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The recovery of apatite from the hydro-separator overflow of copper tailings

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SYNOPSIS

This paper describes pilot-plant tests on the recovery of apatite from a hitherto untapped material. This material—hydro-separator overflow—is very fine (80 per cent smaller than 45 μm and 32 per cent smaller than 10 μm).

The pilot-plant tests proved that the recovery of apatite from this material is economically and technically viable. This was confirmed by plant trials, which, in the early stages, produced concentrate assaying 36,6 per cent P_2O_5 at a recovery of 65,5 per cent.

SAMEVATTING

Hierdie referaat beskryf proefaanlegtoetse in verband met die herwinning van apatiet uit materiaal wat tot nog toe onbenut was. Hierdie materiaal—die oorloop van 'n hidroskeier—is baie fyn (80 persent kleiner as 45 μm en 32 persent kleiner as 10 μm).

Die proefaanlegtoetse het bewys dat die herwinning van apatiet uit hierdie materiaal ekonomies en tegnies lewensvatbaar is. Dit is bevestig deur aanlegproewe wat in die vroeë stadiums 'n konsentraat met 'n essaieerwaarde van 36,6 persent P_2O_5 vir 'n herwinning van 65,6 persent opgelewer het.

Introduction

The Phalaborwa Complex is situated in the Lowveld of the north-eastern Transvaal and comprises the following types of rock: pyroxenite, feldspathic pyroxenite, syenite, olivine–diopside–phlogopite pegmatoids, fenite, foskorite, carbonatite, and later intrusions of dolerite.

Two companies, Phosphate Development Corporation Limited (Foskor) and Palabora Mining Company (PMC), operate open-cast mines and mineral-beneficiation plants at Phalaborwa.

Apatite, the phosphate mineral, is an important constituent of the pyroxenite and foskorite ores and is recovered by means of flotation at Foskor. For almost a decade now, Foskor has been recovering apatite from a third source, namely tailings emanating from PMC's copper concentrator. The tailings are made available to Foskor as underflow from a hydro-separator, and currently an average of 26 kt of solids is processed daily. A hitherto untapped potential source of apatite is some 15 kt per day of hydro-separator overflow. As can be seen from Table I, which gives a typical particle-size analysis of this material, 80 per cent of the material is smaller than 45 μm, while 30 per cent is smaller than 10 μm. The fineness of this product presents problems in the production of an acceptable grade of phosphate concentrate, but it could also offer certain advantages. At present, concentrate that is pumped to Fedmis for the production of phosphoric acid must be remilled to about 65 per cent passing 38 μm to render the product suitable for further treatment. The concentrate produced from fines could be more amenable to acid digestion.

Concentrate produced from fines may also be attractive for export purposes. Many phosphoric acid plants abroad treating phosphate rock of sedimentary origin do not have grinding facilities at their disposal. Concentrate produced from fines should become an attractive feedstock for these plants, requiring no further grinding prior to acid digestion.

TABLE I

TYPICAL PARTICLE-SIZE DISTRIBUTION OF HYDRO-SEPARATOR OVERFLOW MATERIAL

Size fraction μm	Mass %
+45	18,8
+38	6,3
+34	5,0
+26	10,0
+18	12,9
+13	9,9
+10	5,0
−10	32,1

During 1979 a research project was launched on the possibility of recovering the apatite from this fine material.

Most of the work conducted at the research department of Foskor follows an empirical approach. The acquisition of technological knowledge is always an aim of research, but in many cases it is only of a secondary nature, the economic principle being the primary objective. Results are always compared with existing data from the main plant, with emphasis on the economics and practicality of the whole process.

Where flotation is concerned, the aim is always a concentrate assaying at least 36,4 per cent P_2O_5 at an acceptable recovery. Many factors are involved in the determination of an 'acceptable recovery' and, when this term is used, it must be understood that it can be any value as long as the process is economically viable.

In this paper, we discuss early ideas and problems, and how these ideas evolved into a process that was tested on pilot-plant scale and eventually on a production scale.

