Plane Table

Subsequently, the "plane table" was designed in 1949 and was adopted on many treatment plants because, having no moving parts, it required virtually no maintenance. The plane table consists of an inclined steel or wooden table, 1,52 m long and 1,07 m wide, sloping between 8 and 12° with a step halfway down the table. A feed box, together with distributing baffles, is situated at the head, while a launder is provided underneath the step and at the lower end as a receptable for concentrate drippings. The deck is covered with two sections of riffled rubber matting of the same type previously used as a substitute for corduroy cloth but with the riffles running longitudinally instead of transversely. As the pulp flows over the table, gold concentrates collect in the troughs of the rubber matting and, on reaching the mid-way slot or the end of the table, gravitate, i.e. drip into strategically placed launders while the faster flowing pulp passes on into a separate launder. In some recent installations the number of slots in the table has been increased and as many as five have been installed. As with all types of gravity concentrators, turbulent and uneven flow of the pulp tends to lower efficiency. However, the design of the feed box and distributor has been found adequate to ensure an even flow over the table. A slightly diverging stream is produced and to reduce eddy currents to a minimum

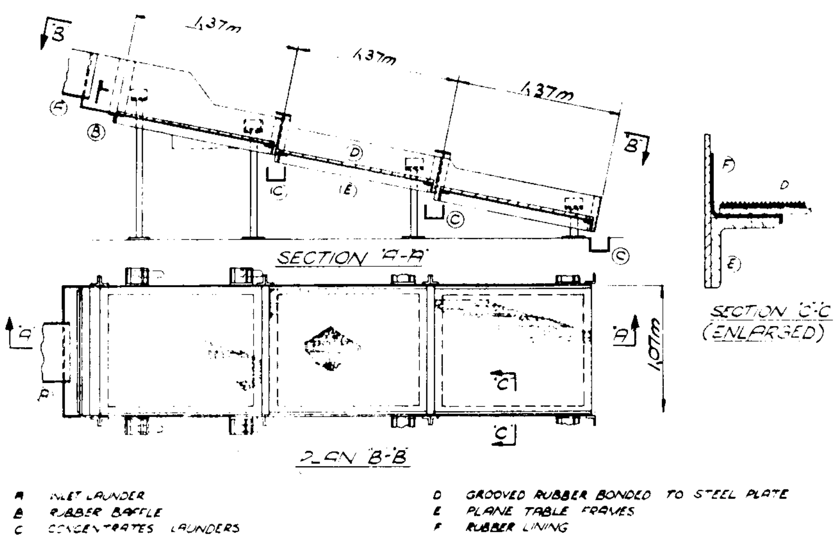


Figure 5. Plane table concentrator in three sections

the two sections of riffled rubber matting are laid with the saw teeth facing outwards. The framework is of wooden or steel construction and the slope can, if desired, be varied. The table is brushed down with a wire brush, usually once or twice per shift, mainly for the removal of fine iron. The concentrates are further treated on James or similar shaking tables prior to amalgamation.

Cylindrical Concentrators

Later, in 1958, a modified type of Johnson concentrator using riffled rubber mats instead of corduroy cloth to line the interior of the cylinder was installed on many mines in the Orange Free State. This apparatus was used as a primary concentrator and the concentrate from three units was re-treated on an endless moving belt concentrator prior to amalgamation in the recovery plant.

Present-day practice is in favour of either the plane table with secondary circuit shaking tables or the combination of Johnson type concentrators and endless moving belt re-concentrators.

