The heavy-minerals plant at Palabora Mining Company—a low-grade, high-tonnage gravity concentator*

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SYNOPSIS

The heavy-minerals plant at the Palabora Mining Company treats 25 to 30 kt of new feed daily. The U_3O_8 value of the ore averages approximately 40 p.p.m. The circuit consists of four stages of upgrading on 68 Reichert cones, followed by a jig and 30 table decks.

The paper describes how feed-distribution problems have been overcome and the methods used for monitoring and controlling the cone performance.

The adaptation of a standard jig for the recovery of fine minerals is discussed, and a method of 'table agglomeration, for the recovery of coarse refractory copper sulphide is described. The tables consist of polyurethane-covered fibreglass decks, and their operation is controlled according to measurements of the uranothorianite content of the concentrates and middlings by means of a nuclear scintillation counter.

The upgrading ratio of the uranium mineral by gravity concentration is 1000:1, baddeleyite (ZrO_2) being recovered as a byproduct.

SAMEVATTING

Die swaarmineraleaanleg van die Palabora Mining Company behandel daagliks 25 tot 30 kt nuwe toevoermateriaal. Die U_3O_8-waarde van die erts is gemiddeld ongeveer 40 d.p.m. Die kring bestaan uit vier opgraderingstrappe op 68 Reichert-keëls gevolg deur 'n wipsif en 30 tafeldekke.

Die referaat beskryf hoe distribusieprobleme met die toevoer oorkom is en die metodes wat vir die monitering en beheer van die werkverrigting van die keëls gebruik word.

Die aanpassing van 'n standaardwipsif vir die herwinning van fyn minerale word bespreek en 'n metode van tafelagglomerering vir die herwinning van growwe refraktêre kopersulfied beskryf. Die tafels bestaan uit veselglasdekke wat met poliuretaan beklee is en hul werking word volgens die metings van die uranotorianiet-inhoud van die sentrate en middelskot deur middel van 'n kernflitsteller beheer.

Die opgraderingsverhouding van die uraanmineraal deur swaartekragkonsentrasie is 1000:1 en baddeleyiet (ZrO_2) word as 'n neweproduk herwin.

Introduction

The existence of the Phalaborwa Igneous Complex was known to Black tribes, who mined and smelted copper and iron in this area long before the arrival of the White man. Dr Hans Merensky's father, the Reverend Alexander Merensky, placed Loolekop on the map in the late 1800s, and Schwellnus wrote a thesis on the mineral wealth of the area in 1910. The first active mining of the phosphate and vermiculite deposits commenced in 1931.

It was the vermiculite that first sparked off Dr Hans Merensky's interest in the area in 1938, and the Transvaal Ore Company started exploiting the vermiculite deposit. It is interesting to note that a subsidiary of Rio Tinto obtained the agency for selling this vermiculite in Europe, and Rio Tinto's Palabora connection results from this association.

In 1952 a geological unit attached to the South African Geological Survey established the presence of the radioactive mineral uranothorianite in the Phalaborwa Igneous Complex. Because of its very low grade, the exploitation of this orebody for uranium was not economically viable. However, the exploration work showed the extent of the copper mineralization and was largely responsible for the subsequent mining of the orebody by Palabora Mining Company Ltd.

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Brief Description of the Orebody

The rocks comprising the Phalaborwa Igneous Complex (Fig. 1) are pyroxenite and felspathic pyroxenite, syenite, olivine, diopside–phlogopite pegmatoid, fenite, and carbonatite, most of which were intruded into the granite–gneiss of the Archaean Complex, and some are considered to be metasomatic.

The pyroxenite was intruded first, followed by syenite and then by a centrally located core of transgressive carbonatite. The pyroxenite, which has the largest area distribution, forms a north–south kidney-shaped body approximately 15 km^2 in extent.

An east–west elongated pipe-like body of carbonatite occurs near the centre of the pyroxenite body. This is surrounded by a serpentine (olivine) magnetite–apatite rock to which the name foskorite has been given.

The carbonatite forms the host for the copper sulphide minerals mined by Palabora Mining Company. Small concentrations of baddeleyite (ZrO_2) are present in the foskorite and carbonatite. Irregularly distributed concentrations of uranothorianite are present, particularly in the transgressive carbonatite.

Viability of Exploitation

Exploration and development work carried out in 1952 showed that the exploitation of the Phalaborwa orebody for uranium recovery was not economically viable. However, with the establishment of Palabora Mining Company as a copper-mining enterprise treating (initially) 40 kt/d, the amount of uranium in the ore became significant.