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Development of a Recovery Process

All the process-development work described in this paper was conducted at Foskor's 4 t/h on-stream pilot plant, which is situated at PMC. This pilot plant is equipped with two thickeners of 5 m diameter that receive their feed from PMC. Each thickener is coupled to three conditioning tanks that feed a section of 21 Denver number 8 flotation cells consisting of 8 roughers, 8 scavengers, 3 cleaners and 2 recleaners. Each section can handle up to 2 t/h.

Early Ideas

The first idea that came to mind when this project was initiated concerned the amenability of the fine material to the standard methods employed for the flotation of the PMC tailings that were received as hydro-separator underflow. When this approach failed, the fines were mixed in various ratios with the normal PMC tailings.

It was soon found that the ultra-fine particles in the minus 10 μm region had a deleterious effect on the flotation process by invariably reporting in the froth phase. This resulted in an unselective and often uncontrollable float. In addition, the consumption of some of the reagents employed became unacceptably high due to the large surface area of the fine particles. Clearly, the ultra-fine particles had to be removed.

First System

After numerous experiments, a classification system consisting of two 40 mm cyclones in parallel was evolved. The cyclones were supplied by Arban Engineering, and were fitted with 14 mm vortex finders and 8 mm spigots.

The pulp density (Pd) of the feed from the hydro-separator overflow varied between 1030 and 1100 g/l. One of the thickeners at the pilot plant was used to thicken this material to a Pd of 1200 g/l before it was pumped to the cyclones. To ensure a constant pressure to the cyclones. The thickened pulp was fed into a feed sump. By means of a valve and a bypass in the feed line between the pump and each cyclone, the pulp level was kept constant in the feed sump and the presssure to the cyclone could be regulated (Fig. 1).

When these two cyclones were operated at a pressure of 200 kPa with a feed Pd of 1200 g/l, each cyclone delivered an underflow of 1800 g/l Pd at a rate of 0,5 t/h with an overflow Pd of 1090 g/l. The D_{50} for the cyclones was 15 μm while the total recovery by mass over the two cyclones was 55 per cent.

The particle-size distribution of the various cyclone products was obtained by means of a Warman Cyclosizer. The cut points referred to above were then determined by the plotting of the well-known Tromp curve for each test run.

The classified material was diluted to a Pd of 1350 g/l and then subjected to flotation. Much better results were obtained, and concentrate grades of 36,4 per cent P_2O_5 at recoveries of 70 per cent could be produced. This classified product was mixed in various ratios with the standard PMC tailings but, although good concentrate grades could be achieved, the recoveries were down by 10 to 15 per cent compared with those from the standard PMC-tailings flotation, even though the dosage of reagent was changed to allow for the large surface area presented by the fines.

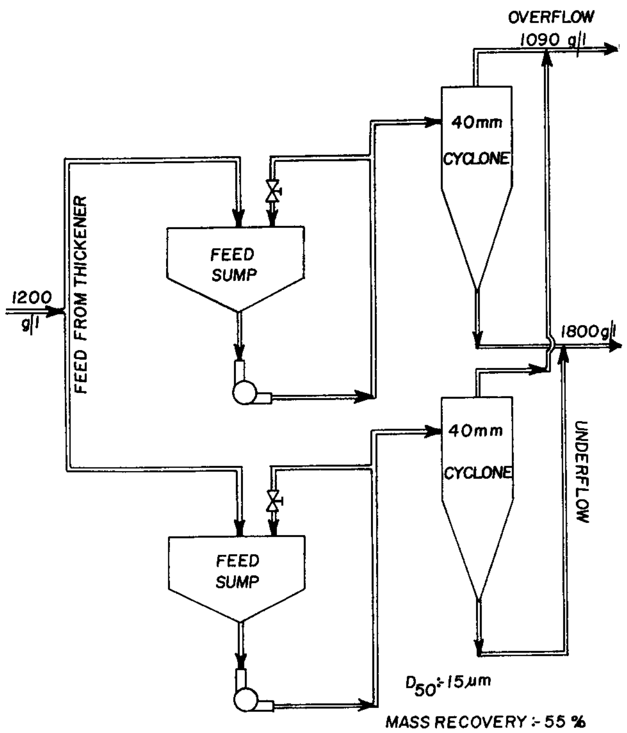

Fig. 1—Classification system consisting of two 40 mm cyclones in parallel

Second and Later Systems

The development of a classification system that would give a higher tonnage throughput and a lower cut point was undertaken. Cutting at 15 μm meant the loss of a substantial amount of phosphate-bearing material.

