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## SOME DEVELOPMENTS IN THE EXTRACTION OF GOLD FROM WITWATERSRAND ORES


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12th December, 1975

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NATIONAL INSTITUTE FOR METALLURGY

NASIONALE INSTITUUT VIR METALLURGIE

REPORT ● VERSLAG

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investigator: P.A. Laxen

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## SYNOPSIS

The extraction of gold from most ores of the Witwatersrand has been consistently high – approximately 95 per cent – for many years. The present high price of gold has stimulated research and development into potential improvements of the standard procedures.

The occurrence of the gold and its association with other minerals is described briefly. The Witwatersrand reefs also contain uranium and pyrite, and some of the improvements to the standard gold-extraction process are associated with the recovery or concentration of one or both of these valuable constituents. For instance, the 'reverse-leach' process, in which acid treatment for the extraction of uranium precedes cyanidation, results in improved gold extraction. Some concentration techniques used on the present residues from cyanidation offer promise in improving the overall gold recoveries. Thucholite is an important carrier of residual gold, and its recovery from some residues by flotation appears to be economically justifiable. Wet high-intensity magnetic separation has given promising results in the laboratory and on the pilot plant, for the recovery of both uranium and gold from a number of cyanide tailings.

The benefits of gravity concentration and of the application of empirical modelling to the gold-extraction process have been demonstrated by a mining group. Some developments in instrumentation for control during cyanidation are discussed briefly.



## SAMEVATTING

Die ekstraksie van goud uit die meeste ertse van die Witwatersrand is al jare lank konsekwent hoog – ongeveer 95 persent. Die huidige hoë prys van goud het navorsing en ontwikkeling in verband met die moontlike verbetering van die standaardprosedures gestimuleer.

Die voorkoms van die goud en die assosiasie daarvan met ander minerale word kortliks beskryf. Die Witwatersrandse riwwe bevat ook uraan en piriet en sommige van die verbeterings aan die standaardgoudekstraksieproses staan in verband met die herwinning of konsentrasie van een van hierdie waardevolle bestanddele of albei. Die omgekeerde loogproses waarin suurbehandeling vir die ekstraksie van uraan sianidisering voorafgaan, lei byvoorbeeld tot beter goudekstraksie. Sommige konsentrasietegnieke wat vir die huidige sianidiseringsresidu's gebruik word, hou belofte in vir die verbetering van die totale goudherwinning. Thucholiet is 'n belangrike draer van residuele goud en die herwinning daarvan lyk of dit ekonomies geregverdig kan word. Nat magnetiese skeiding by 'n hoë intensiteit het belowende resultate in die laboratorium en in 'n proefaanleg vir die herwinning van sowel uraan as goud uit 'n aantal sianieduitskotte gelewer.

'n Myngroep het die voordele van swaartekragkonsentrasie en die toepassing van empiriese modellering op die goudekstraksieproses bewys. 'n Paar ontwikkelings in instrumentasie vir beheer tydens sianidisering word kortliks bespreek.

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## 1. INTRODUCTION

For many years the extraction of gold from Witwatersrand ores has been consistently above 95 per cent[1]. This high extraction can be attributed to the favourable mineralogy of this enormous suite of ores. Many of the flowsheets in these gold plants have been basically similar, differences in the type and size of equipment being among the major differences between various flowsheets.

In the years before World War II, the selection of a process, in over-simplified terms, was made from three flowsheets:

Flowsheet 1 — Milling → Gravity concentration → Tailings → Cyanidation
Gravity concentration → Concentrate → Amalgamation

Flowsheet 2 — Milling → Cyanidation

Flowsheet 3 — Milling in cyanide (rarely used).

The flowsheet that was selected depended, of course, on the particular ore, but depended also to an extent on the policy or preference of the metallurgists in a particular mining group.

