Reichert cones for Witwatersrand gold ores*

by R. GUEST†, B.Sc. Eng. (Met.), M.S.A.I.M.M.

SYNOPSIS

Reichert cone concentrators are used successfully in many applications, but have as yet not found favour in the gold mining industry on the Witwatersrand, because, for optimum operation, the cones need to have specially designed feed-preparation and circuit details. In addition, good liberation of the valuable minerals is essential.

Gravity tests, in which a pilot plant was used on refractory gold ore, showed that 94 per cent of the gold could be extracted after gravity concentration and preferential grinding, roasting, and cyanidation of the concentrate, whereas 96 per cent could be extracted by flotation followed by the same preferential treatment. However, the capital and operating costs for gravity separation are expected to be considerably lower than those for flotation.

SAMEVATTING

Reichert-keëlkonsentreerders word met sukses vir baie doeleindes gebruik, maar het nog nie in die goudmynbedryf aan die Witwatersrand ingeslaan nie omdat die keëls vir optimale werking spesiaal ontwerpte toevoervoorbereiding en kringdetails moet hê. Verder is goeie bevryding van die waardevolle minerale noodsaaklik.

Swaartekragtoetse, waarin 'n weerbarstige gouderts in 'n proefaanleg gebruik is, het getoon dat 94 persent van die goud na swaartekragkonsentrasie en voorkeurmaling -roostering en -sianidering van die konsentraat geëkstraheer kon word, terwyl 96 persent deur flottasie gevolg deur dieselfde voorkeurbehandeling geëkstraheer kon word. Die kapitaal- en bedryfskoste vir swaartekragskeiding sal egter, na verwagting heelwat laer as dié vir flottasie wees.

Introduction

The main types of modern gravity equipment, their feed ranges, capacities, and costs have been tabulated by Moncrieff *et al.*[1] and are listed in Table I, from which it can be seen that the Reichert cone has distinct advantages over the other concentration devices in regard to capacity and cost. Some workers claim that the cone can handle particles as coarse as 3000 μm.

Gravity separation has several advantages over flotation: the capital costs of the equipment are lower, less power is usually required per ton of material treated, and the effluent is considerably less harmful to the environment. In 1981, budget prices were obtained from manufacturers for plants able to produce 100 t/h. The prices included the flotation or gravity units for roughing and cleaning, building erection, civil engineering and electrical engineering installations, pumps, and reagent-feeding equipment for flotation. The price for a flotation plant was given as 1,1 million rands, and for a gravity plant as R300 000, and the operating costs[2] are expected to be about four times higher for flotation than for gravity separation.

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† Council for Mineral Technology (Mintek), Private Bag X3015, Randburg 2125, Transvaal.

Reichert Cones in Southern Africa

The Reichert cone is used successfully in a number of applications in Southern Africa. It is used at Richards Bay in the application for which it was originally designed, viz to concentrate heavy minerals from beach sands; it is used very successfully at Phalaborwa for the concentration of uranothorianite and baddeleyite from flotation tailings; and, finally, a cone has been installed at Tsumeb to recover oxidized copper and lead minerals prior to flotation, thus considerably reducing the consumption of reagents.

The Reichert cone has not yet been used successfully in South Africa's largest mining industry, i.e., gold, uranium, and pyrite, although it has been tried on Witwatersrand material on several occasions. The question arises as to why this should be so, and two of these unsuccessful cases are dealt with here. Much information has been published on the operation of Reichert cones and on their description[3]; the construction and mode of operation of the cone are therefore not described in this paper.

