

# The recovery of sulphur, uranium, and gold from residues

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## SYNOPSIS

The report describes the Amuran project in the Welkom area, which is being conducted by six members of the Anglo American Group. The project comprises three plants for the flotation of pyrite, a twin-stream uranium plant in which 'high-grade' slimes and pyrite can be leached separately, a pyrite-burning sulphuric acid plant, and a plant for the recovery of gold from calcines. Details of these plants including capital costs and estimated production are given.

Mention is made of the adsorption of gold onto activated charcoal and wet high-intensity magnetic separation as possible suitable processes for these residues.

## SAMEVATTING

Die verslag beskryf die Amuran-projek in die Welkom-gebied wat deur ses lede van die Anglo American-groep onderneem word. Die projek behels drie aanlegginge vir die flottasie van piriet, 'n dubbelstroomuraanaanleg waarin "hoëgraadse" slyk en piriet afsonderlik uitgelooë kan word, 'n swawelsuuranaanleg wat piriet brand, en 'n aanleg vir die herwinning van goud uit kalsiene. Besonderhede soos die kapitaalkoste en beraamde produksie word oor hierdie aanlegginge verstreë.

Die adsorpsie van goud op geaktiveerde houtskool en nat magnetiese skeiding by 'n hoë intensiteit word as moontlike geskikte prosesse vir hierdie residu's genoem.

## Introduction

After the two uranium plants at the Welkom and President Steyn Gold Mines were closed in the early 1960s, consideration had to be given to the potential recovery of uranium from residual slimes. It was decided that certain gold-plant circuits should be split into high- and low-grade streams, the split being achieved by a screen separation ahead of fine crushing. Although the streams are referred to as *high grade* and *low grade*, it should be appreciated that both are in fact of low uranium grade by world standards.

Such a separation results in a high-grade uranium stream that is 30 per cent of the total mass and contains approximately 48 per cent of the uranium. These high- and low-grade fractions are milled and treated separately in the gold plants, and the residues are stored separately for future treatment. The gold plants at the Welkom, President Brand, and Free State Saaiplaas Mines in the Orange Free State have operated on this basis since 1967.

In addition to this, a heavy-medium separation plant of 70 t/h capacity was in operation at Free State Saaiplaas from 1963 to 1970. This produced a high-grade product of 26 per cent by mass containing 60 per cent of the uranium in the feed, which was treated at the uranium plant of Virginia mine.

A number of studies were carried out in the 1960s into the production of uranium from gold-plant residues. These culminated in the erection of a uranium plant at President Brand with a monthly capacity of 180 000 t, its feed sources being current and deposited high-grade residues from the split gold plants. The plant was completed during 1971 at a cost of 9 million rands, but was

never commissioned owing to the depressed state of the uranium market at the time.

A 9 t/h bulk uranium flotation pilot plant was operated at President Brand during 1972 as part of an investigation into improving the potential for profitable uranium-plant operations. The pilot plant treated residual slimes from President Brand and Welkom Gold Mines, and produced a concentrate of 30 per cent by mass containing 70 per cent of the uranium, 90 per cent of the sulphur, and between 70 and 90 per cent of the gold in the feed.

As a result of marked increases in the prices of both gold and uranium during recent years, the latter being sparked off by the world energy crisis of late 1973, further studies were carried out into the feasibility of the retreatment of gold-plant residues. The technical and economic viability was again proved.

A decision was taken to commission the President Brand uranium plant as soon as possible, and to erect a 1000 t/d sulphuric acid plant and three pyrite flotation plants with a combined capacity of 1,5 million tons per month. Flotation operations would be on a large scale, both to maximize the production of uranium and gold and to provide sufficient pyrite, and hence acid, to support a possible several-fold expansion of uranium leaching operations. Consideration of the location of feedstocks, as well as of other constraints, led to the choice of three separate flotation plants, which are situated at the Free State Saaiplaas, President Steyn, and President Brand Mines. The plants are of conventional design but make use of large-scale equipment wherever possible. Sulphur would be upgraded from 0,7 to 1,0 per cent in the residues to approximately 33 per cent in the pyrite concentrates, which would be transported to the uranium plant at President Brand Mine by road tanker.

In addition, the uranium plant would be expanded to an additional monthly capacity of 30 000 t to treat pyrite concentrates, and the gold plant at President

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Brand would be expanded to an equal additional monthly capacity to treat calcines.

The concurrent installation of flotation and sulphuric acid plants was considered for the following reasons.

- (i) Additional revenue would accrue from the secondary 'byproduct' gold production. Froth flotation was at the time the best method for the recovery of gold from residues.
- (ii) The recovery of sulphur would provide a captive source of sulphuric acid for the uranium and flotation plants, as well as a surplus for outside sale.
- (iii) Pyrite concentrates must be calcined for optimum gold recovery, and the manufacture of acid from the sulphurous gases would prevent air pollution.
- (iv) Additional secondary uranium would be produced from the pyrite concentrates by the flotation up-grading of uranium values from residues that would otherwise be of too low a grade for economic treatment.

The installations are as follows.

- (i) A 570 000 t/month plant at Free State Geduld to treat residues from Free State Geduld and Western Holdings.
- (ii) A 570 000 t/month plant at President Steyn to treat residues from President Steyn, President Brand, Free State Saaiplaas, and, at a later stage, Welkom Gold Mines.
- (iii) A 360 000 t/month plant at President Brand to

treat uranium-plant residue, as well as residues from the President Brand and Free State Saaiplaas gold plants.

