Some factors influencing gold recovery by gravity concentration

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SYNOPSIS

Recent studies have shown that the overall extraction of gold from Witwatersrand ores is increased as the proportion of gold reporting to the amalgamation circuit is increased. Testwork has therefore been carried out with a view to improving recovery by gravity concentration.

In this paper, the history of gold recovery by gravity concentration is reviewed, and methods and machines at present in use are briefly described. The results of plant and laboratory tests are presented, and the limitation of existing concentrators is discussed.

A pilot plant is being built for the testing of improvements to the existing concentrators and of several new devices, particularly the Reichert-cone concentrator.

SINOPSIS

Onlangse studies het getoon dat die totale ekstraksie van goud uit die Witwatersrandse ertse toeneem namate die hoeveelheid goud in die amalgamasiekring toeneem. Daar is dus toetse uitgevoer met die oog op die verbetering van die herwinning deur swaartekragkonsentrasie.

Hierdie verhandeling gee 'n oorsig oor die geskiedenis van goudherwinning deur swaartekragkonsentrasie en beskryf kortliks die metodes en masjiene wat tans in gebruik is. Die resultate van aanleg- en laboratoriumtoetse word uiteengesit en die beperkings van die bestaande konsentreerders bespreek.

Daar is 'n proefaanleg in aanbou om die verbeterings aan die bestaande konsentreerders en verskeie nuwe toestelle, veral die Reichert-keëlkonsentreerder, te toets.

HISTORICAL

Gravity concentration of gold has been used since antiquity[1] but its importance has decreased since the turn of the century, when the cyanide process was developed. In the early days of the Witwatersrand[2], before the introduction of cyanidation, the ores were stamp-milled and the gold recovered by amalgamation. This practice became obsolete as the oxidized ores became exhausted. The introduction of tube mills with corduroy-blanket strakes on the discharge to collect the coarse gold, which was recovered by amalgamation later, gave rise to the cyanide plant that we know today.

The vastly more efficient cyanide process has caused many plants to dispense with gravity concentration, although they tacitly use longer residence times in the cyanide solution to compensate for the lack of concentration. Dorr and Bosqui[3] emphasize the importance of the early removal of gold from the milling circuit and advocate gravity concentration, especially for those ores in which a significant proportion of the gold is associated with base metal sulphides.

The history of modern gravity concentration on the Rand mines goes back to the corduroy blankets, which were highly labour intensive. As a result, the endless-belt concentrator, the Johnson drum concentrator[4], the plane table, and jigs came into use. Jigs have not gained wide acceptance, but the two concentrators are still in current use on Anglo American Corporation mines. An additional bonus in the form of osmiridium is realized on plants employing gravity-concentration circuits, and this often outweighs the cost of the treatment.

Douglas and Moir[5] reviewed South African gold-recovery practice in 1961, pointing out that two-thirds of the major plants incorporated gravity concentration followed by cyanidation of the bulk ore, up to 73 per cent of the gold being recovered by gravity concentration when the gold was relatively coarse.

In this paper we shall attempt to put the argument for gravity concentration in a more balanced light, especially since computer correlation studies have shown it to have a significant effect on the overall plant recovery.

GRAVITY CONCENTRATORS

In 1961, the following methods of gravity concentration were in use:

(1) corduroy-blanket strakes,
(2) Johnson concentrators,
(3) plane tables,
(4) endless-belt concentrators, and
(5) jigs.

Corduroy-blanket strakes

These, or rubber-blanket strakes, were apparently still in use on 24 plants in 1961. They were placed at the tube-mill outlets, and were manually washed at intervals to recover the concentrates. The method was highly labour-intensive and constituted a security risk and, as far as is known, has now been entirely replaced by mechanical concentrators. This change has been accompanied by circuit modifications to provide selective grinding and by the installation of as few machines as possible.

Johnson concentrator

This mechanical concentrator[6] was developed in 1925 and has since gained wide acceptance. It consists of a continuously rotating cylindrical drum, having its axis inclined at between 2,5 and 5 degrees to the horizontal, and lined internally with riffled rubber. Pulp flows down the length of the drum, and the heavy minerals settle in the riffles and are carried out of the pulp stream to the top, where they are washed into a launder by sprays. In 1961, mines using these machines in conjunction

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with endless-belt concentrators for upgrading the drum concentrate reported an average gold recovery of 38 per cent by concentration and amalgamation, the range being from 30 to 52 per cent.

Plane table

This device, developed at Rand Leases, consists of riffled rubber covering a series of inclined smooth surfaces, with steps between them. The pulp flows down the length of the riffles, and the riffles collect the gold concentrate, which is drawn off through slots at the steps. The faster-flowing pulp is carried across the slots and is collected at the end of the table. At Rand Leases, the plane-table concentrates were reground and reconcentrated on secondary plane tables, followed by a final concentration stage using James tables. In this circuit, 51 per cent of the gold was recovered by amalgamation.

Endless-belt concentrator

The belt concentrator was developed at Brakpan in 1949 to replace corduroy blankets. The unit consists of a flat, endless rubber belt 150 cm wide, with 300 cm between head and tail pulleys, and inclined at a slope of 3 degrees to the horizontal. The upper surface of the belt has saw-tooth riffles, 5 mm deep with a pitch of 10mm, running across the belt. The belt moves continuously against the pulp stream at a speed of 30 cm per minute. A spray water-pipe washes concentrate from the riffles after the belt has passed over the head pulley. The unit has been used as a primary concentrator and, more usually, as a cleaner concentrator for the retreatment of the product from Johnson drums.

Jigs

These machines have never gained wide acceptance in the South African gold industry, probably owing to the large bulk of concentrate. At Durban Roodepoort Deep and Western Areas, Yuba-Richards jigs are used. At Western Areas, the jigs treat secondary mill discharge, producing a concentrate that is dewatered and reground before reconcentration in secondary jigs. Tailings from these jigs are returned to the regrind-mill cyclones, and concentrates are cleaned on endless-belt concentrators and a James table. The overall plant recovery is of the order of 30 per cent by gravity concentration.

ANGLO AMERICAN PRACTICE

Of the Group's 13 plants, 8 practise gravity concentration. Gold recoveries by amalgamation range from 21 to 52 per cent.

Only two types of concentrators are in use, in all cases the Johnson drums acting as roughers and the endless belts as cleaners. Flowsheets are similar in all plants, differences being in the number of concentrators, pulp feed rates, and the ratio of Johnson drums to endless belts (Figs. 1a and 1b). Table I gives mechanical details and operational data for the concentrators in the various plants.

Typically, secondary tube-mill product is pumped to tertiary cyclones whose underflows are distributed among several Johnson drum concentrators. The tailings are returned to the mill circuit, while the concentrates are retreated on endless-belt concentrators, there usually being one belt for every 4 to 8 Johnson drums. The belt concentrate constitutes the feed to amalgamation while the belt tailings are returned to the mill circuit.

As part of a major project aimed at increasing gold recovery in Anglo American plants, improvements in gravity concentration are being investigated both in the laboratory and on the plants. The project is divided

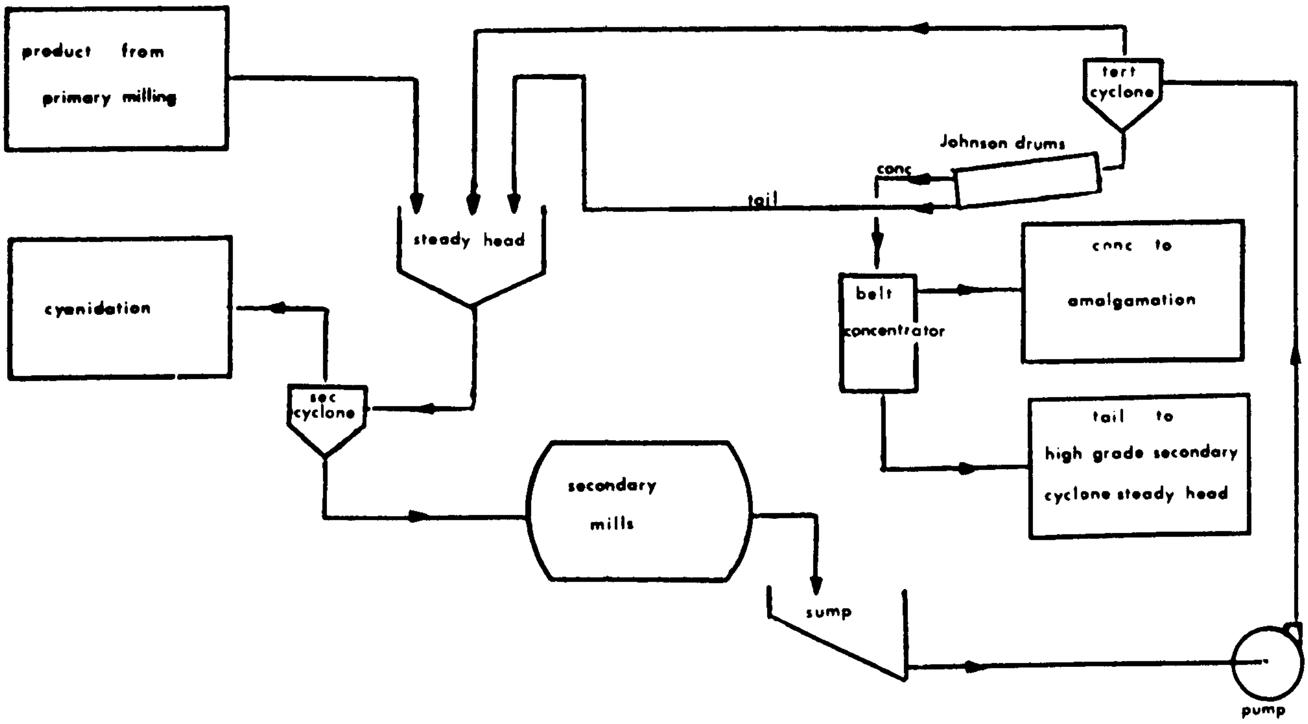

Fig. 1a—Concentration with secondary milling

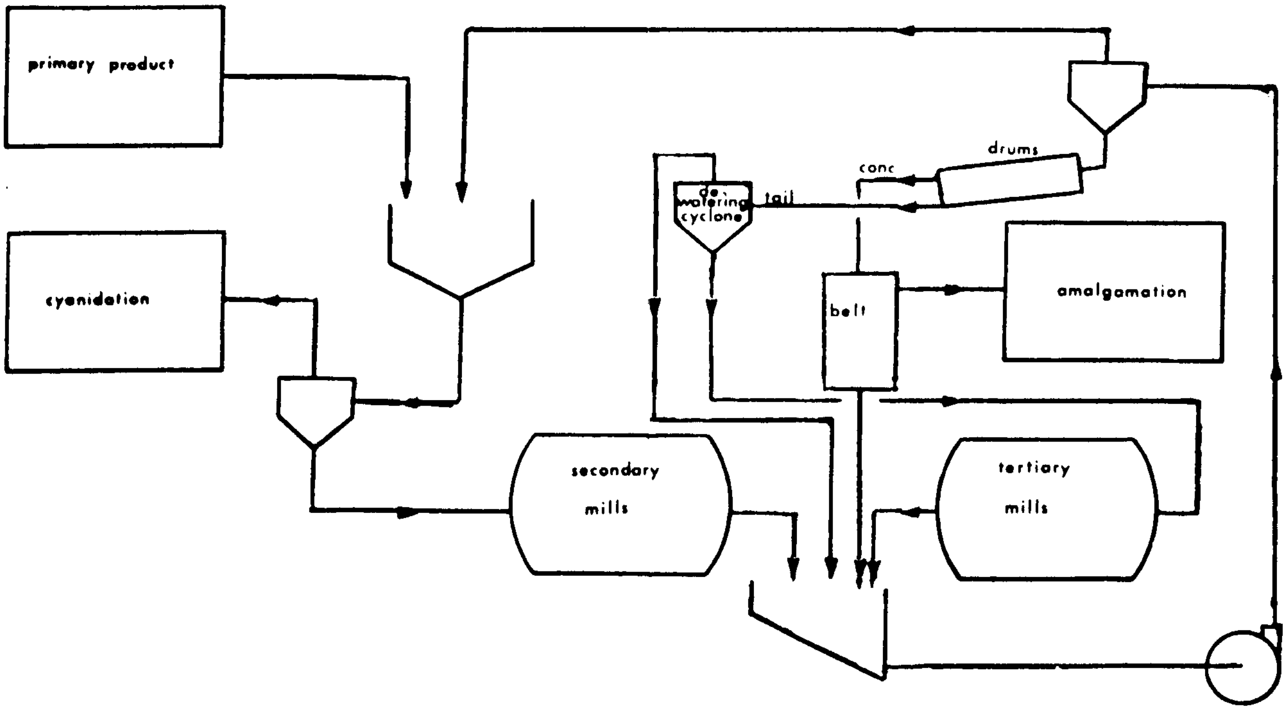

Fig. 1b—Concentration with tertiary milling

into two phases: the first is short term, with its object the optimization and improvement of existing machines and circuits, and the second is longer term—the development of alternative machines and/or circuits. The motivation for this investigation was the finding, during a computer correlation study of plant operating data, that the overall gold recovery is increased as the proportion of gold recovered by concentration and amalgamation is increased. Without going into details this is obviously due to the fact that recovery by amalgamation is more efficient than that by cyanidation.

GRINDING AND STRAKING

Grinding tests were carried out to determine liberation grinds for ores from several of the Group's mines. The term "liberation" as used in this paper is best defined as that particle size which maximizes profit for the subsequent process. Hence, for cyanidation or any leaching operation, the mineral grain need be only partly exposed, whereas, for gravity concentration, it must be almost completely separated from the gangue.

In an ore as unhomogeneous as the gold-bearing reefs, there can obviously be no single liberation size—the larger particles of gold are liberated at a much coarser grind than the finer particles, for example those enclosed within pyrite grains.

