J. S. Afr. Inst. Min. Metall., vol. 84, no. 7.
Jul. 1984. pp. 208-219.

The design, erection, and commissioning of the froth-flotation section of the Grootegeluk coal-preparation plant

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SYNOPSIS

The Grootegeluk coal-processing plant treats 3 kt of raw coal per hour to produce a middlings fuel coal and a coking-coal fraction with ash contents of 35 and 10 per cent respectively. The plus 0,5 mm fraction is treated by heavy-medium processes, the final top size of the coal being 15 mm. About 400 t of minus 0,5 mm coal per hour is subjected to froth flot-ation.

The routes originally considered for the processing of this fine coal were froth flotation, shaking tables, heavy-medium cyclones, and water-only cyclones in combination with froth flotation. The results from tests on the fines arising from a small-trial open-cast mine are discussed.

When the froth flotation route was decided upon, further tests were conducted on the selection of the equipment most suited to the particular requirements of this coal and the rest of the plant. Tests using different types of flotation cells and dewatering equipment are discussed.

During the commissioning, several operational problems were encountered, and the work done to find the best solutions is discussed.

SAMEVATTING

Die Grootegeluk-steenkoolverwerkingsaanleg behandel 3 kt onbehandelde steenkool per uur om 'n tussengraadse brandstofsteenkool en 'n kooksteenkoolfraksie met asinhoude van 35 en 10 persent respektiewelik te lewer. Die fraksie groter as 0,5 mm word deur middel van swaarmediumprosesse behandel en die uiteindelike topgrootte is 15 mm. Ongeveer 400 t steenkool kleiner as 0,5 mm word per uur aan skuimflottasie onderwerp.

Die roetes wat oorspronklik vir die verwerking van hierdie fynsteenkool oorweeg is, was skuimflottasie, skudtafels, swaarmediumsiklone en watersiklone gekombineer met skuimflottasie. Die resultate van toetse op die fynsteenkool afkomstig van 'n klein proefdagboumyn word bespreek.

Toe daar op die skuimflottasieroete besluit is, is verdere toetse uitgevoer om die geskikste toerusting vir die besondere vereistes van hierdie steenkool en die res van die aanleg te kies. Toetse met gebruik van verskillende soorte flottasieselle en ontwateringstoerusting word bespreek.

Daar is verskeie bedryfsprobleme met die inbedryfstelling ondervind en die werk wat gedoen is om die beste oplossings te vind, word bespreek.

Introduction

The feasibility study undertaken by Iscor Limited on all the parameters involved in the long-term planning of Grootegeluk[1,2] indicated that a coal-beneficiation plant with an output of 1,82 Mt of coking coal per annum was a viable proposition in the Waterberg Coalfield.

The beneficiation plant at Grootegeluk had to be designed to accommodate the following:

(a) strict quality requirements as laid down by Iscor Steel Works,
(b) non-selective mining, resulting in a low yield of coking coal and the use of a large beneficiation plant,
(c) large amounts of near-gravity material in the feed, which demands very close control of the separation densities and equipment,
(d) planned annual outputs as shown in Table I, and
(e) a final coking-coal product with a size grading of less than 15 mm and an ash content of 10 per cent.

Flowlines were developed for the plant based on information obtained from the evaluation of borehole and box-cut samples obtained at the site. The basic plant flowline that is currently in use at Grootegeluk (Fig. 1) is described briefly to provide background to the need for the treatment of minus 0,5 mm coal particles.

TABLE I
PLANNED ANNUAL OUTPUTS

Material	Output, Mt
Run-of-mine coal (plant feed)	15
Coking coal	1,82
Power-station coal—middlings	2,6 to 3,5
Power-station coal—flotation tailings	0,9
Plant waste	11,5
Plant waste and overburden	18

Run-of-mine material, as large as 1 m in diameter, is fed at an average rate of 3 kt per hour to three rotary breakers. The oversize (plus 150 mm) fraction, which normally represents 1,7 per cent of the run-of-mine material, is regarded as waste. The minus 150 mm fraction is transported to a dry-screening plant screening at 15 mm. The fraction between 150 and 15 mm (66 per cent of the run-of-mine material) is transported to the Teska (static-bath) plant, which consists of five parallel modules each capable of treating 400 t of material per hour. The minus 15 mm fraction (34 per cent of the run-of-mine material) is transported to the primary-cyclone plant, which consists of five parallel modules each capable of treating 270 t of material per hour. Degradation material (the minus 15 mm underflow from the feed-preparation

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SA ISSN 0038-223X/$3.00 + 0.00.