This project, involving six members of the Anglo American Group in the Orange Free State (O.F.S.), is known as the Joint Metallurgical Scheme or Amuran.

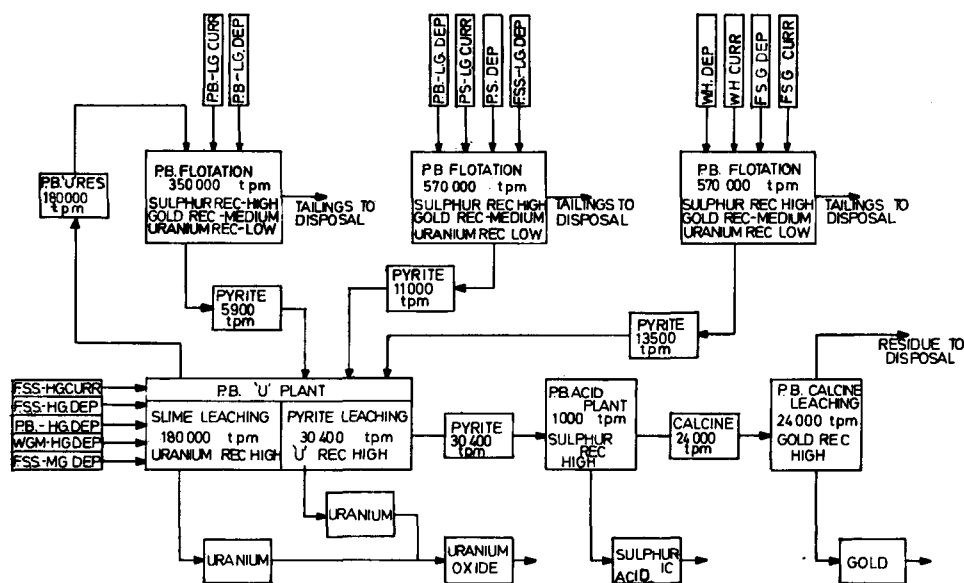
The treatment and production schedules have been based on a 15-year life, covering the period 1977 to 1991. A block diagram of the Amuran scheme, which depicts projected average rates is shown in Fig. 1.

### Sources of Feed

The basic philosophy is that only residues of low uranium grade would be treated in the flotation plants. High-grade residues must first be leached for uranium, or must be stockpiled for future treatment if there is insufficient uranium plant capacity.

Existing slimes dams were surveyed, and their gold, uranium, and sulphur contents were evaluated by means of auger drilling and assaying. These results were used together with projections of future arisings to ensure that the reserves were sufficient to support an operation of at least 15 years. It is predicted that the sulphur and other grades will drop in the future on some mines, and the proportion of current residues and deposited slimes treated will have to be scheduled accordingly to maintain a steady production rate of pyrite.

Slimes deposited in existing dams are recovered by



\*Fig. 1—The O.F.S. Joint Metallurgical Scheme

<b>PB</b>	<b>President Brand</b>	<b>Curr</b>	<b>Current residues from gold plants</b>	<b>FSS</b>	<b>Free State Saaiplaas</b>
<b>FSG</b>	<b>Free State Geduld</b>	<b>HG</b>	<b>High uranium grade</b>	<b>Dep</b>	<b>Deposited slimes on tailings dams</b>
<b>WH</b>	<b>Western Holdings</b>	<b>LG</b>	<b>Low uranium grade</b>	<b>rec</b>	<b>Recovery</b>
<b>PS</b>	<b>President Steyn</b>	<b>MG</b>	<b>Medium uranium grade</b>	<b>tpm</b>	<b>Tons per month</b>
<b>WGM</b>	<b>Welkom Gold Mine</b>				

\*We apologize to readers for the poor lettering, errors, and spelling mistakes in the diagrams illustrating this paper. Unfortunately, there was no time for editing, re-lettering, and amendments.

Hon. Editor.

sluicing down with high-pressure water jets using remote-controlled monitoring guns.

Even with three separate flotation plants, a considerable amount of transfer pumping is involved between mines and between plants.

### Recovery of Pyrite

Extensive bench and pilot-scale tests were carried out on the various residues to be treated. Operations on two pilot plants installed at Free State Geduld and President Brand indicate that the best results are to be obtained by flotation from an acidic pulp (approximate pH 4) with sodium mercaptobenzothiazole as the collector and TEB as the frother. Long acid-conditioning times are required to counteract the depressant effects on the pyrite of the residual lime and cyanide in the current gold-plant residues. Copper sulphate has also to be added.

The presence of variable but significant amounts of pyrophyllite in the ore complicates matters, and a depressant for the shale is required. Yellow dextrin appears to be the most cost-effective reagent for this purpose.

Flotation with xanthate at a slightly higher pH value gave promising results in the laboratory, but these could not be reproduced on the pilot plants. Further testwork has been aimed at improving the low recovery of uranium into the final concentrate, and at increasing the recovery of soluble gold in the feed pulp by adsorption onto activated carbon followed by flotation of the carbon together with the pyrite.

Gold and uranium recoveries are inversely related to the sulphur grade of the pyrite concentrate, and operations will be aimed at producing as low a sulphur grade as can be burnt in the acid-plant roasters.

The flotation plants are of conventional design but make use of large-scale equipment wherever possible.

Incoming pulps are metered for accounting purposes, and are then diluted to the density of the flotation feed and acidified in a series of  $10 \times 20$  m pachucas, providing a total of 5 to 6 hours of acid-conditioning time. The bulk of the flotation reagents are then added in mechanically agitated conditioners, although there is provision for further additions at various points in the circuit.