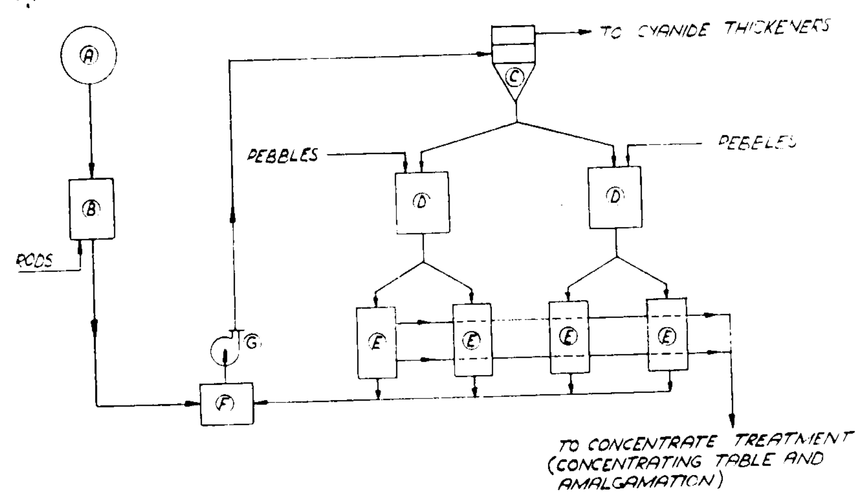
Jigging

Although sporadic experiments had been previously conducted with several types of jigs to effect concentration in milling circuits, no economically satisfactory results were obtained until 1960. A special application had been in operation since 1935 on one reduction plant but the primary object of the installation was the recovery of iron pyrites for the production of sulphuric acid. In this case moving sieve type units, known as Hardy-Smith buddle jigs, with a capacity of 270 tons per 24 hours, produced a primary concentrate containing 15% FeS_2 which was reconcentrated to 30% FeS_2 in secondary jigs. The concentrate from the latter was treated on James shaking tables resulting in a final concentrate grade of 82% FeS_2. The gold content was extracted by milling, amalgamating and cyaniding. The residual pyritic material was dried and despatched to the sulphuric acid plant. The plant has functioned for 36 years, but no claim has been made for an improvement in gold extraction.

During the 1960's two plants were erected which included jigs at the mill outlets with the objectives of eliminating corduroy concentrators, thus mechanising the gold concentration section, and of recovering iron pyrites for sale to sulphuric acid producers. A further advantage offered was that jigs occupied less floor space in the mill building than any other design of gravity concentrator. The type selected was the Yuba Richards mineral jig which comprises two cells, each 1,07 m square, fitted with fixed screens, and rubber diaphragms to provide the necessary pulsations. Operating results do not indicate any noticeable improvement in gold recovery and their application to pyrite recovery is only effective in cases of a high sulphide grade.

Advantages of Gravity Concentration

While the cyanide process has reached a very high degree of efficiency there are still advantages in extracting as much gold as possible in the milling circuit by gravity concentration. The main reasons for continuing with this type of treatment are: selective grinding can be applied to the pyritic concentrate to ensure the release of enclosed gold particles without having to overgrind the quartzitic material; by removing the gold from the pulp circulating through the mills, the amount of gold locked up behind the mill liners is considerably reduced; the reduction of the gold content of the slime by approximately 50% results in a lower grade cyanide solution which has certain benefits in filtration and



A FINES SILO (-15mm PRODUCT)
B ONE × 2,7m DIA × 3,6m LONG ROD MILL
C TWO PRIMARY + TWO SECONDARY CYCLONES (0,76m DIA)
D TWO × 3,6m DIA × 4,8m LONG PEBBLE MILLS
E FOUR × 4,12m × 1,06m TRIPLEX PLANE TABLES
F MILL SUMP
G MILL TAILINGS PUMP

Figure 6. Concentrator circuit flow diagram

precipitation. Furthermore, it permits the recovery of fine particles of osmiridium which occur in varying amounts in the different gold reefs. Without the introduction of a gravity concentration process, this constituent, which occurs at a grade of one gram to between 50 and 500 tons milled, i.e. 0,02 g/t to 0,002 g/t, would not be recovered and would be discharged to the slimes dam. Gravity concentration is of greatest benefit when treating high grade ores, but is applicable to all grades above 10 grams per ton. Over 50% of the gold mines in South Africa practise one or more of the methods described in this chapter.