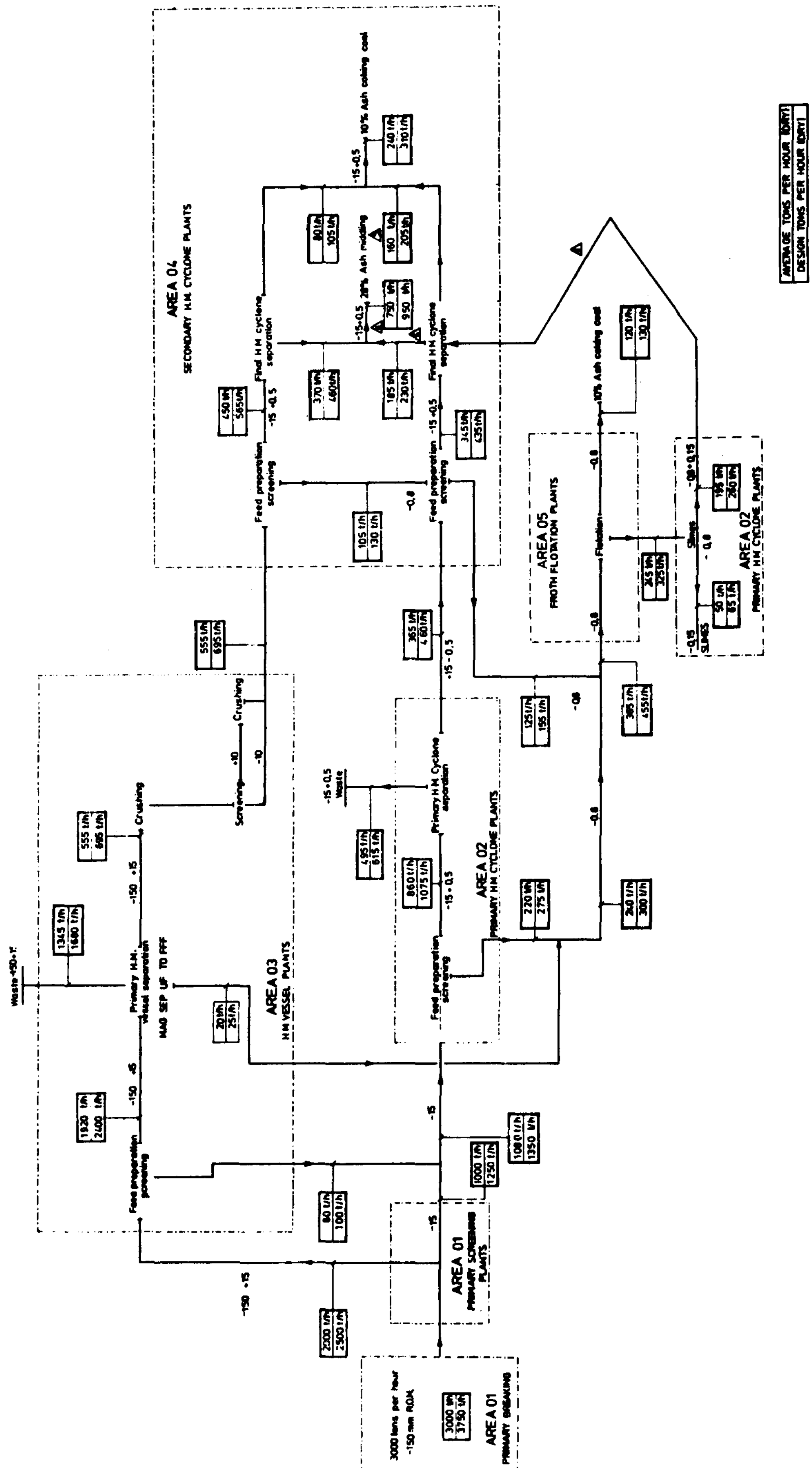

Fig. 1—Basic flowsheet at Grootegeluk beneficiation plant

screens) from the Teska plant is also fed to the primary-cyclone plant. A modification regarding the diversion of some of this degradation material to the secondary-cyclone plant is currently in progress. Primary separation at a separation density of 1,70 to 1,81 g/cm^3 is facilitated in the above-mentioned plants.

Mass yields of 29 and 55 per cent are obtained in the Teska and primary-cyclone plants respectively. Ash values of 26 to 30 per cent in the concentrate and 78 to 82 per cent in the waste are experienced in these sections. The blended concentrate originating from the primary-cyclone plant and the crushed product from the Teska plant are fed to the secondary-cyclone plant (38 per cent of the run-of-mine material), which consists of five parallel modules each capable of treating 230 t per hour of particles between 15 and 0 mm. A separation density of 1,31 to 1,34 g/cm^3 is strictly maintained because of the high percentage of near-gravity material (30 to 40 per cent).

The mass yield of concentrate is between 20 and 30 per cent at an ash value of 9,5 to 10,5 per cent. The ash values in the sink product are 33 to 36 per cent, and this product is classified as middlings for power-station requirements.

It was found that an average coking yield of only 7,6 per cent in the worst cases, and 11,4 per cent in the best, could be achieved if the product between 15 and 0,5 mm were beneficiated.

Investigations showed that approximately 15 per cent of the minus 0,5 mm particles are generated in the above-mentioned processes, containing approximately 33 per cent of the coking coal with an ash content of 10 per cent. The successful beneficiation of this fraction would therefore achieve an improvement of approximately 4,95 per cent in the yield of coking coal, resulting in huge savings in cost. It was therefore evident that the selection of a process to facilitate the recovery of minus 0,5 mm coking coal was a very critical phase in the development of this beneficiation plant.

Eventually a decision was made, based on studies of metallurgical performance and cost feasibility (as will be shown in this paper), to install five Outokumpu-type flotation modules each capable of treating a maximum of 93 t of minus 0,5 mm particles per hour, giving a concentrate with a mass recovery of 30 per cent and an ash content of 10,3 to 11,2 per cent.

Chemical Properties of the Fine Coal

The coal occurring at Grootegeluk contains small amounts of pyrite (FeS_2) and siderite ($FeCO_3$).

The distribution of pyrite in the coal is rather irregular, with more coarse than fine pyrite. Fine pyrite occurs in colonies either as colloidal micrometre-sized crystallites or as framboids of 10 to 40 μm in diameter. The size distribution of the pyrite determines the method used for the removal of sulphur from the coal.

Because the specified chemical analyses of the coking coal to be produced at Grootegeluk are 10 per cent ash, 1,1 per cent sulphur, 38,0 per cent maximum volatile material, and a minimum Roga value of 60, the beneficiation process is aimed at the production of low-ash clean coal, rather than at the removal of pyritic sulphur as a predominant step. The coal at Grootegeluk can be classified as a high-volatile bituminous type of coal of a vitrinitic maceral group with a high-carbon ratio (0,80 to 0,60), and an atomic carbon content of 51 to 62 per cent. In comparison with coals of the exinitic maceral group (which contain an even higher percentage of volatile matter), coals of the vitrinitic maceral group are more prone to oxidation, transforming to humic acids. This oxidation rate or aging is much more pronounced in the finer fractions, especially in the particles smaller than 0,5 mm.

The chemical analyses of Grootegeluk coals compared with European coals are given in Table II.

TABLE II

CHEMICAL ANALYSES OF GROOTEGELUK AND EUROPEAN COALS

Parameter	Coking coal	Middlings coal	Typical run-of-mine coal	European coals
Inherent moisture	3,0	2,2	2,4	4,0
Ash, %	10,2	30,5	65	20
Hardgrove index	53	48		
CO_2, %	0,48	1,25		
C, %	70,79	53,02	81	87
H_2, %	4,78	3,79	4,7	4,6–5,2
N_2, %	1,46–1,6	1,02	1,22	1,5
Organic sulphur, %	0,46	0,17	0,27	0,6
Total sulphur, %	0,82	1,0	0,95	1,0
Oxygen, %	9,31	8,3	8,6	4,0

Physical Properties of the Fine Coal

The ash contents and screening analyses that were originally envisaged for the minus 0,5 mm feed to the fine-coal beneficiation plant are given in Tables III and IV respectively.

TABLE III

ASH CONTENTS OF VARIOUS DENSITY FRACTIONS

Relative density g/cm^3	Mass yield, %		Ash, %	
	Fraction	Cumulative	Fraction	Cumulative
1,30	16,6	16,6	3,5	3,5
1,35	7,7	24,3	7,9	4,9
1,40	6,4	30,7	11,6	6,3
1,45	4,9	35,6	16,5	7,7
1,50	3,1	37,6	24,4	9,2
1,60	5,3	44,0	31,5	11,9
2,60	56,0	100,0	78,5	48,9

TABLE IV

SCREENING ANALYSIS OF FLOTATION FEED

Fraction, mm	Mass, % (cum)
+0,84	2,5
+0,595	10,5
+0,420	22,0
+0,297	35,1
+0,210	50,4
+0,149	67,0
+0,105	74
+0,074	80
+0,053	85
+0	100