In February 1981, a single-stage 100 mm Multotec cyclone replaced the two 40 mm cyclones (Fig. 2). The cyclone parameters were as follows:

Spigot	10 mm
Vortex finder	26 mm
Chamber inlet	16 mm

The D_{50} of this system was 13,75 μm with 11,6 per cent of the material in the underflow smaller than 10 μm.

This still proved to be undesirable for flotation purposes, and a second-stage 100 mm Multotec cyclone was therefore introduced in March 1981 (Fig. 3). The cyclone parameters were as follows:

	Stage 1	*Stage 2*
	mm	mm
Spigot	12	12
Vortex finder	26	26
Chamber inlet	16	9

A substantial amount of water is used to dilute the feed to the first stage, the second stage, and the flotation section. To alleviate this problem, the second-stage

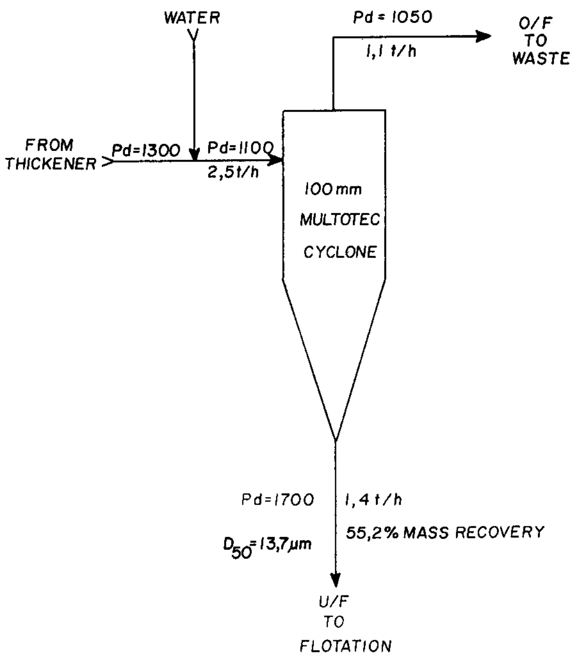

Fig. 2—Single-stage 100 mm cyclone

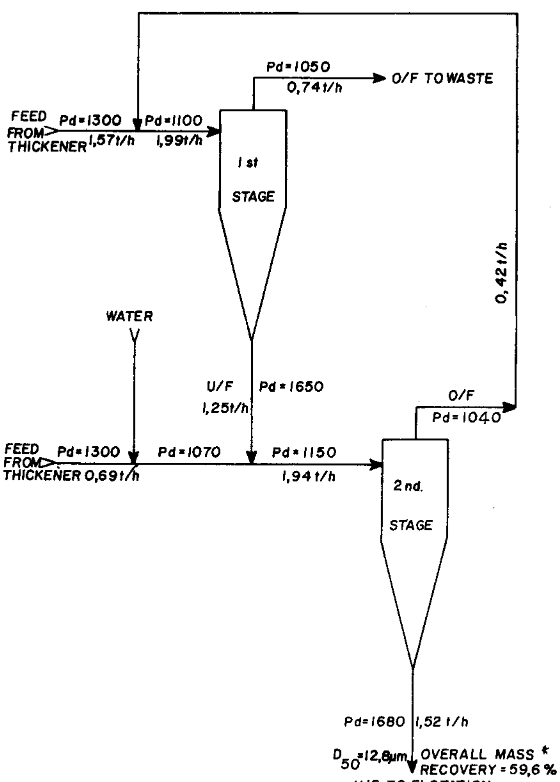

Fig. 3—Two 100 mm cyclones with circulating load
*** Average figure for period 11/3/81 to 30/4/81 (recovery for this system = 67,2%)**

overflow was recycled to the first-stage feed while the second-stage feed was complemented by further feed from the hydro-separator overflow. The D_{50} for this system was 12,76 μm with only 6,8 per cent of the material in the flotation feed smaller than 10 μm.