Laboratory-scale investigations in 1967 led to the

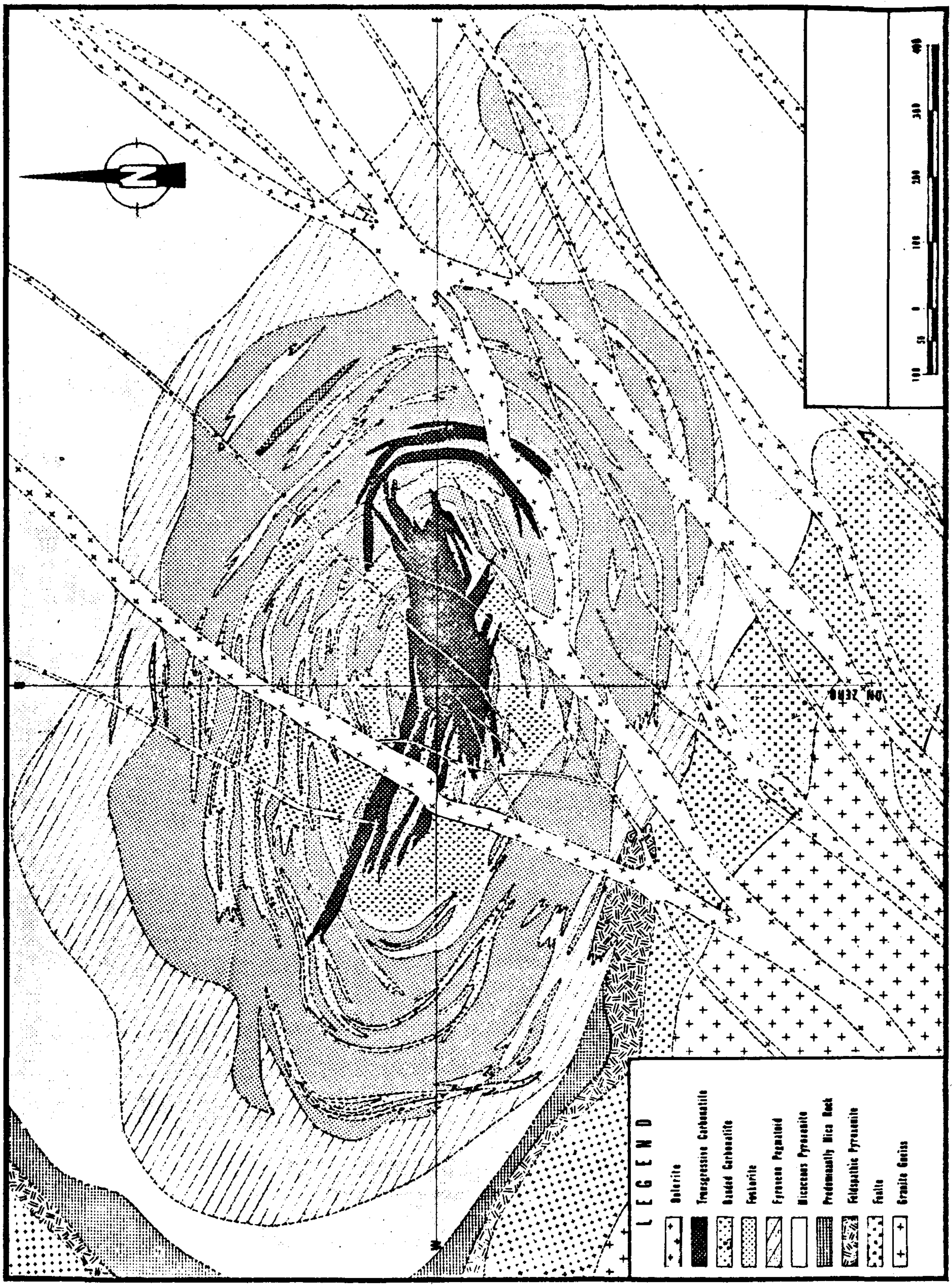

Fig. 1—The Phalaborwa Igneous Complex

TABLE I
TYPICAL PARTICLE-SIZE DISTRIBUTION OF GRAVITY-PLANT FEEDS

Particle size	Cyclone feed				Cyclone underflow				Cyclone overflow	
	PMS tails lo-phos		PMS tails hi-phos		Rougher cone feed lo-phos		Rougher cone feed hi-phos		Lo- + hi-phos	
μm	% mass	Cum	% mass	Cum	% mass	Cum	% mass	Cum	% mass	Cum
+500	6,3		15,5		9,6		12,1		1,0	
+300	12,7	19,0	20,2	35,7	15,7	25,3	20,2	32,2	2,5	3,5
+212	8,2	27,2	11,7	47,4	10,9	36,2	16,5	48,8	2,7	6,2
+150	14,3	41,5	16,1	63,5	17,8	54,0	16,9	65,7	5,5	11,7
+106	18,8	60,3	11,5	75,0	11,7	65,7	13,9	79,6	6,5	18,2
+ 75	6,9	67,2	7,8	82,8	15,2	80,9	10,5	90,1	9,1	27,3
+ 45	11,1	78,3	6,1	88,9	9,0	89,9	6,1	96,2	16,0	43,3
− 45	21,7		11,1		10,1		3,8		56,7	

construction of a pilot plant to recover the heavy minerals from the copper-plant tailings and provide concentrates for tests on chemical extraction. As a result of this test programme, the decision was taken to build a gravity-concentration plant for the concentration of uranium that would be capable of treating the entire tailings from the copper-recovery plant.

Uranium-recovery Circuit

The processing can conveniently be regarded as three broad steps: gravity concentration, chemical extraction, and calcination.

In the gravity-concentration section of the plant, primary magnetic-separator tailings from the copper concentrator are treated in Reichert cones to produce a heavy-minerals concentrate consisting largely of uranothorianite, baddeleyite, and carbonatite gangue. This concentrate is upgraded by tabling and jigging into a uranothorianite concentrate, a baddeleyite concentrate, and table tailings.

Uranium and thorium are extracted from the uranothorianite concentrate by leaching with nitric acid, and are then separated in a solvent-extraction circuit.

The thorium is discarded via the raffinate stream, and the uranium is precipitated as ammonium diuranate and finally calcined for shipment.

Feed Preparation

The tailings from the copper concentrator are pumped to the magnetic-separation plant, where some 20 kt of magnetite are removed daily. The non-magnetic fraction is treated in a bank of feed-preparation cyclones prior to being fed to the Reichert-cone plant. This is necessary for the removal of the 45 μm slime and excess water associated with the magnetite-removal step.

The cyclone assembly consists of 10 operating and 2 standby cyclones supplied by Amberger Kaolinwerke. These units are constructed in cast polyurethane elastomer, and are fitted with 305 mm vortex finders and 152 mm apices.

The particle-size distribution of the cyclone feed, overflow, and underflow is shown in Table I. *Lo-Phos* and *Hi-Phos* refer to the low-phosphate and high-phosphate fractions respectively.

Gravity Concentration

Reichert-cone Circuit

The Reichert-cone circuit is shown in Figs. 2 and 3. The cyclone underflow from the feed preparation is fed to 24 Reichert cones, which comprise the rougher circuit. This feed averages approximately 900 kt/m, and the rougher circuit comprises 6 cones in the Lo-Phos circuit and 18 in the Hi-Phos section.

The Hi-Phos feed is pumped through a two-way pressure splitter, each side feeding a nine-way rotary distributor. The Lo-Phos feed is split in a six-way rotary distributor, and the pulp from these rotary distributors gravitates to the individual cones.

Two metering systems monitor the tonnage and pulp density of the flows to the two circuits. Any deviations in the pulp density of the feed are noted and compensated for by manual adjustment of the feed-preparation pumps.

The pulp density of the feed to the roughers is maintained between 1,65 and 1,69, which is between 62 and 64 per cent solids. (The relative density of the solids is 2,8.)

Of the 24 rougher cones, 22 have a 3 DS (double cone, single cone) configuration and 2 are of the 4 DS type (Figs. 4 and 5).

The first and second concentrates from the rougher cones pass on to the cleaner section, while the third take-off, or in the case of the four DS cones the third and fourth take-offs, are treated as middlings in the scavenger circuit.

On the scavenger cones, only the first concentrate passes on to the cleaner circuit, the two middlings cuts being recirculated. The scavenger cones are also fed via rotary distributors. The concentrates from the rougher and scavenger cones are pumped to a twelve-way rotary distributor to the cleaner cones. These cones have a 2 DSS DSV configuration (Fig. 6) and the first two concentrates are combined and treated in four Krupp double-drum magnetic separators for the removal of any residual magnetite before they are fed to the recleaner circuit. The cleaner-cone middlings are recirculated, and the tailings are returned to eight cleaner tailing scavenger cones, which operate in the same mode as the other scavenger cones.