A nominal roughing and scavenging time of 20 minutes has been allowed for in  $14 \text{ m}^3$  cells. Provision has been made for the routing of the initial rougher concentrates direct to the final product if sufficiently high grades are obtained. Scavenger concentrates are recycled to the head of the circuit, and rougher concentrates are brought up to the required final grade of concentrate in a flexible arrangement of cleaner, recleaner, and cleaner-scavenger cells, all of which are  $8,5 \text{ m}^3$  machines. It is anticipated that the cleaner-scavenger tailings will be of sufficiently low grade to be discarded together with the rougher-scavenger tailings.

With the anticipated head grade of between 0,7 and 1,0 per cent sulphur, a recovery of 80 to 85 per cent, and a concentrate grade of approximately 33 per cent sulphur, a pyrite concentrate constituting only about 2 per cent by mass of feed is obtained.

As water conservation is important in the area, the tailings are thickened in five 107 m-diameter traction thickeners for the three flotation plants and are then neutralized and pumped to large new dams, which have been designed by outside consultants for high deposition rates and maximum water recovery. The thickeners operate under acidic conditions so that the recycling of overflow water to the feed-preparation section will minimize the acid required for pH control.

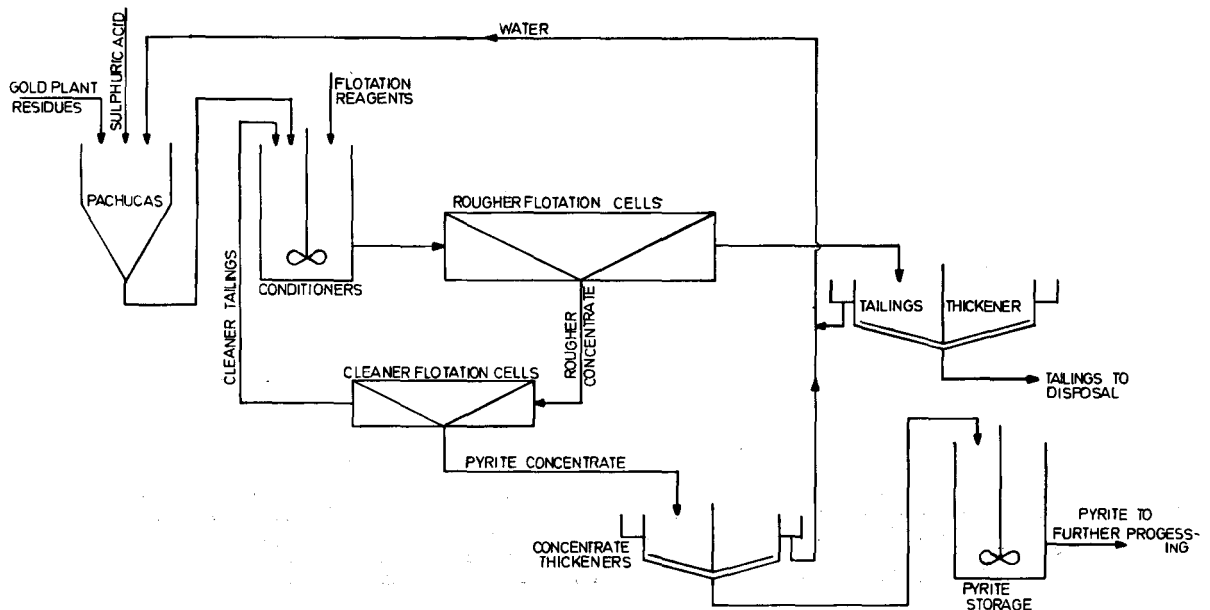


Fig. 2—Simplified flowsheet of the flotation plants

(GOLD PLANT RESIDUES should read DEPOSITED AND GOLD PLANT RESIDUES)

The thickener underflow is transferred to two neutralizing pachuca, where lime is added. Lime is received in the form of lump or pulverized quicklime, which is slaked in a separate section and added to the neutralizing pachuca as a slurry. This reacts with any remaining acid before the residue is pumped away to the slimes dam.

The pyrite concentrates are thickened, and are then stored in mechanically agitated tanks before being transferred to a central collection point at President Brand. Concentrates from the Free State Geduld and President Steyn plants are transported in specially designed road tankers, while those from President Brand are pumped.

A simplified flow diagram is depicted in Fig. 2.

### Recovery of Uranium by Leaching of Slime

The President Brand uranium plant was designed to treat 180 000 t of gold-plant residues per month by the conventional processes of leaching with sulphuric acid, rotary-drum filtration, solvent extraction, and ammonium diuranate precipitation.

Feed to the plant comes from the repulpers of the gold-plant residue in the case of current residues, and from monitoring stations at the slimes dams in the case of deposited residues. These pulp streams are received into three pachuca tanks that are used for metering the plant input and as surge storage ahead of the non-acid filters.

The first step in the process is the filtration and washing of the incoming residues and their repulping with the dilute solution of acid and uranium that comes from the second-stage acid filters. This preliminary filtration has the dual purpose of removing the lime-and-cyanide-bearing solution emanating from the gold plants, and of providing a means of controlling the pulp density to the leaching stage. Filtration is carried out on twelve 5,5 × 5,4 m non-acid rotary-drum filters operating in parallel. The filtrate is returned to the gold plant. Flocculant is added to the pulp ahead of filtration as a filter aid.