After World War II, some of the Witwatersrand reefs were found to contain quantities of uranium that were economically feasible, either when they had been mined previously for gold, or when mining costs were based on both the gold and uranium contents. The pyrite content of some ores could also be recovered economically by flotation for use in the manufacture of the sulphuric acid used in uranium leaching. These factors introduced a new set of flowsheets into the treatment of gold–uranium ores:

Flowsheet 4 (termed 'normal')
Cyanidation → Acid leaching → Flotation and roasting

Flowsheet 5 (termed 'reverse')
Acid leaching → Cyanidation → Flotation and roasting

Flowsheet 6
Coarse grind → Flotation → Concentrate → Very fine grind → Acid leach → Cyanidation
Flotation → Tailings → Slight grind → Acid leach → Cyanidation

Flowsheet 7
Crushing and screening at about 1,27 cm (½ in) → (Fine, high grade) → Acid leaching → Cyanidation
Crushing and screening at about 1,27 cm (½ in) → (Coarse, low grade) → Cyanidation

A gravity-concentration step for gold could be inserted in any of flowsheets 4 to 7. Flowsheet 7 was found to be applicable to certain reefs from which the whole ore can be crushed and screened to yield a high-grade fraction rich in gold and uranium. However, there has been no uniformity in the treatment of Witwatersrand gold–uranium ores, and all four flowsheets are in use at the present time. Because the profit derived from the recovery of uranium has been small in comparison with that from the production of gold, and because the markets for uranium and sulphur have been uncertain, flowsheet 4 has tended to become the usual or normal process.

However, major new possibilities for Witwatersrand ores have emerged because of the spectacular rise in the price of gold, the greatly improved demand and price outlook for uranium, and the shortage of sulphur. These factors have contributed to an upsurge in research into all aspects of the various gold and uranium circuits by the mining industry and by the National Institute for Metallurgy (NIM).

The purpose of this report, which was originally presented to a meeting of AIME (New York, February 1975), is to summarize some of the developments that have arisen, and are likely to arise, from these recent investigations.

## 2. MINERALOGY

The major constituent of all Witwatersrand reefs is the barren, rounded quartz pebble associated with the placer origin of all these reefs. Virtually all the mineralization occurs in the fine-grained matrix between these pebbles. This matrix is mostly quartz and micaceous minerals such as sericite, chlorite, and pyrophillite[2]. Pyrite is a major accessory mineral in all the reefs. Gold, although of detrital origin, is held to be present mainly as 'secondary' gold replacing matrix minerals and occupying spaces and crevices. The gold is actually a gold–silver alloy, the silver content being about 10 per cent. The size distribution of the gold grains varies between individual reefs, and even between individual mines on the same reef[3].

Whereas gold is frequently found attached to, or in close proximity to, pyrite, the latter does not contain important quantities of gold or uranium as inclusions. A hydrocarbon mineral, thucholite – commonly known as 'carbon' – is present to some extent in all the reefs. In some reefs, e.g., the Carbon Leader Reef, this mineral is an important carrier of gold and uranium as inclusions[3].

Uranium in Witwatersrand ores occurs to a small extent as grains of primary uraninite and, in some ores, as secondary uraninite. Uraninite grains can contain gold particles as inclusions. In both head and leached-residue samples, the uranium values have been found to concentrate in the finest sizes. Small, but variable, amounts of osmiridium are present in almost all the ores.

## 3. MINING

Underground reef mining takes place at depths usually between 1000 and 3000 metres. Stoping heights, which depend on the width of the particular reef, can be as low as 1 metre. Mining is done by drilling and blasting. With a narrow, rich, friable reef, blasting causes a scattering of the fine valuable ore. Some of the most important recent improvements in gold recovery have, in fact, been due to the recovery of this fine, scattered ore by improved procedures of sweeping and washing of stopes[4]. Any change in mining technique to the use of a reef-cutting machine – as seems likely from the work of the Chamber of Mines Research Laboratory[4] – would have a marked effect on recoveries from the reefs that are narrow and friable.

## 4. DEVELOPMENTS IN METALLURGICAL CIRCUITS

It would be a fair prediction to say that more changes will be introduced into the gold circuits within the next two years or so than have been introduced in the past forty years. These will come as improvements to existing processes or circuits, or as additions to the circuits for the recovery or scavenging of additional gold and associated minerals from present residues.