OPERATING DATA

Table 1

PLANE TABLE CONCENTRATORS

Table size	1,52 m × 0,76 m	1,52 m × 1,07 m	1,52 m × 1,22 m
Number in series	2	3	2
Slope, degrees	8	10	10
Slot width mm	25	10	30
Feed source	sec. mill outlet	sec. mill outlet	sec. mill outlet
Tons solids per hour	55	33	60
% moisture	35	50	33
Gradings % + 48 Tyler	2	—	3
+ 100 ,,	37	23	25
+ 200 ,,	31	40	35
− 200 ,,	30	37	37
Head value g/t	57	420	90
Tail value g/t	17	285	40
% concentrate	1,0	1,0	1,0
% recovery	70	32	55
Reconcentration by	James tables	Belt conc.	Belt conc.

Riffle dimensions are usually 3,2 mm wide at the surface, 3 mm deep and 3,2 mm apart. The flow rate of the pulp over the tables is the most significant parameter in determining gold recovery efficiency. The velocity of flow depends on the quantity of solids passing over the table, the moisture content of the pulp, the slope of the table and the effective width of the table over which the pulp flows. To measure and control pulp velocity directly would be extremely difficult and would require expensive instrumentation. However, with good distributor design that utilizes the full width of the table and with a fixed slope (determined experimentally), gold recovery becomes dependent on the tonnage of solids per unit time and pulp moisture content. In the normal operation of grinding mills, the quantity of solids passing through the milling circuit is most consistent and thus the pulp velocity over the table or tables is dependent only on the moisture content which is a parameter that is easily determined and adjusted. It will be noted that there is a considerable variation in the characteristics of mill pulps that can be treated by means of the plane table. The operating conditions for a particular application can only be approximately determined from Table 1—optimum settings have to be decided empirically once the tables are installed.

The concentrate from the plane table is too bulky for amalgamation and therefore reconcentration either in one or two stages is conducted with shaking tables or belt concentrators.

Table 2

JOHNSON TYPE CYLINDRICAL CONCENTRATORS

Dimensions (Diam. × length)	0,76 m × 3,66 m	0,91 m × 3,66 m	0,61 m × 3,66 m
Slope, degrees	4	2,5	5
Speed, r.p.m.	0,125	3,00	0,10
Feed source	cyclone underflow	sec. mill outlet	cyclone underflow
Tons solids per hour	40	125	5
% moisture	49	30	73
Gradings % + 48 Tyler	15	14	18
+ 100 ,,	35	34	34
+ 200 ,,	34	32	34
− 200 ,,	16	20	14
Head value g/t	46	30	530
Tail value g/t	10	20	300
% concentrate	0,27	2,0	1,5
% recovery	78	33	43
Reconcentration by	Belt conc.	Belt conc.	Belt conc.

Power consumption is a minor consideration in the operation of the Johnson concentrator, as a motor of 1 kW rating is adequate to rotate the cylinder. The saw-tooth riffled rubber lining of the drum lasts for several years, the only attention required being a periodic brushing to remove a build-up of lime scale in the riffles. The latter are usually 3 mm deep by 3,5 mm in width and 3 mm apart. They run longitudinally and parallel to the axis of the drum. Rotation of the drum has to be in the direction opposite to the sharp edge of the saw tooth and not as might be expected opposite the blunt edge. Alternatively if the lining is suitably spiralled the orientation of the teeth can be with the sharp edge leading. The concentrator is fairly elastic regarding the feed rate, but the weight of concentrate washed out of the riffles is closely associated with the moisture in the feed, a high moisture content producing a low weight of concentrate.

As in the case of the plane table, reconcentration is necessary before amalgamation and for this purpose the riffled moving belt concentrator is usually preferred for secondary concentration.