Leaching is carried out in three parallel streams of four pachuca each, which provide a leaching time of approximately 24 hours. Controlled amounts of sulphuric acid, manganese dioxide, and steam are added to the first pachuca, the sulphuric acid being obtained from the contact acid plant. It is added in sufficient quantity to maintain a required residual acid concentration in the pulp, the addition being automatically controlled on the basis of pulp conductivity. Manganese dioxide is supplied as lump pyrolusite ore, which is ground and slurried in a separate section. It is an oxidant, which is added to ensure that all the uranium is oxidized to a state that is amenable to sulphuric acid leaching. The addition rate is automatically controlled on the basis of the pulp redox potential. Steam is produced in coal-fired boilers and is injected live into the pulp to increase the temperature to between 60 and 65°C and thus improve the dissolution of the uranium.

The dissolution of uranium from these slimes is of a high order, viz 90 to 92 per cent. The role played by thucholite during the dissolution is not fully understood

and is being investigated, especially as affected by the mode of agitation during leaching.

After leaching, the pulp is transferred to a surge pachuca ahead of the primary acid filters. Primary acid filtration is carried out on twelve 5,5 × 5,4 m acid-duty rotary-drum filters operating in parallel. The wash liquor on these filters is the filtrate from the second-stage acid filters, which consists of dilute acid and uranium. Guar flocculant is added to the pulp ahead of the filtration as a filter aid.

As the soluble losses would be too high if the primary filter cake were discarded, it is repulped with fresh water and refiltered. There is a second surge pachuca between the primary and secondary filtration; and secondary filtration is carried out on a further twelve 5,5 × 5,4 m acid-duty filters. Fresh water is used as the wash on these filters.

The filter cake from the secondary filters is repulped with calcine thickener overflow from the acid plant and is transferred to the flotation plant.

Filtrate from the primary acid filters contains most of the uranium dissolved in the leaching stage, and is the valuable product from the leaching and filtration operations. It is referred to as pregnant solution. Apart from uranyl sulphate and residual sulphuric acid, the pregnant solution contains a number of dissolved impurities that would contaminate the final product if it were precipitated directly. The solution is therefore purified and upgraded by solvent extraction or, more correctly, liquid ion exchange.

The pregnant solution carries some fine solid particles through from the acid filtration stages, and these must be removed since they would otherwise cause problems in the solvent-extraction process. Clarification is carried out by gravity sedimentation, followed by polishing in Niagra pressure filters precoated with diatomaceous filter-aid.

The organic phase used in the solvent-extraction process is a mixture of an alkyl amine, which is the active constituent, iso-decanol, which prevents the formation of a third phase, and paraffin as a diluent and carrier. The amine selectively forms a complex with the uranyl sulphate and extracts it from the pregnant solution when the organic and aqueous phases are mixed. When the phases are then allowed to separate, this complex remains in the organic phase.

To ensure a high recovery of uranium, the extraction step is carried out in four mixer-settler stages operating with countercurrent flow of solvent and pregnant solution. The barren aqueous solution from this process, termed the *raffinate*, contains the bulk of the impurities and very little uranium. It joins the final residue in the neutralizing pachuca for the flotation-plant tailings.

The loaded solvent from the extraction stages carries through some impurities, which are scrubbed out in another section of four mixer-settlers using dilute ammonium hydroxide and sulphuric acid. As some uranium is also stripped from the organic phase during scrubbing, the aqueous phase is recycled to the head of the extraction section.

The purified solvent is then stripped of its uranium content in a further section of four mixer-settlers