### *4.1. IMPROVEMENTS WITH REVERSE LEACHING*

In most of the first uranium plants, the acid-leaching circuit followed the cyanidation circuit, i.e., the normal leach of flowsheet 4. The main advantage seen for this circuit was that it involved no change to the important gold circuit. With the demand for uranium as uncertain as it was, the insertion of an initial acid leach ahead of the cyanidation – flowsheet 5 – could have been unwise, despite the resulting small improvement in gold extraction.

This latter process, first indicated by laboratory-scale tests under Professor Gaudin at the Massachusetts Insitute of Technology[5], was developed to plant scale at Hartebeestfontein Gold Mine by metallurgists of the Anglo-Transvaal group[6,7]. The improved recoveries with reverse leaching obtained at Hartebeestfontein are given in Table 1.

The prior acid leach improves the extraction of gold by exposing the gold in acid-soluble minerals like uraninite and partly soluble silicates like chlorite, and also, it is presumed, by cleaning the surfaces of the gold grains through removing films and tarnishes. With the relatively small amount of extra gold recovered, it is difficult to say with certainty what the major reason for the improvement is. The acid-leaching step would not dissolve the pyrite or the thucholite in the ore, although it would attack pyrrhotite and some secondary sulphides, which can contain gold.

After the introduction of reverse leaching at the Hartebeestfontein Gold Mine, a number of mines on the West Rand – those that were treating the Carbon Leader Reef – introduced a size split on the crushed ore followed by reverse leaching of the fine fraction, as represented by flowsheet 7. The fine fraction, approximately 30 per cent by mass, contained two-thirds of the gold and more than half of

*TABLE 1*

***Improvements with reverse leaching – Hartebeestfontein Gold Mine****

| Undissolved gold, g/t | Normal leach | Reverse leach | Benefit | Benefit: R1,13 per g (i.e., $45 per oz) Rands per t | Benefit: R4,03 per g (i.e., $160 per oz) Rands per t |
|---|---|---|---|---|---|
| Laboratory results | 0,806 | 0,429 | 0,377 | 0,39 | 1,38 |
| Plant results | 0,677 | 0,350 | 0,327 | 0,33 | 1,20 |

*The monetary values were based on the Rand Dollar parity of January 1975

the uranium in the ore. This fine fraction it was found, benefited from an intensive treatment consisting of a fine grind and a reverse leach. The results from some recent laboratory tests[8] by NIM at one of these mines indicate the benefit to gold dissolution from reverse leaching of the high-grade fraction. Undissolved gold after a reverse leach was 0,67 g/t, as compared with 1,27 g/t after the normal leach.

The low-grade ore from the above split is low in uranium, and on the plant it is only cyanided. Laboratory tests that compared normal and reverse leaching on this low-grade ore fraction showed practically no benefit for gold extraction from a prior acid leach (0,29 g/t of undissolved gold in the normal leach and 0,27 g/t in the reverse leach). This indicates that the acid-soluble silicates and the uraninite must go largely into the high-grade friable ore.

In the course of a recent investigation by NIM on the possibility of improving the uranium extraction being obtained by a uranium producer, laboratory tests showed that gold extraction benefited from a conversion of the existing normal leach to a reverse leach[9]. An indication of the results obtained in the laboratory testwork is given in Table 2.

*TABLE 2*

***Buffelsfontein Gold Mine – improvements resulting from reverse leaching****

| | Normal-leach residue | Reverse leach residue | Benefit | Benefit: At R1,13 per g (i.e., $45 per oz) Rands per t | Benefit: At R4,03 per g (i.e., $160 per oz) Rands per t | Annual benefit at 200 000 t per month (before tax) Rands |
|---|---|---|---|---|---|---|
| Undissolved gold, g/t | 0,49 | 0,25 | 0,24 | 0,25 | 0,88 | 2 108 000 |

*The monetary values were based on the Rand Dollar parity of January 1975

These laboratory results were confirmed by small-scale tests on the plant, and the mine authorities have decided to go ahead with the alteration of their circuit to that of a reverse leach. The change-over is imminent. The gold plant residue will almost certainly show improvements similar to those obtained in the laboratory. A rough estimate of the cost of change-over was $400 000 (i.e. R290 000).