Because of difficulties in the control of the circuit, and after consultation with PMC over water-balance considerations, it was decided in May 1981 to resort to a two-stage circuit as shown in Fig. 4. As can be seen, no circulating load was introduced to either of the cyclones while the feed to the first-stage cyclone was not thickened. The cyclone parameters were as follows:

	Stage 1	*Stage 2*
	mm	mm
Spigot	14	10
Vortex finder	34	26
Chamber inlet	16	9

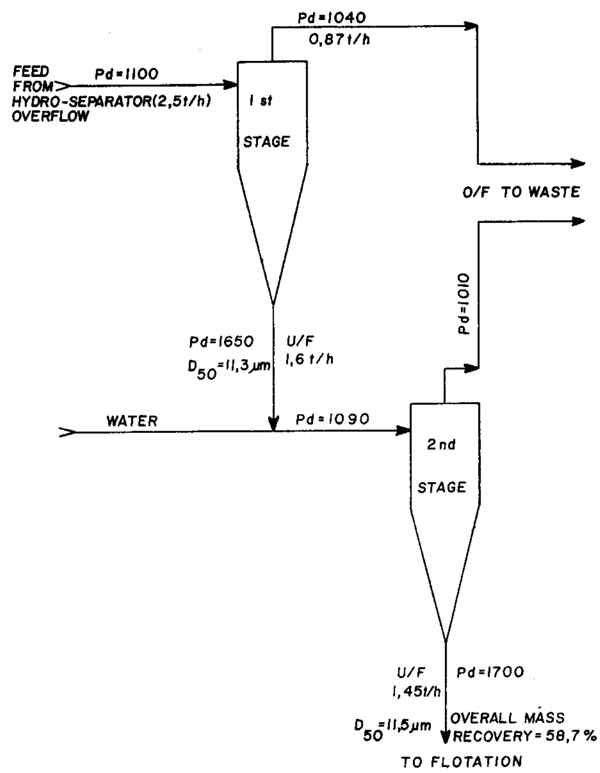

Fig. 4—Two 100 mm cyclones in series

The D_{50} of the first-stage cyclone was 11,34 μm and that of the second-stage cyclone was 11,5 μm. Only 6 per cent of the minus 10 μm material reported in the flotation feed.

At that time, the feed rate to the first cyclone was 2,5 t/h, and the second-stage cyclone was giving an underflow of 1700 g/l Pd at a rate of 1,45 t/h. The classification system was considered to be adequate for the time being, and more attention was given to the flotation circuit. Up to then, the average grade of concentrate was 36,2 per cent P_2O_5 at a recovery of 52,1 per cent.

It was soon found that the flotation rate of this product was much lower than that of normal PMC tailings. An increase in the retention time from about 20 to 35 minutes in the rougher-scavenger system resulted in a remarkable improvement. Flotation recoveries of more than 70 per cent were maintained at an average concentrate grade of 36,4 per cent P_2O_5 over a two-month period.

The air inlet was reduced to a minimum in all the stages of flotation since it had been found that both the grade and the recovery were jeopardized if too much air was admitted. The sand-release settings of the cells were also reduced to a minimum.

When the head grade fell below 9 per cent P_2O_5, the concentrate assayed in the lower thirties. The low grade of the concentrate was overcome by the introduction of a re-recleaner (third cleaner) stage in the flotation process, but at the detriment of the recovery, which fell by some 10 per cent.

The characteristics of the concentrate are shown in Table II.

TABLE II

ASSAY/SCREEN ANALYSIS OF THE FLOTATION CONCENTRATE—AVERAGE FIGURES FROM 16th FEBRUARY TO 15th MAY 1981

Size fraction μm	Mass %	P_2O_5 %	MgO %
+45	28,1	38,7	0,9
+38	21,1	37,3	4,1
+34	1,7	39,0	1,0
+26	13,1	36,7	1,3
+18	18,9	34,0	1,7
+13	7,1	31,7	2,2
+10	1,2	29,2	2,7
−10	8,8	N.D.	N.D.
TOTAL	100,0	36,4	1,3

This product, PALFOS 80 PMF, has excellent filtering characteristics. With a Delkor 1 m^2 belt filter, a throughput of about 3 t/h per square metre could be maintained, producing a filter cake with a moisture content below 10 per cent.