Tests were carried out to determine what percentage of the gold could be recovered at various degrees of grinding. Ore samples were milled for various periods and the products passed over a small vibrating strake covered with riffled rubber. Curves for recovery versus grinding are shown in Fig. 2 for three different samples of ore. Recoveries in the range 75-80 per cent were achieved at relatively coarse grinds. The Vaal Reefs ore, in particular, yielded a gold recovery of 75 per cent without any milling at all (crushed to pass 1,5 mm). As milling was continued, recovery increased slightly to 79 per cent at 93 per cent minus 74 μm. The curves for Western Deep Levels high-grade and low-grade ores show increasing recovery up to about 75 per cent minus 74 μm, after which there is a marked decrease in recovery with finer grinding. This effect is thought to be due to a flattening of the liberated gold particles as the result of overgrinding. The flattened particles settle far less easily than spherical grains, and are more likely to be lost to the tailings. The fact that the same effect was not observed with the Vaal Reefs material is probably due to the much coarser gold in this ore—even when flattened, the grains are still effectively caught in the riffles.

The results of these tests suggest that gravity concentration could be effectively incorporated much earlier in the circuit than it is at present, and that high recoveries can be obtained with simple equipment very similar in principle to that already in use. Early recovery of liberated gold has long been advocated but somehow seems to have been forgotten over the years. The deleterious effects of overgrinding—not only flattening but also comminution—of the gold, would thus be avoided.

TESTWORK ON THE JOHNSON DRUM CONCENTRATORS

A series of tests was carried out on one of the low-grade Johnson drum concentrators at Western Deep Levels Mine. This drum had been modified to facilitate sampling, and a variable-speed drive was installed.

Composite samples of the feed, concentrate, and tailing were taken under various conditions of solids feedrate, feed pulp density, and drum speed. The least varied was the pulp density, which was adjusted only to ensure that the pulp did not sand out in the drum, the highest possible pulp density and hence the longest residence time possible being thus achieved. The gold content of the feed was an interfering variable, ranging from 91,5 to 197,6 g/t, but no correlation was found between this and the concentrate grade, which varied from 920 to 5468 g/t. The mass of concentrate was 0,44 t/h at the lower figure and 0,07 t/h at the higher. The solids feedrate was varied from 5,1 to 26,4 t/h. The percentage recovery was found to be inversely proportional to the solids feedrate (Fig. 3), while the actual yield of gold (kg/h) was roughly constant (Fig. 4).

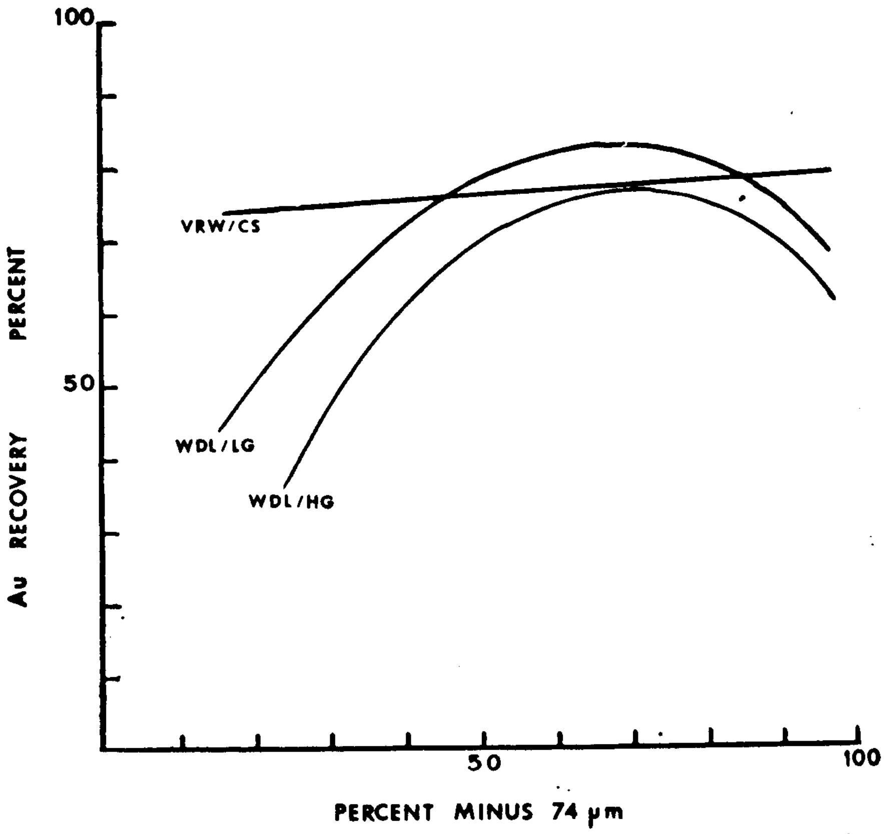

Fig. 2—Strake tests—recovery versus grind

Fig. 3 is a least-squares fit of percentage gold versus feedrate and shows gold recovery to increase hyperbolically with decreasing tonnage. This is to be expected since the residence time of the pulp in the drum is increased when the tonnage is decreased. What is not so expected is that the actual mass of gold recovered per hour remains constant, as can be gauged from Fig. 4. Here the masses of gold recovered per unit time are plotted on normal probability paper in order of increasing magnitude against the cumulative observed frequency. The fact that a reasonably straight line is obtained indicates that the differences observed are due to experimental error and the values are not distinguishable from the mean. The explanation for the constant mass recovery is to be found in the fact that, although the residence time is increased with decreased tonnage, the mass of heavy minerals, including gold, is reduced proportionately.

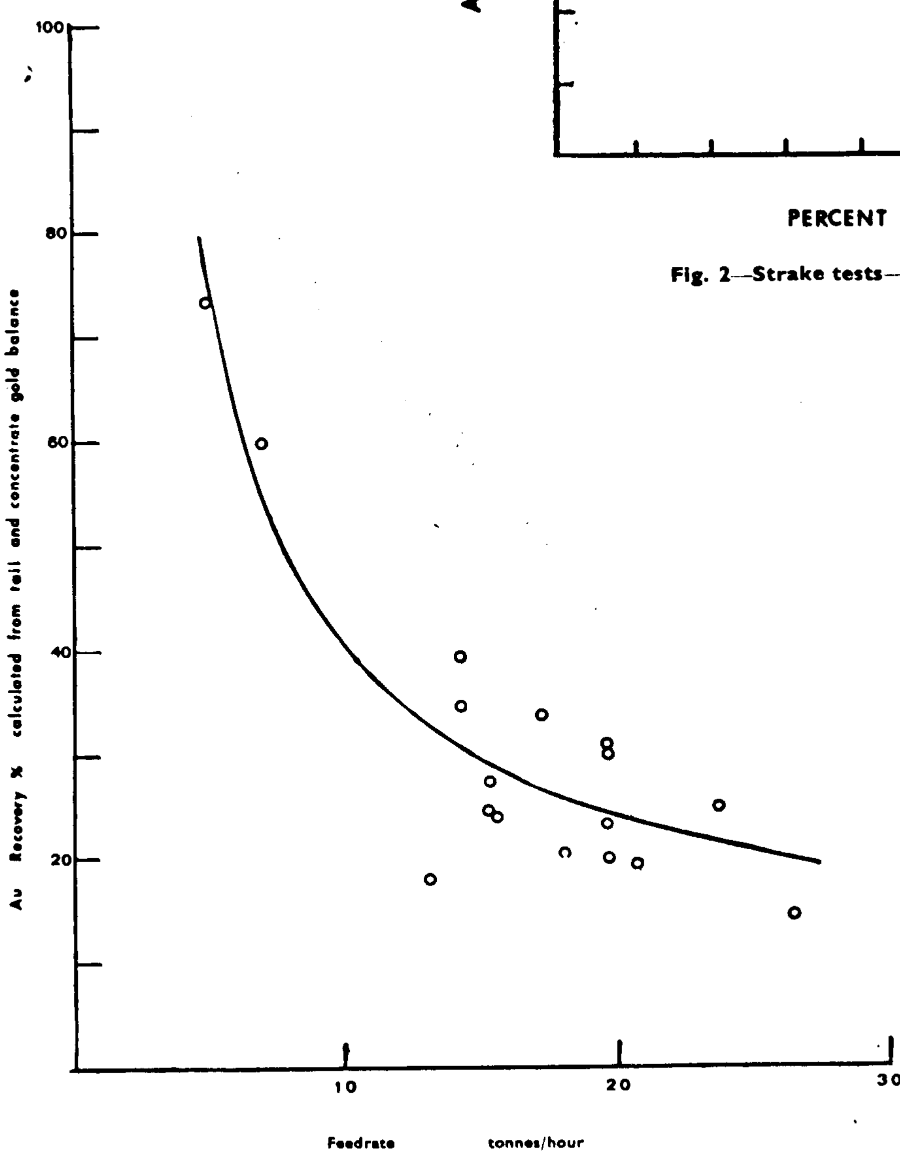

Fig. 3—Johnson drum concentrator—recovery versus feedrate

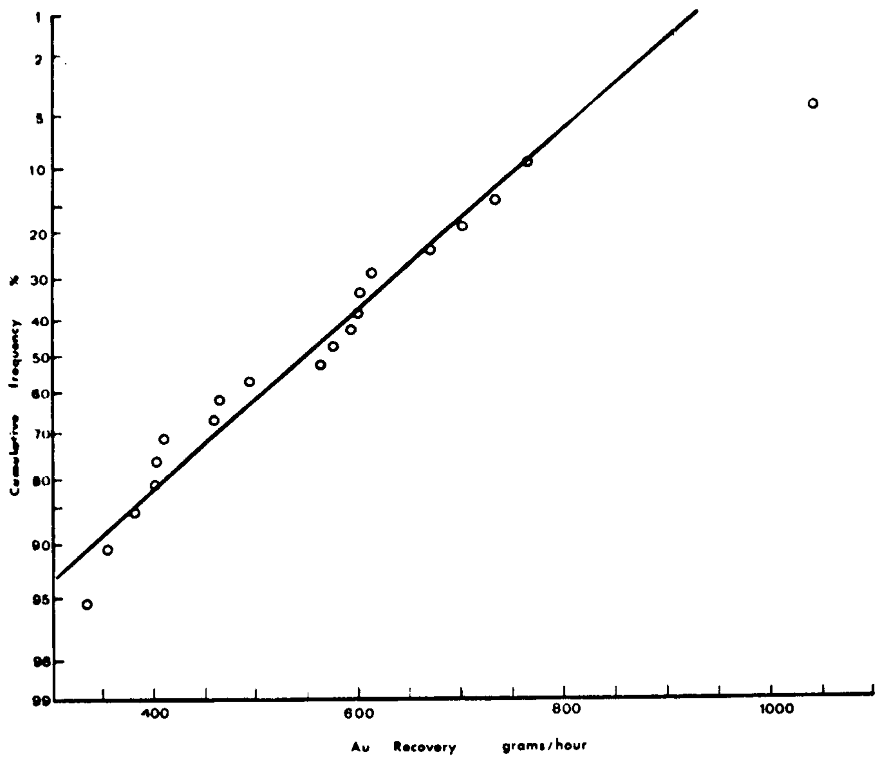

Fig. 4—Johnson drum concentrators—Gold recovery versus cumulative frequency, showing the recovery to be normally distributed

The constant mass recovery of gold per unit time is also a feature of the high-grade drums, where about 1 kg of gold was recovered per hour as compared with the 0,55 kg of gold per hour on the low-grade drums. This is attributed to the much higher heavy-mineral content of the high-grade circuit, the mass of gold being assumed negligible in parts per million.

The Johnson drum concentrator is a cheap, easily run machine. Although it is not very efficient as a unit, it nonetheless, because of a large recirculating load, recovers up to 70 per cent of the gold entering the plant. Screen analyses showed the recovery of gold to be poorest in the minus 43 μm fraction and best for the coarse sizes. This is in common with all gravity-concentration machines and is due to the lower settling rates of the fine particles.

TESTWORK ON THE ENDLESS-BELT CONCENTRATORS

Both plant[7] and laboratory results have been obtained for these concentrators, the former for two machines in routine service and the latter for a small-scale belt concentrator specially made for the purpose.

Plant machines

The two plant machines are each fed by the combined concentrates of three high-grade and three low-grade Johnson drum concentrators. The results represent day-shift samples taken over a period of six months and are hence inherently more reliable than those obtained in the laboratory. The average gold recovery was 63 per cent, iron recovery (including tramp iron) was 29 per cent, and uranium recovery was 9 per cent. The concentrate mass was 3,8 per cent.

The uranium recovery is considered to be of interest since some of this uranium comes from the low-grade circuit via the Johnson drum concentrators, and an improvement in this could be of considerable economic benefit. Doubling the speed of one of the belts resulted in an overall increase in total gold recovery by amalgamation of 2,25 per cent, but at the expense of in-

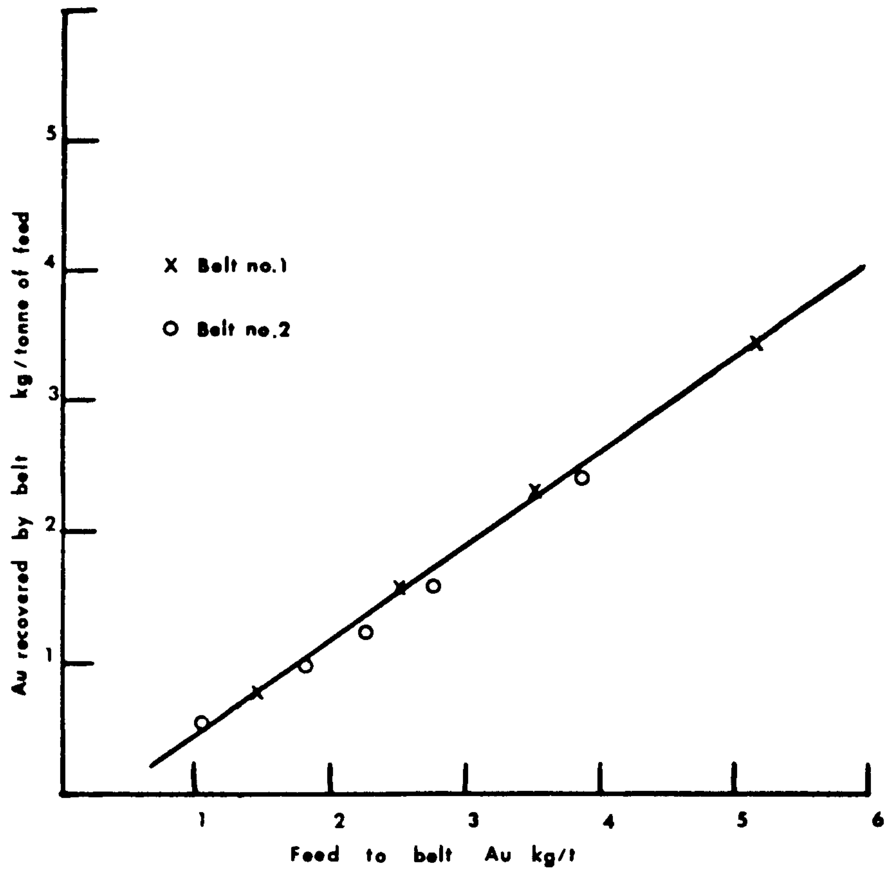

Fig. 5—Endless-belt concentrators—recovery as a function of feed grade

creasing the concentrate mass by nearly 50 per cent.