A further, and potentially useful, benefit that could be derived from reverse leaching is shown in the laboratory results obtained at NIM (Table 3). These are comparative tests[8] on a single ore at various grinds.

*TABLE 3*

*Comparisons at different grinds*

| Grind < 200 mesh, % | Undissolved gold, g/t | |
|---|---|---|
| | Normal leach | Reverse leach |
| 58 | 1,08 | 0,56 |
| 68 | 0,64 | 0,50 |
| 79 | 0,56 | 0,24 |

Therefore, if an acid leach is applied first, high extractions of gold can be obtained at coarse grinds. Uranium extractions are not improved by very fine grinding.

Not only does reverse leaching yield better gold extractions by cyanidation and allow coarser grinds to be considered for high gold extractions, but flotation subsequent to reverse leaching results in much better overall recoveries than those obtained from flotation subsequent to normal leaching. This is shown by the tailing values in the laboratory results[8] given in Table 4.

*TABLE 4*

*Flotation of residues after acid and cyanide leaching*

| | Grind < 200 mesh | Cleaner concentrate | | | Tailing | Head |
|---|---|---|---|---|---|---|
| | % | Total mass % | Gold value g/t | Gold in residue % | Gold value g/t | Calc. gold g/t |
| Normal leach | 58 | 4,6 | 14,2 | 52,3 | 0,63 | 1,26 |
| | 68 | 3,7 | 8,0 | 66,6 | 0,15 | 0,44 |
| | 79 | 3,3 | 6,3 | 59,1 | 0,15 | 0,35 |
| Reverse leach | 55 | 3,1 | 9,8 | 67,3 | 0,16 | 0,46 |
| | 68 | 3,7 | 8,0 | 66,6 | 0,15 | 0,44 |
| | 79 | 3,3 | 6,3 | 59,1 | 0,15 | 0,35 |

***4.2. DEVELOPMENTS IN CONCENTRATION***

For many years, gold concentration was synonymous with gravity concentration. However, the introduction of uranium recovery brought with it a novel and highly effective form of gold concentration, namely flotation.

***4.3. FLOTATION***

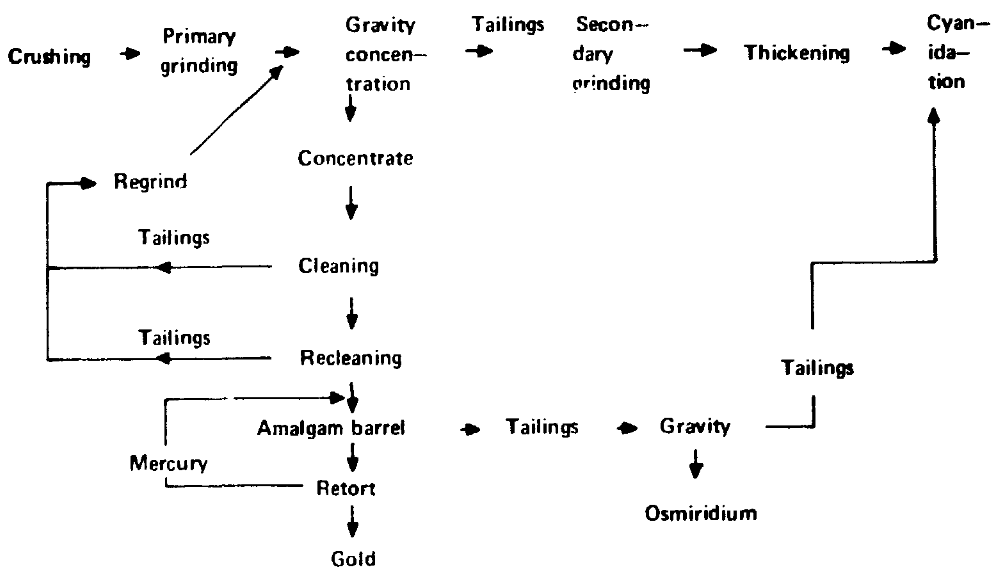
One of the major innovations in the role played by concentration in improving the recovery of gold from Witwatersrand ores was the introduction, again at Hartebeestfontein Gold Mine, of gold-sulphide flotation prior to reverse leaching, as shown in flowsheet 6. Approximately 80 per cent of the gold, 25 per cent of the uranium, and more than 80 per cent of the sulphur report in the flotation concentrate, which is approximately 3 per cent of the original mass. This circuit was introduced at Hartebeestfontein after the change from normal to reverse leaching[6].