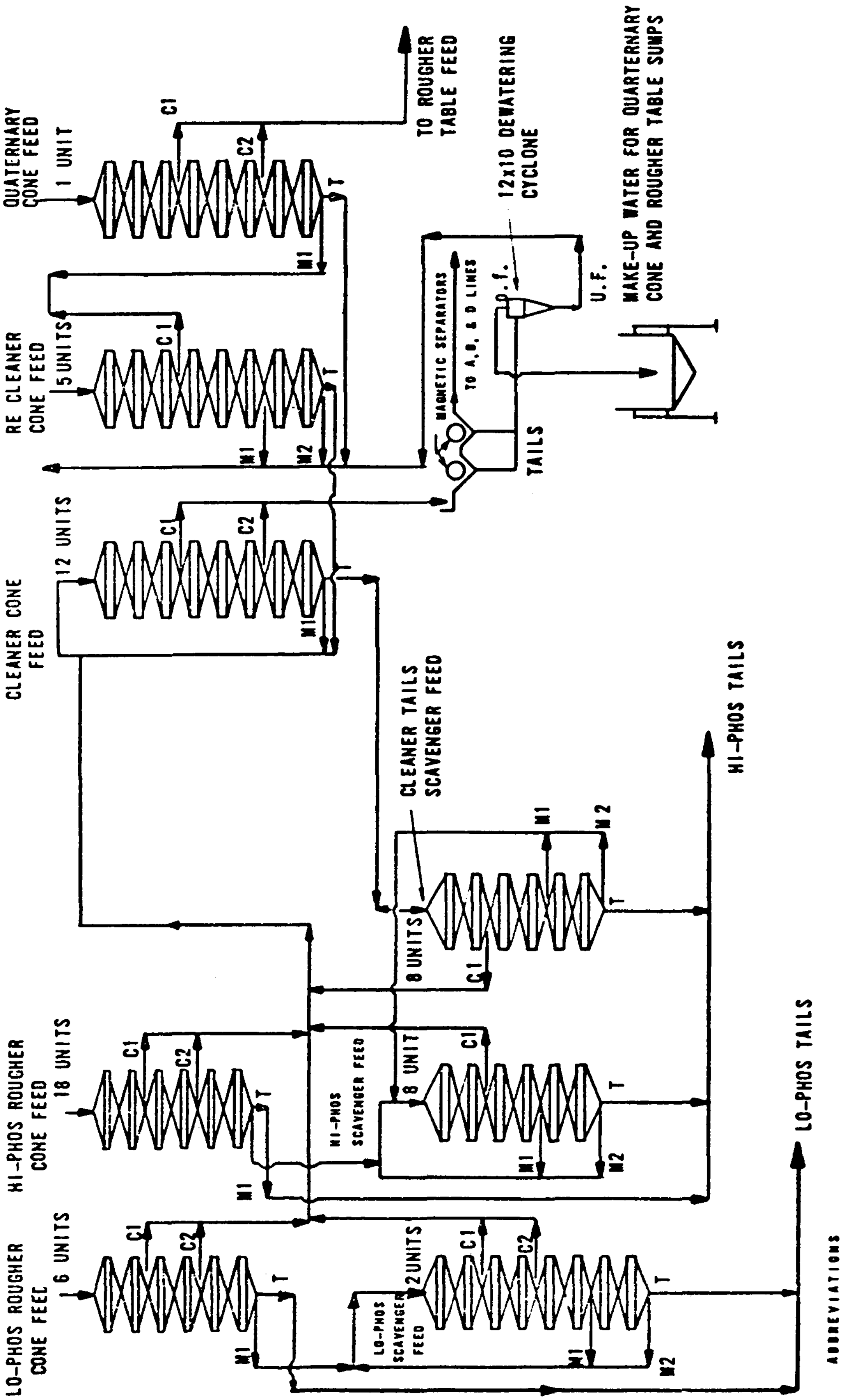

Fig. 2—The Reichert-cone circuit

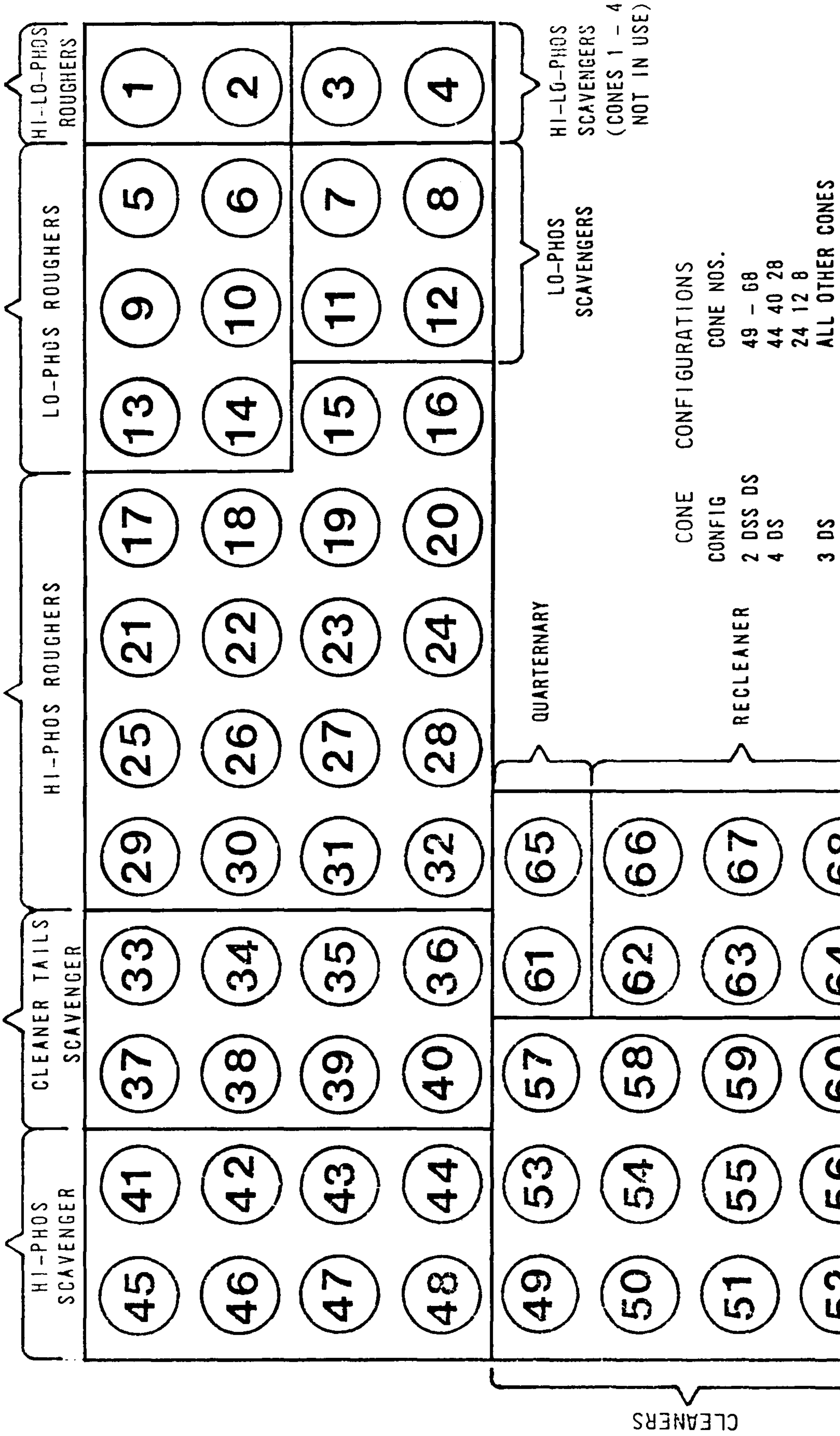

Fig. 3—The Reichert-cone arrangement

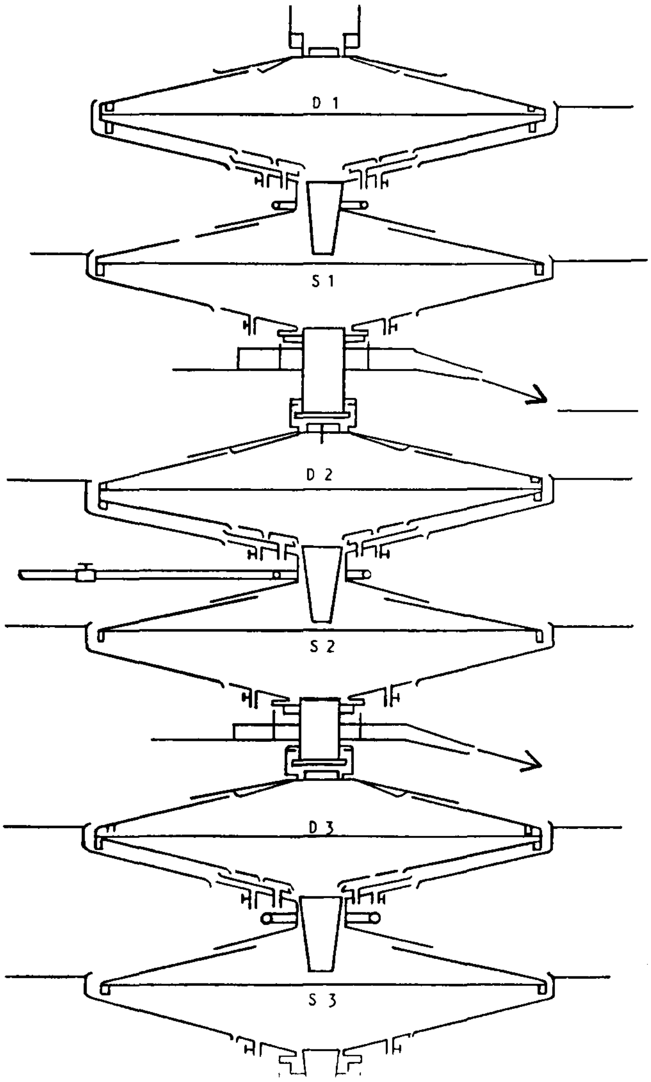

Fig. 4—The 3 DS Reichert-cone configuration

The third stage of the cleaner cones has an adjustable tulip insert, which can be set to adjust the circulating load in this circuit (SV—single cone with adjustable collection ring).

The recleaner circuit consists of 5 cones that have a 2 DSS DSV configuration, the only difference being that the concentrates from the second stage pass over trays that further upgrade and reduce the volume of this product.