The effect of doubling the speed of both belts would be to increase the total gold recovery by amalgamation to 41,5 per cent (an increase of 4,55 per cent), but such a move would double the concentrate mass, which would overload the existing amalgamation facilities.

When the feed grade to the belts is plotted against the mass of gold recovered per tonne of feed, a most interesting result is obtained. The recovery increases in direct proportion to the feed grade (Fig. 5), and a high feed grade therefore implies a high recovery. This is contrary to the situation on the Johnson drum, where a high feed grade automatically follows from a decreased tonnage (but a constant mass of gold), and the residence time on the belt increases.

Laboratory small-scale belt

This machine has a riffled-rubber belt 12,5 cm in width, with 60 cm of exposed surface, and was driven up a plane inclined at 15 degrees to the horizontal. The feed was presented about 10 cm down the belt from the head pulley, and water flowed onto the belt in a sheet immediately behind the feed point to wash gangue out of the concentrate.

Tests were run under various conditions of wash-water flowrate and feed pulp density. A constant volume of pulp was delivered so that the solids feedrate was a function of the pulp density. The concentrate is expressed as a concentration ratio to suppress the effect of feed grade. Fig. 6 shows logarithmic plots of concentration ratio against the percentage mass of concentrate and percentage recovery, the latter passing through the Western Deep Levels figure of 63 per cent recovery at a concentration ratio of 17. A computer analysis of the results (Table II) showed the concentration ratio to be a function of the feedrate and the wash-water flowrate.

Conclusion

From the combined results of the plant and the laboratory, it seems that, given a lower feedrate at a higher grade, it would be possible, by adding wash-water to the belt

TABLE I
JOHNSON CYLINDRICAL CONCENTRATORS
FEED SOURCE: TERTIARY CYCLONE UNDERFLOW

	VAAL REEFS		WESTERN DEEP LEVELS		PRESIDENT STEYN	WESTERN HOLDINGS	WELKOM GOLD MINE		FREE STATE GEDULD	PRESIDENT BRAND	
Size (length × diameter, m)	3,658 × 0,762		3,658 × 0,762		3,658 × 0,762	3,658 × 0,762	3,658 × 0,762		3,658 × 0,610	3,658 × 0,762	
Number in use	4 HG	6 LG	6 HG	6 LG	8	12	5 HG	5 LG	12	10 HG	6 LG
Slope	4°		2,5°		5°	2° to 2,5°	2°		5°	2,5°	
Speed, min/rev	8,17		3,9		5,47	8 at 5,4 4 at 8,5	7,5		11,2	7,85	
mm/min	293		614		438	443 283	319		171	305	
Solids feedrate, t/h	HG 42,5	LG 39,0	HG 6,947	LG 19,713	14,9	9,36	HG 9,072	LG 11,527	5	HG 11,5	LG 21,7
Moisture in feed, %	39	49	49	43	50	69	56,6	44,3	73	73	49
Feed gradings +300	15,3	15,1	5,4	15,6	19,4	7,6	6,0	8,2	18,4	2,0	6,7
+150	24,7	35,0	22,3	39,9	33,2	28,4	29,6	31,8	33,6	26,2	34,3
+74	33,9	34,1	45,8	30,2	33,0	33,3	46,6	43,6	34,2	37,7	37,6
−74	26,1	15,8	26,5	14,3	14,4	30,7	17,8	16,4	13,8	34,1	21,4
Head value, g/t	207,5	46,5	213,2	126,6	93,3	205	129,5	57,8	532,0	681,2	93,3
Tailings value, g/t	96,0	10,1	177,4	111,3	59,1	169	85,7	33,4	303,0	376,4	56,0
Concentrate, %	0,271		1,79		4,0	1,9	1,369	1,077	1,5	8,696	6,15
Recovery, %	53,7	78,3	34,0		36,67	17,6	33,8	42,2	43,0	44,7	40,0
Concentrate value, g/t	1 848,0		2 838		842,9	5 815	1 329,3	786,5	22 195,0	5 172,5	547,4
Reconcentration by	Riffled-belt concentrator		Riffled-belt concentrator		Riffled-belt concentrator	Riffled-belt concentrator	Riffled-belt concentrator		Riffled-belt concentrator	Riffled-belt concentrator	

HG—High Grade LG—Low Grade

TABLE I
RIFFLED BELT CONCENTRATORS
FEED SOURCE: CONCENTRATES FROM JOHNSON CONCENTRATORS

	VAAL REEFS	WESTERN DEEP LEVELS	PRESIDENT STEYN	WESTERN HOLDINGS	WELKOM GOLD MINE		FREE STATE GEDULD	PRESIDENT BRAND	
Size (length × width conc. areas, m)	3,048 × 1,524	3,086 × 1,524	3,048 × 1,524	3,048 × 1,524	3,048 × 1,524		3,048 × 1,524	3,353 × 1,524	
					HG	LG		HG	LG
Number in use	2	2	2	3	1	1	3	2	1
Slope	10°	10°	10°	10°	5°	7,5°	10°	10°	10°
Speed, mm/min	152	191	(1) 578 (2) 562	(1) 152 (2) 406	430	285	356	254	254
Solids feedrate, t/h	0,548	1,439	2,384	0,72	0,621	0,621	0,3	5	8
Moisture in feed, %	94,8	95,8	47,0	94	94,1	94,4	96	95	91
Feed gradings +300	6,4	7,4	3,0	12,4	7,6	12,6	7,2	6,0	5,3
−150	11,7	24,1	47,5	10,9	22,8	27,2	19,3	16,8	24,5
+74	50,4	47,1	43,0	48,8	47,2	43,9	45,6	44,5	45,0
−74	31,5	21,4	6,5	27,9	22,4	16,3	27,9	32,7	24,2
Head value, g/t	1 848,0	2 838	842,9	5 815	1 329,3	786,5	22 195,0	5 172,5	547,4
Tailings value, g/t	343,9	1 069	248,8	1 362	309,2	213,6	1 009,0	2 251,9	90,2
Concentrate, %	4,865	3,77	3,33	8,7	6,32		13,3	0,7	0,677
Recovery, %	81,39	63,2	70,48	76,6	76,7	72,8	95,5	56,5	83,5
Concentrate value, g/t			89 590	16 670	13 556		63 800	32 963	16 269
Total gold recovery after amalgamation, %	20,7	36,9	42,5	33,6	30,9		51,5	29,2	

behind the feed point to increase the concentration ratio without losing on recovery.

THE PILOT PLANT

The varied operation of gravity-concentration plants and their absence on some of the Group's mines prompted us to design a portable pilot plant. This plant consists of four modules: the first contains devices for measuring the incoming feed and presenting it to the other units; the second holds a sieve bend and facilities for various flow schemes to include or bypass the cyclones on the third module, which also holds a Johnson drum concentrator; and the drum concentrate is fed to a half-width endless-belt concentrator on the fourth module. This belt is of novel design in as much as it can be vibrated at 30 Hz in two directions. A flowsheet of the plant is shown in Fig. 7. All streams are capable of being sampled automatically, and the major flows can be diverted into a tonnage box on the second module. The cyclones on the third module will also act as a test-bed for cyclones of novel design (to be described later). This plant is due to be commissioned at Vaal Reefs North in February 1973.

While the design work was in progress on the first pilot plant, we carried out laboratory gravity-concentration testwork on a Wilfley table. Table III shows the results of Reichert-cone simulations carried out on the Wilfley table, as well as the tabling of some belt-concentrator tailings. These results proved so encouraging that we have ordered a Reichert-cone, which is to be installed in a second portable pilot plant that will be commissioned at Vaal Reefs South. Fig. 8 shows the flowsheet split into its modules. The cone concentrates will be fed to a triple-deck shaking table, and the middlings and, possibly, the lower-grade concentrate will be recycled.

CYCLONES AS CONCENTRATORS

Reference to Figs. 1a and 1b shows that the function of the tertiary cyclones in typical Anglo American circuits is that of concentrators rather than classifiers. This is particularly true of the Western Deep

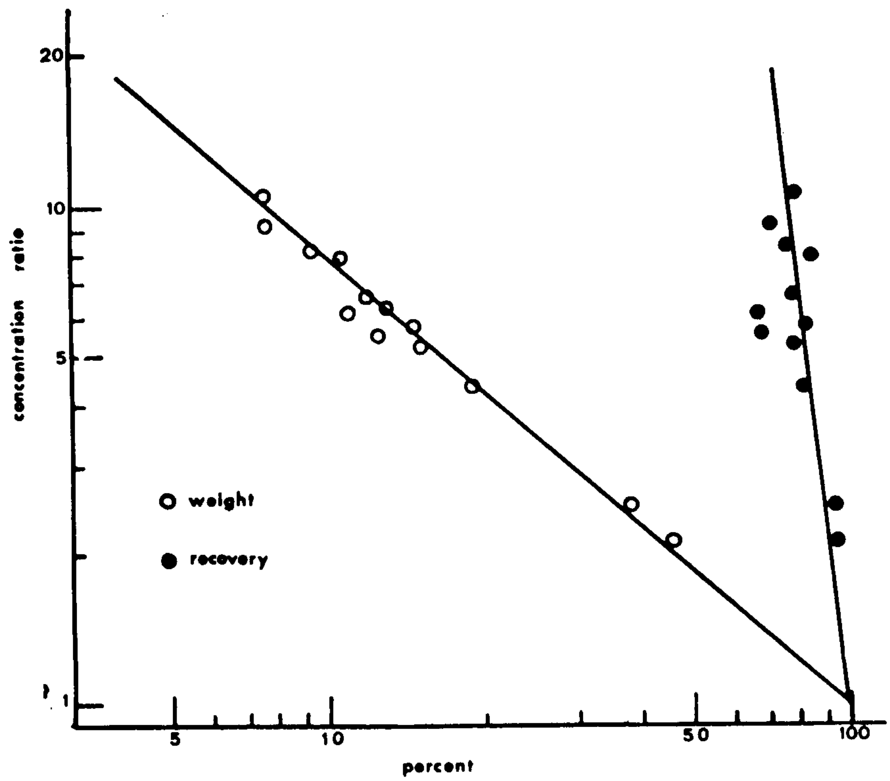

Fig. 6—Endless-belt concentrator—recovery and mass fraction of concentrate as a function of concentration ratio

TABLE II

RESULTS OF TESTWORK ON THE LABORATORY BELT CONCENTRATOR

Test no.	Solids feed %	Wash-water flowrate l/min	Feed-rate relative units	Belt speed cm/min	Conc. ratio	Mass %	Re-covery %	Calc. head Au g/t
1	23	1,8	4,31	40	8,2	9,1	74,4	365,0
2	23	1,0	4,31	40	2,5	37,7	92,5	513,7
3	30	1,6	5,91	40	10,3	7,4	76,5	708,8
4	29	1,6	5,67	40	7,9	10,5	82,8	858,9
5	35	1,6	7,18	40	9,3	7,5	70,0	362,4
6	6	0	2,00	40	2,1	45,4	93,5	1311,7
7	35	1,1	7,18	40	6,6	11,7	76,8	365,3
8	25	1,4	4,75	40	6,2	12,7	79,3	533,7
9	15	1,6	2,66	40	5,5	12,2	67,4	204,5
10	6	0	2,00	40	4,3	18,8	81,1	1149,2
11	20	1,6	3,67	40	6,1	10,9	66,1	207,1
12	30	1,4	5,91	80	5,2	14,9	77,9	247,4
13	25	1,8	4,75	80	5,7	14,4	81,8	282,5

$$\text{Concentration ratio} = \frac{\text{concentrate grade}}{\text{feed grade}}$$

Computer result of regression analysis:

Concentration ratio = 0,555 (feedrate) (wash-water flowrate) + 2,578

Standard error of concentration ratio = 1,489

Standard error of variable = 0,123

t-test = 4,50

Levels circuit, where there is no tertiary milling. In other circuits (Fig. 1b), the tertiary cyclones are also concentrators for the feed to the Johnson drums, but the Johnson drum tailings are dewatered in another cyclone before being milled in the tertiary mills, thus receiving some classification. It is not possible for a given cyclone to perform both functions simultaneously at maximum efficiency—an increase in classification efficiency automatically results in a decrease in concentration efficiency. This correlation between concentration inefficiency and classification efficiency is understandable when one considers that, for high concentration efficiency, the cyclone is expected to make a separation in which coarse, light particles report to the overflow, and fine, heavy particles (gold-pyrite) report to the spigot. This is contrary to the normal tendency of the cyclone to act basically as a classifier. This point will be discussed in more detail later.