Table 3

RIFFLED MOVING BELT CONCENTRATORS

Surface dimensions length × width	3,05 m × 1,52 m	3,20 m × 1,52 m	3,35 m × 1,52 m
Slope, degrees	10	10	10
Speed, m/min	0,36	0,10	0,25
Feed source	Johnson conc.	Plane table conc.	Johnson conc.
Tons solids per hour	0,3	2,0	8,0
% moisture	96	90	91
Gradings % + 48 Tyler	7	1	6
+ 100 ,,	19	25	25
+ 200 ,,	46	41	45
− 200 ,,	28	33	24
Head value g/t	22 200	110	550
Tail value g/t	1 010	40	90
% concentrate	13	2	0,67
% recovery	95	64	84
Reconcentration by	Amalgamation	Amalgamation	Amalgamation

The riffled moving belt concentrator also has a low power drive of a one kilowatt motor similar to the Johnson concentrator. Here again the orientation of the saw tooth riffles is important. The flow of pulp over the belt should be first up the inclined face of the saw tooth and then down the steep face. The riffled belt has to be perfectly level across the flow and care taken to prevent slippage on the head pulley. With a slow movement, of as low as 0,10 metre per minute, a 50% loss in speed by slipping is often not noticed. An even feed over the full width of the belt is also required to obtain maximum efficiency. Scale deposits in the riffles are soon apparent and can be removed by transverse brushing.

The riffled moving belt concentrator is not as efficient as the Johnson cylinder in primary concentration, but in the secondary stage, with a feed rate of a few tons of solids per day at 90% moisture, it is highly effective.

Corduroy blanket strakes, derived originally from the practice of the ancients of recovering alluvial gold on sheep or goat skins, act as excellent traps for the coarser particles of gold in mill pulps. There are, however, two major drawbacks to their use. Firstly if not changed at the correct intervals their efficiency falls away rapidly. Secondly they have to be changed manually and the adhering concentrates have to be washed out by hand, thus necessitating a "washing gang". Two lesser objections are that the blankets wear and have to be replaced every 6 to 8 months and that

Table 4
CORDUROY STRAKES

Size:			
Length × width	2,85 m × 1,45 m	2,44 m × 1,32 m	2,59 m × 1,22 m
Slope, degrees	7	10	10
Feed source	mill outlet	sec. mill	sec. mill
Tons solids per hour	3,5	12,5	6,3
% moisture	75	48	47
Gradings % + 48 Tyler	12	—	6
+ 100 ,,	33	—	38
+ 200 ,,	22	58	29
− 200 ,,	33	42	27
Head value g/t	4,4	28	16
Tail value g/t	2,0	19	11
% concentrate	0,08	0,10	0,10
% recovery	55	32	31
Reconcentration by	Wilfley table	F & C table	Wilfley table

the floor space occupied by corduroy tables is generally greater than in the case of other types of concentrators. A further drawback is that in the course of changing and washing the blankets, with the relatively large labour force involved, the opportunity for gold theft is increased. Before the development of automatically operated concentrators, the above disadvantages had to be accepted and corduroy strakes have certainly played a notable part in gold recovery in South Africa. However, in all modern gold plants they have been displaced by one of the mechanical concentrators, or by the installation of all-cyaniding treatment plants in which gravity concentration has been discarded.

Table 5
JIGS

Type of jig	Yuba Richards	Yuba Richards
Cells per unit	2	2
Size of cell	1,07 m × 1,07 m	1,07 m × 1,07 m
Power	2kW	2 kW
Stroke	13 mm	16 mm
Strokes/min.	200	136
Hutch water	-	0,82 m^3/min.
Ragging	6 mm steel shot	6 mm natural grit
Screen aperture	13 mm × 3 mm	13 mm × 3 mm
Hutch spigot diam.	19 mm	13 mm
Feed source	Pebble mill outlet	Pebble mill outlet
Tons solids per hour	60	90
% moisture	28	45
Gradings % + 48 Tyler	18	42
+ 100 ,,	36	27
+ 200 ,,	26	16
− 200 ,,	20	15
Head value g/t	21	7
Tail value g/t	12	4,5
% concentrate	—	2,2
% recovery	43	36
Reconcentration by	Cleaner jigs	James table