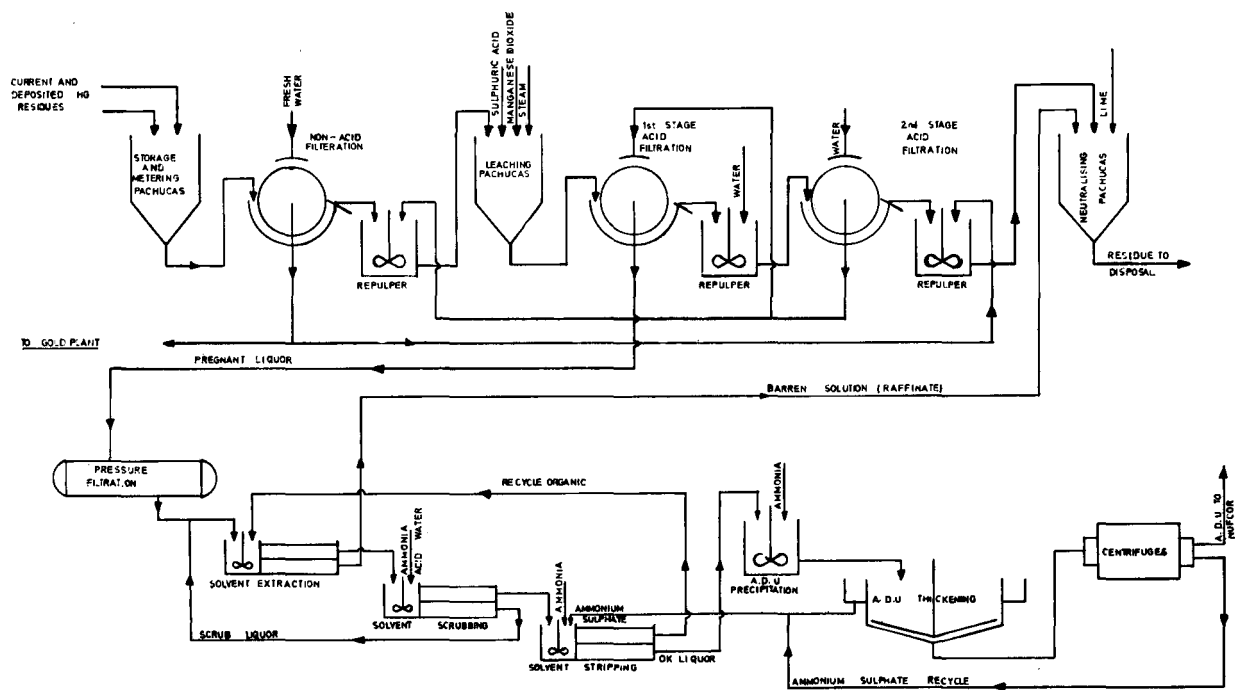


Fig. 3—Simplified flowsheet of the uranium plant at President Brand

using a solution of ammonium hydroxide and ammonium sulphate. The stripped solvent is subsequently treated with a solution of caustic soda and sodium carbonate, which removes any solvent poisons, and is then recycled to the extraction section. The organic phase thus operates in a closed circuit, a small amount of make-up being necessary to compensate for dissolution and entrainment losses to the raffinate.

The aqueous phase from the stripping section carries the purified uranyl sulphate and is termed the *O.K. liquor*. Uranium is precipitated from this solution in the form of ammonium diuranate (ADU) by the addition of gaseous ammonia. The precipitate is thickened, firstly in a conventional thickener and then washed in a two-stage centrifugal process. The ammonium sulphate liquor from these operations is recycled to the solvent stripping stage, and the thickened ADU slurry is transported by tanker to NUFCOR for calcining (Fig. 3).

The major sections of the plant may be doubled or even trebled in the future, depending on the uranium market. Studies have again been carried out to ensure that there will be sufficient high- and medium-grade residues available to satisfy the feed requirements.

### Recovery of Uranium by Pyrite Leaching

Sections of the plant have been extended to provide for the recovery of uranium from the pyrite concentrates. Concentrates from the three flotation plants are metered and mixed at a central receiving station, which also acts as a surge storage ahead of leaching. The leaching operation is similar to that for slime, although the higher grades and lower tonnages involved make it economic for the pyrite concentrates to be subjected to a more intensive treatment.

Extensions to the acid-filtration section differ from the

original plant in that four 60 m<sup>2</sup> horizontal-belt filters have been installed. The entire filtration and counter-current washing operation is carried out in a single stage on these filters.

Pregnant solution from the pyrite section is treated together with the solution from the original plant in the existing solvent-extraction plant, which has been up-rated to handle the increased volume. Fig. 4 is a simplified flow diagram of the pyrite-treatment section. The testwork carried out to date indicated that uranium can be recovered from the pyrite concentrates at acceptable levels and rates without any undue problems.

### Production of Sulphuric Acid

The acid plant, which is a Lurgi design, is capable of treating approximately 1000 t of pyrite concentrate per day to produce 1000 t of acid per day. (Fig. 5).

Leached pyrite from the extension to the uranium plant will be stored ahead of the acid plant in large mechanically agitated tanks. There is also provision for the stockpiling of pyrite in a dry form, and for its reclamation and the introduction of purchased pyrite into the system.

The pyrite is fed as a slurry into two fluidized-bed roasters operating at 800 to 850°C, where it reacts with the oxygen in the fluidizing air to form sulphur dioxide and an iron oxide calcine. The roasters have been specially designed to provide a long contact time and to minimize carry-over of calcine, so that the subsequent recovery of gold can be optimized.

Bed overflow calcines are cooled in water-jacketed rotating drums before being quenched and transferred to a thickener. Roaster gases pass through dry cyclones, and are then quenched and scrubbed in venturi scrubbers. Sulphur dioxide dissolved in scrubber effluent is recovered in Peabody strippers.

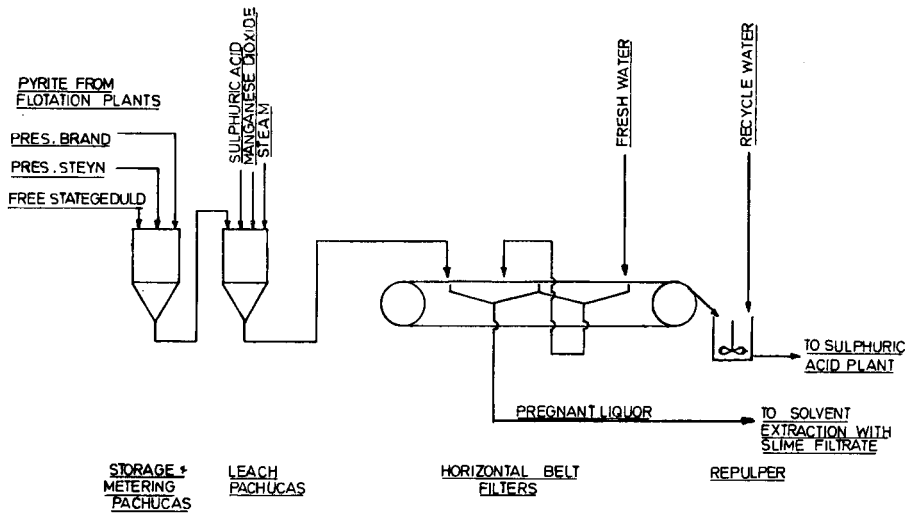


Fig. 4—Simplified flowsheet of the leaching of pyrite in the uranium plant at President Brand