If the above is accepted, a decision must be made on which of the two functions—classification or concentration—is the more important for the tertiary cyclones. Since the tertiary overflow is reclassified in the secondary cyclones, low classifying efficiency in the former is not serious. Any coarse, light gangue, (possibly containing unliberated gold) that may overflow the tertiaries will be returned to the secondary mills via the secondary cyclone underflow. It would seem reasonable therefore, to aim at increased concentration efficiency in the tertiary cyclones at the expense of classification efficiency. The aim should be maximum recovery of gold in the cyclone underflow at maximum grade. Obviously, once this is accepted, it would theoretically be possible to replace the cyclones with any other more efficient concentrator. However, the cyclones offer so many advantages from a maintenance and operating viewpoint that it seems logical to retain them, and to attempt to improve their concentrating efficiency. As far as is known, no serious attempt has been made to improve the operation of the tertiary cyclones *as concentrators*, and th present investigations were motivated by the considerable scope for improvement in this section of the plants.

Recent years have seen the development of various cyclones specifically for concentrating purposes. Their physical design is such that classification effects are suppressed and the influence of particle specific gravity is maximized. These are not heavy-medium cyclones, which have been in use in the mineral-processing industry for some years, but true hydrocyclones. They are characterized by blunt cones (i.e., large included apex angles) in the range 60 to 180 degrees and long, large-diameter vortex finders. These developments are largely a result of work in the coal industry[8-12], where cyclones operating with a water medium are now in wide use for the upgrading of fine coal. Investigations have also been carried out into the use of cyclones for the beneficiation of cassiterite[13, 14] and iron ores, and a recent Russian paper[15] describes the concentration of gold from milled conglomerate ores by use of a short-cone hydrocyclone.

As mentioned above, classification cyclones are of slender design, with a narrow cone angle usually 15 to 30 degrees. In these cyclones, the particles must discharge either through the vortex finder or the apex orifice, and the basic requirement for separation is the achievement of balance between the centrifugal forces accelerating the particles radially outward, and the centripetal drag forces. For this purpose, relatively large vortex-finder clearances are common in classification cyclones, because this assists in providing the time for particles in the central flow region to reach their terminal free-settling velocity relative to the entraining water.

In contrast, concentrating cyclones are of squat design, with a wide cone angle—in the range 60 to 180 degrees and are operated to suppress classification phenomena in favour of gravity-concentration effects. Briefly, they are based on two hypotheses[10]: (a) a particle bed stratifies according to density along the conical wall of the lower cyclone section, and (b) particles entering the central spiral flow separate during the initial phase of their acceleration in a radial direction. The former hypothesis has led to the selection of wide-cone angles for concentrators, because particles stratify more reliably along the shallower wall of a wide cone than would be possible along the steep wall of a slender cone. The latter hypothesis has led to the preference for relatively short vortex-finder clearances in concentrators, because this design assists in separating particles that swirl towards the vortex-finder before the centrifugal mass and the centripetal drag forces acting on these particles attain equilibrium. Since it is the drag force that increases the relative influence of particle size on separation, this force is to be suppressed in the operation of cyclones as concentrators.

The Compound Water Cyclone

The Compound Water Cyclone (CWC) was invented by J. Visman and developed by him at the Mines

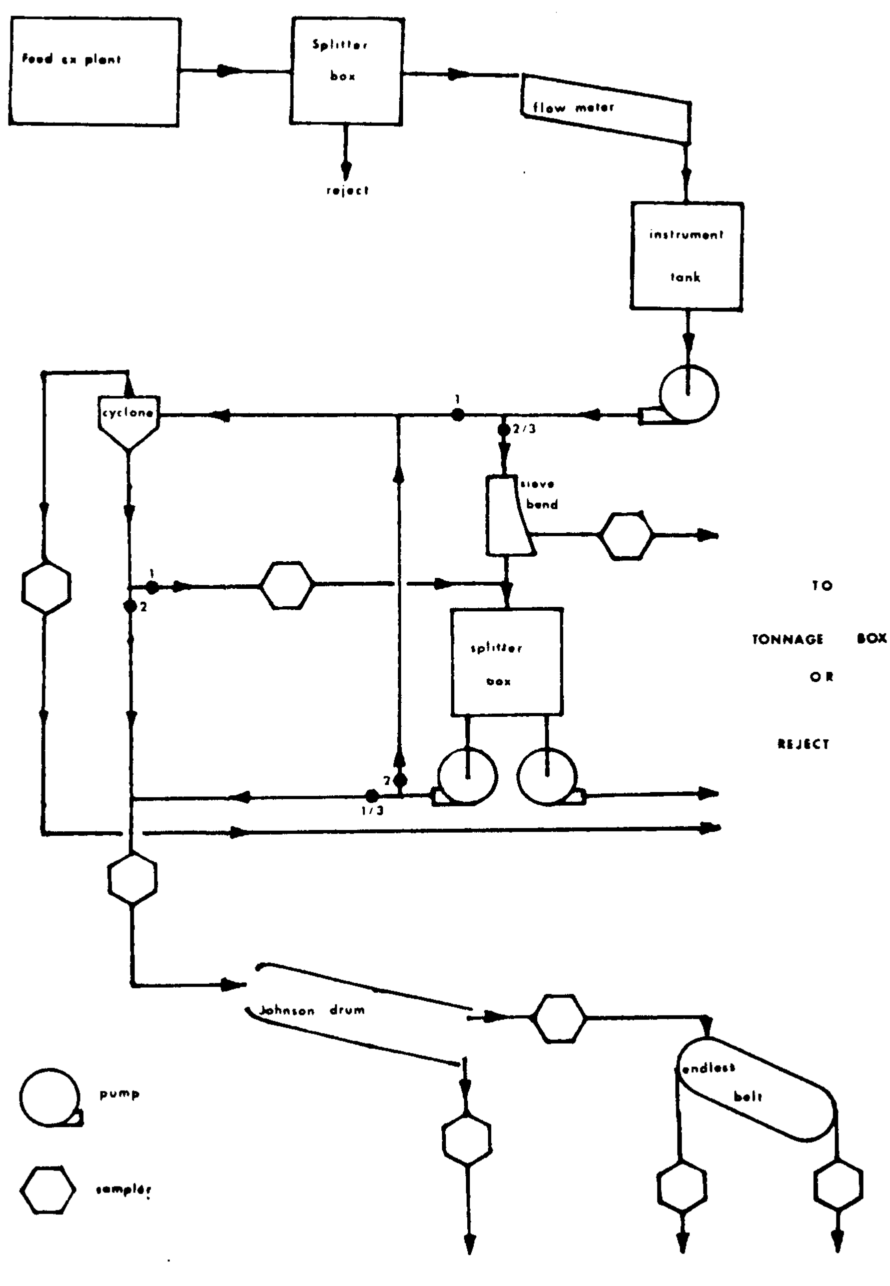

Fig. 7—Pilot plant no. I

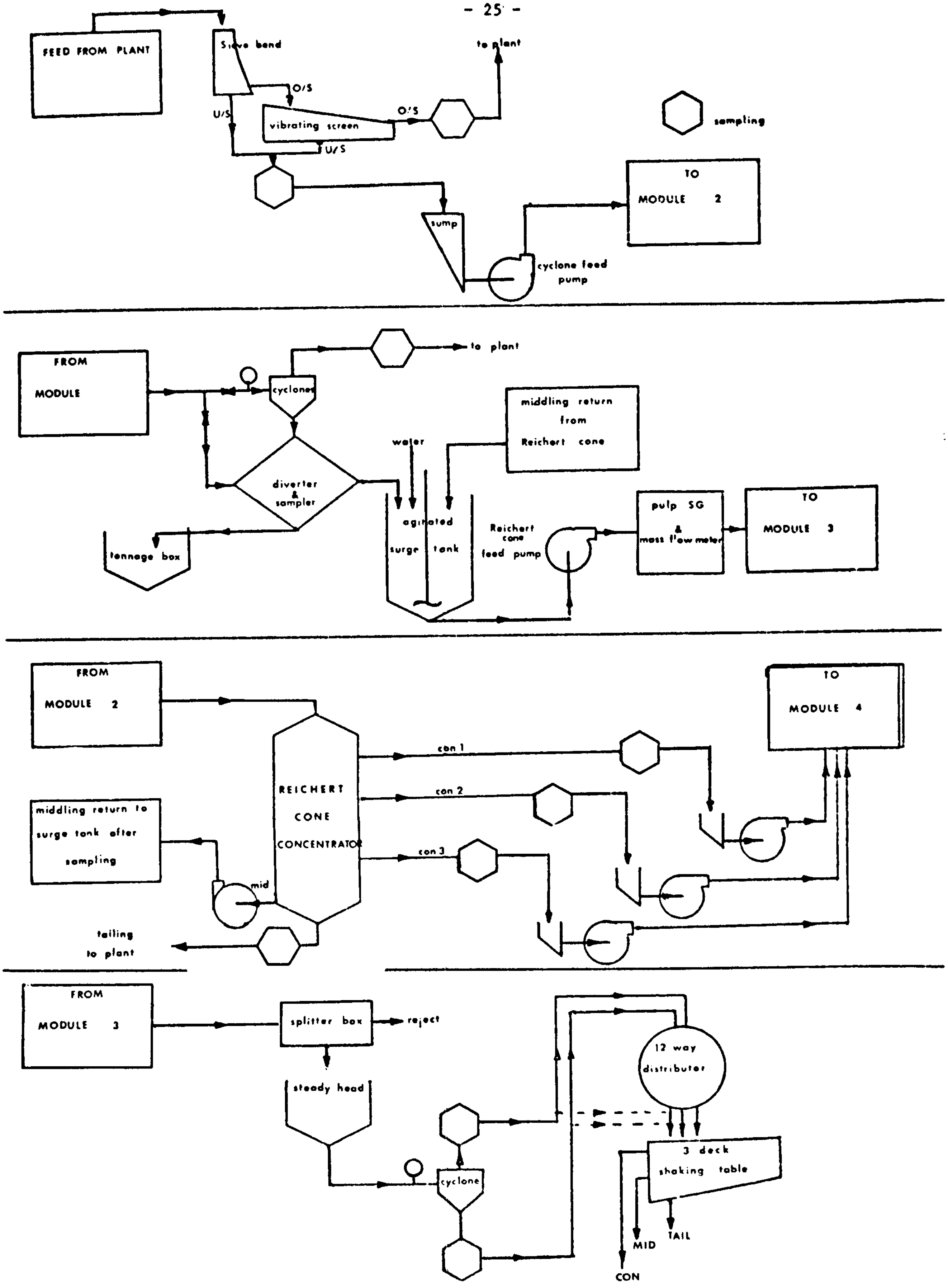

Fig. 8—Pilot plant no. 2

TABLE III
REICHERT-CONE SIMULATION AND OTHER TESTWORK

Ore/Test	Fraction	Mass %	Analysis Au, g/t	Analysis S, %	Distribution % Au	Distribution % S
Vaal Reefs West 2 mill discharge high-grade Reichert-cone simulation	Conc.	10,2	467,6	37,4	88,4	92,6
	Tailing	77,7	6,8	0,18	9,8	3,4
	Slimes	12,1	8,25	1,36	1,8	4,0
	Calc. head	100,0	54,0	4,12	100,0	100,0
Vaal Reefs West 2 mill discharge low-grade Reichert-cone simulation	Conc.	10,0	442,0	39,1	88,7	89,4
	Tailing	68,4	5,9	0,25	8,1	3,9
	Slimes	21,6	7,45	1,35	3,2	6,7
	Calc. head	100,0	49,8	4,37	100,0	100,0
Western Deep Levels 2" mill discharge high-grade Reichert-cone simulation	Conc.	4,7	581,3	20,9	66,9	88,8
	Tailing	87,2	14,35	0,1	30,6	7,9
	Slimes	8,1	12,65	0,45	2,5	3,3
	Calc. head	100,0	41,00	1,10	100,0	100,0
Western Deep Levels 2" mill discharge low-grade Reichert-cone simulation	Conc.	3,8	409,3	23,5	64,9	89,6
	Tailing	89,4	9,0	0,09	33,4	8,1
	Slimes	6,8	7,1	0,34	2,0	2,3
	Calc. head	100,0	24,1	1,00	100,0	100,0
Western Deep Levels belt-concentrator tailing one pass on table	Conc.	0,9	30 198,6		66,2	
	Middlings	7,6	1 419,25		26,3	
	Tailing	91,5	33,8		7,5	
	Calc. head	100,0	410,6		100,0	
	Conc. & mid.	8,5	4 469,9		92,5	

Branch, Department of Mines and Technical Surveys, Canada[10, 16]. It consists of a rather short cylinder, fitted with one of three alternative compound cones (Fig. 9). Each compound cone has three sections, with included angles of 120, 75, and 20 degrees. In each case, the widest angle is nearest the cylinder and the smallest at the spigot, but the three types differ in the relative areas of their inner cone surfaces. The inlet is conventional, but the vortex finder is rather larger and longer than that of an ordinary classifying cyclone.

Visman[16] explains the concentrating action taking place in his cyclone as follows (see Fig. 10).

Particles of different sizes and specific gravity form a hindered settling bed in Section I of the compound cone. Light, coarse particles are prevented from penetrating the lower strata of this bed by the coarse, heavy fractions and by the fine particles filling the interstices of the bed. Consequently, the water passing from the periphery of the cyclone chamber towards its main outlet (the vortex finder) erodes the top of the stratified bed and substantially removes the light, coarse particles via the "central current" around the air core.

The remainder of the bed is forced into the second conical section (II) by new feed entering the cyclone, substantially without losing its stratified character. Here, the central current is much stronger and erodes the top of the bed, where the middlings are now exposed. The light middlings are swept up and discharged through the vortex finder. The heavy middlings that spiral upward in the central current may bypass the orifice of the lower vortex finder owing to their higher specific gravity. Consequently, the coarse heavy-middlings fraction tends to recirculate to the stratified bed and finally enters the third conical section. In this last section (III), the bed is finally destroyed as coarse particles fan out along the cyclone wall in a single layer, exposing the small particles that so far have been protected from being washed out. The central current in Section III is relatively weak, as it has nearly spent itself in the previous sections. The upward current that remains separates the small particles from the remainder of the material, with preference for those of low specific gravity. Thus the fine, light particles are finally discharged through the vortex finder by a process of elutriation. The heavy particles, fine as well as coarse, are discharged through the apex. The separation thus takes place in three steps, as shown diagrammatically in Fig. 10.