The advantages claimed for flowsheet 6 include all the advantages of reverse leaching, the advantages of flotation at a coarse grind, the saving in grinding costs on the bulk of the ore, and the advantages of pyrite flotation without the disadvantages resulting from the presence of cyanide. Further advantages are found in the intensive treatment (namely, very fine grinding to material approximately 93 per cent smaller than 325 mesh and reverse leaching) that can be given to the small mass of concentrate that is rich in uranium and contains the bulk of the gold. By this treatment, more than 99,5 per cent of the gold in the flotation concentrate is dissolved, representing the highest percentage recovery from any product in the gold industry. This gold concentrate is given a double filtration and washing – in place of the normal single-stage filtration and washing – so that the loss of dissolved gold is reduced. The loss of dissolved gold in this overall flowsheet was better than that for a normal flowsheet.

One disadvantage is that the flotation process is least efficient on the very fine and the very coarse gold particles, with the result that two plants use gravity concentration after flotation to recover the coarse gold. This practice also results in the recovery of osmiridium.

### 4.4. GRAVITY CONCENTRATION

Gravity concentration has been, and still is, widely used on Witwatersrand gold circuits. This process has reduced the time of cyanidation by removing the large gold grains, and has taken a load from the cyanidation and zinc-precipitation steps. Whether amalgamation is more efficient than cyanidation is open to doubt. A simplified flowsheet for a gravity-concentration step on a gold mine would be as shown in flowsheet 8.




As much as 70 to 80 per cent of the total gold can be recovered in an initial rougher concentrate that contains about 10 per cent by mass of the initial feed. The proportion of gold recovered in the final gold concentrate, before amalgamation, is usually between 20 and 50 per cent of the total gold. The amount depends on the nature of the gold and its association in the ore.

The very efficient corduroy-blanket strakes, which were labour-intensive and offered poor security, were replaced by mechanical devices some ten years ago. In view of the major role played by gravity in gold circuits, the Anglo American Research Laboratories (AARL) recently undertook a detailed investigation of many of the devices used[10]. A pilot plant is being erected that is aimed at the improvement of existing circuits and the testing of new devices, including various types of cyclones and the Reichert cone. The latter device could give recoveries of up to 90 per cent of both the gold and the sulphur in a 10 per cent mass of concentrate on some ores. This is in line with testwork at NIM, which is aimed, in addition, at obtaining a bulk gravity concentrate from a coarse grind in open circuit[11]. Such a concentrate should recover, at low cost, a large proportion of the gold and sulphur as well as some of the uranium.

Therefore, the direction in which gravity concentration appears to be moving is towards bulk rougher concentrates for maximum recovery, followed by direct treatment of these concentrates. Several methods of treatment are being investigated, one of which would involve very fine grinding before acid leaching and cyanidation. All grades of both gravity and flotation concentrates have been shown to benefit from a prior acid leach[11].

### *4.5. DEVELOPMENTS IN AMALGAMATION*

Amalgamation still forms a part of every gravity concentration circuit, and improvements, in the form of prior treatment of the gold concentrate so that the surfaces are more readily available, are being tested. These procedures include the removal of films by treatment with nitrates, by fine grinding, and by fine grinding in cyanide. Because the amalgamation circuit treats up to 50 per cent of the total gold, any improvement in amalgamation efficiencies can have a marked effect on the overall efficiencies and recovered values. However, because of the poisonous nature of mercury, there is a strong move towards the elimination of amalgamation. Direct smelting of the concentrate is one of the procedures at present being tested as an alternative[12]. However, this procedure would mean that the osmiridium would be included in the smelted gold.