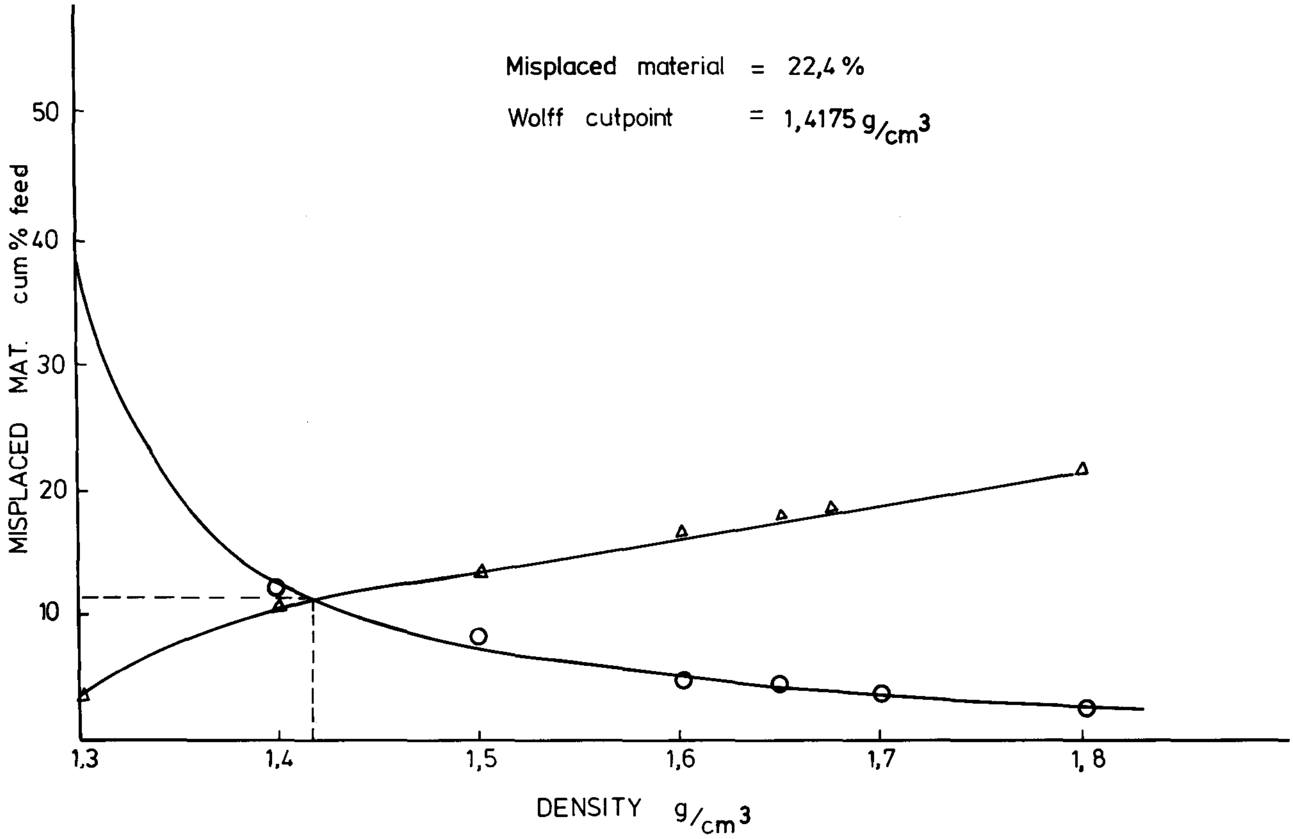

Fig. 2—Wolff curve

Selection of a Suitable Process

Grootegeluk coking coal is generally regarded as blend-coking coal because of its low swelling index and high volatile content. It was therefore decided that a target ash value of 10 per cent should be aimed for, and that a process giving maximum mass yields and economical operating costs should be used.

The following beneficiation processes were considered: Deister concentrating tables, water-only cyclones, heavy-medium cyclones, and flotation. Because of the large amount of near-gravity material present in the feed, the use of pulsating jigs was not investigated.

Deister Concentrating Tables

A series of tests were conducted on a quarter-size Deister table to indicate the operating conditions and maximum metallurgical performance that could be expected from the use of such tables for the upgrading of the coal smaller than 0,5 mm.

The following results were obtained. Detailed results are given in Tables V to VII, and Figs 2 and 3.

(a) Optimum conditions were achieved at 0,87 t/h.
(b) The solids in the feed should be maintained at between 25 and 40 per cent, which would prevent the misplacement of the lighter fractions to the waste zone on the table and the fine heavy fractions at the feed end.
(c) The ash value of clean coal at 0,87 t/h was 12 per cent at a yield of 64 per cent.
(d) As the minus 0,143 mm material present represented approximately 8 to 15 per cent, a lot of high-ash

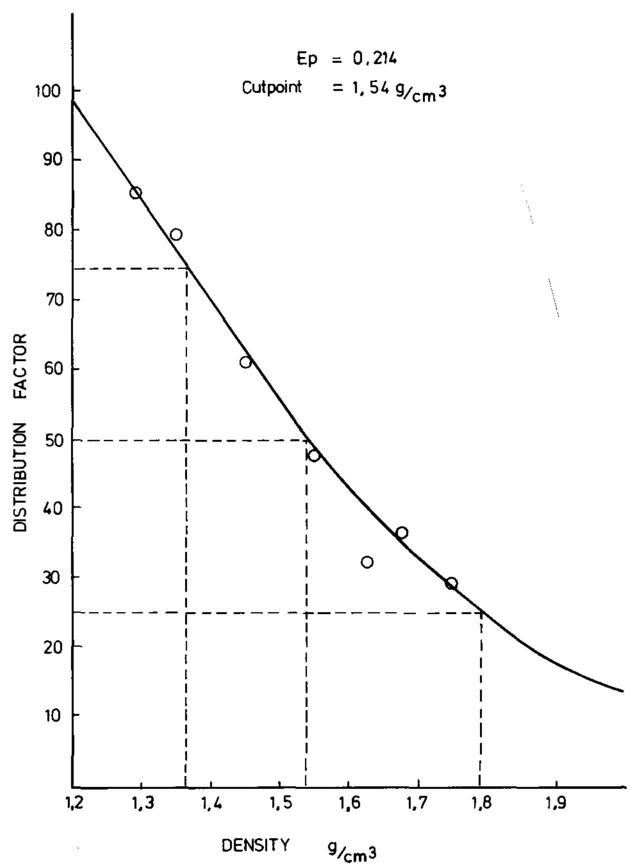

Fig. 3—Tromp curve

TABLE V

SIZE GRADING OF MINUS 0,84 mm FEED TO DEISTER TABLE

Fraction, mm	Mass, %		Ash, %	
	Fraction	Cumulative	Fraction	Cumulative
Head sample	—	—	26,9	—
+0,595	18,4	18,4	25,1	21,5
−0,595 + 0,420	19,8	38,2	24,5	25,2
−0,420 + 0,297	15,4	53,6	24,8	25,1
−0,297 + 0,210	12,0	65,5	24,9	25,1
−0,210 + 0,149	8,8	74,4	25,1	25,1
−0,149 + 0,105	6,0	80,4	25,2	25,1
−0,105 + 0,074	5,3	85,7	27,4	25,2
−0,074 + 0,044	5,8	91,5	29,2	25,5
−0,44	8,5	100,0	41,6	26,9
Total	100,0	—	26,9	—

TABLE VI

WASHABILITY FIGURES FOR DEISTER-TABLE CONCENTRATION

Density fraction g/cm³	Mass, %		Ash, %	
	Fraction	Cumulative	Fraction	Cumulative
Head sample	—	—	25,9	—
Float 1,4	52,4	52,42	6,0	6,0
Sink 1,4 Float 1,5	12,4	64,8	19,0	8,5
Sink 1,5 Float 1,6	7,3	72,1	29,0	10,6
Sink 1,6 Float 1,65	1,8	73,9	35,4	11,2
Sink 1,65 Float 1,7	1,5	75,4	39,8	11,7
Sink 1,7 Float 1,8	3,7	79,1	46,8	13,4
Sink 1,8	20,9	100,0	72,2	25,7
Total	100	—	25,7	—

fine material was misplaced to the low-ash region, which resulted in higher ash values for the clean coal.