TESTWORK AND RESULTS

Tests were carried out using both the CWC 2M (50 mm) unit and the CWC4 (100 mm) unit. The test cyclone was fed from a centrifugal pump, and the overflow and underflow were returned direct to the pump feed tank. Sampling was carried out by cutting both product streams simultaneously, generally for a period of 5 seconds. The cyclone was operated in a vertical position for all tests.

All tests were performed on a slurry of Western Deep Levels secondary mill discharge, and the operating conditions included variations in feed pulp density, feed pressure, cone type, and vortex finder clearance.

The results are presented in Table IV, which shows the experimental conditions, gold assays of overflow and underflow, and the distribution of gold in these products. In addition, the table includes calculated values of the Gaudin "Selectivity Index" (S.I.)[17]. This is defined as "the geometric mean of the relative recoveries and relative rejections of two minerals, metals, or groups of minerals or metals"

$$\text{i.e.,}\quad \text{S.I.} = \sqrt{\frac{R_a J_b}{R_b J_a}}\,.$$

In the present case,

R_a = recovery of gold in underflow,
R_b = recovery of gangue in underflow,
J_a rejection of gangue to overflow,
J_b = rejection of gold to overflow,
all values being percentages.

The index is used here as a convenient way of comparing the results of tests under various conditions.

In Fig. 11, the value of S.I. is plotted against inlet pressure for the three different cone types CWC 2 unit. The curves all pass through a maximum at about 0,6 kg/cm², and the value of the S.I. increases with cone type in the order $M < L < S$. The value 4,0—the highest observed in any of the tests—is mainly due to the very low underflow mass of 3,6 per cent.

In Fig. 12, similar data have been plotted for the CWC 4 (100 mm) cyclone, with the extra variable "vortex finder clearance" included (see Fig. 10). It appears that the pressures employed were not high enough to show the maximum observed in tests on the CWC 2M cyclone, and which presumably exists for the larger unit. However, the L-type cone again gives higher values of the S.I. than does the M type. This is true regardless of the inlet pressure. An increase in vortex-finder clearance also increases the value of S.I.

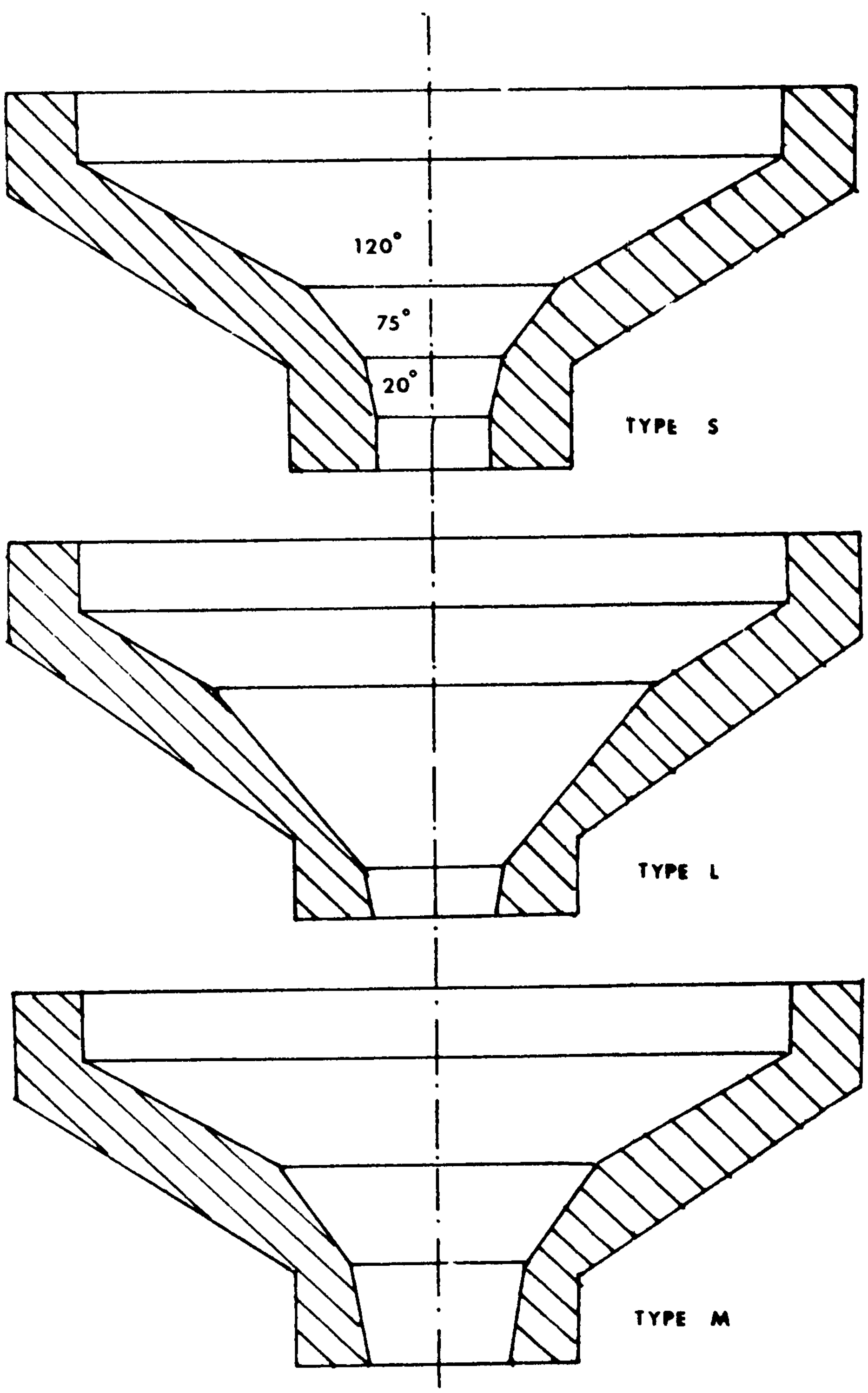

Fig. 9—The types of cone used in the Compound Water Cyclone

Fig. 13 shows that, in the CWC 2M (50 mm) cyclone, the value of S.I. decreases as the vortex-finder clearance is increased, at pressures above 0,5 kg/cm². Below this pressure, the value of S.I. at first increases and then decreases as the vortex-finder clearance is decreased.

Screen analyses and gold assays of the fractions were carried out on the products from selected tests. The results are shown in Table V, together with the gold recovery in each size range. It is apparent that recovery is lowest in the finest (minus 44 μm) fraction, and, since the major proportion of the gold exists in this size range, it is this recovery that determines the overall gold recovery to the spigot product.

Discussion

Although the Selectivity Index is a useful criterion for comparison of the concentrating efficiencies of various machines and experimental procedures, it is necessary to look at the results of the present tests rather more critically, bearing in mind the potential applications to existing milling and concentrating circuits. A high S.I. value can result from a low mass of underflow at high grade and low recovery, or, alternatively, from high recovery in a large underflow mass at low grade. Neither of these conditions is ideal from the viewpoint of the present investigation, the highest values of both grade and recovery being desirable.

Examination of the results shows that gold recovery to the underflow of 80 per cent is easily achieved with either cyclone and under a variety of operating conditions. If this is taken as the minimum acceptable recovery, the "best" result is that which yields this recovery in the lowest underflow mass. In three tests with the CWC 2M unit, this recovery was achieved in underflow masses of 37 per cent. With the CWC 4 unit, recoveries in excess of 80 per cent were obtained in underflow masses of 45 per cent. The L-type cone was used in the tests on the CWC 2 unit, and the M type in the tests on the CWC 4 unit. However, a test on the

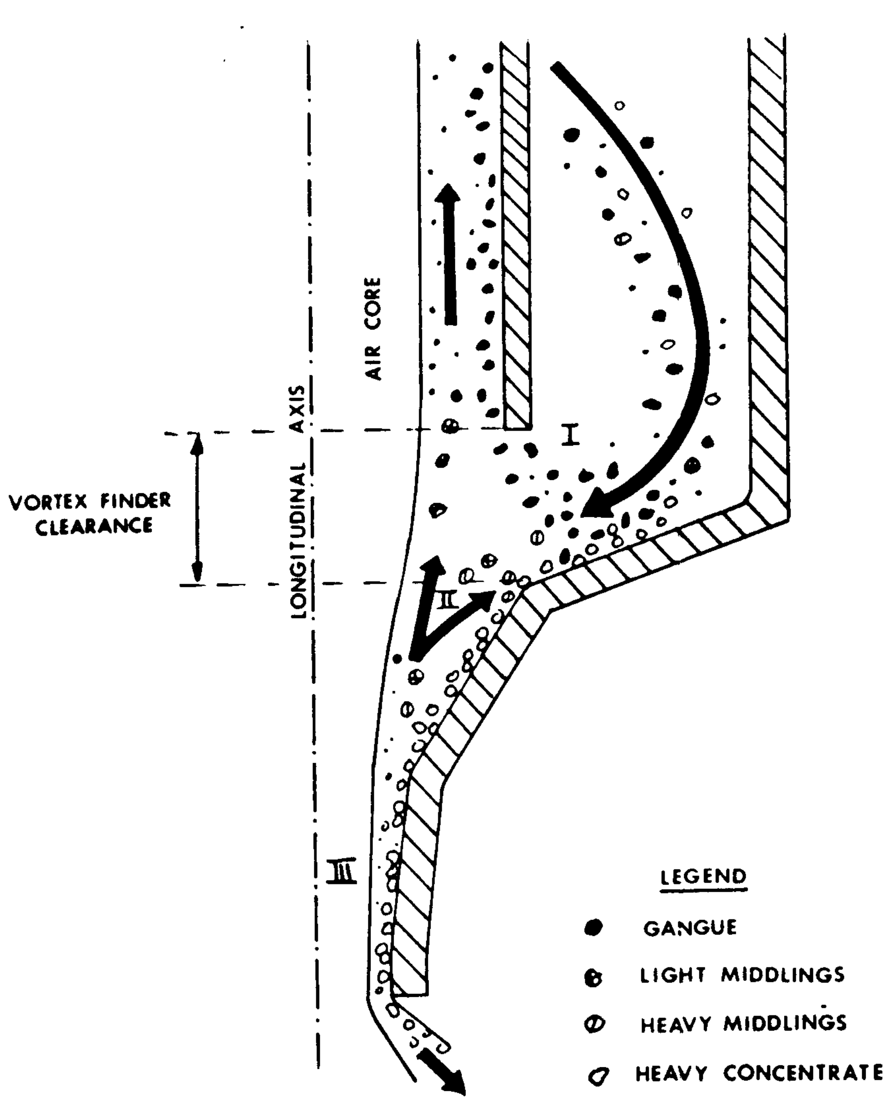

Fig. 10—Separating process in a Compound Water Cyclone (after Visman)

CWC 4 unit with the L-type cone gave a recovery of 78 per cent in an underflow mass of 34 per cent.

The recoveries achieved appear to be higher than those obtained in the tertiary cyclones in Anglo American plants. However, data on the latter are rather scanty, lacking tonnage measurements in particular. In addition, the test cyclones are small units, operated under ideal conditions at low feed pulp densities. Further laboratory tests that could be carried out on these units include investigation of the effect of feed pulp density and the determination of their d_{50} point[18] for gold, which was precluded in the present investigation because of lack of size data below 44 μm. Furthermore, these cyclones are at a disadvantage under typical gold-plant conditions, because the small size dictates an unrealistic level of operator attention. Therefore, despite the incomplete nature of the laboratory investigation, it is felt that further tests should be carried out on CWC 8 (200 mm) and/or CWC 12 (300 mm) cyclones under plant or pilot-plant conditions. The gravity-concentration pilot plant at present being commissioned provides ideal conditions for these tests.

A blunt-cone cyclone is already operating at Western Holdings Limited with apparent success. The unit was designed by the Anglo American Research Laboratories and is shown in Fig. 14. It has a cone included

TABLE IV.

SELECTED RESULTS FROM CONCENTRATION TESTWORK USING COMPOUND WATER CYCLONES

Test no.	Cyclone diam.	Cone type	Vortex finder clearance mm	FEED Pressure kg/cm²	FEED Solids %	FEED Solids g/l pulp	FEED Au Calc. g/t	UNDERFLOW Solids %	UNDERFLOW Solids g/l pulp	UNDERFLOW Mass %	UNDERFLOW Conc. Ratio	UNDERFLOW Recovery Au %	UNDERFLOW S.I.
1	50	M	13	0,60	25	297	108,1	62,0	1017	17,4	3,39	59,0	2,6
2	50	M	13	0,35	25	297	108,6	60,4	975	23,4	2,68	62,8	2,3
3	50	M	38	0,15	25	297	81,3	44,6	620	54,8	1,63	83,1	2,0
4	50	S	13	0,40	20	230	99,2	54,5	831	3,6	10,20	36,7	4,0
5	50	S	13	0,20	20	230	98,1	44,7	622	17,4	2,95	51,3	2,2
6	50	S	38	0,25	30	370	133,5	63,1	1048	25,5	2,53	64,6	2,3
7	50	L	13	0,05	20	230	71,1	26,9	324	57,1	1,37	78,4	1,6
8	50	L	13	0,60	30	370	90,9	62,1	1018	36,6	2,26	82,6	2,9
9	50	L	38	0,20	30	370	133,0	45,8	644	57,0	1,51	86,3	2,2
10	100	M	35	1,30	12	130	90,5	43,5	599	26,2	1,95	51,1	1,7
11	100	M	55	1,30	12	130	87,3	61,9	1015	37,5	2,05	76,4	2,4
12	100	M	90	1,30	12	130	98,8	64,9	1098	45,7	1,85	85,3	2,6
13	100	L	35	1,05	12	130	91,1	61,7	1011	27,4	2,31	63,3	2,1
14	100	L	55	1,05	12	130	105,6	61,7	1011	33,5	2,33	78,0	2,6

$$\text{Concentration ratio} = \frac{\text{underflow grade}}{\text{calc. feed grade}}$$