Jigs are very robust and withstand arduous conditions with minimum mechanical maintenance. The main advantages of this type of concentrator are the immediate response to control variables and the flexibility with which local fluctuations are absorbed. They do require certain attention but provided bed mobility is well attended to, a clean concentrate can be consistently recovered at low cost. Tramp metal, mainly wire from underground sources, is the most significant nuisance encountered.

Natural ragging comprised of quartz and pyrite can be readily formed on the jig bed screens and indications are that in some instances such ragging produces fewer operating problems than a bed of either steel shot or haematite. Apart from the economy, natural ragging, by virtue of a lower bed density, provides easier dilation and requires less hutch water. Operational control of jigs is based largely on visual inspection of the concentrate discharged from individual hutch spigots.

HEAVY MEDIA SEPARATION

A recent introduction in the sphere of gravity concentration on South African gold mines has been the heavy media separation process. The reason for its adoption was to separate the ore, prior to milling, into high grade and low grade fractions with respect to uranium oxide concentration. By this means the bulk of the uranium content can be segregated in one portion of the ore, which after milling and cyaniding, can be passed to a uranium treatment plant for profitable recovery of the U_3O_8. The success of the operation is due to the sympathetic association of gold, uraninite and iron pyrites in the matrix of the reef. The pyrite is by far the predominant mineral and although it has a much lower specific gravity than the other two, its presence in sufficient quantity can have an appreciable effect on the relative density of the reef. For example, a content of 5% pyrite results in an increase of ore density from 2,70 to 2,76 and therefore if broken ore is passed through a liquid medium with a relative density of 2,72 the pyritic reef will sink and the lighter portion, containing little if any pyrite, and probably low in uranium, will float. Consequently the reef will be separated into high and low density fractions, the high density fraction containing the bulk of the pyrite, gold and uraninite. Also most waste rock, which generally has a specific gravity of 2,68 or lower, will report with the float, thus segregating a diluent of both gold and uraninite grades.

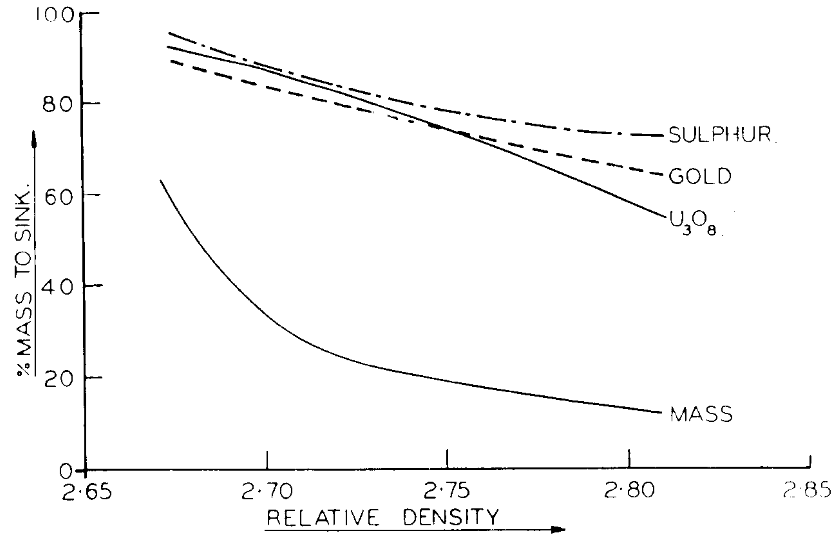


Figure 7. Relative density distribution of mass, gold, uranium and sulphur in a typical pyritic gold ore

In practice the minus 13 mm mill feed is screened at 3 mm, since the undersize is already of payable uranium grade and moreover would be costly to upgrade. Fine size material in the—3 mm range would require extensive and expensive washing to avoid undue medium consumption.

