MIBC to paraffin, the dosage rate of the mixture, and the use of other frothers as paraffin-carriers were examined. The conclusions were that the 20:80 mixture was about optimal, and that a dosage of 50 g/t was adequate. A larger

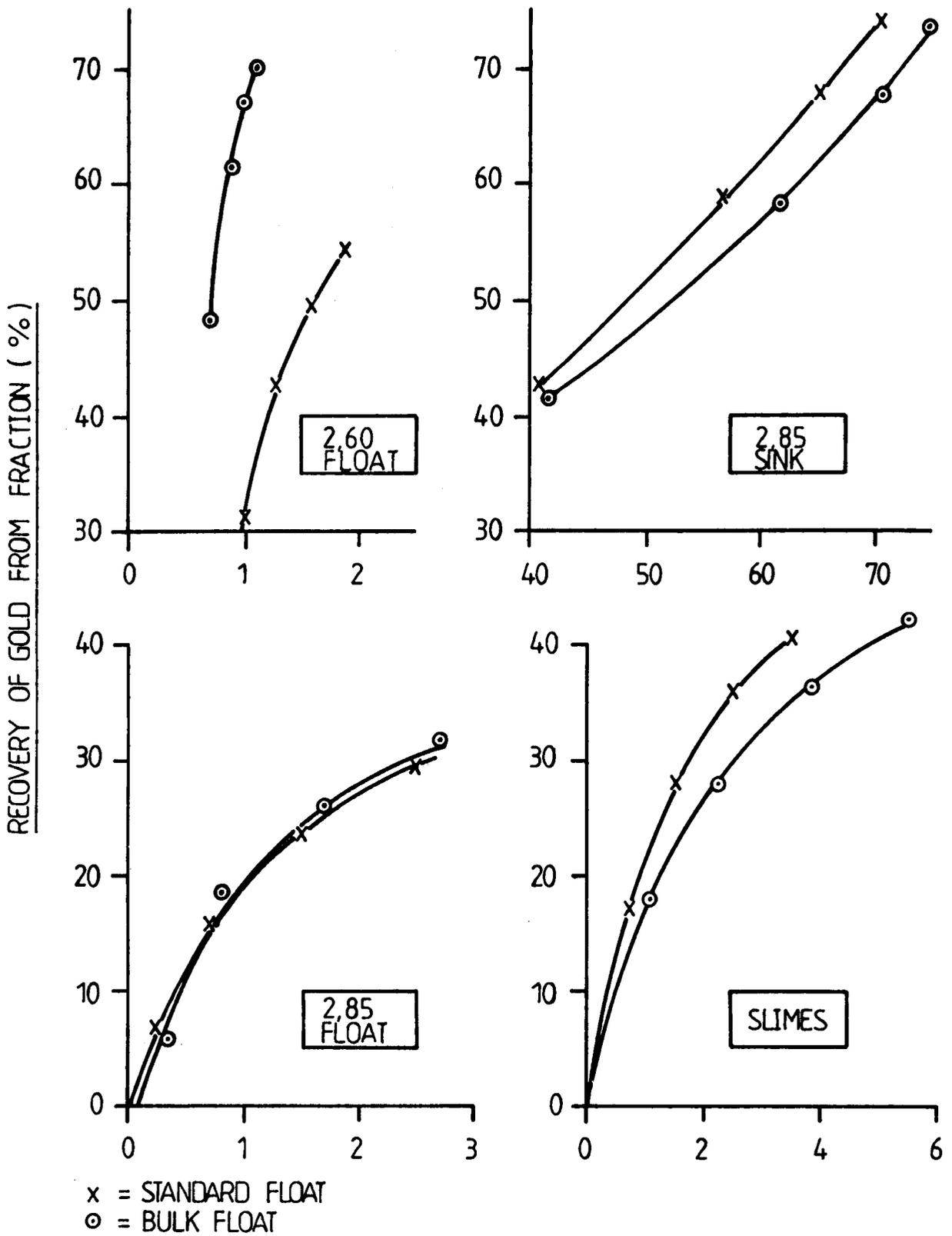


Fig. 3—The effect of paraffin (bulk float) on the recovery of gold from heavy-liquid fractions

dosage tended to destabilize the froth and reduce the initial sulphur grade. A lower dosage reduced the effect of the reagent, and, clearly, the level will depend upon grade control in the plant.