As in the case of the cleaner cones, the first two concentrates provide the feed to the quaternary cone, while the middlings are recirculated and the tailings returned to the feed sump of the cleaner cone.

The recleaner cone concentrates are fed to the quaternary cone, which has a 2 DSSV DSV configuration. The variable concentrate take-offs are set to provide a constant tonnage of feed (about 28 t/h) to the jig and table circuits. Again, the middlings are recirculated and the tailings returned to the recleaner circuit.

Jig and Table Flotation Circuit

This circuit (Fig. 7) is new and has been commissioned only recently.

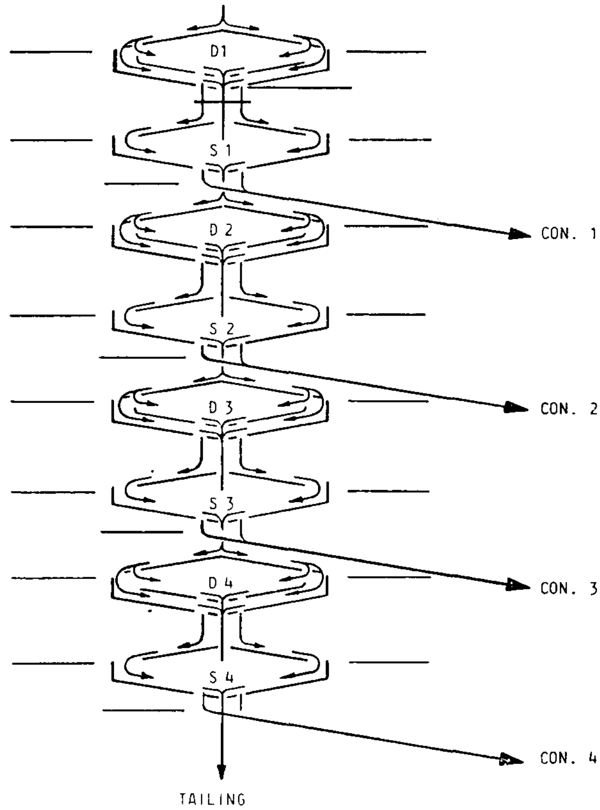

Fig. 5—The 4 DS Reichert-cone configuration (cones 44, 40, 28 24, 12, and 8)

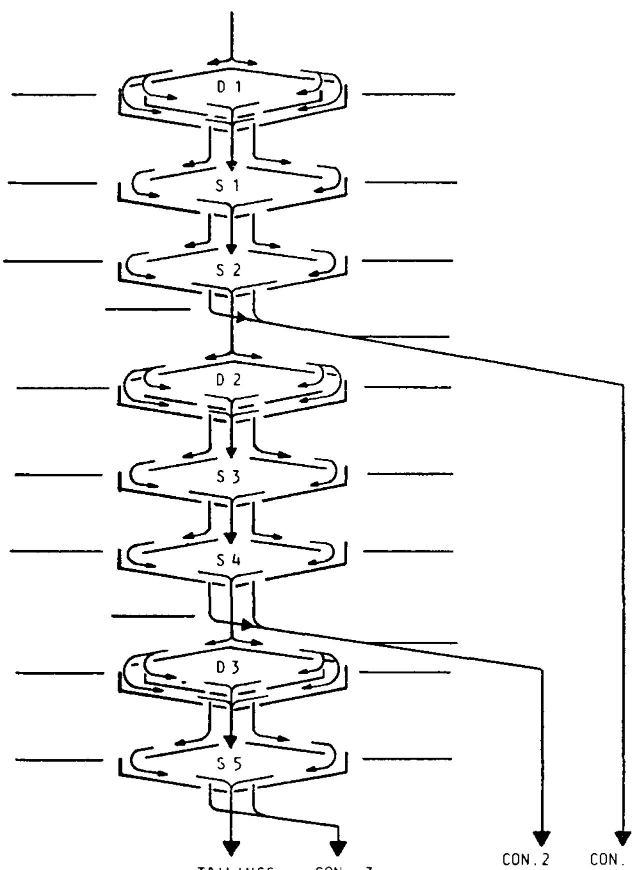

Fig. 6—The 2 DSS DSV Reichert-cone configuration (cones 49 to 68)