(e) When a desliming cyclone was used to deslime the table feed at 0,143 mm to resolve the above problem, better results were achieved: 11,6 per cent ash at a yield of 60 per cent.

(f) Particles between 0,143 and 0,043 mm represented 18 per cent of the minus 0,143 mm fraction, with an average ash content of 8,8 per cent.

The overall metallurgical efficiency of the Deister table was not encouraging in that the Tromp cut-points were 1,54 g/cm³ at an *epm* value of 0,214, misplaced material representing 22,4 per cent. In any event, Deister tables could not achieve ash values of 10 per cent in the clean coal at the required yield, and therefore did not meet the required performance standards.

Water-only Cyclones

The use of hydrocyclones as a final cleaning stage was not feasible owing to the high ash values of the raw coal. The possible use of hydrocyclones as a pre-cleaning stage to froth flotation proved to be more successful because less cell colume would be required for the primary-rougher cells, and less impeller and cell wear could be expected. However, the increase in flotation efficiency for a pretreated feed over a normal feed was not pronounced enough to make pretreatment feasible.

Heavy-medium Cyclones

At the time that a decision had to be made, there was no proven record of the successful use of heavy-medium cyclones in the beneficiation of fine coal. It was therefore decided that this method should not be pursued.

However, a study on the use of heavy-medium cyclones on Grootegeluk fine coal conducted in 1982 indicated that good results could be achieved: 10 per cent ash in clean coal at a yield of approximately 64 per cent. The discard ash values were 62 per cent. This yield was obtained from a deslimed feed, the desliming being done at 140 μm, with 40 per cent of the material reporting to the slime fraction. An effective yield of only 38,4 per cent of the total feed could thus be achieved. A cost study showed that a capital expenditure of R11,8 million would be required for the implementation of this system.

This exercise was done when the flotation process was not yet giving satisfactory results. However, it soon became evident that the froth-flotation problems could be solved, and the idea of changing to heavy-medium beneficiation was abandoned.

Froth Flotation

As Grootegeluk coal can be described as a blend-coking coal, it was decided that it should be upgraded to an ash content of 10 per cent. The results obtained for samples

TABLE VII

FLOAT RESULTS FOR TROMP AND WOLFF CURVES (FIGS. 2 AND 3)

Sink-float fraction g/cm³	Concentrate 60,3% of feed: Mass, %: Fraction	Concentrate 60,3% of feed: Mass, %: Feed %	Concentrate 60,3% of feed: Cumulative sink material in concentrate %		Waste 39,7% of feed: Mass, %: Fraction	Waste 39,7% of feed: Mass, %: Feed %	Waste 39,7% of feed: Cumulative float mateial in waste %		Constituted feed: Fraction	Constituted feed: Cumulative	Actual feed: Fraction	Actual feed: Cumulative	Tromp distribution factor: Average r.d.	Tromp distribution factor: Factor %
Float			*Sink*				*Float*							
1,30	34,0	20,5		60,3	9,0	3,6	1,30	3,6	24,1	24,1	27,4	27,4	1,29	85
1,40	46,1	27,8	1,3	39,8	18,3	7,2	1,40	10,8	35,0	59,1	20,1	47,5	1,35	79
1,50	6,5	3,9	1,4	12,0	6,2	2,5	1,50	13,3	6,0	65,5	9,8	57,3	1,45	60
1,60	4,5	2,9	1,5	8,1	8,1	3,2	1,60	16,5	6,1	71,6	7,7	65,0	1,55	47
1,65	1,5	0,9	1,6	5,2	3,6	1,4	1,65	17,9	2,3	73,9	3,1	68,1	1,625	32
1,70	1,1	0,7	1,65	4,3	3,0	1,2	1,70	19,1	1,9	75,8	0,9	69,0	1,675	36
1,80	1,6	1,0	1,70	6,0	6,0	2,4	1,80	21,5	3,4	79,2	3,9	72,9	1,75	29
Sink														
1,80	4,4	2,6	1,80	2,6	45,8	18,2	—	—	20,8	100,0	27.1	100,0	2,15	12
Total	100,0	60,3	—	—	100,0	39,7	—	—	100,0	—	100,0	—	—	—

Results			Products		
				Mass, %	Ash, %
Tromp curve cutpoint:	1,54 r.d.	*Wolff curve cutpoint:*	Concentrate	60,3	12,5
75% separation	1,365 r.d.	1,4175 r.d.	Waste	39,7	42,8
25% separation	1,794 r.d.	Total misplaced material 22,4% of feed			
epm	0,214				
			Constituted	100	25.1

TABLE VIII

FLOTATION RESULTS (IN %) FOR VARIOUS COAL SAMPLES

Flotation products	DNC				Grootegeluk				Soutpansberg							
	Plant		Laboratory		Zone 10		Zone 11		Natural −0,84 mm		Primary middlings		Secondary middlings			
													Dry grinding (−0,21 mm)		Wet grinding	
	Mass	Ash	Mass	Ash	Mass	Ash	Mass	Ash	Mass	Ash	Mass	Ash	Mass	Ash	Mass	Ash
Concentrate	47,9	10,2	48,2	10,1	39,7	9,7	28,6	10,5	74,2	11,5	50,0	11,9	10,8	13,6	26,7	12,1
Middlings 2	—	—	—	—	—	—	—	—	—	—	—	—	—	—	18,3	25,5
Middlings 1	—	—	—	—	23,2	27,8	19,1	29,6	5,2	49,3	22,4	28,5	12,8	29,6	19,9	42,2
Tailings	52,1	38,1	51,7	38,3	37,1	66,8	52,3	75,7	20,6	63,9	27,6	41,8	76,4	39,6	35,1	52,2
Total	100,0	24,7	100,0	24,6	100,0	35,1	100,0	48,3	100,0	24,3	100,0	23,9	100,0	35,5	100,0	34,0
Flotation efficiency					74,9		59,0		93,3		71,9		33,4		64,6	

from Zones 10 and 11 are reported in Table VIII, which shows that flotation recoveries of about 39,7 and 28,6 per cent for Zones 10 and 11 material respectively were obtained.