A few cleaner tests demonstrated a loss of some of the additional gold recovered. However, these tests were not regarded as being representative of plant conditions, and the rougher tests were considered to be sufficiently encouraging to justify plant tests. It is important to note that the selectivity of the float for gold was improved. It should also be noted that the total time for flotation remained the same, i.e. an extension to the existing flotation plants is not necessary.

Tests on feed sources other than President Steyn and President Brand did not produce comparable improvements, and further work must await successful implementation on these mines.

Plant Trial

Facilities for the addition of the MIBC-paraffin mixture were installed at the President Brand flotation plant. The first plant trial started on 11th October, 1982, and continued until 17th November. After the latter date, the plant received material from a different source, which required adjustments to the suite of reagents. Tests on the use of MIBC-paraffin were suspended until the problems associated with the new feedstock were overcome.

The addition of 50 g/t of the MIBC-paraffin mixture to the standard float produced a voluminous, dirty froth, which required cleaning. An adjustment of the air flow and a reduction of the dosage to 43,5 g/t improved the appearance and grade of the froth, making possible a return to the 'normal' plant configuration, in which the concentrate from the first four cells went direct to the final product. The dosage of TEB frother (tri-ethoxy

butane) was initially reduced from 25 to 22,5 g/t, and

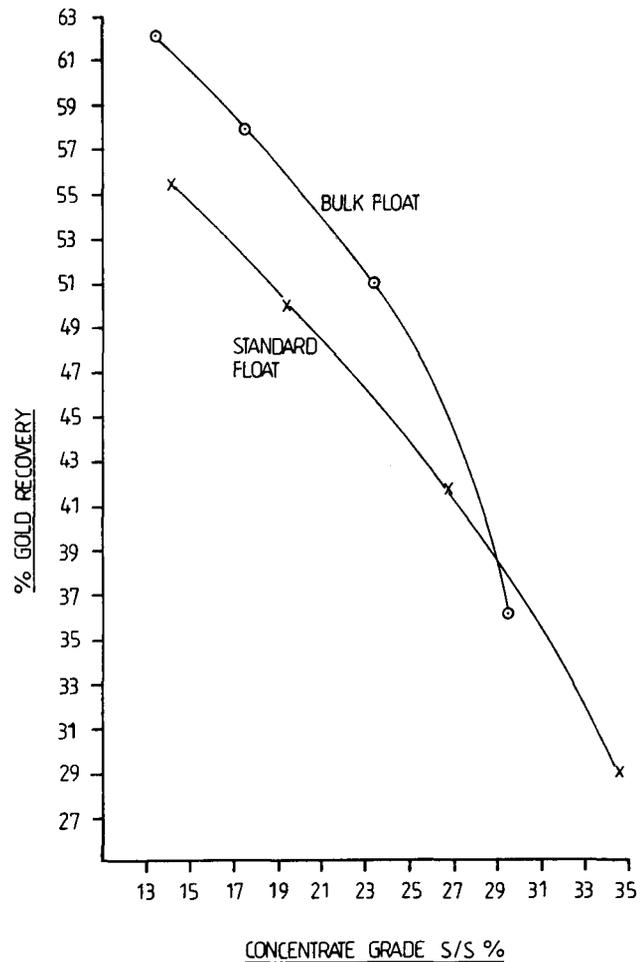


Fig. 2—Grade of sulphide sulphur versus gold recovery in the standard and bulk-flotation tests

TABLE II
WEIGHTED AVERAGE RESULTS OF ROUGHER FLOTATION—CUMULATIVE RESULTS
COMPARISON BETWEEN STANDARD AND BULK FLOTATION, PRESIDENT STEYN FEED