Blyvooruitzicht

A Reichert cone concentrator was installed at the Blyvooruitzicht mill as a test unit. The purpose of these tests was the recovery of a little extra uranium from the low-grade circuit in order to increase the amount of uranium being extracted from the high-grade circuit. The interest therefore lay in uranium rather than in gold

TABLE I

EQUIPMENT FOR GRAVITY CONCENTRATION OF CASSITERITE (AFTER MONCRIEFF *et al.*[1])

Purpose	Equipment	Feed range μm	Capacity t/h	Capital cost*
Roughing	Jig, coarse	6350 to 1675	12	414
	Jig, fine	1675 to 600	6	828
	Spiral	600 to 75	2	345
	Reichert cone	500 to 50	70	254
	Bartles–Mozley table	100 to 5	2,25	2 560
Cleaning	Sand table	500 to 75	1	3 490
	Slimes table	100 to 10	0,25	13 960
	Vanner (Field House)	100 to 10	0,25	21 780
	Vanner (Bartles–Mozley)	100 to 10	0,45	9 950

* Calculated in 1976 as pounds (sterling) per ton per hour. Installation costs were not taken into account.

or pyrite. The unit operated in 1970 on the discharge from the rod mill, from which the material coarser than 3 mm had been removed. The overall recovery of uranium was only 17,7 per cent, but recoveries of over 36 per cent were obtained from the material between 106 and 45 μm in size. This seemed to indicate that the uraninite in the coarser size fractions had a low degree of liberation.

A mineralogical investigation of the products indicated that there was a marked difference between the uraninite-to-carbon ratio for the thucholite recovered in the concentrate and that lost in the tailing, the ratio of the latter being noticeably lower than that of the former.

Because the inefficient operation of the Reichert cone appeared to be due to poor liberation and feed material that was too coarse, a new circuit was constructed in which the cone was placed in the circulating load of a ball mill–classifier circuit. This circuit is shown in Fig. 1.

It can be seen that the total feed rate to the cone was 57,7 t/h, while the rate at which new feed was introduced into the circuit was only 15,6 t/h; there was, therefore, a large recirculating middling fraction diluting the feed.

A calculation of the distribution of the three products from the Reichert cone shows that the recovery of uranium in the concentrate was only 21 per cent. However, if one disregards the middling fraction and takes only the two final products into account, it can be seen that the concentrate was over 50 per cent of the new feed and the uranium recovery 62,2 per cent (Table II). This result is at first glance again very discouraging but, because some of the uranium was present as unliberated uraninite or as thucholite, it seemed that the grade and recovery of U_3O_8 was a poor indication of the performance of the cone, and that it would be more accurate for the performance to be judged on the recovery of the high-density material.

TABLE II
CALCULATION OF URANIUM RECOVERED FROM THE BLYVOORUITZICHT TEST

Product	Rate t/h	Mass %	U_3O_8 Concentration p.p.m.	U_3O_8 Distribution %
Concentrate	8,6	55,1	150	62,2
Final pulp	7,0	44,9	112	37,8
Head	15,6	100	133	

Heavy-liquid separation was carried out on four sieve fractions at relative densities of 2,95 and 3,31, and the three products obtained from each size fraction were designated as 'light', 'intermediate', and 'heavy' material. The efficiency of the Reichert cone in concentrating these materials is shown in Table III, which shows that the cone recovered 92,8 per cent of the uranium contained in the heavy material coarser than 600 μm, and 76,1 per cent of the uranium in the intermediate material of the same size.

At this stage it should be noted that the amount of material reporting to the light fraction was over 95 per cent, and this fraction contained between 60 and 80 per cent of the total uranium. The amount of material reporting to the heavy fraction was only between 0,51 and 1,89 per cent. This is a very small amount of the total sample, but the point of interest was whether the