The following findings were made.

(i) The best results were obtained when paraffin was used as the collector with starvation quantities of a frother such as aliphatic alcohol or a polyglycol type.

(ii) A dual-stage flotation process was necessary to achieve the required ash values in the concentrates. Ash values between 9,7 and 10,5 per cent were obtained during these tests.

(iii) Flotation efficiencies were established from a comparison of the flotation and washability results. The washabilities obtained from sink–float analyses were compared with flotation results for the same feed. The flotation efficiency for a certain grade of ash can be obtained from curves such as those in Fig. 4 (based on Table IX) and is expressed as a percentage of the theoretical value. The flotation efficiencies were low (74,8 and 59,9 per cent) because of the fine intergrowth of particles in Grootegeluk coal.

(iv) The regulation of air is an absolute necessity.

(v) Flotation was metallurgically superior to the use of Deister tables and water-only cyclones in that it achieved the required ash values at feasible recoveries, and it has an inherently higher selectivity.

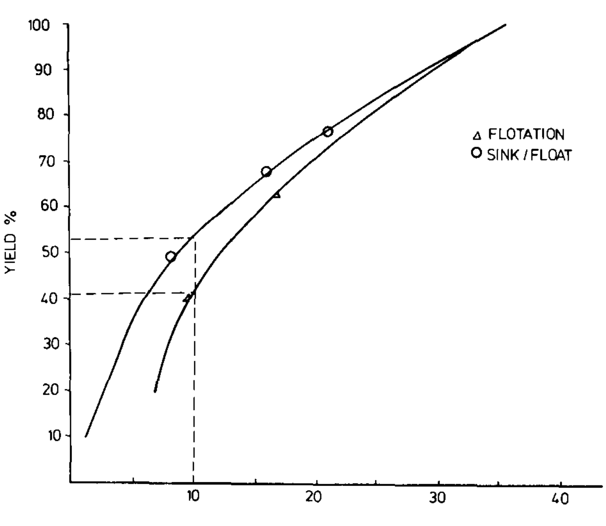

Fig. 4—Washability versus flotation, Grootegeluk Zone 10

It is a well established fact that the floatability of coal increases as the rank increases from lignite to bituminous coal of low and medium volatile content, and then decreases as the rank further increases to anthracite. The low floatability of high-rank coals such as anthracite is due to the graphite-like structures it contains, instead of hydrocarbon-like structures as in bituminous coals. Grootegeluk coal is of the high-volatile bituminous type.

The following equation, as compiled by Sun[3], expresses coal floatability, F_c:

$$F_c = x\frac{H}{2{,}08} + y\frac{C}{12} - \frac{H}{2{,}08} + 0{,}4\frac{S}{32{,}06} - z\frac{M}{18} - 3{,}4\frac{O}{16} - \frac{N}{14},$$

where H, C, S, N, O, and M designate the mass percentage of hydrogen, carbon, sulphur, nitrogen, oxygen, and

TABLE IX

DATA FOR FIG. 4

Flotation Feed Sink/Float:

Density	Mass % (cum)	Ash % (cum)
Float 1,4	49,5	8,1
	(54,1)	(10,0)
Float 1,5	67,1	15,3
Float 1,6	76,0	20,6
Sink	100,0	34,8

Flotation Results:

Product	Yield %	Ash (cum) %
Float concentrate	39,7	9,7
Middlings	62,9	16,4
Tailings	100,0	35,1

(Cum) = cumulative

moisture respectively. The assumed numerical values for x, y, and z are 2,5, 0,8, and 3 respectively. $\frac{H}{2{,}08}$ represents the number of the hydrocarbon $CH_{2,08}$.

The floatability indexes for South African and European coals are 4,78 and 6,05 respectively, which show the relative difficulty in the flotation of South African coals.

South African coals differ from British and European coals in that they contain a high percentage of middlings coal, i.e. coal with a high content of fixed ash. This problem demands finer crushing for liberation, resulting in the formation of superfine material.

The investigation of Grootegeluk coal started on a laboratory scale, and was followed by pilot-scale work at Iscor's pilot plant in Pretoria and on site at Iscor's 25 t/h Hoornbosch pilot plant. Investigations were also conducted on commercial-size cells.

Design of the Flotation Plant

After the pilot-plant and laboratory results had been evaluated, a set of design parameters was compiled. The process requirements resulting from the flowline for the total plant were stipulated to facilitate the transformation of the pilot-plant results to a full-scale application.

The following process requirements were used during this phase:

(a)	Design capacities:	Maximum throughput	495 t/h
		Normal throughput	375 t/h
(b)	Expected mass yield of contrate	Maximum	48 %
		Normal	30 %
(c)	Moisture content of final concentrate		18% (maximum)

(d) Treatment of flotation tailings by classification cyclones with a cutpoint of 0,15 mm maximum
The minus 0,15 mm fraction was to be transported to the tailings thickeners, and the spigot product had to be dewatered and blended with waste from the Teska-vessel and cyclone plants.

The laboratory and pilot-plant test results stipulated the following design parameters.

(1) A two-stage flotation facility using a roughing and a cleaning stage had to be designed.
(2) Provision had to be made for two-stage conditioning for the collector, frother, and feed-dilution requirements.
(3) Flotation cells using an external air supply had to be used owing to the critical effect of the air flow on performance.

During the laboratory and pilot-plant tests, the following technical parameters were determined for optimum recovery of coking coal.

(i) Dosage rates of 4 kg/t of kerosene and 0,04 kg/t of frother (MIBC) were envisaged.
(ii) Optimum flotation performance was expected at a pulp density of 15 per cent solids (by mass) in the feed.
(iii) Since flotation-feed thickeners produced slurries at a pulp concentration of 35 per cent solids (by mass), dilution water was necessary at the secondary conditioners. (The Grootegeluk coal-beneficiation plant was the first plant to treat the total minus 0,5 mm fraction in thickeners.)
(iv) No desliming of feed was required by the water-only cyclones because of an expected coking-coal loss of about 4 per cent of the flotation feed.
(v) The retention time for the roughing stage was 10 minutes, and that for the cleaning stage 6 minutes.
(vi) The cell productivity was 0,75 $t/m^3/h$ for roughing and 1,5 $t/m^3/h$ for cleaning.

Calculations regarding pulp flowrates, dilution-water flowrates, and required cell volume, cell turbulence, etc. were made. During this stage, the general trend in flotation was towards the use of larger cell units, resulting in a lower power usage (kilowatts per ton of feed solids). Various tests were conducted with commercial-type units, and it was eventually decided that seven 16 m^3 modules should be used for the first-stage cleaning (roughing) and seven 3 m^3 modules for the second-stage cleaning. Five Outokumpu-type modules each treating 75 to 93 t/h solids were proposed.

Large cells were selected for the following reasons.