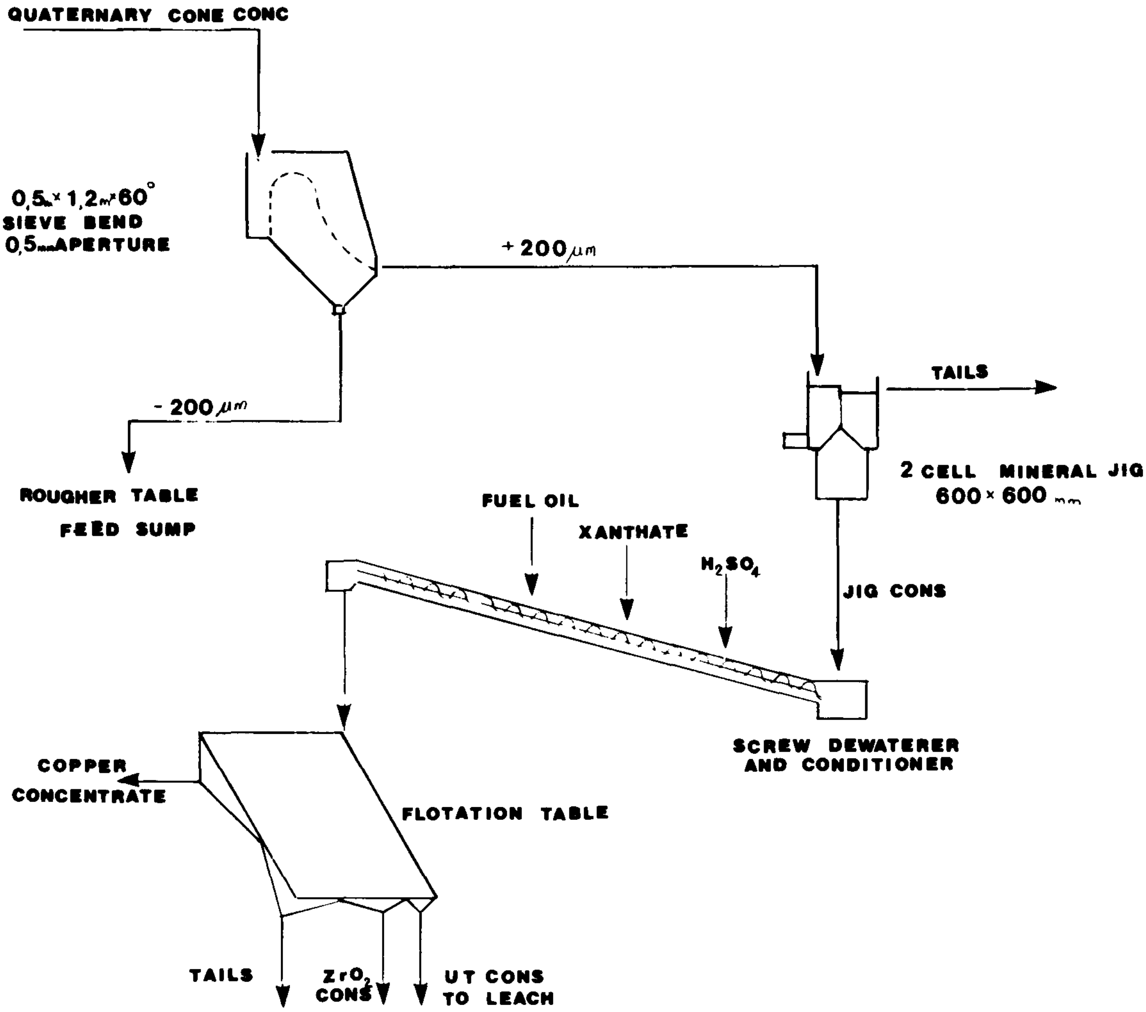

Fig. 7—The jig and table flotation circuit

The quaternary cone concentrates pass over a 0,5 m by 1,2 m 60° sieve bend with a 0,5 mm aperture, the object being to size the concentrates into plus 0,25 and minus 0,25 mm fractions. The oversize from the sieve bend contains 90 per cent of the refractory copper minerals not recovered in the copper concentrator, and gravitates into a 600 by 600 mm two-cell mineral jig. The feed to the jig is approximately 6 t/h of all minus 0,5 mm material and contains 10 per cent heavy minerals.

The jig concentrate is conditioned with sulphuric acid, xanthate, and fuel oil in a screw conveyor, and is fed onto a standard James table operated with 'dry spots' along the deck. When the conditioned coarse copper mineral enters the air–water interface in these 'dry spots', it floats off into the tailings launder.

The James-table concentrates join the table concentrates for leaching, and the middlings should provide a high-grade baddeleyite concentrate. At this stage, the value and degree of liberation of the mineral in the jig and James-table tailings are not known, and their fate will be decided only when this information is available.

Table Circuit

The undersize from the sieve bend is pumped to a six-way rotary distributor, and each fraction gravitates to a hydraulic classifier, the underflow of which provides the feed to the six triple-deck tables (Fig. 8).

The classifier overflows are combined and thickened in a hydro-separator.

Three products are separated on the 18 rougher table decks:

(i) a uranothorianite concentrate, which is sent to leaching,
(ii) a middling, which is fed to the scavenger tables, and
(iii) a tailing, which joins the Lo-Phos rougher-cone middlings, and is fed to the two Lo-Phos scavenger 4 DS cones.

Rougher-table middlings are pumped to a dewatering cone, the underflow passing through two stages of classification, while the overflows join the overflows from the rougher-table classifiers in the hydro-separator.

The underflows from the table-feed classifiers feed the two triple-deck scavenger tables. Four products are removed from the scavenger tables:

(a) a uranothorianite concentrate for leaching,
(b) a middling, which passes on to the cleaner table,
(c) a second middling, which will be further upgraded for the production of baddeleyite, and
(d) a tailing, which is recirculated through the recleaner cones.

The first middling joins the fines-table middling in a dewatering cone feeding the cleaner table.