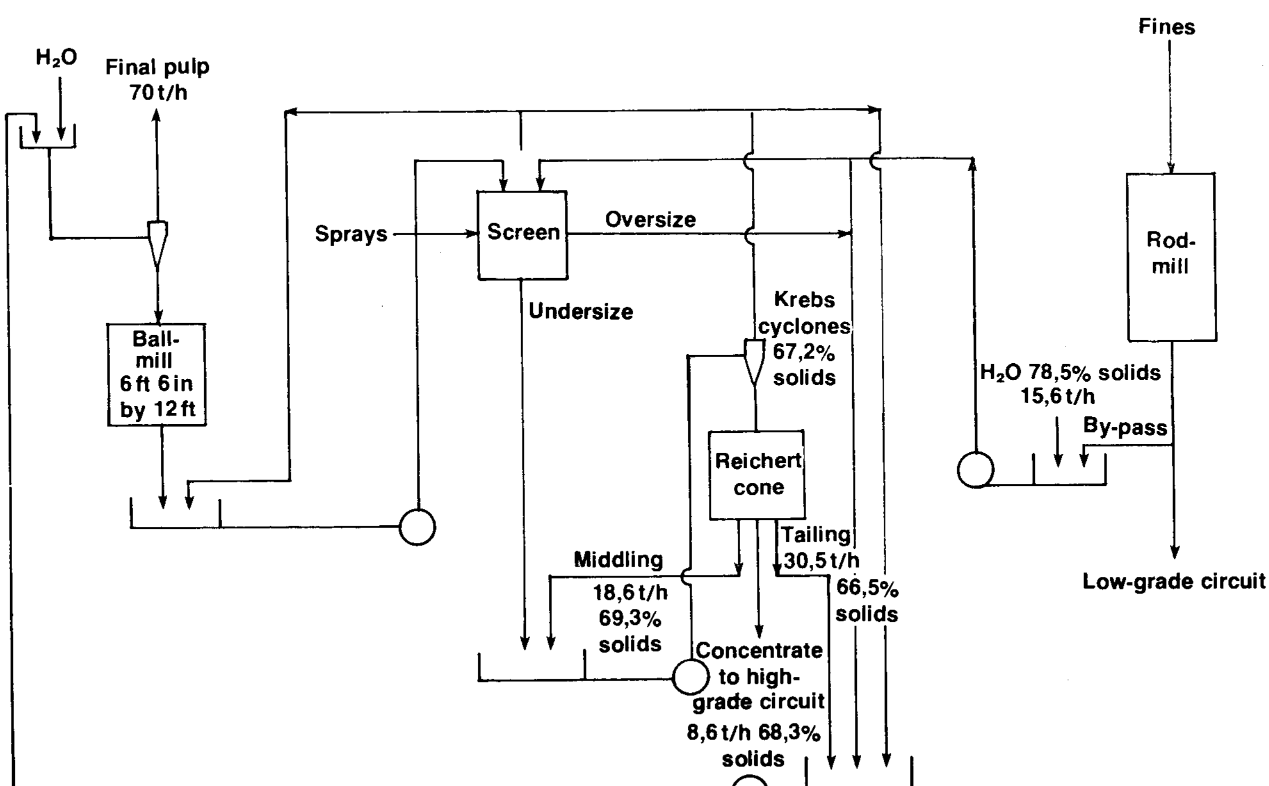

Fig. 1—The circuit for the Reichert cone at Blyvooruitzicht

TABLE III

THE EFFICIENCY OF THE REICHERT CONE FOR VARIOUS SIZE FRACTIONS AND RELATIVE DENSITIES

Size fraction μm	Product	Heavy material			Intermediate material		
			U_3O_8			U_3O_8	
		Mass %	Concn p.p.m.	Distn %	Mass %	Concn p.p.m.	Distn %
>600	Concentrate	74,51	1550	92,83	65,92	543	76,09
	Tailing	25,49	350	7,17	34,08	330	23,91
	Total	100,00	1244	100,00	100,00	470	100,00
600 to 212	Concentrate	56,52	1510	50,55	25,49	576	25,79
	Tailing	43,48	1920	49,45	74,51	567	74,21
	Total	100,00	1688	100,00	100,00	569	100,00
212 to 106	Concentrate	50,00	1350	52,53	30,86	822	31,58
	Tailing	50,00	1220	47,47	69,14	795	68,42
	Total	100,00	1285	100,00	100,00	803	100,00
106 to 38	Concentrate	51,32	2590	67,41	54,61	636	42,75
	Tailing	48,68	1320	32,59	45,36	1026	57,25
	Total	100,00	1972	100,00	100,00	813	100,00

Reichert cone was inefficient in recovering uranium, or just inefficient in recovering uranium that had a low relative density.

The recovery of uranium seemed to drop for the material between 600 and 212 μm and increase once more for the finer sizes. The Reichert cone therefore seems to have two regions of maximum recovery under the conditions of the test: one coarser than 600 μm, and the other between 106 and 34 μm or possibly even finer. It is possible that the small amount of uranium in the coarsest fraction is in the form of thucholite of high relative density, and that the uraninite becomes liberated only in the fraction between 106 and 38 μm.