(1) There is less turbulence, resulting in the formation of a more stable froth at lower aeration rates.
(2) The selectivity is higher. Cell sedimentation due to coarse, high-ash particles resulted in an effective decrease of 12 per cent in cell volume. The sedimentation kinetics to some extent could be controlled by aeration rates.
(3) An external supply of air is vital, and the Outokumpu cells are appropriate for such an application.

Various drying methods for the flotation concentrate were considered: disc vacuum filters, drum vacuum filters, horizontal-belt vacuum filters, and centrifuges. However, owing to the stable nature of the froth in the concentrate, the vacuum filters, and even the dewatering screens, yielded a high moisture content in the product (approximately 25 per cent).

The vacuum filters had the disadvantages of high maintenance, high power consumption, large space requirements, and clogging of the filter cloth.

Tests with a screen-bowl centrifuge at the Iscor Durnacoal plant indicated that moisture contents as low as 12 to 16 per cent could be achieved, and it was thus decided that a 900 mm by 1800 mm long screen-bowl centrifuge should be installed on each of the five flotation modules. These centrifuges were each designed to treat 30 t of flotation concentrate per hour.

Modular flow diagrams were compiled specifying capacities, flowrates, etc., and tender enquiries were processed. The flow diagram is given in Fig. 5.

Commissioning and Development of the Froth-Flotation Plant

Flowline Restrictions

The performance of the froth-flotation plant was hampered by the following problems:

(a) variations in the pulp density of the feed,
(b) variations in the flowrate of air to the flotation cells,
(c) uneven pulp distribution at the five-way distributor,
(d) sedimentation in the conditioners due to the presence

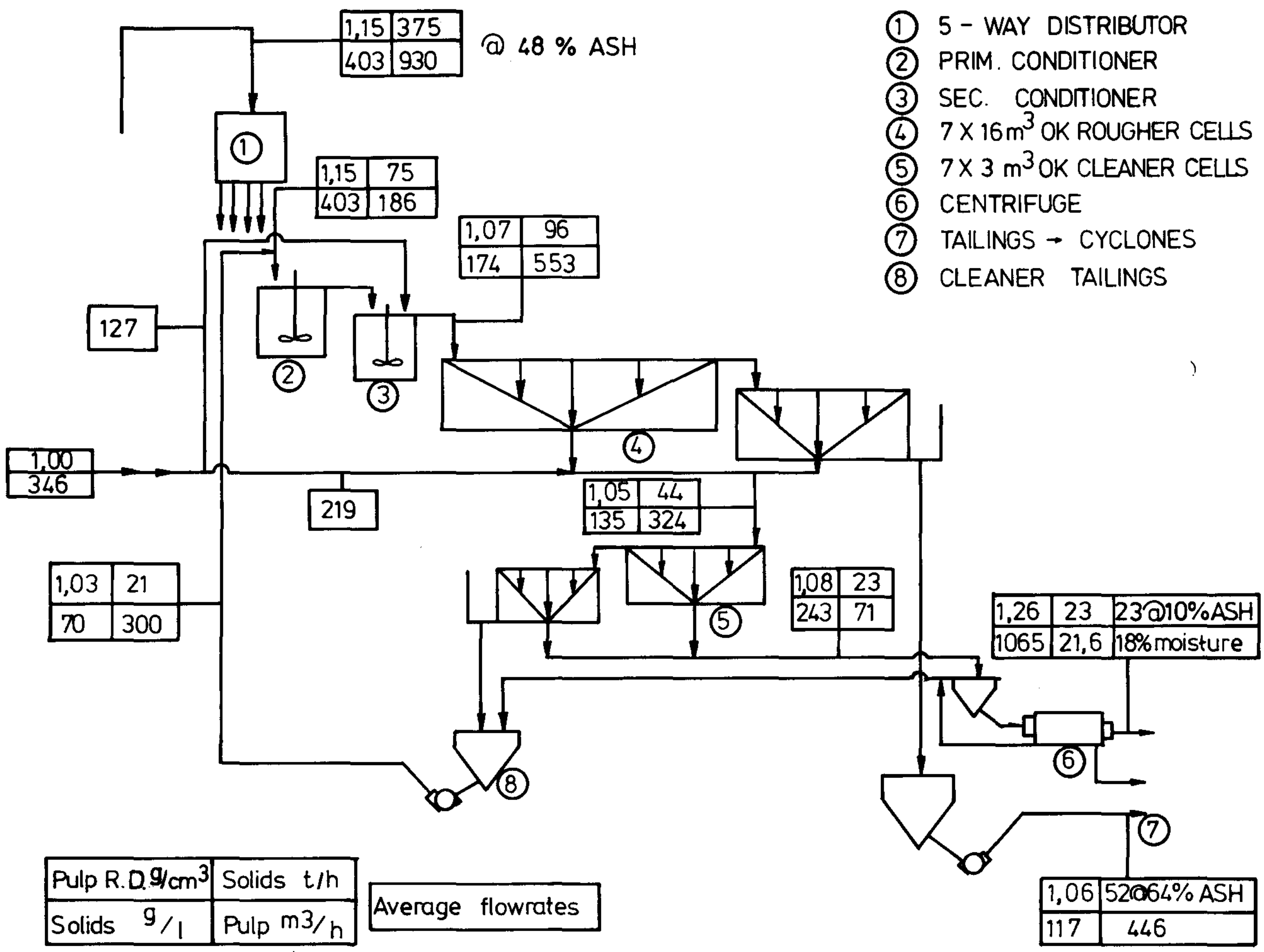

Fig. 5—Proposed flotation flow diagram

of particles larger than 0,50 mm in the feed,

(e) inaccurate control in the dosage rates of reagents,

(f) inefficient performance of the pulp-level control system in the cells, and

(g) variations in pulp density.

During the first month of operation, the flotation recoveries varied from 2,3 to 61,5 per cent by mass, and the ash in the concentrate varied from 7,5 to 14,8 per cent. During the same period, the ash values in the flotation feed varied from 18,5 to 46,8 per cent, with the result that absolutely no quality-control measures could be achieved.

Variations in the Densities of the Feed Pulp

During the pilot-plant tests, it was confirmed that pulp densities of 15 per cent solids in the feed should be maintained if optimum flotation performance were to be achieved. However, variations in pulp density from 7 to 19 per cent were experienced owing to the following:

(i) inefficient density control of the underflow from the flotation-feed thickener due to short circuiting in the thickeners and inadequate control instruments,

(ii) variations in the solids content of the recirculating cleaner tailings,

(iii) wrong electrical sequencing resulting in unstable start-up procedures after the tripping of the product conveyors, and

(iv) problems with the feed rate of dilution water to the cleaner stage.

Variations in the pulp feed resulted in excessive variations in the dosage rates of reagents, the retention times, and the pulp levels in the cells, impeding the recovery of coking coal.

Rectifying actions regarding (i) and (iii) were taken and a density-control system using water additions to the underflow of the feed thickeners was installed. To resolve the problem of varying flowrates of dilution water, the installed steady-head tank was bypassed, direct feed being sent to the re-flotation stage.