Further Optimization of Classification System

During most of 1982, different makes and sizes of cyclones were tested, and different makes of flotation cells were tried in the flotation circuit. In the end, the 100 mm Multotec cyclones and the Denver number 8 cells were retained.

In January 1983, attention was once more turned to the classification system, and the set-up was changed to that depicted in Fig. 5.

This circuit differs somewhat from the preceding one (Fig. 4) in that about 50 per cent by volume of the second-stage cyclone overflow is recycled to the first-stage underflow. This was done to reduce the amount of water needed for dilution to the second-stage feed. The cyclone parameters were as follows:

	Stage 1 mm	*Stage 2* mm
Spigot	16	12
Vortex finder	34	26
Chamber inlet	16	9

An increase in the spigot diameter in the first stage resulted in an appreciable decrease in the underflow Pd from 1650 to 1530 g/l, thereby curbing even further the amount of water needed in the second-stage feed dilution (a highly desirable situation).

With the classification system, the overall recovery was

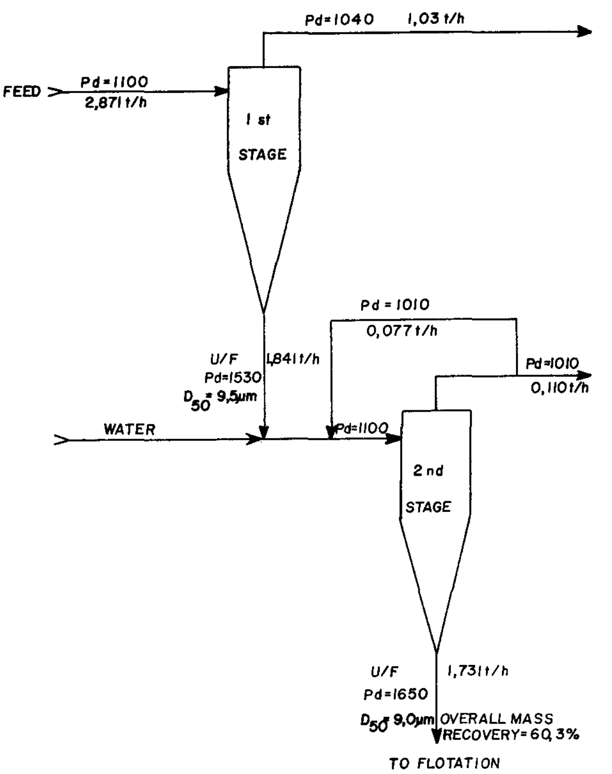

Fig. 5—Final system for 100 mm cyclones

increased from 58,7 to 60,3 per cent by mass. The throughput was increased from 2,5 to 2,8 t/h, while the second-stage underflow had a Pd of 1647 g/l at a rate of 1,73 t/h. The D_{50} of the first- and second-stage cyclones dropped to 9,5 and 9,0 μm respectively, with 6 per cent of the material in the second-stage underflow smaller than 10 μm.

The particle-size distribution of the flotation feed, or the cyclone underflow, is shown in Table III.

The flotation behaviour was unchanged. Over a period from January to February 1983, the average head grade was 9,2 per cent P_2O_5, the concentrate assayed 36,9 per cent P_2O_5, and the recovery was 66,8 per cent.

The pilot-plant trials proved that the recovery of phosphate concentrate from hydro-separator overflow material is economically and technically viable.