The test gave unsatisfactory results because between 60 and 80 per cent of the uranium was in the light material, which contains the unliberated uraninite and low-density thucholite. The figures for the recovery of gold and sulphur would have been of interest but were not available at the time.

President Steyn

The operation of the Reichert cones that were used at President Steyn Gold Mine was found to be unsatisfactory for the following reasons.

(1) Thin steel flakes from broken grinding balls, nails from underground, bits of wire, wood chips, etc., found their way through the slotted scalping screen ahead of the cone and entered the cone feed. They then became cemented onto the surface of the cone by pyrite that had oxidized, and possibly also by carbonates that had precipitated out of the water. These surface obstructions acted as traps for the liberated gold. Up to 6 kg of gold could be trapped in this manner, and this constituted a security risk.

(2) The original configuration of the cone was 3DS (3 double single), but the slots feeding the double cones became blocked with wood and steel rubble, which prevented one of the double sections from operating. The double cones were subsequently altered to single cones.

(3) The inspection hatches were too small for the deck surfaces to be inspected or cleaned properly. When larger hatches were cut, the feed-distribution cones became distorted, leading to uneven feeding of the concentrating cones. The hatches were also responsible for the loss of some of the pulp as a result of splashing.

(4) The relative density of the pulp could not be adjusted to the optimum very frequently because no provision was made for density readings to be taken near the feed pumps, which were four storeys below the tops of the cones.

(5) The location of the pumps led to surging of the feed.

(6) No water for dilution was added to the cleaning cones (the water pipes had been disconnected), and the feed to the cleaners was therefore possibly too dense for efficient operation.

Pilot-plant Tests

A gold mill in the Orange Free State (owned by the Gencor group) has ore supplied from two different locations. One of these ores is partially refractory. The Council for Mineral Technology (Mintek) — then the National Institute for Metallurgy (NIM) — undertook to investigate the feasibility of gravity separation to recover the refractory gold. The circuit of the pilot plant that was used is shown in Fig. 2.

The dimensions of the rod mill were 400 by 840 mm, and those of the ball mill – a rubber-lined Sala mill – 1000 by 1400 mm. A rake classifier was used instead of a cyclone because cyclones that are small enough to treat the feed at the pilot-plant flowrates give a size cut that is too fine for the purposes of this investigation. On the