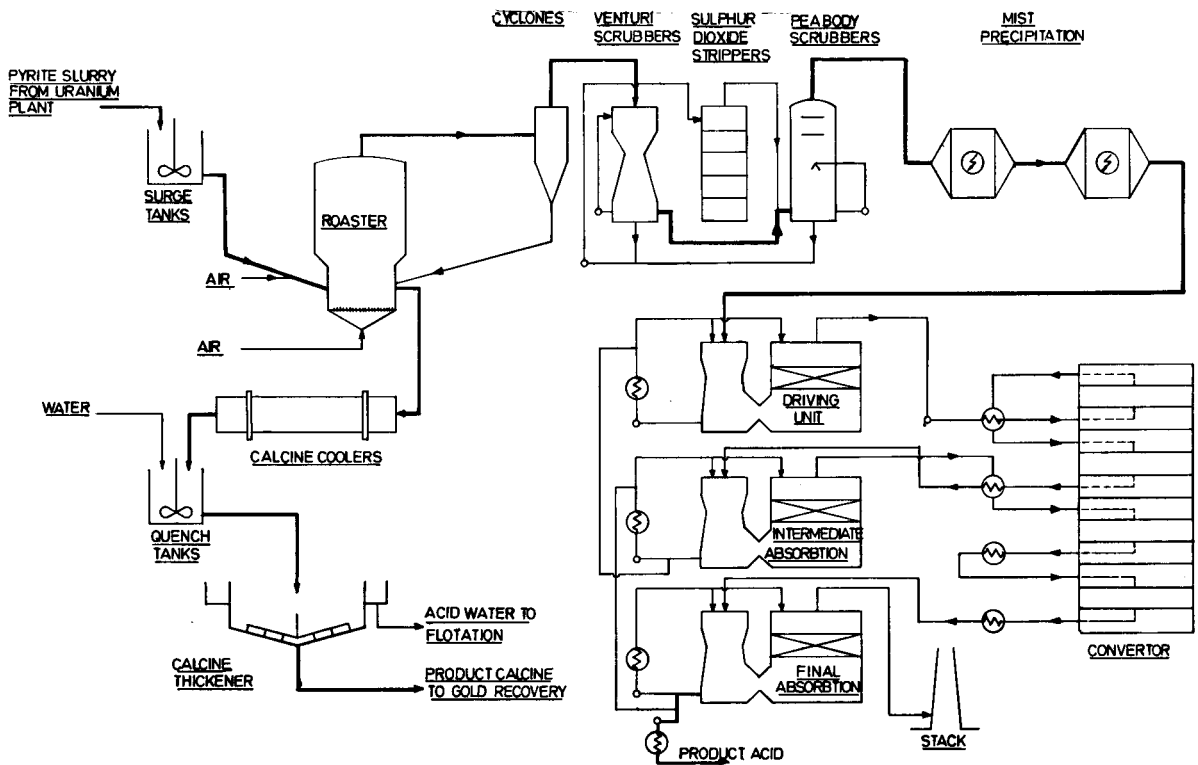


Fig. 5—The sulphuric acid plant at President Brand  
(DRIVING UNIT should read DRYING UNIT)

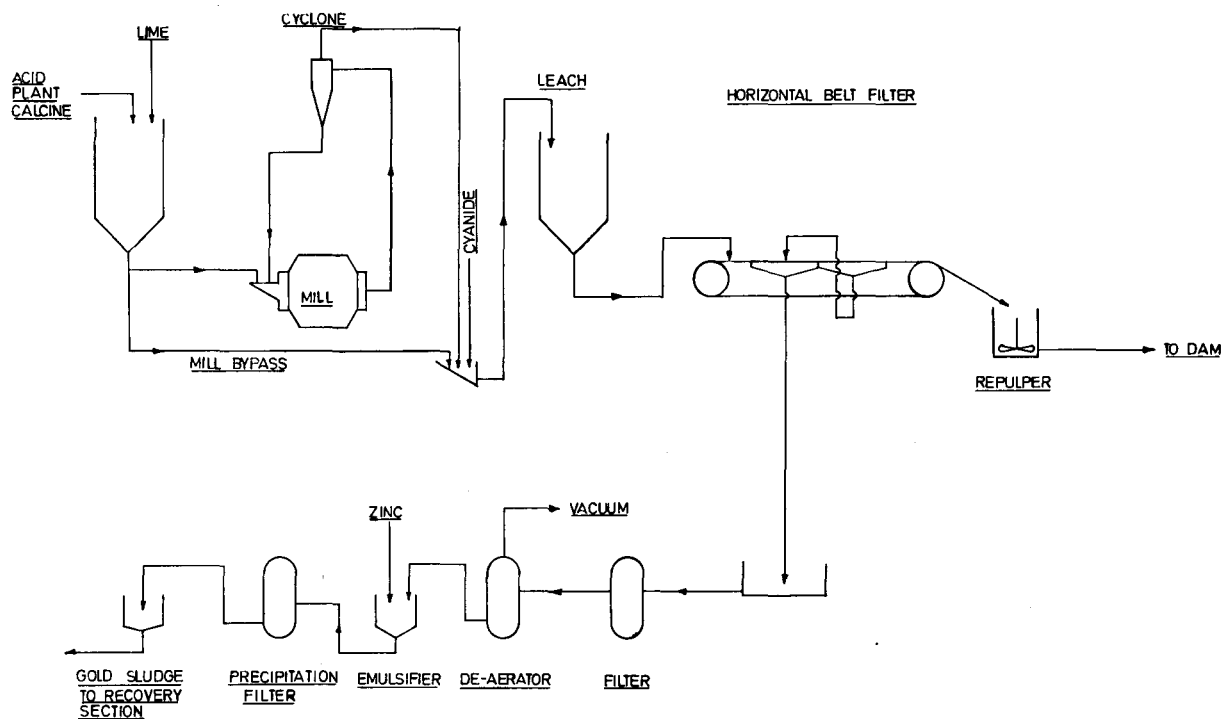


Fig. 6—Simplified flowsheet of the extension to the gold plant at President Brand