TABLE III

SIZE FRACTIONS IN FLOTATION FEED

Size fraction μm	Mass %
+45	43,0
+33	5,3
+25	14,4
+18	16,5
+13	10,9
+9,7	4,3
−9,7	5,6

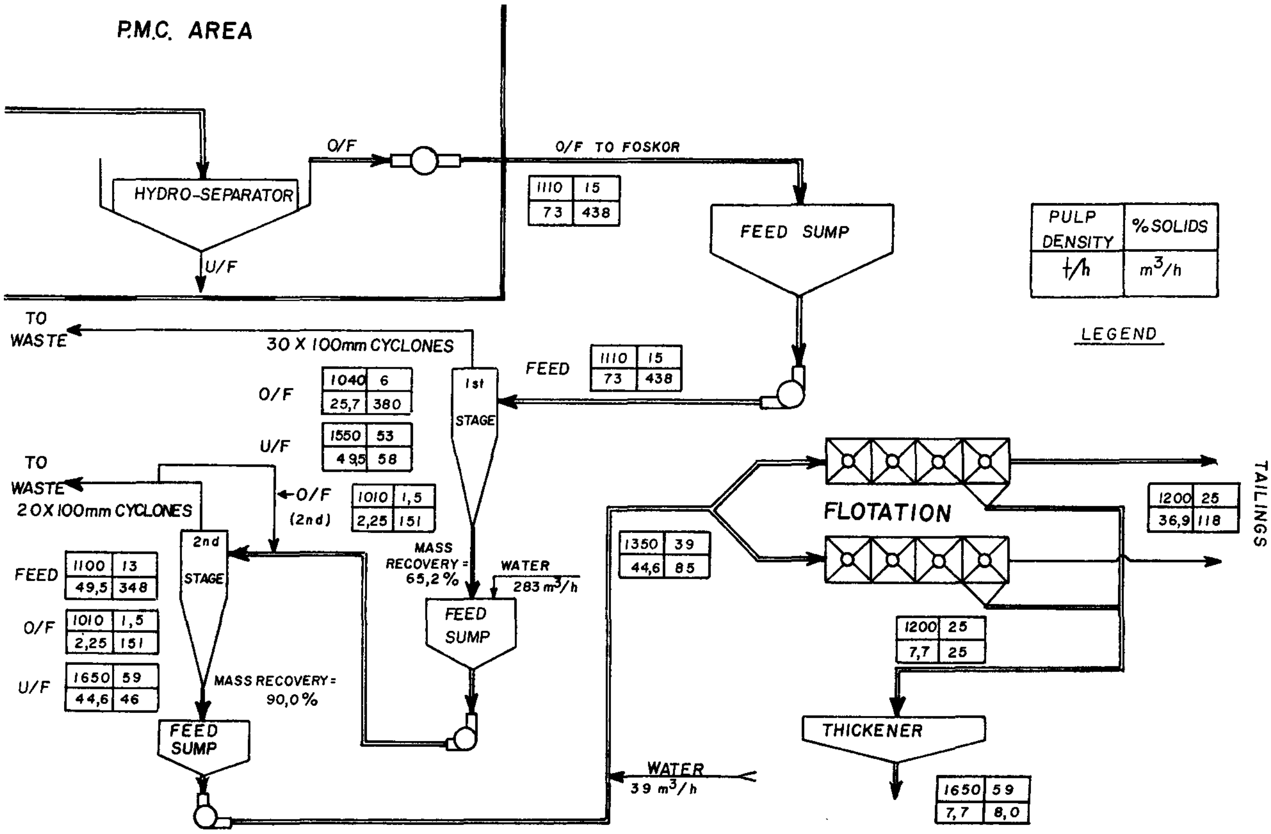

Fig. 6—Trial plant for the production of about 170 t/d PALFOS 80 PMF

Plant Trials

The next phase of the project necessitated the production of relatively large tonnages of fine concentrate so that the drying and handling properties could be assessed, and trial shipments could be dispatched to potential customers for product-acceptability tests.

Plans were drawn up for a trial classification section to handle 73 t/h of hydro-separator overflow material, meaning that about 40 t/h would be fed to the flotation section. One section of flotation cells on the plant was modified so that the air intake could be regulated, and the sand-release settings on the cells were reduced to a minimum. Fig. 6 gives a schematic representation of the whole circuit.