### *4.6. IMPROVEMENTS TO CYANIDE-TREATMENT PLANTS*

Every mine has a great deal of operating data on all aspects of the cyanidation process. Empirical mathematical models based on this accumulated data were established by metallurgists from the AARL for a number of plants[13]. These models have been used for the examination of operating variables like grind, specific gravity of leach pulps, cyanide concentrations, and pH values and temperatures of pulps, so that optimum values for each factor could be determined. This work, which has resulted in increased gold production (i.e., less gold in the residues) and in conservation of reagents, is being extended. The models have also been useful when plants were being enlarged and for 'trouble-shooting', and they have proved to be a useful basis for cooperation between research staff and operators.

A useful improvement to the existing cyanide process has been the recent development by NIM of a continuous monitor of the cyanide concentration in a continuous leaching system[14]. This instrument uses or records the electrochemical potential generated by a silver electrode immersed in an agitated cyanide pulp. A Lazaran reference electrode is used in conjunction with a silver electrode, both of which are immersed in the pulp in a specially designed holder. A gold electrode was found to be unsuitable, and the reason for the suitability of the silver electrode is not yet clear. The formation of a film on the silver electrode appears to be important, as is shown by the need to condition an electrode (i.e., to stabilize its output) when the environment changes, and by the ability of the electrode to withstand prolonged immersion. The electrode is suitable for use in a stable continuous-leaching system, but not in a batch system, where the environment is changing continuously. These KEGOLD electrodes have operated successfully on a plant scale, and are being marketed by a commercial organization.

Research work[15] is being undertaken by the Chamber of Mines Research Laboratory on the use of solvent extraction and ion exchange for the recovery of gold from pregnant solutions and pulps. Although this work is still in the investigational stage, a pilot plant that used solvent extraction on pregnant solutions was operated for a short period on one of the gold mines. An ion-exchange process is being investigated by NIM[16] for the recovery of the major part of the gold at present lost as soluble gold in the residual pulp and in barren solutions of grades higher than normal.

### *4.7. CONTRIBUTIONS BY MINERALOGICAL INVESTIGATIONS*

Mineralogists[3] at the Chamber of Mines Research Laboratory have studied in detail the distribution of gold in various reefs of the Witwatersrand by dissolving quartz and silicate minerals from reef samples in hydrofluoric acid. The size distribution of the gold in its original shape was studied, and its association with other minerals was deduced. The gold was found to be associated mainly with the silica minerals. Extremely fine gold was found associated with thucholite, and another major association was in the form of gold-coated pyrite grains. Gold grains in ground products were also studied by this technique.

Mineralogical investigations have always played an important role in explaining recoveries from the cyanidation process. Films on gold particles have frequently been suspected of causing trouble. Work at NIM with an electron microprobe identified many components of the films that were forming on gold particles held in a mill lining[17]. A recent major investigation by AARL[18] stressed the presence of films on the surfaces of gold particles in the total gold circuit. Films, composed mainly of iron oxide, were even identified on gold particles in the original untreated ore. With continued processing (e.g., milling and contact with iron surfaces), these films increased, confirming the belief that gold grains should be removed from the circuit (e.g., by concentration) as early as possible if contamination is to be avoided. Mineralogists frequently observe grains of 'free' gold in residues, but the proportion of free gold present is a matter of controversy. Laboratory tests at AARL showed that films of sulphide formed on pure silver and on silver gold alloys, but not on pure gold[18]. The Witwatersrand gold, which is gold with a silver content of about 10 per cent, could have been contaminated by sulphides during its history, which would have led to the formation of films on some particles. These films are considered to be seeds for the further growth of films, which could then act to prevent cyanidation and amalgamation.

It has been shown that part of the gold remaining in gold residues (about 0,1 g/t) is attributable to adsorption or re-precipitation of gold from cyanide solutions onto gangue minerals – mainly heavy minerals and clay minerals[19].

***4.8. SCAVENGING OPERATIONS AT THE END OF PRESENT CIRCUITS***

The end of the circuit is a much favoured area for the improvement of gold recoveries, mainly because any modifications there will not affect current operations, and also because scavenging operations there would recover gold that is apparently not amenable to, or cannot be recovered by, the existing cyanidation–amalgamation circuits. NIM has been actively involved in research on this aspect, and some instances are presented here.