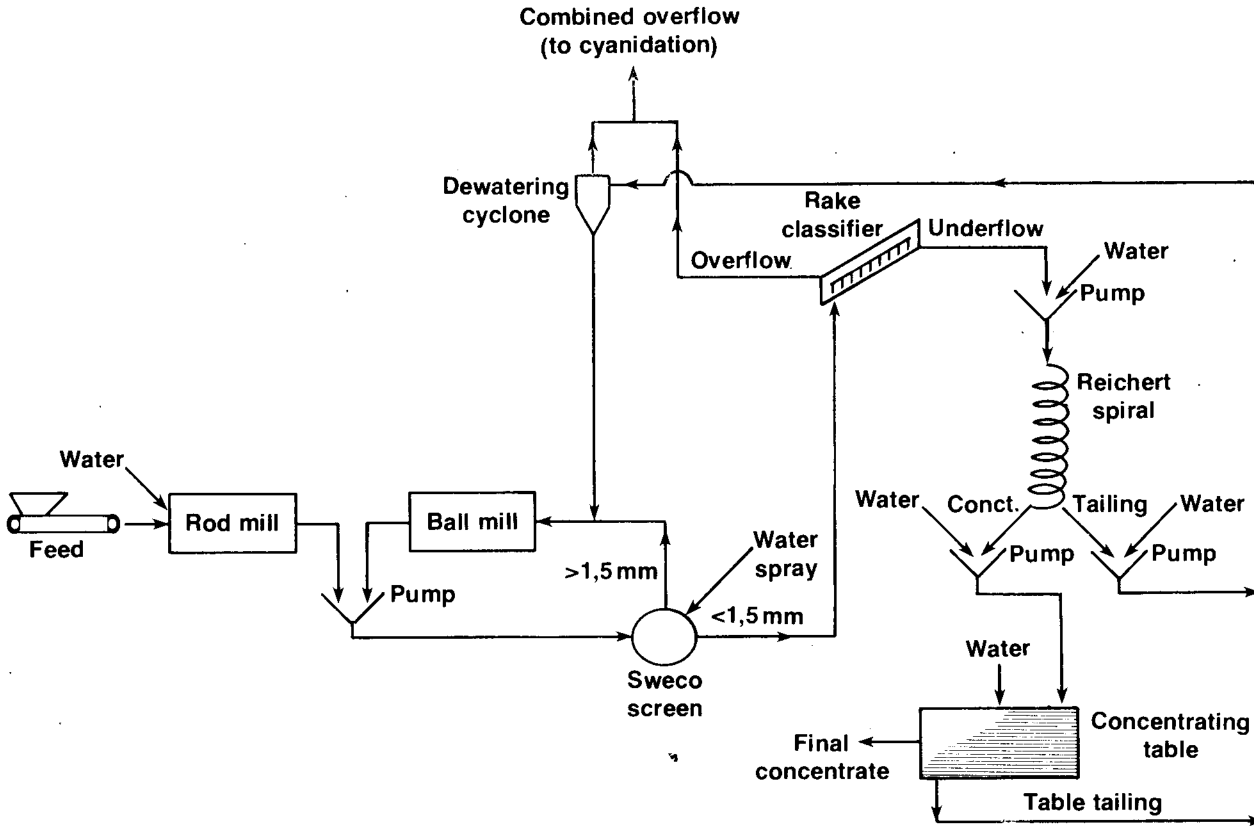

Fig. 2—The pilot-plant circuit

other hand, a cyclone that gives the correct cut requires a feed rate that is too high for the pilot plant. The Reichert spiral was used as the rougher concentrator and the shaking table as the cleaner.

The feed rate to the pilot plant was set at 5,2 kg/min or 312 kg/h. Occasionally the mass balance of the circuit did not agree with the feed rate, but it was found that the samples of the combined overflow had been taken over relatively short times and were sometimes inaccurate. The standard sampling procedure involved the collection of all the final concentrate for a period of approximately 1 h while a known amount of material was fed to the plant. Many small samples of the combined overflow were taken over this period.

The relative density of the pulp in the mills was kept between 1,90 and 1,92, and that of the material fed to the spiral at about 1,35.

Gravity Concentration

A number of tests, the results of which are given in Table IV, were carried out for the production of high-grade and low-grade concentrates.

The particle size of the combined overflow material became coarser as the project progressed because the number of balls added to the ball mill was insufficient to keep the ball charge constant.

The sulphur grade of the final concentrate ranged from 23,5 to 45,1 per cent, but the recovery of sulphur did not vary very much over that range. Fig 3 gives the graph for sulphur of grade versus recovery. The results are rather scattered, the best being obtained for runs 4 and 11.

The assay laboratory had a certain amount of trouble assaying the concentrates and even the head samples for gold (probably because of the presence of native gold), and the U_3O_8 value of the head also varied to some extent. However, the average head values of the sample were 6,0 g/t for gold, 16 g/t for U_3O_8, and 2,3 per cent for sulphur. From Table IV it can be seen that the calculated head values for gold and uranium were unsteady over all the runs, but that those for sulphur were fairly steady.

Most of the concentrates were analysed for cobalt, and the results, which varied from 107 to 405 g/t, are also shown in Table IV. Some of the combined overflow samples were also analysed for cobalt, but in all instances the result was below 15 g/t, which is the lower limit of determination of the X-ray method used for the analyses. The cobalt recoveries are not shown, but they were above 50 per cent when the overflow was taken as 15 g/t.

Size analyses of the concentrates showed that, in three cases out of four, only approximately 1 per cent of the material was finer than 38 μm. In the fourth case (run 8), the amount of this material was 8,7 per cent. This concentrate from run 8 was well below an acceptable grade of sulphur, probably because the shaking table was set at a very flat angle for that run.