The classification system consists of fifty 100 mm Multotec cyclones with the same parameters as the test

TABLE IV

FLOTATION RESULTS FOR PALFOS 80 PMF

Day	Feed to cyclone (1st stage)	Flotation feed		Reagent dosage, g/t				Concentrate	Recovery
July 1983	P_2O_5 %	P_2O_5 %	Pd g/l	Water-glass	Acrol LG-21	Fatty acid	PGE	P_2O_5 %	%
7	7,5	8,8	1 328	276	36	171	68	33,5	63,1
8	10,8	12,4	1 358	115	23	135	43	39,8	62,7
9	10,7	12,4	1 340	453	40	413	74	37,3	64,0
10	11,2	12,7	1 315	944	71	775	133	36,1	65,2
11	10,3	11,6	1 330	535	45	421	93	37,9	62,1
12	7,3	8,8	1 333	347	56	347	90	34,4	64,4
14	9,2	11,2	1 370	197	27	178	38	37,5	61,7
15	8,6	10,3	1 339	630	72	437	107	34,9	67,5
17	10,3	11,7	1 339	640	78	441	127	36,7	68,2
18	10,0	11,2	1 300	692	83	473	138	37,2	67,0
19	10,2	12,0	1 350	333	40	228	67	37,5	76,0
Average	9,6	11,2	1 336	469	52	365	89	36,6	65,6

Acrol: A guar-based depressant
PGE: Polyglycol ether

circuit (Fig. 5). The first stage has two clusters of 15 cyclones each, and the second stage one cluster of 20 cyclones.

First trials on this plant started on 1st July, 1983. After the usual teething problems associated with the commissioning of a new plant, meaningful results started coming in on 7th July. The results from 7th to 19th July are reported in Table IV.

During this period, concentrate with an average assay of 36,6 per cent P_2O_5 was produced at a recovery of 65,6 per cent, which compares very well with the pilot-plant results. An interesting feature is the degree of upgrading that occurs during the two stages of classification. Typically, the underflow of the second stage was 1 to 2 per cent higher in P_2O_5 content than the feed from PMC. As shown in Table IV, the average feed from PMC was 9,6 per cent P_2O_5, while the flotation feed after classification assayed 11,2 per cent P_2O_5.

On the whole, the scale-up of the whole process proceeded exceptionally well, and the flotation results compare favourably with the results obtained in the pilot plant, even though at that stage the plant operators were still experimenting with reagent dosages to optimize the flotation system.

Up to 19th July, the average Pd values over the cyclones were as follows:

	1st stage	*2nd stage*
	g/l	g/l
Feed	1 120	1 140
Overflow	1 030	1 033
Underflow	1 720	1 820

Although the classification system had still to be thoroughly checked for particle-size distribution, cut point, and mass recovery, the flotation-plant results show that it was doing a creditable job of fines removal despite the fact that the Pd of the underflow in the first and second stages was on the high side compared with the results obtained in the pilot plant.

Acknowledgement

The initial choice of operating parameters for the cyclones was made by Mr J. Engelbrecht of Multotec, whose assistance was invaluable during the whole project.

Noble metals

The International Precious Metals Insitute is to hold a seminar on 'Noble metals Fabrication and Technology' in Jerusalem from 20th to 25th April, 1985.

'The objective of this Seminar is to present the State of the Art, identify accomplishments and problems, present answers through professional papers, and to encourage open exchange of ideas concerning Noble Metals technology and fabrication'—Dr Leah Gal-Or, Seminar Chairperson, Israel Institute of Metals.

This seminar will focus on the following topics:

- State of the Art Technology—Materials, Equipment, Processes
- Jewelry—Fabrication, Electroplating
- Noble Metals in the Electronics Industry—Wire and Die Bonding
- Surface and Substrate Technology
- Refining and Recovery of the Noble Metals
- Advanced Technology in the Working of Noble Metals
- Noble Metals in the Telecommunications Industry
- Financial and Economic Considerations
- Electroplating of Noble Metals—Equipment, Processes, High-speed and Pulse Plating
- Management and Security

You are invited to submit a comprehensive abstract on your accomplishments or problems appropriate for a presentation of approximately 30 to 45 minutes to IPMI, 2254 Barrington Road, Bethlehem, PA 18018, U.S.A. The abstract deadline is 15th April, 1984.

Advance information on the programme, seminar, and hotel registration, and optional Holy Land Tours is to be released at the 8th International Precious Metals Conference in Toronto, Canada, 3rd to 7th June, 1984.

Contact Robert W. Steinmetz, IPMI, at the above address.