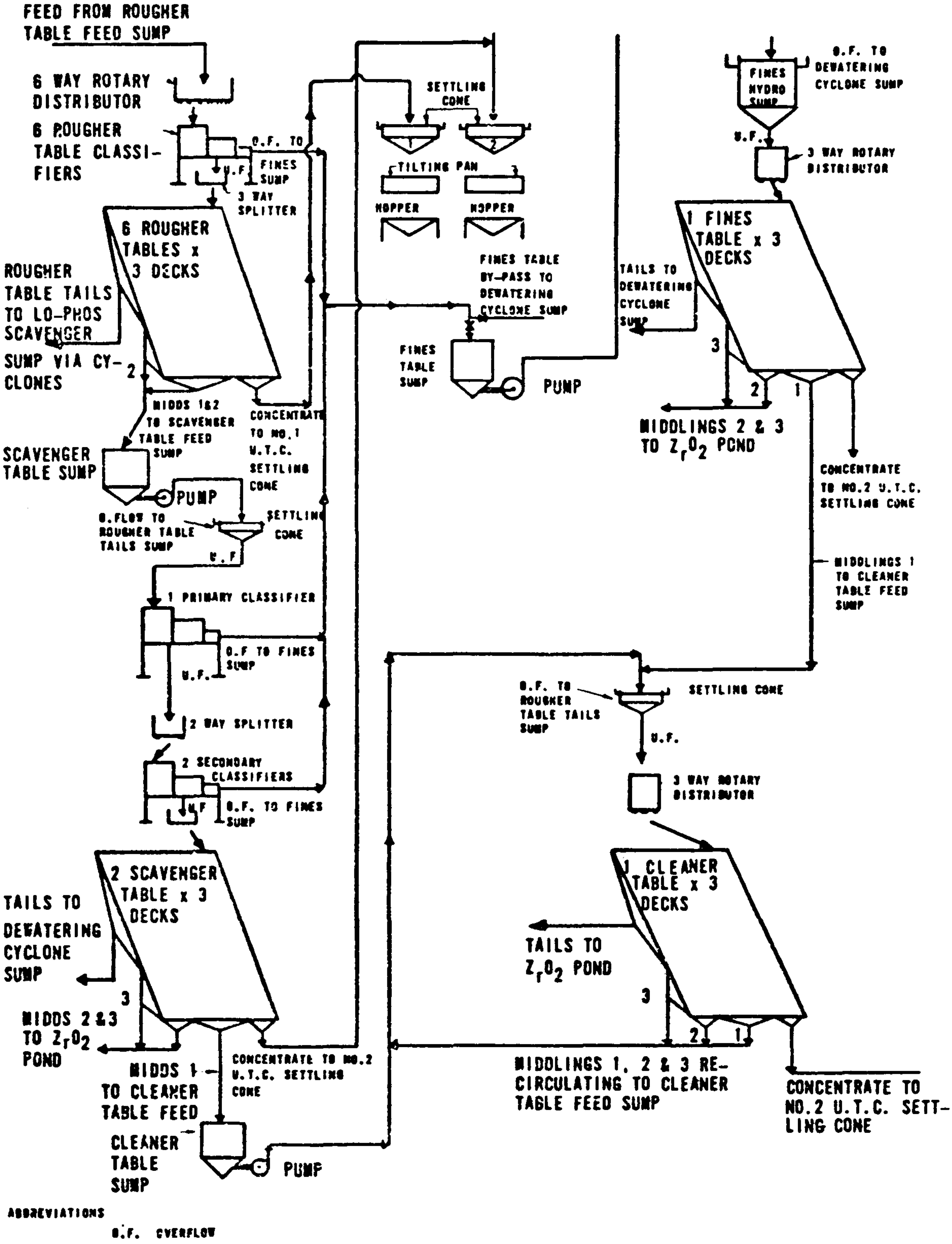

Fig. 8—The table circuit

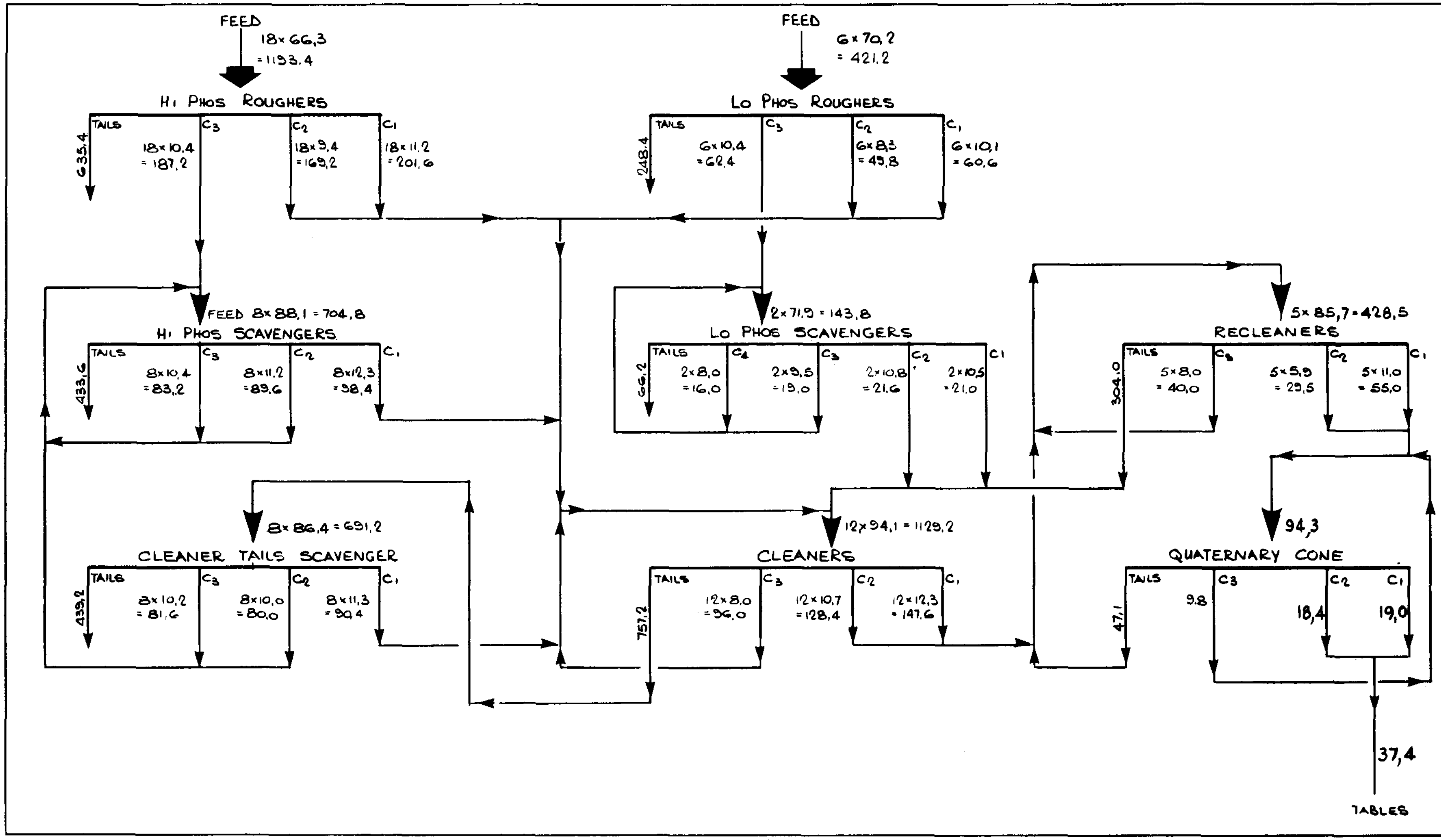

Fig. 9—Balance for the heavy-minerals circuit

All the overflows from the table classifiers are dewatered in a 2 m hydro-separator the underflow of which feeds one triple-deck fines table. The table has the same four products as the scavenger tables.

The first middlings from the scavenger and fines tables are dewatered and are fed to the cleaner table, and these concentrates join the concentrates from the other tables as feed to the chemical plant. The middlings are recirculated, and the tailings go to the production of baddeleyite.

Testwork has just been completed on the upgrading of the baddeleyite from the table middlings, and at this stage it appears that the circuit will consist of a ball mill to regrind the pulp, spirals as roughers, and hydraulic classification and upgrading on tables.

Plant Performance

The flow tonnages in the various sections of the cone plant are shown in Fig. 9.

Upgrading and Recovery

As in most large-scale open-pit mines today, it is not possible to carry out complex ore blending on the mining side. It is therefore the responsibility of the metallurgist to ensure optimum recovery from plant feeds that vary considerably over short periods of time.

The grain-size distribution of the heavy minerals in the feed to the heavy-minerals plant also varies. A typical size analysis is given in Table II.

Another factor that has far-reaching effects on the recovery of uranothorianite is the baddeleyite content of the ore, which can vary from 0,15 to 0,45 per cent. High baddeleyite loading causes a build-up of middlings

TABLE II

GRAIN-SIZE DISTRIBUTION OF HEAVY MINERALS IN PLANT FEED AND CONCENTRATES

Particle size μm	Rougher-cone feed mass %		Uranothorianite table concentrate mass % U_3O_8
	U_3O_8	ZrO_2	
+425	3,3	2,0	
+300	4,0	4,2	2,1
+212	5,6	8,2	3,6
+150	9,8	17,6	13,4
+106	18,3	25,0	21,5
+ 75	12,9	14,2	16,4
+ 45	17,8	15,4	26,1
− 45	28,3	13,4	16,9
	100,0	100,0	100,0

recirculation in the cleaner and recleaner sections of the cone plant, and sometimes squeezes valuable minerals into the tailings.

Table III shows typical uranium and zirconium levels in the various product streams of the gravity plant.

The overall recovery of uranium and zirconium in the gravity-concentrating section of the plant varies considerably in direct relation to the head grade. Tables IV and V show these recoveries for July 1981. For the balance shown in Table IV, the U_3O_8 recovery in the heavy-mineral concentrate is 69,57 per cent, and the uranium recovery is 59,81 per cent. For Table V, the zirconia recovery in the heavy-mineral concentrate is 67,64 per cent.

Process Control

Generally, simple methods of control are employed. Pulp densities at the various Reichert-cone stages are monitored by means of timed tonnage samples and checking on a pulp-density scale. Control of the various table streams is effected by means of a nuclear counter.