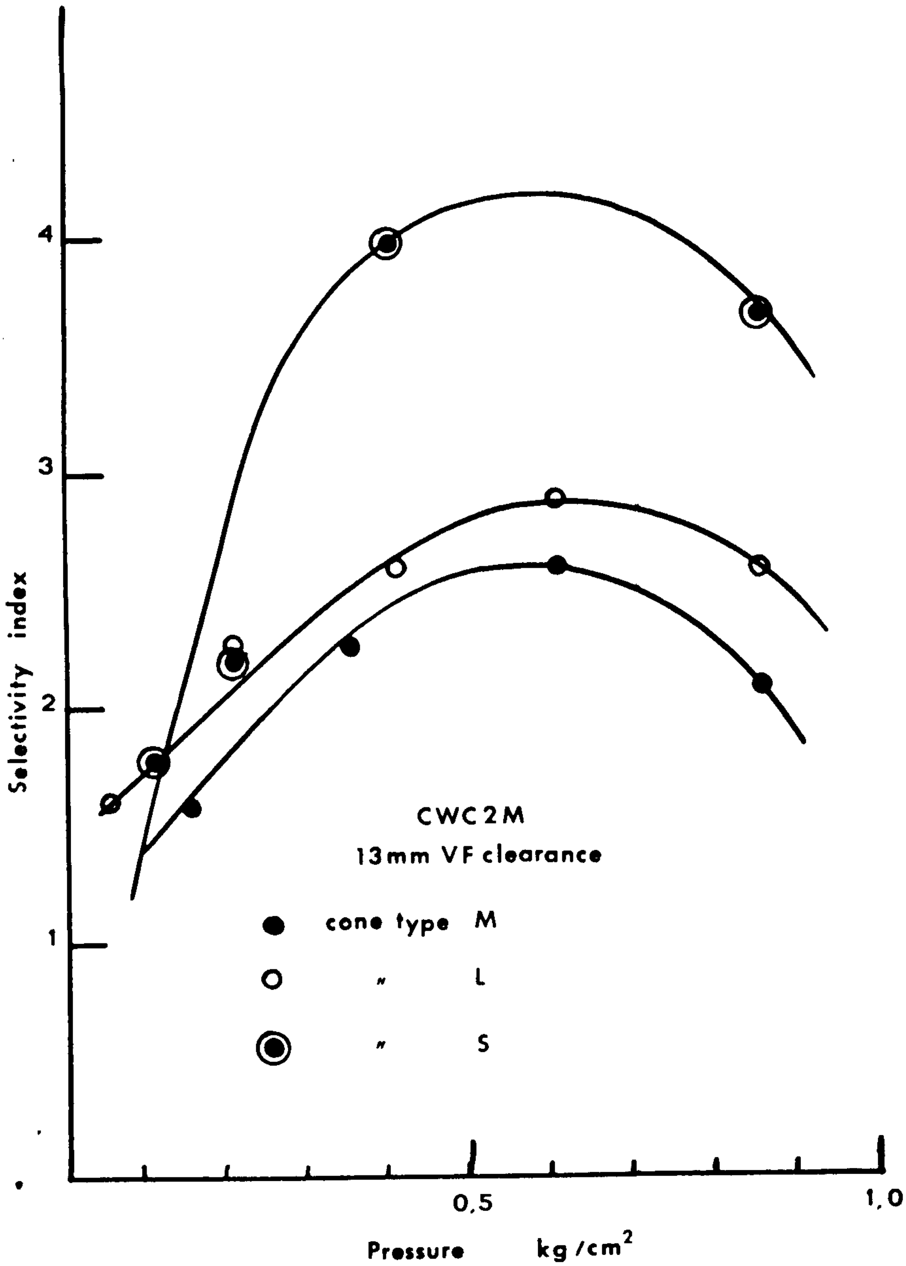

Fig. 11—Selectivity Index versus inlet pressure for the Compound Water Cyclone 50 mm unit

angle of 120 degrees and is fed with tertiary cyclone underflow, thus replacing a Johnson drum concentrator. The inlet pressure is of the order of 0,2 kg/cm². Recoveries of 70 per cent in some 30 per cent by mass are currently being obtained.

Close liaison is being kept with a firm in Germany that is experimenting in the field of cyclones used as concentrators. This firm makes cyclones from cast polyurethane, which has definite maintenance advantages. They are currently building concentrating cyclones with wide-angle cones for test purposes and achieving similar results to our own.

CONCLUSIONS

Any improvement in the overall recovery of gold by gravity concentration and subsequent amalgamation relies on the efficiency of each concentration stage in the removal of gold from the pulp. Since no machine can be 100 per cent efficient, one must aim for the highest unit efficiency in terms of the presented pulp, as well as the highest unit efficiency in terms of the plant as a whole. On those plants having both gravity concentration and cyanide circuits, the gravity-concentration circuit must complement the cyanide circuit in that the gold entering the cyanide circuit must be of such a nature as to respond with maximum efficiency to cyanidation. Feather and Koen[19] point out that the majority of the gold in the residues is free but is coated with iron oxides and has gangue minerals pressed into the surface. These grains are those which have bypassed the gravity-concentration circuit for one reason or another, presumably "floating" into the secondary-cyclone overflows after being flattened by repeated passes through the mill.

The essence, then, of efficient gravity concentration lies in selective classification of the gold to the gravity-concentration circuit, accompanied by a reduction of the recirculating load of gold in the mills. It is felt that blunt-cone cyclones will prove to be of considerable efficacy here, provided dilution of the mill circuit is not experienced. At the same time, considerably improved recovery of the gold by the concentrators is called for. This can be achieved in two ways: firstly, an increase in the concentrator capacity to relieve the overloaded conditions that exist (this is currently being done on most Anglo American plants); and, secondly, the installation of novel machines that are inherently more efficient than the existing concentrators, which are possibly obsolete in terms of modern practice. The work currently being undertaken, coupled with the vastly superior analytical tools available today (such as the electron microprobe and high-speed computers), should elucidate the problems involved and indicate the most practical method of attaining optimum and efficient operations. It is hoped that the two pilot plants will be very useful in optimizing the existing concentrators and improving the concentrator capacity. This applies especially to the Reichert-cone concentrator plant, which uses the principle of the pinched flume without introducing wall effects and overcomes the problem of choked riffles by removing the concentrates through a slot.

The ultimate result of this work will be to reduce the inventory of gold held in the mill circuit and to reduce cyanide losses by lessening the probability that gold will become coated while in the mill circuit.

ACKNOWLEDGEMENTS

We much appreciate the invaluable assistance and co-operation of the various mines whose operations we investigated, and of those companies who lent us equipment and did testwork on our behalf.

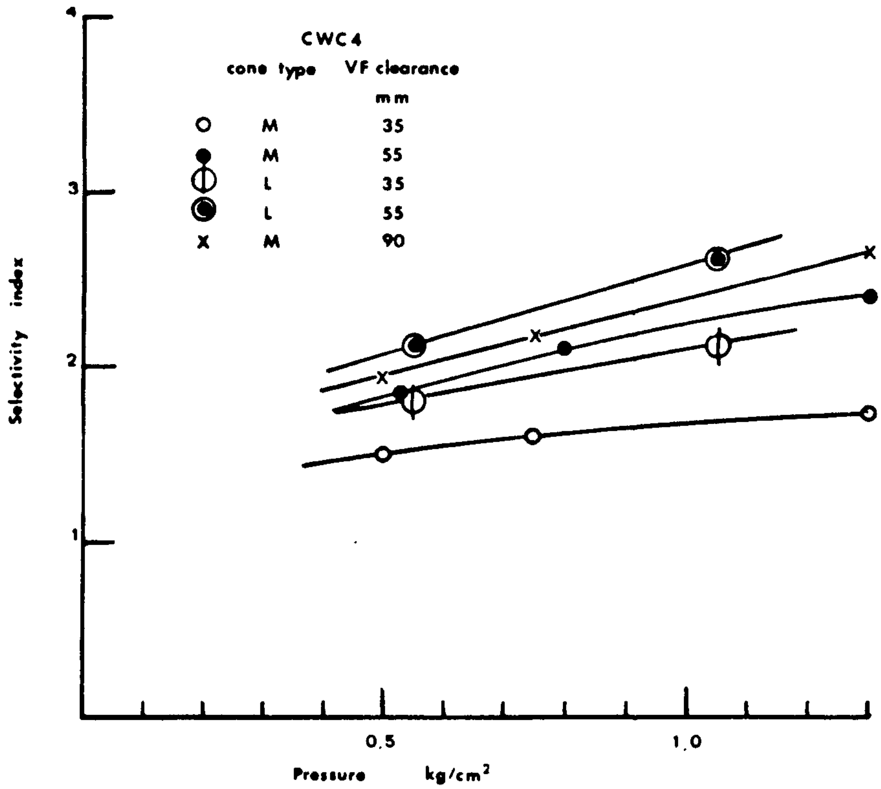

Fig. 12 Selectivity Index versus inlet pressure for the Compound Water Cyclone 100 mm unit

TABLE V

SCREEN ANALYSES AND GOLD DISTRIBUTION OF PRODUCTS FROM SELECTED TESTS

Test no.	Size fraction μm	Overflow			Underflow			Au recovery to u/f in size %
		Mass %	Au g/t	Au distrn. %	Mass %	Au g/t	Au distrn. %	
6	297	2,9			3,7	120,60	3,5	92
	210		22,64	0,5	4,1	67,60	2,1	
	149	10,4	7,84	0,6	6,5	52,64	2,7	82
	105	15,6	9,52	1,2	4,4	98,92	3,4	74
	74	12,0	11,88	1,1	2,9	178,20	4,0	78
	53	12,0	18,06	1,7	1,9	325,56	4,8	74
	44	5,6	41,52	1,8	0,9	834,80	5,8	76
	−44	16,0	254,86	31,6	1,1	4 138,60	35,2	53
	Original	74,5	66,95	38,5	25,5	316,36	61,5	
13	105	3,5	14,05	0,6	8,6	29,3	2,9	83
	74	9,8	5,85	0,7	8,3	32,30	3,0	81
	53	14,4	6,30	1,1	5,8	64,80	4,4	80
	44	8,3	11,30	1,1	2,6	186,10	5,7	84
	−44	36,6	78,30	33,5	2,1	1 914,50	47,0	58
	Original	72,6	42,47	37,0	27,4	197,57	63,0	

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DISCUSSION

1. S. K. DE KOK*, B.Sc. (Eng.) Rand (Fellow)

I wish to comment on the subject of the recovery of gold before cyanidation, the beneficial effect of which, to my mind, has emerged as the most significant single finding of the Anglo American Research team. Their conclusion supports the long-held convictions of the Anglovaal Group, which have found expression in the inclusion of gravity concentration within the grinding circuits and in the employment of flotation, either in the grinding circuit or in the milled-pulp stream on its way to the cyanide plant. This contribution is made in the hope that our experience in the field of concentration may be of some value to the Anglo American investigators in their further efforts.

On reading between the lines of the paper by Messrs Bath, Duncan, and Rudolph and also those of the related papers by other members of the team, I get the impression that they are possibly on the verge of repeating the error of omission that was made some twenty years ago when we failed to realize the full potential of the plane table as a

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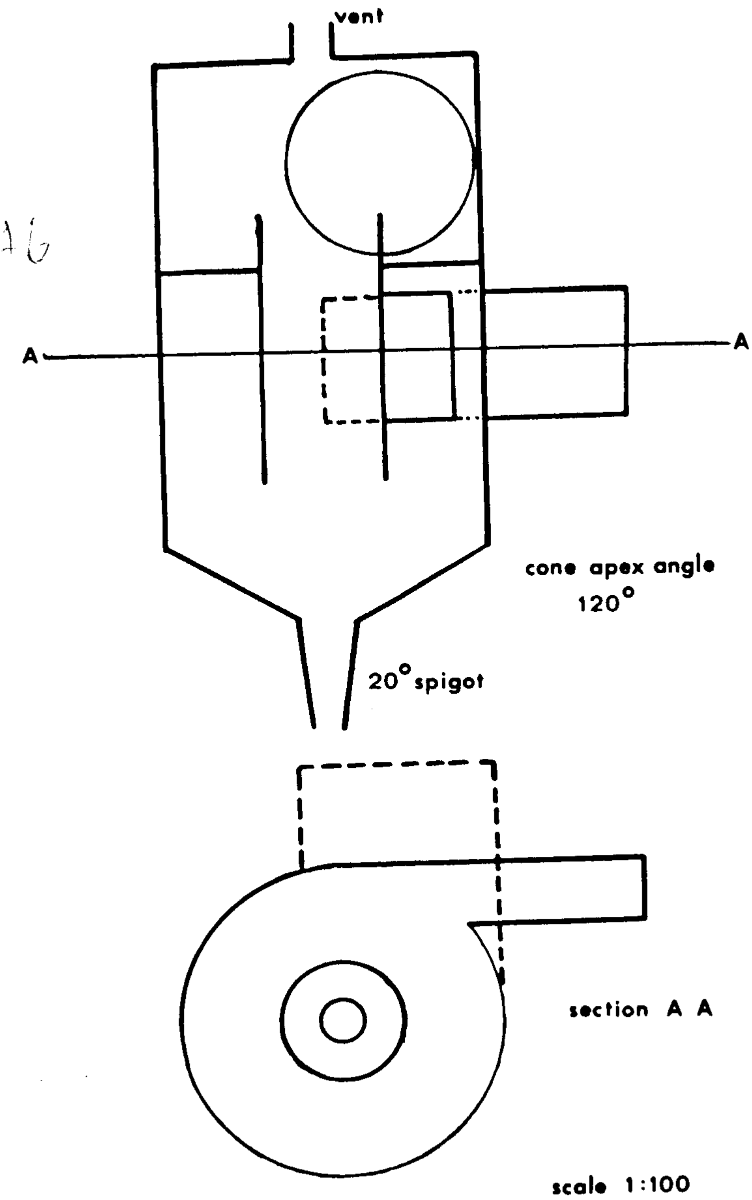