Cyanidation

The final products from run 4 were tested for the extraction of gold by direct cyanidation and by grinding or roasting before cyanidation at the Union Corporation Research Laboratories. The results are given in Table V.

The extractions of gold by cyanidation given in Table

TABLE IV

TEST RESULTS

Run no.	Product	Combined O/F %< 75 μm	Mass %	Cobalt concn g/t	Gold Concn g/t	Gold Distn %	U_3O_8 Concn g/t	U_3O_8 Distn %	Sulphur Concn %	Sulphur Distn %
3	Conct.		3,4		68,3	61,1	99	14,2	45,1	67,6
	Overflow	79,6	96,6		1,53	38,9	21	81,8	0,76	32,4
	Head		100,0		3,8	100,0	24	100,0	2,27	100,0
4	Conct.		3,8	405	118	76,3	64	24,0	45,0	72,9
	Overflow	76,6	96,2		1,44	23,7	8	76,0	0,66	27,1
	Head		100,0		5,87	100,0	10	100,0	2,35	100,0
7	Conct.		4,7	107	129	83,4	107	24,8	36,0	74,8
	Overflow	62,4	95,3		1,27	16,6	16	75,2	0,6	25,2
	Head		100,0		7,27	100,0	20	100,0	2,26	100,0
8	Conct.		7,8	245	53,8	77,2	54	31,3	23,5	75,9
	Overflow	67,1	92,2		1,34	22,8	10	68,7	0,63	24,1
	Head		100,0		5,44	100,0	13	100,0	2,41	100,0
9	Conct.		4,4	364	85,9	72,5	77	22,8	38,3	74,4
	Overflow	58,9	95,6		1,5	27,5	12	77,2	0,61	25,6
	Head		100,0		5,20	100,0	15	100,0	2,27	100,0
10	Conct.		4,2	355	96,4	75,3	74	31,7	38,9	73,1
	Overflow	56,0	95,8		1,39	24,7	7	68,3	0,63	26,9
	Head		100,0		5,38	100,0	10	100,0	2,23	100,0
11	Conct.		4,9	268	70,1	78,4	71	36,2	28,3	78,8
	Overflow	55,8	95,1		1,39	21,6	9	63,8	0,55	21,2
	Head		100,0		6,00	100,0	13	100,0	2,41	100,0

V were used in the calculation of the total gold extraction from run 4:

Extraction from ore = 87,4 per cent
Extraction from combined overflow = 81,0 per cent of 23,7 per cent
= 19,2 per cent
Extraction from concentrate after grinding, roasting, and cyanidation = 97,3 per cent of 76,3 per cent
= 74,2 per cent
Total extraction = 19,2 per cent + 74,2 per cent
= 93,4 per cent

The same extractions of gold were used in the calculation of the possible recovery of gold from the other runs (Table VI).

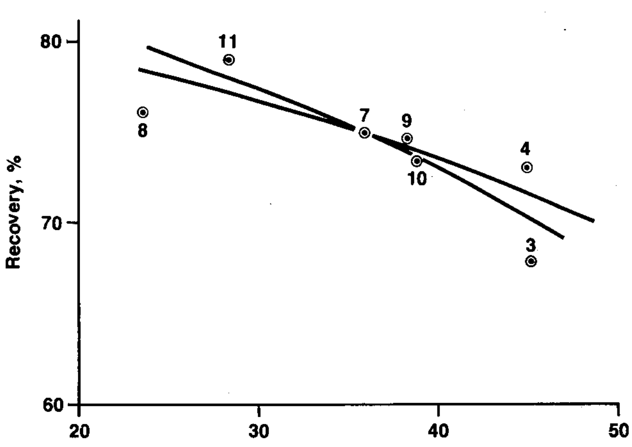

Fig. 3—Grade versus recovery for sulphur

After gravity separation, the gold extraction by cyanidation (94 per cent) may not be as high as that obtained by flotation (96 per cent). A possible explanation for this difference is that there is an intermediate pyrite particle that is too fine to be recovered by gravity separation and yet is still coarse enough to contain locked gold. However, there is a possibility that these results could be improved on a large-scale plant. The ore had to be crushed to a relatively fine size for the testwork on the pilot plant, and this crushing, together with the grinding in a rod mill, produced fine particles. The feed to the rod mill contained material 9,0 per cent finer than 75 μm, some of which would be lost in the classifier overflow without having the opportunity of being concentrated on the spiral.