***4.8.1. Flotation***

An investigation was conducted by NIM[20] on the residues from a mine that is operating on the Carbon Leader Reef on the Far West Rand. This ore is split into high- and low-grade fractions, as in flowsheet 7, with a reverse leach on the high-grade fraction. The thucholite, which is relatively light, concentrates mainly in the high grade section and tends to escape fine grinding. This is illustrated in Table 5 by the results of the assays done on sized fractions from the high-grade residue after acid leaching and cyanidation.

*TABLE 5*

***Analysis of size fractions of high-grade residue***

| Size fraction mesh | Total mass in fraction % | Gold | | $U_3O_8$ | |
|---|---|---|---|---|---|
| | | Assay g/t | Total amount in fraction % | Assay p.p.m. | Total amount in fraction % |
| > 65 | 0,63 | 17,7 | 23,62 | 584 | 4,94 |
| > 100 | 1,31 | 5,5 | 15,27 | 204 | 3,59 |
| > 150 | 7,85 | 0,88 | 14,64 | 65 | 6,85 |
| > 200 | 11,78 | 0,50 | 12,48 | 44 | 6,96 |
| < 200 | 78,43 | 0,20 | 33,99 | 74 | 77,66 |
| Calc feed | 100,00 | 0,47 | 100,00 | 75 | 100,00 |

The high gold values in the coarse fraction of this ore are attributable to the presence of thucholite particles. Screening of the total residue at 100 mesh would 'remove' a fraction containing approximately 40 per cent of the gold in the residue. Because it is carbonaceous, the thucholite can be concentrated by flotation using a frother alone. Sulphide flotation, after an initial flotation with frother, recovers further gold values mainly in the form of thucholite chats on sulphide particles. The results from a frother (i.e., 'carbon' flotation) and sulphide flotation are given in Table 6.

Thus, over 50 per cent of the gold in the residue could be recovered in a bulk flotation concentrate that is relatively low in pyritic sulphur. These results were confirmed on a pilot-plant operation based on combined carbon and sulphide flotation. Re-cleaning of the bulk concentrate resulted in a loss of gold values. Just how the gold values are concentrated in the coarse fractions of the residue is demonstrated in Table 7, which shows the distribution of values in size fractions in the flotation concentrate from the first of the eight cells of the pilot plant.

*TABLE 6*

***Products from 'carbon' and sulphide flotation***

| | 'Carbon' concentrate | Sulphide concentrate | Combined concentrate | Flotation tailing | Calculated feed |
|---|---|---|---|---|---|
| Total mass in product, % | 1,63 | 2,13 | 3,76 | 96,24 | 100,00 |
| Gold, g/t | 9,70 | 4,5 | 6,75 | 0,216 | 0,46 |
| Gold, % of total | 34,2 | 20,8 | 55,0 | 45,0 | 100,0 |
| $U_3O_8$, p.p.m. | 778 | 306 | 509 | 54 | 71 |
| $U_3O_8$, % of total | 17,8 | 9,2 | 27,0 | 73,0 | 100,0 |
| Sulphur, % | 9,62 | 15,70 | 13,1 | 0,21 | 0,69 |
| Sulphur % of total | 22,6 | 49,2 | 71 8 | 28,2 | 100,0 |

*TABLE 7*

***Size fractions of concentrate from Cell no. 1***

| Size fraction mesh | Percentage of total mass in fraction % | Gold Assay g/t | Gold Percentage of total in fraction % | $U_3O_8$ Assay p.p.m. | $U_3O_8$ Percentage of total in fraction % |
|---|---|---|---|---|---|
| >65 | 0,73 | 251,00 | 22,4 | 9600 | 14,0 |
| >100 | 0,99 | 206,00 | 24,9 | 9900 | 19,6 |
| >150 | 1,47 | 94,20 | 16,8 | 5780 | 17,0 |
| >200 | 2,08 | 36,90 | 9,4 | 2200 | 9,2 |
| >325 | 6,94 | 13,60 | 11,5 | 744 | 10,3 |
| <325 | 87,79 | 1,40 | 15,0 | 170 | 29,9 |
| Calc. feed | 100,00 | 8,20 | 100,0 | 500 | 100,0 |

Therefore, 73,5 per cent of the gold in the cell concentrate could be concentrated in 5,27 per cent by mass of the material larger than 200 mesh, or 85 per cent recovered in 12,21 per cent by mass of the material larger than 325 mesh.