TABLE III

TYPICAL URANIUM AND ZIRCONIUM LEVELS IN PRODUCT STREAMS OF GRAVITY SECTION

Product stream	U_3O_8	ZrO_2
Rougher cone feed	0,0035	0,40
Rougher and scavenger cone tailings	0,0010	0,12
Rougher cone concentrate	0,0070	1,6
Scavenger cone tailing	0,0013	0,13
Cleaner cone feed	0,0060	1,0
Cleaner cone tailing	0,0022	0,15
Recleaner cone feed	0,0170	4,2
Recleaner concentrate (quaternary cone feed)	0,037	8,7
Quaternary cone concentrate (rougher table feed)	0,082	17,5
Uranothorianite table concentrate	3,0	77,4
Zirconia table concentrate	0,17	40,0

Control of Reichert Cones

The Reichert-cone arrangement is shown in Fig. 3, together with various configurations in Figs. 4 to 6, and should be read in conjunction with Fig. 2, which shows the slurry flows of the gravity circuit. Four stages of upgrading are achieved in the 68 cones, the total upgrading achieved being in the region of 1000:1 including the tabling circuit. The importance of proper cone operation and control cannot therefore be overstressed.

The following observations are made on every cone on each shift:

(1) feed condition to the cones, i.e. surging, cone overloading or underloading, cone messing,
(2) top-stage distributors, squarely positioned,
(3) condition of all D-deck splitter rings and their position in relation to the edge of the deck,
(4) condition of all cone concentrate 'curtains', settings of all concentrate collection rings,
(5) check which dilution-water rings are running and whether water feed is uniform,
(6) interstage distributors to be level and central,
(7) check for other defects such as loose hatch covers, trash or waste, particularly on D1 deck,
(8) check for excess water in the annular collectors,
(9) regular pulp density measurement of the various cone feeds.

The need for a regular cone-cleaning programme is vital because of algal growth and build-up on cone parts.

The above observations are well complemented by the 'Cone Tonnage and Density Maintenance Program' which quantifies the pertinent aspects of Reichert cone operation. The various measurements are carried out by the operator, who completes the calculation, follows up with any checks, and finally presents the data in a

TABLE IV

THE URANIUM BALANCE FROM DAILY FEED ASSAYS AND TONNAGES

		U_3O_8, t	
1. *Feed*			
Hi-Phos	662 994 t at 0,003766 % U_3O_8		24,970
Lo-Phos	250 207 t at 0,004864 % U_3O_8		12,170
Total	913 201 t at 0,004067 % U_3O_8		37,140
2. *Tailings*			
Hi-Phos	647 844 t at 0,001602 % U_3O_8	10,378	
Lo-Phos	244 469 t at 0,001464 % U_3O_8	3,579	
Total	892 313 t at 0,001564 % U_3O_8	13,955	
3. *Concentrates*			
Uranothorianite	708,925 t at 3,133 % U_3O_8	22,212	
Zirconium	2 603,060 t at 0,139 % U_3O_8	3,628	
Magnetics	17 576,260 t at 0,009 % U_3O_8	1,596	
Total	20 888,245 t at 0,131 % U_3O_8	27,436	
4. *Total accountable*		41,391	
5. *Unaccountable (dewatering cyclone overflow, etc.)*		Gain −11,45 %	−4,251

TABLE V

THE ZIRCONIUM BALANCE FROM DAILY FEED ASSAYS AND TONNAGES

		ZrO_2, t	
1. *Feed*			
Hi-Phos	662 994 t at 0,309 % ZrO_2		2 045,869
Lo-Phos	250 207 t at 0,207 % ZrO_2		517,055
Total	913 201 t at 0,281 % ZrO_2		2 562,924
2. *Tailings*			
Hi-Phos	647 844 t at 0,095 % ZrO_2	615,145	
Lo-Phos	244 469 t at 0,017 % ZrO_2	174,579	
Total	892 313 t at 0,089 % ZrO_2	789,726	
3. *Concentrates*			
Uranothorianite	708,925 t at 72,46 % ZrO_2	513,705	
Zirconium	2 603,060 t at 46,86 % ZrO_2	1 219,811	
Magnetics	17 576,260 t at 0,00 % ZrO_2	0,000	
Total	20 888,245 t at 8,30 % ZrO_2	1 733,516	
4. *Total accountable*		2 523,242	
5. *Unaccountable loss*		1,55 %	39,682

formal manner for transfer to the master cone sheet. The approach as designed means that each cone in the plant receives a complete tonnage and density description once a month, i.e. for each observable deck and concentrate. At the end of the month like section data are averaged, and a cone circuit balance is presented. In this way, anomalous trends can be detected and corrected, or on a daily basis gross deviations can be adjusted immediately. Fig. 10 shows a typical cone-density tonnage profile.

Control of Deister Tables

As stated previously, uranium concentrates leaving the table circuits have generally undergone an upgrading of 1000:1 compared with the feed values. It is therefore easy to understand how uranium losses to the middlings and tailings can run very high if strict operational practice and uniform control are not maintained. Furthermore, the major mineralization of baddeleyite (ZrO_2) and uranothorianite (UO_2. ThO_2. Pb), which have to be separated from each other and the gangue material, are both black in colour, and display wide size ranges and a small density difference.

The following tabling operations are important.

(i) The table feed must not surge.

(ii) The density must be correct. Low density is indicated by a rush of water from the feed box, and high density by islands of stranded material (usually about 1:3).

(iii) The deck loading should be 1,3 to 1,6 t per deck-hour.

(iv) The correct concentrate position is continuously controlled by operation of the table-tilting mechanism and not adjustment of the wash water. Obviously dry spots within the concentrate–

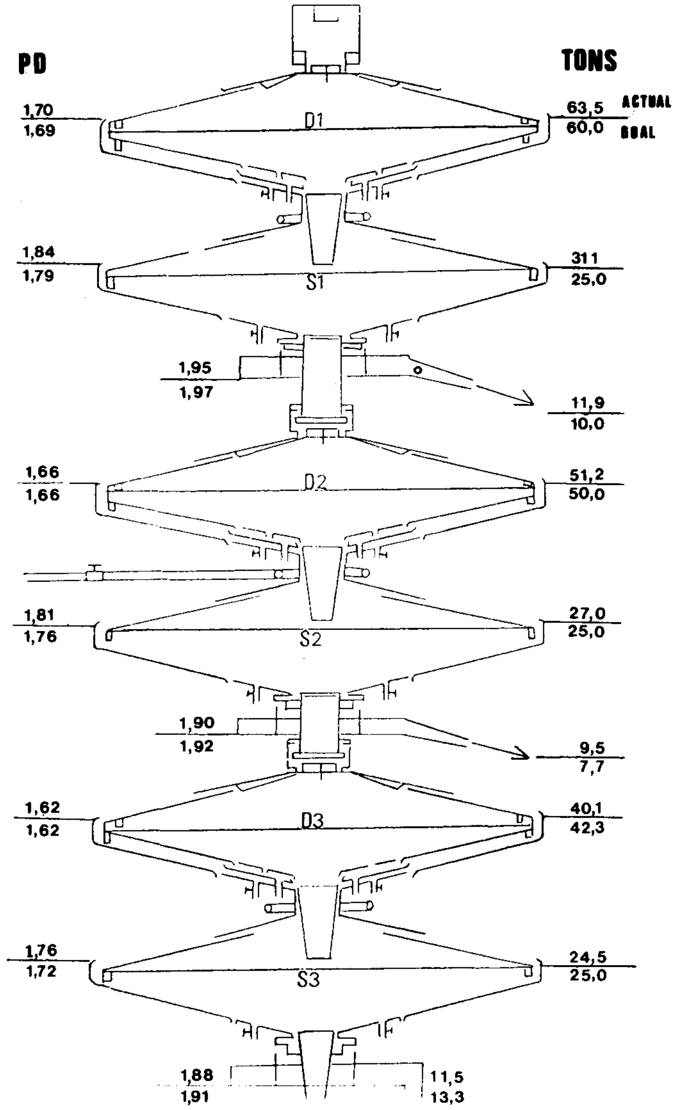

Fig. 10—A typical profile showing pulp densities and tonnages

middlings spread must be avoided.