Variations in the Flowrate of Air to the Flotation Cells

As mentioned before, the regulation of air to the flotation cells is crucial. Air requirements for the seven 16 m^3 rougher stages and the seven 3 m^3 cleaner stages were found to be a minimum of 2,2 m^3/m^2 and 2,1 m^3/m^2 respectively. (The surface area used is that of the froth area in the cells.)

Blowers were designed according to these requirements, but constant-volume, instead of constant-pressure, type blowers were installed. Each cell was fitted with its own air-regulating valve, but effective regulation of the airflow could not be achieved because of the variations in pressure. A makeshift counterweight pressure-relief valve, calibrated at 19,11 kPa, was installed in the air supply, and this proved to be successful.

After optimization, it was found that the actual air requirements were between 0,7 and 0,9 $m^3/m^2/min$ for both the roughing and the cleaning stages.

Uneven Pulp Distribution at the Five-way Distributor

Uneven pipe lengths from the central distributor to the five flotation modules resulted in marked segregation between the different modules (Table IX), aggravated by the fact that plus 0,5 mm particles were present in the feed reporting primarily to the three inner-most modules.

A new five-way distributor using a rotating mechanism that ensured equal opportunities to all the particles to be diverted to each of the five modules was designed.

Sedimentation in the Conditioners

Owing to an excessive amount of plus 0,5 mm material in the flotation feed, orginating primarily from the conveyor-belt washing system, sedimentation occurred in the conditioners. A sieve bend/screen protection system was installed to remove plus 0,5 mm material from the belt washings. Investigations are being carried out on the removal of all the plus 0,5 mm material from the flotation feed.

Studies on the performance of the plus 0,5 mm particles in the flotation cells indicated that the recovery of that fraction was 18 per cent, compared with 32 per cent for the minus 0,5 mm and 57 per cent for the minus 0,1 mm fractions.

Inaccurate Control of Reagent Dosages

Metering pumps were used to feed reagents from the ground floor to the conditioners, resulting in very difficult control by the operating personnel. The dosage pumps were therefore moved to the same level as the conditioners.

Although the pilot-plant tests had predicted a paraffin dosage of 4,0 kg/t and a frother dosage of 0,04 kg/t, it was found that the paraffin dosages could vary from 1,4 to 4,8 kg/t for the required results to be obtained. During the initial stages, collector dosages of up to 5 kg/t were necessary for the desired results to be obtained.

A three-stage dosage system was implemented to increase the recovery on the cleaner cells and decrease the ash-value gradients on the rougher cells.

Inefficiency of the Level-control Instruments

Originally, each bank of cells was equipped with one hand-operated plug valve and one automatic plug valve fitted with a pneumatic actuator controlled from a float. Owing to the malfunctioning of the pneumatic actuators, as well as large fluctuations in flow volume, the level-control system did not function. Foxboro pneumatic actuators were then installed on all the plug valves, which resulted in efficient functioning of the system.

From this point onwards, the froth thickness could be controlled with positive results by means of proper regulation of the aeration. Performance tests conducted during this period indicated clean coal yields of 35 to 40 per cent at ash values of 12,6 to 13,1 per cent. The consumption of paraffin at 2,12 kg/t at pulp densities of 1,10 g/cm^3 and 1,06 g/cm^3 in roughers and cleaners respectively were experienced. Both these pulp densities were found to be too high.

High Pulp Densities

The flowrates of dilution water were restricted to 2 m^3/min owing to low flow capacities in the rougher and cleaner cells. This resulted in high pulp densities, which caused a higher consumption of reagents and decreased the cell selectivity, with a resultant high-ash content in the concentrates. To facilitate the addition of more dilution water, the passage ways between the cells were enlarged.

Training of Operators

If one considers that the Grootegeluk flotation facility was designed to treat 9 kt of fine coal daily and to produce 30 per cent of the final coking-coal tonnage, it is evident that this area should be under competent supervision to maintain highly efficient performance.

Shortcomings in this area were eliminated by proper training of all the plant operators, who have all now successfully completed the Coal Preparation Course of the South African Coal Processing Society. This gave them a conceptual idea of the delicacy of a flotation facility, and resulted in considerable improvements in production and process control.

Quality-control Functions

Because no provision was made for routine sampling, there was a lack of feedback to the process-control section. Automatic samplers of various types have been tested, but none was found to be successful. Eventually it was decided that injection-type pipe samplers should be installed for the rougher tailings and the cleaner concentrate. The currently designed five-way distributor makes provision for an automatic feed-sample compartment.

Currently, hand sampling is done and will continue until these automatic samplers have been installed. Quality-control strategy in the flotation plant entails the following.

(i) After start-up, the dosages of reagents and the airflow rates are set at a norm.
(ii) The ash values of the feed, concentrate, and tailings are reported at 30-minute intervals.
(iii) The dosages of reagents are adjusted according to prescribed values, depending on the type of coal and the ash value.
(iv) The froth stability and froth loading are inspected visually at regular intervals by plant operators specifically assigned to the flotation section.
(v) The froth level is adjusted depending on the characteristics of the froth carry-over.
(vi) The following parameters are monitored for process and quality-control purposes:
   (a) the underflow density of the feed thickener (automatic),
   (b) pulp density of the flotation feed (by hand),
   (c) ash values of the feed, concentrate, and tailings (ash monitor by hand),

(d) the tonnage rate (automatic), and
(e) the consumption of reagents (by hand). The calibration of the reagent-dosage pumps is checked regularly by hand.

It is important to note that, although the flotation plant had been designed for an average ash content of 48 per cent in the feed, yielding a waste ash value of 64 per cent, it was found in practice that the ash in the feed varied from 28 to 38 per cent, the waste ash values varying from 34 to 48 per cent.

Because of this, the flotation waste was diverted from the waste to the middlings product. An investigation is currently under way on the diversion of the coarse plus 0,5 mm flotation tailings to the secondary heavy-medium cyclone plant so that any coking coal lost in this fraction can be reclaimed.

Current and Future Developments

The following typical flotation performances are being achieved:

Average ash content of feed	28 to 38 %
Average ash content of concentrate	9 to 12 %
Average ash content of tailings	39 to 47 %
Average mass recovery	28 to 44 %.

With material originating from Zones 9 and 10, mass recoveries of 46 to 48 per cent were experienced in the flotation plant, with the final coking-coal product containing some 27 per cent of the flotation concentrate.

Average consumption of paraffin	1,1 to 1,9 kg/t
Average consumption of frother	0,01 to 0,02 kg/t

The following development work is currently in progress.