Fig. 14—Blunt-cone cyclone

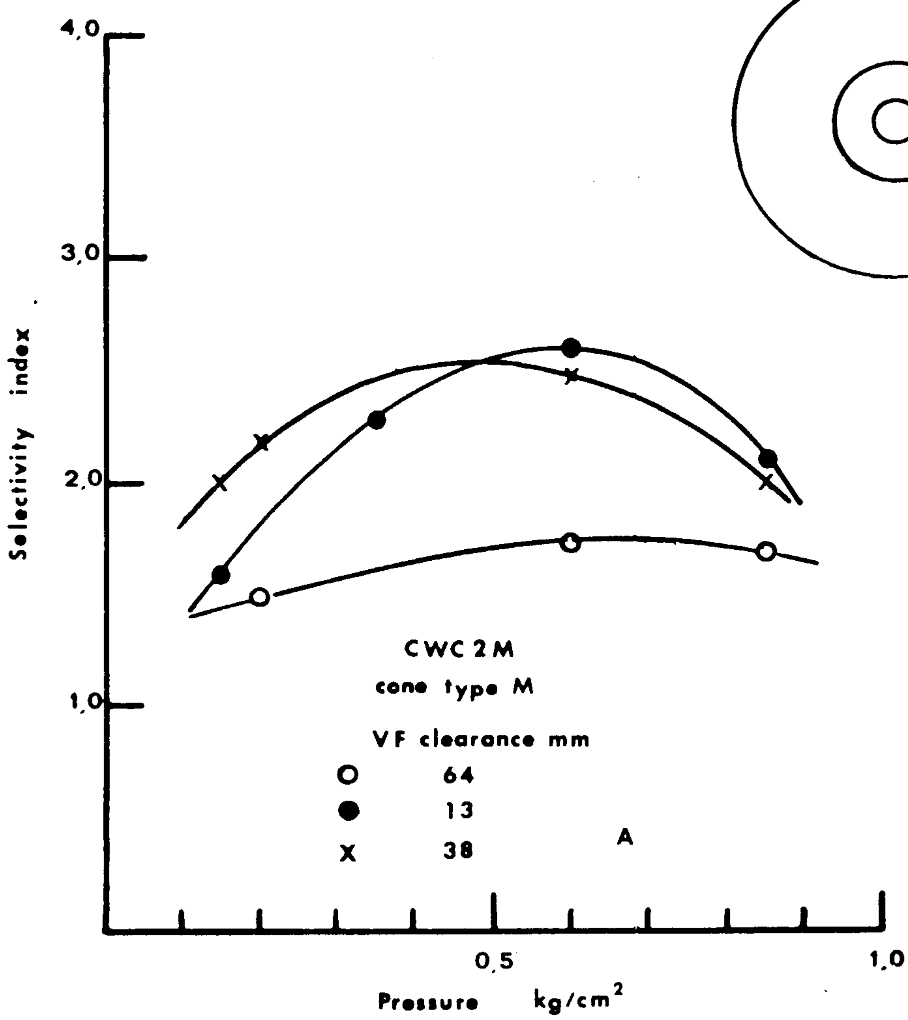

Fig. 13—Compound Water Cyclone 50 mm unit-effect of variation in the vortex-finder clearance

concentrator of refractory, as well as of free, gold. To substantiate this view, I should first like to elaborate on the application of this device on a few mines within our group and then attempt to draw a parallel with the authors' present work.

The plane table, functioning essentially on the flowing-film principle, is an exceptionally high-capacity roughing concentrator, which is a decided advantage in view of the trend towards large grinding units and high circulating loads. At the usual pulp dilution, it has a throughput of some 1500 tonnes of solids per day per metre width, which means that it does not lend itself easily to laboratory or even pilot-plant investigation, although I understand that NIM has had some success in this regard. It is extremely simple to operate, but requires careful initial adjustments to get it functioning; once working, it never stops. It was invented at Rand Leases in 1949 by D. McLean for the sole purpose of replacing corduroy strakes that were installed in the secondary grinding circuits[1]. Now the corduroy cloth is probably the most selective concentrator of free-milling gold that has ever been used because it produces a small bulk of very high-grade concentrate that is easily dressed on a shaking table before being charged into an amalgam barrel. The plane table, on the other hand, produces a very much greater bulk of lower-grade pyritic concentrate, which, to a far greater extent requires reconcentration so that the quantity is reduced to that which can be conveniently handled in the barrel. But the removal of total gold from the grinding circuit is far higher than that achieved by corduroys. At Rand Leases, the plane tables recovered some 90 per cent of the gold being fed to the plant, which was double that previously recovered in amalgam form by the cloth. However, because of the constraint imposed on the system by the small capacity of the amalgam barrels and because not all the gold recovered by this primary or "roughing" device was amalgamable, some 50 per cent of the value removed was returned to the mill in the middlings and tailings products from the shaking tables and in the barrel tailings; the net recovery of gold in amalgam form was virtually unchanged from that achieved by the corduroy cloths. Of the gold contained in the shaking-table middlings some 30 per cent proved to be amalgamable. In other words, it was shown that the plane tables were more effective than the corduroy in recovering fine free gold, and that the shaking tables were not as effective as the plane tables in retaining the flaky grains. The remainder of the gold returned to the circuit was either encased in sulphides or tarnished by iron oxide stains. The circuit finally adopted was a stratagem devised to reduce the load on the shaking tables and to subject the concentrate, grading 36 per cent minus 200 mesh, to a separate fine grind. The concentrates from ten tables installed in the secondary grinding circuits were fed to a pebble mill (2 m by 6 m (6 ft 6 in by 20 ft)) and reconcentrated on a single plane table treating the mill effluent. The concentrate from this table, approximately 15 t/d, was delivered to the shaking table and thence to the amalgam barrel, while the classifier overflow, amounting to 3,6 per cent of the total plant feed and grading 96 per cent minus 200 mesh, joined the secondary-mill products for delivery to cyanidation. Following the reduction in feed tonnage to the shaking table and the release of more gold by the finer grind, total recovery by amalgamation increased by 5 per cent to 50 per cent. The net effect of the circuit was that 3,6 per cent by mass of the ore containing 40 per cent of the total gold, including the major portion of the refractory gold, was allowed to proceed to the cyanide plant, together with the remaining 96,4 per cent of the ore containing only 10 per cent of the gold, without any preliminary treatment other than finer grinding. The mistake I have referred to lay in not subjecting this small tonnage of high-grade material to separate treatment such as more-intensive cyanidation or pretreatment with acid. It is not irrelevant to note that, in the treatment of the far more refractory ores of the three mines of the Eastern Transvaal Consolidated Group, the plane table concentrates after regrinding and removal of the free gold and siliceous gangue, are added to the flotation concentrates for roasting and further grinding (in cyanide solution) prior to cyanidation.

Subsequent experience at Hartebeestfontein has confirmed the benefit of a preliminary acid treatment. On this plant, plane tables were initially installed in the secondary-grinding circuits, as at Rand Leases, but in this case only 60 to 70 per cent of the gold in the mill feed was removed, the lower recovery almost certainly being due to the association of much of the gold with the light mineral, thucholite, and to the extreme fineness of the free gold. After re-dressing of the plane-table concentrates on an endless-belt concentrator and a shaking table, the net gold recovery by amalgamation was reduced to 11 per cent. Concentration was consequently abandoned. The cyanide residue value was far higher than desirable, even at an extremely fine grind, but it was subsequently reduced by 50 per cent on the introduction of the reverse-leach process of uranium-gold recovery, which subjected the total pulp to acid treatment before cyanidation[2]. It is, of course, not claimed that the reverse-leach procedure is the economic solution to the problem in all cases, but it does point the way to the more effective treatment of concentrates containing gold that either is filmed with coatings, for which expensive scrubbing or prolonged cyanidation may be the only alternative answer, or is encased in acid-soluble minerals, which would not otherwise permit cyanide attack. After all, uraninite was first detected in our ores from an examination of corduroy concentrates.

I turn now to the authors' work on the more effective application of concentration in an attempt to improve overall recovery. The Anglo American plants employ the Johnson concentrator as the primary device; this is a fairly low-capacity, mechanized form of the corduroy cloth, although it produces a greater bulk of concentrate. For cleaning of the Johnson concentrate, use is made of endless-belt concentrators, in which the belt movement is against the direction of pulp flow. For this purpose, I think consideration should be given to the use of the Gallagher table, which is extensively used in Rhodesia but is hardly known in this country; in this device, the belt moves transversely at right angles to the pulp flow. I fully agree with the authors' statement that early removal of the gold is most desirable to avoid the deleterious effects of overgrinding, but the Anglo American grinding and concentration circuits as at present constituted are not conducive to efficiency in this respect. Concentration is confined to the tertiary stage of classification/milling, at which stage much of the liberated but maltreated gold has already escaped to the cyanide circuit via the secondary cyclones. For an increased concentration recovery, reliance must therefore be placed on the concentrating effect of the cyclones to deliver as much of the gold as possible to the Johnson concentrators. As a consequence, the authors are having to investigate wider-angled cyclones, which tend to suppress the classifying and accentuate the concentrating effects of settling. Earlier removal of the gold would seem to be the logical method, but this would require the concentrators to be placed within the circulating loads of the primary- and secondary-milling circuits. The low capacity of the standard-sized Johnsons is not a factor in their favour for this purpose, and the authors are investigating devices such as the blunt-cone cyclone and the Reichert cone, which is but a version of the plane table without its side-wall effects. Experiments have been carried out at the Anglovaal Laboratory with laboratory-sized vessels lined with riffled rubber. These are somewhat similar to the Johnson concentrator but are conical in shape and run at supercritical speeds; the centrifugal forces developed give, in open circuit, 90 per cent recovery of the gold at high ratios of concentration, but, as with all centrifugal machines of this type, continuous discharge of the concentrate presents difficulties. The development of a concentrator utilizing centrifugal forces for the recovery of the finer sizes of the heavy minerals is, I feel, one of the fields of research in which our industry should be actively engaged.

Arithmetical averages of the figures presented by the authors, which are sufficiently accurate for illustration purposes, indicate that the endless-belt concentrators recover 70 to 75 per cent of the gold delivered to them by the Johnsons, and that the amalgamation of the concentrate so produced results in a net recovery of 35 per cent of the gold fed to the plant. If, as stated, the Johnson concentrators recover some 70 per cent of the total gold, it is obvious that the endless-belt tailings contain some 15 per cent of the gold and, in the circuit as at present composed, this is merely returned to the grinding circuit for further defacement and delivery to the cyanide circuit; at least, at Rand Leases this material was given a separate finer grind. After emphasizing the importance of a high recovery of gold before cyanidation, the authors have quite rightly applied themselves to improving the primary recovery, but so far have neglected to investigate the correct treatment of the gold that is lost in the process of reducing the bulk to suit the amalgamation process. They admit that doubling the speed of the endless belts increased total gold recovery by amalgamation by 2.5 per cent but at the expense of increasing concentrate mass by 50 per cent. I submit that there is a far greater potential pay-off in investigating the non-amalgamable gold, which the primary concentrators have handed to them on a plate, than in an improvement in the amalgamation process itself. Indeed, the complete elimination of amalgamation, besides removing the restrictions imposed by the limitations of the plant, would obviate the very real danger of mercury escaping with the pulp and inhibiting cyanidation.

This prompts me to ask whether the improvement in total recovery following an improvement in amalgamation is not in part due to a reduction in mercury loss. Alternative methods of treatment of the total primary concentrate could be devised that would provide for the recovery of the osmiridium and free gold, possibly in the form of a "clean-cut" for direct smelting or by flotation. The reduction in cyanide residue value resulting from improved amalgamation, noted by the authors, is not, in my opinion, due to an increase in amalgam recovery *per se*, but rather to a reduction in the quantity of refractory gold in the tailings from the amalgam barrel that proceeds to cyanidation. There is undoubtedly far more of this material present in the endless-belt tailings than was ever present in the barrel tailings before changes were made in the techniques of amalgamation. Contrary to the authors' assertion that recovery by amalgamation is inherently more efficient than that by cyanidation, I believe that amalgamable gold is the gold that is also most easily cyanided. If special treatment is required to render the gold amalgamable, that same treatment will render it more amenable to cyanide. It is not important that the gold recovery as amalgam be 30 per cent or 50 per cent or any other figure; what is important is that, if concentration has been considered necessary to relieve the cyanide plant, advantage be taken of the opportunity to correctly prepare for cyanidation the gold in the primary concentrate that has not reached the barrel, as well as the gold that amalgamation has failed to recover. In failing to make full use of this opportunity, the authors are perpetuating the myth that concentration is synonomous with amalgamation, just as we did until we saw the light. It is not surprising that, in the days when concentration was regarded merely as a means to amalgamation, the proponents of all-cyanidation were not convinced of the value of concentration.

The advantages of separate treatment of the high-grade, more-refractory portion of the ore are highlighted at three of our mines on which flotation plants were installed for the recovery of pyrite required for acid manufacture for the uranium industry[3]. Anglovaal broke with the traditional practice of recovering this pyrite from the uranium-plant tailings by interposing the flotation plants between grinding and cyanidation. At one of the Hartebeestfontein plants, the flotation cells are installed within the grinding circuits, whereas, at Loraine and the other Hartebeestfontein plant (formerly Zandpan), flotation is carried out on the final mill pulp. Gravity concentration in the grinding circuit has been retained at Loraine, both for osmiridium recovery and because of our experience, verified on the Eastern Transvaal Consolidated Group of mines, that flotation is least successful in recovering the very finest and the very coarsest fractions of the gold particles. On all the mines, the recovery of gold into the flotation concentrates exceeds 80 per cent. It is of interest to note that well over 50 per cent of the total gold can be floated with the addition of frother only and without the addition of xanthate or any other pyrite promoter, the main dilutent in the high-grade concentrate being fine siliceous gangue. The separate regrinding and intensive cyanidation of the pyrite concentrates result in 99 per cent recovery of the contained gold. The net results have been either increased overall recoveries or unchanged recoveries with increased tonnage throughputs, the natural consequence of decreased dependence on fine overall grinding. At Hartebeestfontein, the combination of reverse-leach treatment of flotation tailings, reground concentrates, and calcines has resulted in an increase in recovery of some 2½ per cent at a grind coarser by 18 per cent minus 200 mesh.