Comparison between Spiral and Cone

A large plant using Reichert cones would have the advantage over a small pilot plant using a spiral in that the Reichert cone has a finer feed range than a spiral. Moncrieff[1] gives the lower size ranges of the feed for spirals and Reichert cones as 75 and 50 μm, respectively. When treating iron ore, Chong[4] found that Reichert cones gave higher recoveries than did spirals, and Ferree[5] states that Reichert cones are metallurgically more efficient for recovering material finer than 100 μm than are spirals, probably because there is less turbulence on cones. This is confirmed by the testwork at Blyvooruitzicht on Reichert cones that had shown an increase in recovery at a particle size of about 34 μm. Sijrak[6] reports

TABLE V
CYANIDATION TESTS ON PRODUCTS FROM RUN 4

Product	Gold g/t	Procedure	Residue g/t	Gold extraction %
Feed	5,71	Grinding to 70% < 75 μm and leaching with cyanide	0,72	87,4
Concentrate	102	Cyanidation	44,4	56,5
Concentrate	102	Grinding to 95% < 45 μm and leaching with cyanide	5,28	94,8
Concentrate	102	Grinding to 99% < 45 μm and leaching with cyanide	4,45	95,6
Concentrate	102	Roasting at 650°C for 16 h and leaching with cyanide	5,11	95,0
Concentrate	102	Grinding, leaching with cyanide, roasting, and final leaching	2,72	97,3
Combined overflow	1,16	Cyanidation	0,22	81,0

that, when three-deck shaking tables were replaced by Reichert cones, the recovery of scheelite increased, even though the cone theoretically falls short of tables as a concentrating device. It seems that the cones ironed out fluctuations in the process.

Conclusions

The tests showed that it is possible for the recovery of gold to improve from 87,4 to about 94 per cent if the pyrite is removed with a spiral from the circulating load of the mill and given preferential treatment. This recovery is not as high as that obtained by flotation tests carried out at the Union Corporation Research Laboratories, but increased recoveries may be possible from a Reichert cone plant.

From ore with a head grade of 6 g/t, flotation will recover 5,76 g and gravity separation 5,64 g. At a gold price of about R13 per gram, the return per ton of ore milled would be R74,88 for flotation and R73,32 for gravity separation, a difference of R1,56 per ton.

The capital cost of a flotation plant is expected to be three to four times that of a Reichert cone plant to treat the same quantity of feed, and the operating cost of a flotation plant about four times that of a gravity plant.

The results show that gravity concentration of Witwatersrand ores should be investigated further, particularly if the recovery of pyrite as a byproduct is being considered.

Some of the benefits from the gravity concentration of Witwatersrand ore bear repetition: the recovery of pyrite, as well as a considerable proportion of the gold, some uranium, and possibly some osmiridium. The concentrate could be subjected to very fine grinding and a lengthy cyanidation for recovery of the refractory gold that is locked in the pyrite. Alternatively, a reverse leach could be carried out to recover uranium and gold before the pyrite is used in the manufacture of acid.

However, it should be remembered that a gravity concentrator performs better when the valuable constituent of the ore is liberated and has a density sufficiently higher than that of the gangue material for good separation to be possible. It is also essential for the feed to be correctly prepared by the removal of any rubble that could block concentrate ports and other openings on the concentrator. Other important factors are the feed rate and the pulp density of the material being fed to roughing and cleaning cones: these should be kept constant at the optimum values recommended as a result of testwork or by the manufacturers.

TABLE VI
FINAL GOLD EXTRACTION

Run no.	Total gold extraction %
3	91,0
4	93,4
7	94,5
8	93,6
9	92,8
10	93,3
11	93,8

Acknowledgements

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