Thickened calcine is pumped to the gold-plant extension for gold recovery, while the slightly acidic thickener overflow is used for dilution of the feed on the President Brand flotation plant, the surplus being neutralized and used for monitoring.

The cooled sulphurous gases are cleaned further in two stages of electrostatic mist precipitators before entering the contact section. The plant has two parallel streams up to the precipitators, and a single stream thereafter.

A double catalysis-process contact section has been installed, giving an overall sulphur dioxide conversion of 99,7 per cent. Other notable features of this section of the plant are venturi spray-type drying and absorption towers, stainless-steel shell-and-tube acid coolers, and a 100 m-high stack installed to comply with air-pollution regulations.

The product acid is stored in four 5000 t tanks before being pumped to the uranium plant and transported by road to the flotation plants or loaded into rail tankers for delivery to outside customers.

#### Recovery of Gold from Acid-plant Calcines

Calcine from the acid-plant thickener is reground in a ball mill before being leached. The leaching process used is similar to the conventional cyanidation process for the dissolution of gold, but the consumption of reagents is considerably higher than for the treatment of slime. Two 60 m<sup>2</sup> horizontal-belt filters have been installed to separate the residue from the gold-bearing solution. A sand pressure filter is employed for clarification and a pressure filter for zinc-dust precipitation (Fig. 6). The gold precipitate is then treated by conventional means.

The calcine residue will initially be disposed of together with the tailings from the President Brand flotation plant, although investigations are being made into the

possible recovery of the small amounts of base metals contained in these residues.

#### Materials of Construction

Owing to the very high content of chlorides in the underground waters of the Orange Free State (2000 p.p.m.), the choice of suitable materials of construction for metallurgical plants rates a high priority. The stainless steel commonly used (grade 316L) is not resistant to acidic liquors having a high chloride content in the presence of ferric iron. The attack of liquors with a chloride content of 1000 p.p.m. on lead also presents a problem.

In the sulphuric acid plant it was necessary to employ special types of materials of construction to counter the effect of chloride corrosion. The vanadium pentoxide catalyst can also be deleteriously affected by the hydrochloric acid that is carried over by the converter gases if the chloride content of the cooling water exceeds certain acceptable limits.

#### Capital Costs

The overall cost of the project, excluding the original cost of the President Brand uranium plant, is approximately 88 million rands, which is split roughly as follows:

Monitoring and feed pumping . . . . .	6,5
Flotation plants . . . . .	41,0
Rehabilitation and commissioning of existing uranium plant . . . . .	6,5
Pyrite reception . . . . .	1,5
Extension of uranium plant . . . . .	4,5
Acid plant . . . . .	25,0
Calcine-leaching plant . . . . .	3,0

TOTAL . . . . . R88,0 million

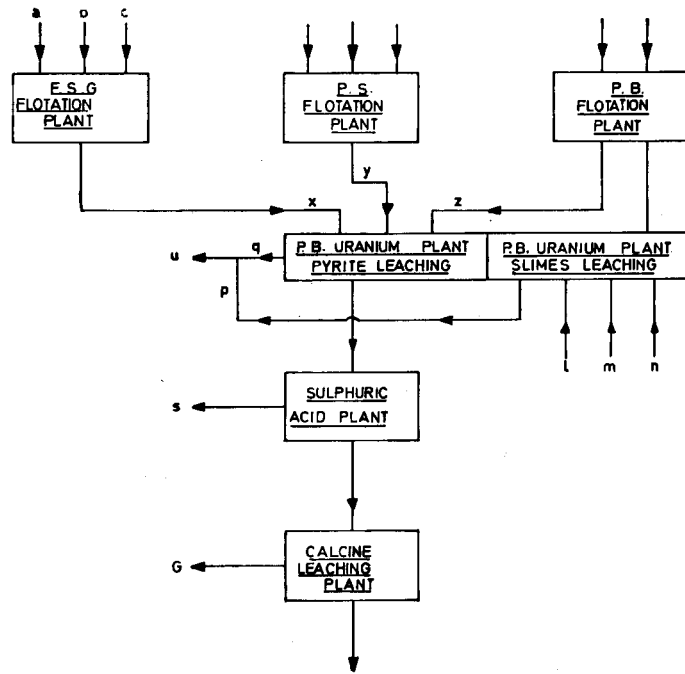


Fig. 7—The allocation of the end products from the Joint Metallurgical Scheme

### Metallurgical Accounting

The metallurgical accounting system has been designed to distribute the costs and revenues from the complex as a whole, without bias, among the various prime sources of slime. All the incoming slimes to the three flotation plants and the uranium plant from the different contributors are individually measured and sampled by mass flow meters and automatic samplers so that the amounts of gold, uranium, and sulphur emanating from the different suppliers can be assessed.