Fraction	Mass %	Cum. mass %	Sulphide/Sulphur				Gold			
			Assay %	Cum. assay %	Distn %	Cum. distn %	Assay g/t	Cum. assay g/t	Distn %	Cum. distn %
<i>Standard Float</i>										
RC1	1,3	1,3	34,50	34,50	56,1	56,1	16,08	16,08	28,8	28,8
RC2	0,9	2,2	15,96	26,79	18,5	74,6	10,07	13,58	12,8	41,6
RC3	1,2	3,4	5,07	19,24	7,5	82,2	5,06	10,62	8,3	49,9
RC4	1,4	4,8	1,99	14,16	3,5	85,7	2,88	8,32	5,5	55,4
RT	95,2	100,0	0,12	0,80	14,3	100,0	0,34	0,73	44,6	100,0
Assay head values			0,81				0,72			
<i>Bulk Float</i>										
RC1	1,5	1,5	29,49	29,49	56,4	56,4	17,25	17,25	36,0	36,0
RC2	1,1	2,6	15,14	23,39	21,5	77,9	9,62	14,00	14,9	50,9
RC3	1,3	3,9	5,28	17,54	8,4	86,3	4,05	10,79	7,0	57,9
RC4	1,4	5,3	1,67	13,35	2,9	89,2	2,15	8,51	4,1	62,1
RT	94,7	100,0	0,09	0,79	10,8	100,0	0,29	0,72	37,9	100,0
Assay head values			0,80				0,74			

TABLE III
DISTRIBUTION OF GOLD IN DENSITY FRACTIONS
STANDARD BATCH FLOTATION OF PRESIDENT STEYN FEED

Fraction	Head			Tailings			Gold * recovery %
	Mass, %	Au, g/t	Distn, %	Mass, %	Au, g/t	Distn, %	
Slimes	18,87	0,48	25,1	18,16	0,24	22,2	41,0
2,6 F	0,26	17,0	12,2	1,39	3,13	22,2	54,5
2,85 F	79,03	0,17	37,0	79,75	0,09	36,4	29,5
2,85 S	1,85	5,00	25,7	0,71	5,40	19,2	73,6
Total	100,0	0,36	100,0	100,0	0,20	100,0	52,7

* Calculated from the sum of flotation concentrate fractions and tailings fractions

finally to 17,5 g/t. Towards the end of the trial, the MIBC-paraffin mixture was mixed with slime by the use of water eductors to ensure efficient dispersion.

The recovery of gold improved from an average of 44,1 per cent for the previous four months to 48,7 per cent during November while the head grade remained the same. It was concluded that the MIBC-paraffin was responsible for the improved recovery.

Conclusions

The improvement in gold recovery obtained in the laboratory by the use of a mixture of MIBC and paraffin was substantial. However, the effect on the initial sulphur grade is a limitation, particularly if the concentrate from the first few cells is not cleaned. The fact that the recovery of uranium did not change in sympathy with the increased recovery of gold was a surprise. However, heavy-liquid separations showed that the recovery of thucholite had been improved substantially, and that the proportion of uranium associated with this mineral was small.

The results of the one-month trial were very encouraging, indicating an increase in gold recovery of 4,6 per cent (i.e. gold production increased by about 10 per cent). Plant tests are continuing.

The applicability of the new reagent to only some ores is not clearly understood, but it appears to be related to the use of gangue depressant and/or the nature of the thucholite.

Acknowledgements

The sponsorship of the research work by the Gold and Uranium Division of the Anglo American Corporation is gratefully acknowledged. This paper is published by permission of the Anglo American Corporation of South Africa Limited and the President Brand Gold Mining Company Limited.