A method of control is applied in order to regulate the grade of the uranothorianite concentrate and minimize the loss of uranium to the middlings and zirconium table concentrates. The method depends on the measurement of the radioactivity emanating from the uranothorianite mineralization by a nuclear scintillation technique. Under the circumstances, the technique provides a very good guide, requires no sample drying or preparation, and can be operated by unskilled workers. The resulting data indicate to the operator how he should adjust his tables in the event of a change in feed condition according to the criteria already outlined. A table control sheet is employed and shows the concentrate count values from the rougher, scavenger, and cleaner tables, together with the appropriate tray settings in centimetres. These individual values are compared with the bulk concentrates ultimately collected in the hoppers. The final scavenger and cleaner middlings are adjusted to minimize uranium losses.

Modifications in Reichert-cone Circuit

Feed Distribution

The standard Reichert-cone feed distributor is a multiple-outlet pressure type of splitter fitted onto the pump discharge line. At Palabora, however, it was decided to use electrically driven rotary distributors for dividing the pulp stream into 4, 6, 8, 9, and 12 fractions. From these distributors the pulp gravitates to the individual cones. As a result of this design it was found that the feed to the cones was not a steady stream but was pulsing owing to vortexing in the distributor outlet.

To combat this, polyurethane spigots were fixed to the end of the feed pipe with the object of building up a head in the feed pipe to ensure steady-flow conditions, the entrapped air being released to the atmosphere through an elevated breather pipe. This resolved the pulsing problem, but it was then discovered that, owing to settlement and segregation taking place in the feed pipe, the feed distribution around the periphery of the cone was extremely erratic, varying in some cases as much as 2:1 on opposite sides of the same cone. (It is the author's opinion that this condition always exists to a greater or lesser extent when sand pulps are fed to splitters or cones.)

To combat this problem, a 'scroll' feeder was developed that causes the feed to swirl, mix, and distribute evenly over the circumference of the cone feed box. The scroll feeder is made from cast polyurethane and can also be fitted with any spigot size to control pulsing and ensure a constant, even flow of feed. The longest-serving scroll feeders have been in service for more than a year, having handled over 0,5 Mt, and the tonnage variation over the periphery of the cone feed decks is in every case less than 10 per cent and in most cases less than 5 per cent.

Dilution-water Distributors

When operating at the high densities necessary for the recovery of the finest uranothorianite (smaller than 75 μm), the proprietary dilution-water rings caused channelling on the distributor decks that continued through the concentrator deck right up to the insert of the concentrator collector rings. The concentrate take-off lip was therefore not being presented with a uniform depth or velocity of pulp around its circumference, but rather with streams of alternately fast- and slow-flowing pulp caused by the channelling. This was resolved by the attachment of saw-tooth skirts on the lower side of the water rings, causing an even water distribution and consequently a uniform sheet-flow pattern over the cone.

Viscosity as a Cone-operating Criterion

The baddeleyite content of the feed to the plant can vary from less than 0,2 per cent to more than 0,4 per cent. As a result of the large circulating loads carried in the recleaner and quaternary cone circuit, a build-up of this mineral will cause the relative density of the dry solids to vary by as much as 20 per cent. Cone control by pulp-density measurements alone are therefore inadequate. Investigations are now being carried out to determine whether the viscosity of the pulp can be measured and used as an operating control to ensure optimum cone operation.

Initial tests carried out using a modified Stormer viscometer show that this approach might well be feasible, especially in the pulp-density range of 60 to 70 per cent at which the cones are operating.

Jig and Table Flotation Circuit

Justification for this Circuit

As mentioned before, in large-scale open-pit operations it is not economically feasible to blend the ore and provide the plant with the ideal feed. There are certain areas in the pit where cubanite is the predominant copper mineral, and it so happens that these particular areas also contain more uranium. When this ore is loaded with a 17 m^3 power shovel into 150 haul-trucks, the production of uranium concentrates soars but the copper content of the feed to the leach can also increase to as high as 10 per cent mainly owing to the presence of hard, refractory cubanite. This condition increases the consumption of nitric acid to double the normal amount.

Testwork has shown that more than 90 per cent of the copper mineral that escapes the flotation circuit and ends up in the concentrates from the heavy-minerals plant is coarser than 250 μm. It was decided to screen out this coarse fraction from the cone concentrates and treat it separately by first upgrading in a jig and then dressing the jig concentrates on a table, where the sulphides can be removed by table flotation. The removal of the coarser fraction in this way will mean that the copper mineral will be recovered and at the same time closer sizing of the table feed will aid the table circuit.

Operating Variables for Jig

The jig will treat between 6 and 8 t/h of very closely sized material. Depending on the efficiency of the sieve bend, most of the grains will be between 0,5 and 0,2 mm. The relative densities of the products to be separated are gangue 3,0, cubanite 4,2, baddeleyite 5,8, and uranothorianite 9,0.

A shallow bed was chosen (50 mm including ragging) for this operation, and the jig will operate at a high speed (about 260 r/min) with a stroke 3 to 5 times the diameter of the largest particle.

Removal of Coarse Sulphide by Table Flotation

The jig is expected to produce approximately 0,6 t of concentrates per hour. It is proposed to dress this material on a James table to produce a uranothorianite concentrate for leaching, a high-grade baddeleyite product, and a tailing, and at the same time float off the coarse copper sulphides.

For successful table flotation the conditioning must be done on dewatered solids with a water content of approximately 10 per cent. Testwork carried out on $\frac{1}{4}$-scale equipment showed that, by the addition and mixing in sequence of 2,5 l/t of 50 per cent sulphuric acid, 1,2 l/t of 50 per cent xanthate, and 2,5 l/t of fuel oil, more than 90 per cent of the copper sulphides can be floated and recovered along the tailings board of the table.

Modifications to Table Circuit

A Maintenance-free Table Deck

The most important development in this section has been the development of a fibreglass table deck with a cast polyurethane cover with integral riffles.

Suspended triple-deck tables are used at Palabora Mining Company because of the space-saving factor and because this type of table can be installed in the upper floors of the plant without causing structural vibration problems. The original wooden decks deteriorated and soon started working loose and becoming less effective, while the rubber deck covers were attacked by algae and became soft and tacky.

A local modification comprising steel decks with cast polyurethane covers with integral riffles was tested, but unfortunately the higher mass attributable to the steel construction affected the table performance adversely. As a result of this, locally designed decks constructed in fibreglass-reinforced plastic (FRP) covered with cast polyurethane were installed. These decks have now been in operation for two years and have been entirely maintenance-free. In fact, if the steel sub-frame is properly protected against corrosion, there is no reason why these tables should not provide many years of trouble-free operation. The new James table for dressing the jig concentrates is also constructed in FRP.

Conclusion

The Palabora Mining Company's heavy-minerals plant is certainly among the largest-tonnage gravity plants in the world. Because of the problems associated with the sampling of these enormous pulp streams, the sample preparation required to reduce the samples to assayable quantities, and the standard deviation in the assaying procedure, it is very difficult to establish optimum operating parameters on the plant. Often 'seat of the pants' metallurgy is the best method to apply.

The efficiency of the plant has improved considerably over the past few years and, as more information becomes available, it should be possible to improve recoveries still further.

The unique chemical section also has its operational problems, and this plant with gravity concentration, flotation, leaching, filtration, clarification, solvent extraction, precipitation, and calcining all under one roof offers a continuous technical challenge to the metallurgist.

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