In conclusion, I wish to congratulate the authors on their stimulating paper. Of their overall philosophy and approach there can be no criticism. They are tackling the problem in a methodical sophisticated manner; I venture to predict that the evidence they are still to gather from their pilot-plant will soon result in a revision of their priorities.

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2. J. LEVIN*, B.Sc. (Eng) Rand (Member)

The National Institute for Metallurgy is engaged on tests on the concentration of uranium, and, because gold and uranium are closely associated in the ores of the Witwatersrand, some of the observations that have been made have a bearing on the problems of gold extraction. Currently, an ore from the Vaal Reef is being examined, and the information obtained so far points to the following tentative conclusions, which are based on the experimental results shown in the accompanying tables.

(1) Gravity concentration for the recovery of gold and uranium can be carried out advantageously in an open circuit, instead of in the usual closed grinding-classification circuit.

(2) The proportion of concentrate made by gravity or flotation concentration should be fairly high—between 5 and 10 per cent of the ore—so that the maximum amount of gold and uranium can be recovered.

(3) Extraction of gold by cyanidation is increased significantly when cyanidation follows acid-leaching for the dissolution of uranium; a significant increase is observed even when the tailings are subjected to prior acid leaching.

(4) Satisfactory extraction of gold from gravity concentrates can be obtained by cyanidation after fine grinding and acid leaching, without resort to amalgamation. (Amalgamation would not be applicable to the relatively large proportion of concentrate being recommended.)

(5) Flotation effects a higher recovery of gold than does gravity concentration.

(6) When gravity concentration is used, the tailing is the largest source of gold not extracted by cyanidation. When flotation is used, the first xanthate concentrate is the largest source of gold not extracted by cyanidation.

Gravity concentration

When concentration in a closed grinding-classification circuit is practised, the quantity of material fed to the concentrate is generally several times that of the new feed entering the circuit, and the possibility of applying concentration in an open circuit is worth considering. To ascertain if recoveries from an open circuit would be substantially lower than from a closed circuit, laboratory tabling tests were done on material ground to various extents, and the results were compared with those obtained from a test in which closed-circuit grinding and concentration were simulated. Table I summarizes the results obtained. The figures actually suggest that, if the optimum degree of grinding for open-circuit concentration is used, the recoveries in open circuit will be higher than from a closed circuit, but it is preferred merely to draw the conclusion that the recoveries of gold and uranium from an open circuit should not be markedly lower than from a closed circuit. Table I also shows that substantially increased recoveries are effected when the mass of concentrate is increased from 5 to 10 per cent of the feed.

Gold extraction from products of gravity concentration

A batch of coarsely ground ore was concentrated by use of a Humphreys spiral and a shaking table to provide three concentrates and a tailing. The four products obtained were ground to various degrees, acid leached, and cyanided. Fig. 1 shows how the concentration was effected and the results obtained. Table II shows the results of cyanidation of the individual products of concentration. The very high extractions of gold obtained from the first concentrate are noteworthy, and it appears that this concentrate does not include gold in a form that needs intensive treatment for its extraction by cyanidation. However, all the other products need to be finely ground and acid leached to enable satisfactory extractions of the gold to be achieved. The high gold content of the tailings should be noted.

Concentration by flotation

Fig. 2 illustrates the procedure used in concentrating the ore by flotation, and Table III gives the results of the pretreatment and cyanidation tests done on the flo-

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tation products. It can be seen that 57,6 per cent of the gold was recovered by flotation with only a frother. Cyanidation of the frother concentrate was very effective, although acid leaching is essential if maximum extraction is to be achieved, and acid leaching in this instance appears to be more important than fine grinding. Cyanidation of the xanthate concentrate was far less satisfactory, and it is evident that these concentrates contain gold in a form that requires intensive treatment for its extraction. The residues from the flotation tailing are markedly lower than those obtained from gravity concentration, but even it benefits from acid leaching prior to cyanidation, the gold assay value being reduced by approximately 0,14 g/t for all degrees of grinding tested.

Mineralogical examinations

Approximately half the gold observed in the first two gravity concentrates was in the form of free grains, the remainder being associated in roughly equal amounts with sulphide or uraninite. In the third gravity concentrate, free gold is less abundant, and gold associated with uraninite is more abundant; this concentrate also contains a marked proportion of grains in which thucholite is attached to uraninite. Although most of the gold grains appeared to be discoloured or stained to various extents, the considerable proportion of grains in which gold is associated with uraninite suggests that much of the benefit obtained from acid leaching prior to cyanidation can be ascribed to the association of gold and uranium. However, there is no definite evidence of the magnitude of the benefit that might have resulted from the removal of coatings from the gold grains by the acid leaching step.

The frother concentrate obtained by flotation contained numerous free grains of gold and numerous grains in which the gold was associated with pyrite or uranium minerals; the latter were predominantly thucholite, but there were also numerous grains in which uraninite was associated with the thucholite, and some grains that appeared to be wholly uraninite.

TABLE I

COMPARISON OF GRAVITY CONCENTRATION IN OPEN AND CLOSED CIRCUITS

Circuit simulated	Grind, % – 75 µm	Gold recovery, % at 5 % mass of conc.	Gold recovery, % at 10 % mass of conc.
Open	40	43,0	63,8
Open	50	51,0	62,0
Open	61	48,2	65,4
Closed	3 stages; 1st stage 40 % – 75 µm	45,5	58,0

TABLE II

CYANIDATION OF PRODUCTS OF GRAVITY CONCENTRATION

(See Fig. 1).

Concentration Product	Grind µm	Gold in cyanide residue g/t		Distbn. of gold, % of gold in: Prod. of conc.		Original ore	
		Dir.*	Rev.†	Dir.*	Rev.†	Dir.*	Rev.†
1st conc.	– 150 µm	7,0	3,1	0,30	0,20	0,09	0,06
	– 100 µm	4,5	4,0	0,21	0,14	0,06	0,06
	– 53 µm	10,0	4,9	0,54	0,20	0,16	0,06
2nd conc.	– 150 µm	8,4	3,6	9,0	3,8	2,49	1,05
	– 100 µm	5,0	2,1	5,4	2,3	1,49	0,64
	– 53 µm	3,7	2,4	3,5	2,5	0,98	0,69
3rd conc.	– 150 µm	10,5	3,9	9,1	3,4	1,11	0,42
	– 100 µm	7,5	2,6	6,5	2,2	0,79	0,27
	– 53 µm	5,2	2,7	4,2	2,4	0,51	0,29
Tailing	53 % – 75 µm	0,84	0,52	13,5	8,4	4,04	2,50
	65 % – 75 µm	0,74	0,37	11,9	6,0	3,56	1,78
	76 % – 75 µm	0,47	0,25	7,6	4,0	2,26	1,20
	87 % – 75 µm	0,39	0,26	6,3	4,2	1,87	1,25

*Direct cyanidation
†Cyanidation after acid leaching

TABLE III

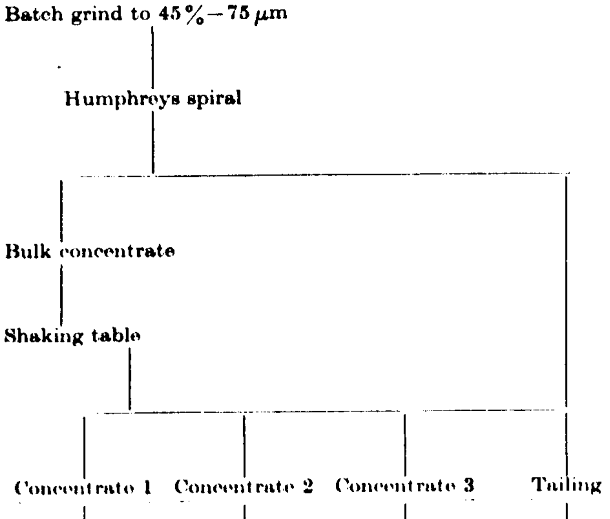
CYANIDATION OF PRODUCTS OF FLOTATION

(See Fig. 2).

Concentration Product	Grind	Gold in cyanide residue g/t		Distbn. of gold, % of gold in: Prod. of flotn.		Original ore	
		Dir.*	Rev.†	Dir.*	Rev.†	Dir.*	Rev.†
Frother conc.	– 150 µm	17,8	4,7	1,20	0,30	0,69	0,17
	– 100 µm	13,7	2,1	0,90	0,14	0,52	0,08
	– 53 µm	11,1	3,1	0,72	0,20	0,42	0,11
1st xanthate conc.	– 150 µm	15,7	10,8	10,70	7,40	3,30	2,78
	– 100 µm	10,3	6,9	7,10	4,70	2,19	1,45
	– 53 µm						
2nd xanthate conc.	– 150 µm	11,4	8,5	26,8	20,0	1,34	1,00
	– 100 µm	7,0	4,4	16,5	10,4	0,82	0,52
	– 53 µm						
Tailing	57 % – 75 µm	0,40	0,27	29,0	19,7	1,91	1,30
	67 % – 75 µm	0,30	0,22	21,4	16,2	1,42	1,05
	75 % – 75 µm	0,31	0,17	22,6	12,2	1,49	0,80
	85 % – 75 µm	0,24	0,10	17,39	7,0	1,15	0,46

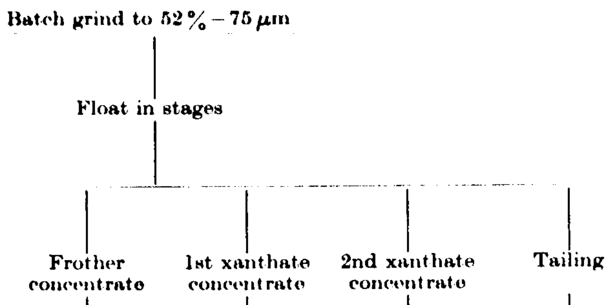
*Direct cyanidation
†Cyanidation after acid leaching

xamination of the two xanthate concentrates indicates that they contained less free gold than the frother concentrate and that more gold was associated with sulphides than with uraninite or thucholite.



	Concentrate 1	Concentrate 2	Concentrate 3	Tailing
Mass, %	0,29	5,71	2,01	91,99
Gold, g/t	2005	92,5	116,0	6,2
Gold distribution, %	30,4	27,7	12,2	29,7
U_3O_8, g/t	4126	1258	1976	230
U_3O_8 distribution, %	3,6	21,4	11,9	63,1

Fig. 1—Procedure and results for gravity concentration test



	Frother concentrate	1st xanthate concentrate	2nd xanthate concentrate	Tailing
Mass, %	0,74	4,12	2,29	92,85
Gold, g/t	1520	146	42,6	1,38
Gold distribution, %	57,6	30,8	5,0	6,6
U_3O_8, g/t	5284	1748	1945	212
U_3O_8 distribution, %	11,1	20,4	12,6	55,9

Fig. 2—Procedure and results for concentration by flotation

3. E. A. D. RUBIDGE* (Fellow)

The paper reported many factors relating to the efficiencies attainable in amalgamation.

At one stage the paper defines the word "liberation" when applied to gold particles and states that the mineral grain "for gravity concentration . . . must be almost completely separated from the gangue." In his address relating to this parameter, Mr Rudolph remarked that hydrated iron oxides often covered the surfaces of the gold particles and prevented or delayed the process of amalgamation.

It is on this point that I wish to report on some experiments carried out in the Rand Leases Assay Department in 1954. They dealt with concentrates from the Virginia Mine. The metallurgist in charge found that barrel tailings repeatedly contained "floured" mercury, which, of course, carried much gold. He asked that tests be carried out to indicate the cause of the trouble and, if possible, provide information that would lead to effective amalgamation.

The material contained many particles that were described as "rusty gold", which was, of course, caused by the "hydrated iron oxide" referred to in the paper under discussion.

The line of investigation undertaken on that occasion was mainly directed to conditioning of the concentrate by chemical means before proceeding with the amalgamation tests.

Only five of the many significant tests will be reported here, with the purpose of high-lighting the part played by the use of cyanide to clean the gold particles before the addition of mercury and rolling for a very brief period without further grinding.

Weighed portions of concentrate, with water added, were dealt with as follows: (see table on page 384)

Note: In all these tests, no grinding balls were added. It was assumed that the grinding that usually accompanies amalgamation work adds to the build-up of hydrated iron oxide film. This would not, of course, occur to any great extent in the

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Test No.	Treatment given	Observations
1	Washed, CaO and $KMnO_4$ added, and rolled on laboratory rolls	Much "floured" mercury
1a	Washed out all CaO and $KMnO_4$ from No. 1 above, added CaO and KCN, and rolled for 30 minutes with Hg	98 % of gold found in amalgam
2	Washed three times with water, then added CaO and Hg, and rolled for 15 minutes	68,3 % of gold recovered
3	Treated as in 2 above, added CaO, KCN, and Hg, and rolled for 15 minutes	92,6 % of gold in amalgam
4	Treated with CaO & KNO_3 and, without washing, added Hg and rolled for 15 minutes	74,1 % of gold in amalgam
5	Treated as in 4 above but added KCN before rolling for 15 minutes	95,4 % of gold in amalgam

short time of rolling reported in the above tests. None-the-less, it is of importance to note that, with the gold surfaces cleaned (and thus "almost completely separated") by cyanide, the amalgamation appears to take place almost immediately.

Many other tests were carried out, and, wherever cyanide was omitted, the extraction was much poorer in these short-time runs.

It will be seen that cyanide would dissolve the very fine or flattened particles of gold (probably a desirable feature!), and the transfer of this gold-bearing solution would have to be attended to in any plant designed to follow the suggested procedure.

The need for the further grinding of the resulting barrel tailing might, of course, be necessary if inspection or assay proved the need for such a move, but returning the tailings to circuit (or other further treatment) after more than 90 per cent of the coarse gold has been removed would not, I am sure, raise any anxieties.

Perhaps the authors of the paper would find it useful to apply their very thorough skills to proving the value or otherwise of the above findings.

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