References

1. KING, A. Gold metallurgy on the Witwatersrand. Johannesburg, Chamber of Mines of South Africa, 1949. pp. 190-196.
2. RUMMER, W. T., BOTHA, F., and ADAMS, J. S. Recovery of sulphur, uranium and gold from residues. *J. S. Afr. Inst. Min. Metall.*, vol. 78. 1977. pp. 134-142.
3. LLOYD, P. J. D. The flotation of gold, uranium, and pyrite from Witwatersrand ores. *J. S. Afr. Inst. Min. Metall.*, vol. 81. Feb. 1981. pp. 41-57.
4. VAN ZYL, A. The separation of kerogen from pyrophyllite by flotation. M.Sc. thesis, University of Cape Town, Jul. 1982.

Cost engineers

On 5th August, 1983, 24 people met at the offices of Anglo American in Johannesburg to establish the South African Section of the American Association of Cost Engineers (AACE).

The following Founder members will serve on the first Board of the Southern African section: Messrs J. M. Bredenkamp (President), C. Grindlay (Vice President), C. J. Sutton, P. Povall, B. Walker, H. Crail, and A. D. Milne (Acting Secretary/Treasurer). Membership of the South African Section is limited to members of the AACE, although people who are in the process of applying for membership of the American body will be welcome to participate in the activities of the local Section. Those interested should contact A. D. Milne on (011) 638-2552.

Cyanide and the environment

A Conference on the above subject is to be held from 4th to 6th December, 1984, in Tucson, U.S.A.

The regulations controlling the use and disposal of cyanide are highly restrictive. Cyanide is used in the mining, plating, agricultural, and other industries for various purposes. The concentrations of cyanide employed differ widely in these industries. Waste-disposal methods involve a variety of operations such as treatment of the waste stream prior to disposal. The disposal techniques include chemical and biological treatment in closed systems, and land treatment in more dilute forms for biological oxidation. Neutralization and impoundment are commonly used for the disposal and control of mining-related cyanide waste.

Although the chemistry of cyanide is well known and the literature substantial, certain aspects of cyanide reactions as they pertain to its ultimate fate in the environment require further evaluation. These aspects are closely related to the complex chemistry of toxicity that needs to be further clarified.

A Workshop on Cyanide from Mineral Processing was held on 2nd and 3rd February, 1982, at the Utah Mining and Mineral Resources Research Institute. This was sponsored by the National Science Foundation in cooperation with the U.S. Bureau of Mines and the mineral industry. The proceedings of this workshop as well as other meetings on cyanide will form the departure point for the present Conference.

The purpose of the Conference is to provide an open forum for the presentation of new developments on the environmental aspects of cyanide use and disposal. It will serve to update and extend the information con-

The AACE was formed 27 years ago in Morgantown, West Virginia, with its original objective to set estimating standards for the construction industry. The Association's activities now extend to the education and certification of cost engineers, the publication of cost-engineering data and methods, and the holding of regular meetings that cost engineers attend to review and update their profession.

Members of the AACE work as contractors, consulting engineers, equipment suppliers, educationalists, and project-managers, and in all parts of the construction industry. This broad base of membership permits a wide range of experience and knowledge to be freely exchanged between members.

tained in the Proceedings of the Workshop on Cyanide from Mineral Processing.

It is further the purpose of this conference to bring together the most up-to-date information on the fundamental aspects of cyanide chemistry and toxicity as a basis for the assessment of environmental impacts.

The topics to be discussed are as follows:

- Cyanide Chemistry and Toxicity
- Cyanide Analysis
- Cyanide Treatment
- Chemical
- Biological
- Cyanide Recovery
- Cyanide Disposal and Use
- Mining Industry
- Plating Industry
- Electronics Industry
- Other Industries
- Alternatives to Cyanide
- Cyanide in the Environment
- Naturally Occurring Cyanide
- Agricultural Use
- Attenuation
- Natural Oxidation
- Thiocyanate
- Regulatory Aspects of Cyanide

Further information is available from Dr Dirk van Zyl, Cyanide and the Environment, Department of Civil Engineering and Engineering Mechanics, The University of Arizona, Tucson, Arizona 85721, U.S.A. Telephone (602) 621-2459 or 621-2266.