The flotation of old accumulated tailings is a potential procedure that is being examined for the recovery of gold and sulphides for retreatment. It is of interest to note here that the Anglo American Corporation is reported to be erecting a flotation plant (having a monthly capacity of one million tonnes) at South African Lands Gold Mine to treat the old slimes dams in this area. The pyrite (for the manufacture of sulphuric acid) and the recovered gold justify the large capital expenditure required.

***4.8.2. Wet High-intensity Magnetic Separation***

Uraninite and some of the host minerals for uranium and gold are feebly magnetic, and dry high-intensity magnetic separation has been used in the laboratory[21] to obtain concentrates for mineralogical examination. However, there were no prospects for the application of high-intensity magnetic separation on an industrial scale until the advent of the wet high-intensity separator[22], which has now opened up interesting possibilities. Laboratory tests have been done on several ores at NIM, and some promising results have been obtained. The work has been done mostly on residues resulting from the cyanidation of gold ores, and it has been shown that wet high-intensity magnetic separation can effect a considerable recovery of both the residual gold and the uranium.

The amount of material reporting in the magnetic concentrate varies from ore to ore. The minerals from Witwatersrand ores that are likely to report in the concentrate are uraninite, chloritoid, chlorite, and minerals with iron stains. Quartz, pyrite, sericite, and pyrophyllite are mainly rejected. Gold itself is non-magnetic, although iron – metallic or oxide – embedded in a gold grain could possibly result in its capture. Concentrates of between 5 and 20 per cent by mass have been obtained from different ores, with a recovery of both the uranium and the residual gold of between 40 and 70 per cent.

A pilot-plant operation was undertaken on a low-grade ore fraction with a small Carpco wet high-intensity magnetic separator using a matrix of iron balls[8,23]. It was found necessary, before the pulp went to the machine, to screen off the woodchips and coarse material at about 20 mesh, and to remove the fine iron from grinding by use of low-intensity magnets. From a residue with an initial gold value of 0,47 g/t, the following concentrate was obtained:

| | |
|---|---|
| Percentage of total by mass | 19,3 per cent |
| Gold content | 1,4 g/t |
| Percentage of total gold | 57,4 per cent |
| Percentage of total $U_3O_8$ | 49,0 per cent |

These results are less favourable than some that have been obtained on other residues, but are quoted because the conditions prevailing at the mill concerned are such that serious consideration is being given to the adoption of wet high-intensity magnetic separation as a part of the programme to improve recoveries of gold and uranium.

Grinding and reverse leaching of these magnetic concentrates give recoveries of up to 90 per cent of the contained gold and uranium.

The selection of the most suitable machine and matrix material is the next important step in this development, to be followed by the installation of such a machine for large-scale tests at one of the mines. Further recent laboratory tests have shown that there is scope for a substantial increase in the efficiency of concentration by wet high-intensity magnetic separation, and work is being done at NIM[24] in attempts to obtain this increase.

## 5. SUMMARY AND CONCLUSIONS

Any improvements to the various types of gold circuits in use on the Witwatersrand are limited to the recovery of the small amount of gold (approximately 3 to 5 per cent of the total) at present left in the residues, which amounts to approximately 0,2 to 0,8 g/t. However, the present high price of gold has made even small improvements in overall recovery worth while. At \$160 (i.e. R114) per ounce, the amounts of gold left in the residues quoted previously have a value of between \$1,03 (i.e. R0,74) and \$4,11 (i.e. R2,93) per tonne of ore. Recent investigations into the improvement of existing circuits have centred mainly on gravity and amalgamation circuits. Other investigations have shown that improved overall extractions can be obtained by acid leaching for uranium before cyanidation, or by the addition of a flotation step or wet high-intensity magnetic separation (for the recovery of gold and uranium) to the end of existing circuits.

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