(a) Installation of extra paddles to promote the removal of froth.
(b) Optimization of the centrifuge performance, the aim being to implement a central feed cone from which all five centrifuges will be fed, eliminating the fluctuations in the feed rate to the centrifuges, which result in high moisture contents in the concentrates.
(c) Finalization of the automatic-sampling devices.
(d) A study of the use of the last three rougher cells as cleaners or implementation of a three-stage system.
(e) Removal of coarse material (plus 0,5 mm) from the flotation feed by means of a protection screen (Fig. 6).
(f) Dewatering of the flotation tailings by means of a horizontal-belt vacuum filter.
(g) Optimization of independent dart-valve action to optimize level control.
(h) Installation of a pulp-density control system on the cleaner cells.
(i) Elimination of the short-circuiting of material in the flotation-feed thickeners and minimization of variations in the density of the thickener underflow by means of flow deflection.

Vacuum Filtration of Flotation Tailings

At present, some 60 t/h of minus 0,5 mm flotation-tailings slimes with an ash content of 45 per cent are pumped to the slimes dams. Dewatering of the tailings-thickeners underflow and its addition to the middlings product, would increase the middlings production and some 250 m^3/h of water may be saved.

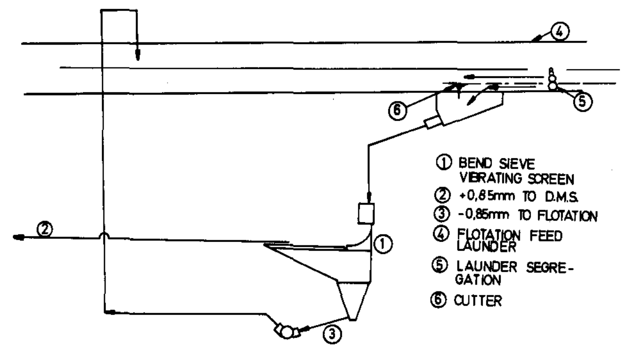

Fig. 6—Flotation-plant protective system

A 1 m^2 filter test unit was installed on the underflow from the tailings thickeners so that the relevant operating parameters could be determined. Owing to the fine particle size (35 to 70 per cent minus 0,043 mm), low filtration rates were experienced, but, by the addition of optimum amounts of anionic and cationic flocculants, marked increases in filtration rates were achieved. The filtration rates varied from 0,7 to 1,2 t/m^2/h with dosage rates of 50 g of anionic flocculant and 15 g of cationic flocculant per ton. A filter cake total moisture content of 25 per cent (i.e. 23 per cent surface moisture) was produced, which was acceptable for conveyor-handling and blending purposes.

The following proposals were made by the mine's metallurgical staff after the test run on the filter had been completed.

(1) A filter facility consisting of two 80 m^2 filters should be installed to filter the combination of flotation tailings and conveyor-belt washing material at a normal flowrate of 73 t/h. The flowline is shown in Fig. 7.
(2) The operating costs would be
R0,49 per ton of dry filter cake for the flocculants,
R0,20 per ton of dry filter cake for the power, and
R0,10 per ton of dry filter cake for the maintenance.

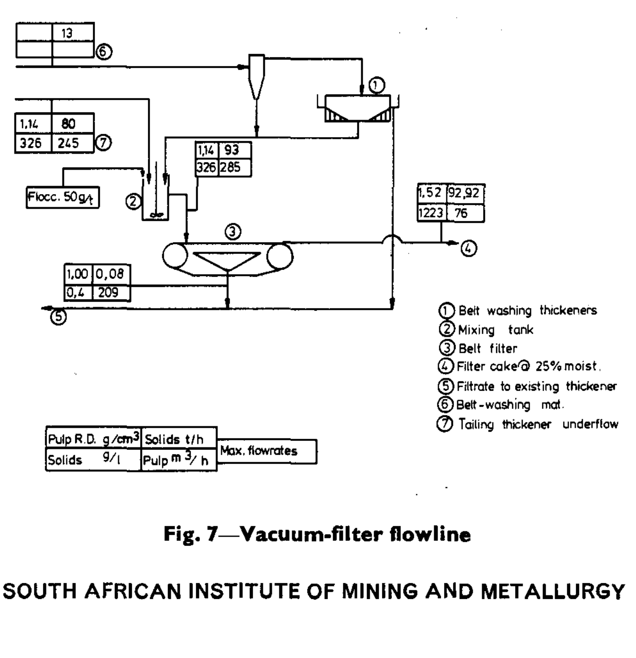

Fig. 7—Vacuum-filter flowline

(3) A discounted cash flow of 45 per cent, i.e. a pay-off time of 30 months, with a capital expenditure of R2 million could be achieved.

Acknowledgements

The authors thank the personnel of Grootegeluk and Iscor for their co-operation in the preparation of this paper. Gratitude is also expressed to the Iscor Mining Department for its permission to publish this paper.

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Extractive metallurgy

Extraction Metallurgy '85 is the fifth in the IMM series of international symposia devoted to advances in extraction metallurgy. It will be held from 9th to 12th September, 1985, at Imperial College of Science and Technology, London, England. The Symposium is being organized by The Institution of Mining and Metallurgy, in association with The Australasian Institute of Mining and Metallurgy, Benelux Metallurgie, the Canadian Institute of Mining and Metallurgy, the Société Francaise de Metallurgie, GDMB, Germany, Norsk Metallurgisk Selskap, Norway, the Metallurgical Society of AIME, the Society of Mining Engineers of AIME, and The South African Institute of Mining and Metallurgy.

Adaptation to change is the theme of the Symposium. New orebodies may require new processes or the adaptation of old techniques. Social pressures and government regulations may demand reductions in effluents. Customers may demand higher-quality products. Cost escalation and other factors may change the relative merits of alternative processes and the relative economics of different operations. New materials (often developed elsewhere) become available. These and many other factors, combined with a professional desire to innovate and improve, bring about changes in the technology of extraction metallurgy.

Sometimes the changes are radical; more normally, however, they are gradual modifications of old technology. Some are quick to react, others are more conservative. But, wherever a change is successfully developed, it is usually not long before it is available for use elsewhere.

Extraction Metallurgy '85 provides a forum for discussion of why, how, when, and where such changes have been, or may be, developed.

It is expected that sessions will be held on the following topics:

- Production of the light metals (Al, Mg, Ti)
- Plasma technology and uses
- Extraction and recycling of electronic/IT metals
- Analytical techniques and quality control
- Complex sulphide metallurgy
- Chloride metallurgy applications
- Effluent treatment and by-product recovery
- Recent developments in gold metallurgy
- Waste-heat recovery
- Process development and design.

Papers are invited on research work, process innovations, or changes in plant practice related to the above or other topics that reflect the industry s adaptation to changing circumstances.

Abstracts (250 to 300 words) of proposed papers should be submitted to the Conference Office, The Institution of Mining and Metallurgy, 44 Portland Place, London W1N 4BR, England, before 1st October, 1984. Authors of approved abstracts will be requested to submit their completed manuscript in March 1985.

The preprinted volume of papers will be distributed to registrants before the symposium.

Technical visits to research establishments and metallurgical plants in the United Kingdom, Europe, and Scandinavia are planned during the week following the Symposium.

Social events will include a welcoming reception and Symposium dinner, at which accompanying persons will be welcomed.