The production during a treatment month is allocated *pro rata* to the input amounts of minerals from prime sources for that same treatment month. No consideration is given to variations in stockpiles or plant retentions.

A number of cost centres have been established within the complex, and the costs incurred under these centres during a treatment month are allocated to the various suppliers *pro rata* to the slime tonnages emanating from the suppliers.

The calculations are simple and repetitive, and have been applied to a computer by use of a terminal at President Brand Mine.

### Allocation of Products

Fig. 7 gives a simplified flowsheet and shows the principle of allocating the production of gold, uranium, and sulphuric acid.

For the three flotation plants, the minerals are allocated on the same basis. For the production of U as  $U_3O_8$  from an indicated production of  $p$  kg originating from slimes leaching and  $q$  kg originating from pyrite leaching, the allocated production from slimes is

$$\frac{p}{p+q} \text{ kg.}$$

and the allocated production from pyrite is

$$\frac{q}{p+q} \text{ kg.}$$

Then the allocation to slimes source  $l$  from the slimes-leaching section is

$$\frac{p}{p+q} \cdot \frac{l}{l+m+n} \text{ kg.}$$

and the allocation to source  $a$  via the pyrite leaching is

$$\frac{q}{p+q} \cdot \frac{x}{x+y+z} \cdot \frac{a}{a+b+c} \text{ kg.}$$

The small amounts of uranium recovered in the President Brand flotation plant from the uranium-plant residual slimes complicates the calculations, but the allocation is still similar to the principle stated.

### Allocation of Costs

In any one of the cost centres within the complex, the basis of allocation of costs is on prime inputs.

For a single cost centre supplied by three prime sources, the costs charged to that centre are allocated in accordance with the percentage utilization of the plant by each of the three sources. In the cases where the cost centre is not a plant receiving material from prime sources, the allocation of costs from that centre is done by the application of percentage utilization factors in series from the cost centre back to the prime source.

Thus, the costs under a particular cost centre can be allocated across any number of prime sources. The Cost Centre President Brand Flotation is taken as an example:

Sources	Uranium-plant slimes ( $D$ ) themselves consisting of tonnages $A+B+C$ from three slime sources
	Two other sources $E$ and $F$ .
Cost	$C_{PBF}$



Allocation to source *A*

$$= \frac{D}{D+E+F} \cdot \frac{A}{A+B+C} \cdot C_{PBF}$$

#### Utilization of Computer Facilities

It can be seen from the above that the accounting calculations are ideally suited to computerization. Representatives from the Mines, Head Office, and Anmercosa Computer Services have compiled programmes suitable for the calculations and run manual checks on these programmes. It is expected that the accounting for the complex will be done wholly by the computer towards the end of the year.

A terminal is to be installed at President Brand Mine, connected to the Welkom computer facility of Anmercosa Computer Services Limited. This facility will be available to staff from the complex towards the end of the year.

#### Production Estimates

The studies have been based on a 15-year project life, covering the period 1977 to 1991. For the recovery factors and grades used in the studies, the average annual production will be approximately

Acid (total)	350 000 t
Uranium (ex slime)	345 t of U <sub>3</sub> O <sub>8</sub>
Uranium (ex pyrite)	350 t of U <sub>3</sub> O <sub>8</sub>
Uranium (total)	695 t of U <sub>3</sub> O <sub>8</sub>
Gold	3 570 kg.

These estimates are based on predicted recoveries of

Sulphur	85 % for flotation 0,36 t of sulphur per ton of acid.
Uranium	85 % for leaching of slime 80 % for leaching of pyrite 32 % for flotation 10 % for flotation of uranium-plant residue.
Gold	51 % for flotation 90 % for leaching of calcine.

The unit operating costs of the various processes have been estimated to be as follows in mid-1977:

Flotation	R0,63 per ton of feed
Acid	R15,91 per ton of acid (including pyrite charged at R25 per ton of sulphur)
Gold	R3,71 per ton of calcine.

#### Recovery of Soluble Gold

Another worth-while aspect of these intended operations is the recovery of the soluble-gold content of the slurries fed to the flotation plants. This will be done by the adsorption of this gold onto activated carbon and the flotation of the carbon with the pyrite.

The roasters of the acid plant have been specially designed to ensure a long residence time for the carbon and calcines in the roasters so that there will be complete combustion of the carbon and thucholite.

#### Wet High-intensity Magnetic Separation

Wet high-intensity magnetic separation (WHIMS) is currently being tested on bench and pilot-plant scale with three different types of separators, and the results have been encouraging.

There are indications that a uranium recovery of 40 per cent into a 7 to 10 per cent mass of concentrate can be obtained, i.e., from the residue of a flotation pilot plant, an overall uranium recovery of 30 per cent of the uranium fed to the flotation plant.

The very fine grain sizes of a large proportion of the uranium minerals present another major problem since this fine size dictates a closely spaced matrix. This type of matrix blocks easily, and tonnage rates and recovery efficiencies are therefore drastically reduced. Hence, an extensive feed-preparation flowsheet is required to eliminate tramp oversize and strongly magnetic materials, and thus economic recoveries are still in the balance.

Leaching at high temperature and pressure is no better than conventional leaching at 65°C, and it is doubtful at this stage whether this new method will find application in the Orange Free State.

#### Conclusions

The project described represents a step forward in that gold, pyrite, and additional uranium are recovered from gold-plant residues, many of which were considered valueless until recently. This additional production will contribute significantly to the profitability of the various mines that furnish the material to be processed.