With hydrocyclones as the separatory vessels and a mixture of ferrosilicon and magnetite powder in a water emulsion as the heavy media, a separation of 70% float and 30% sink has been obtained. This percentage split can be varied by slight changes in the specific gravity of the medium. At any predetermined density of separation, however, the float still has a gold content sufficiently high to require milling and cyaniding before disposal to tailings. The sink fraction, containing the bulk of the gold, uraninite and pyrite in the plus 3 mm ore, is combined with the minus 3 mm fines from the crushing and washing plant. The combination is milled and cyanided in a circuit separate from the low grade ore. If considered economic, it can be milled to a finer size or agitated for a longer period than would be the practice for run-of-mine ore. After cyanidation the highgrade slime is acid-leached to extract the uranium oxide and then subjected to flotation to concentrate the pyrite. The pyrite concentrate is subsequently roasted to provide sulphur dioxide for sulphuric acid manufacture.

Basically this application of heavy media separation in gold plants is a uranium beneficiation operation. It does, however, offer the opportunity for the preferential treatment of the high value pyritic gold ore and thus reduce the overall grinding and cyaniding costs. A pilot plant was installed in 1969 on a gold plant in the Orange Free State and this was followed a year later by a full scale plant in the Western Transvaal. A flow diagram of the latter is shown in Figure 8 and relevant data appears in Table 6.

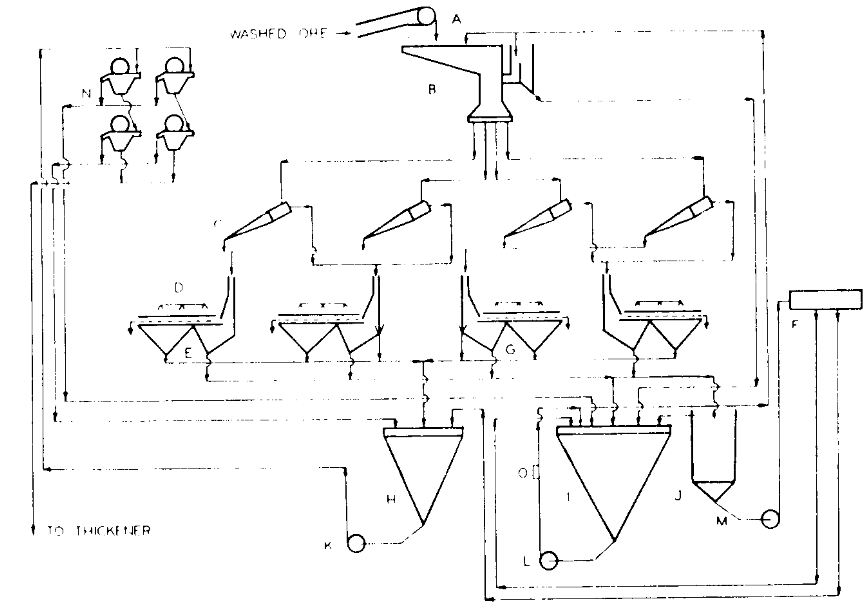


Figure 8. Heavy media circuit flow diagram

WESTERN TRANSVAAL HEAVY MEDIUM 175 TON PER HOUR HYDROCYCLONE PLANT

A. Common mixing launder.
B. Distributor.
C. D.S.M. Cyclones – Four gravity feed 356 mm diameter units.
D. Wash water sprays.
E. Concentrate drain and rinse screens complete with sieve bend fixed – pre-drain screen.
F. Centrifugal densifier.
G. Reject drain and rinse screens complete with sieve bend fixed-pre-drain screen.
H. Dilute medium sump.
I. Circulating medium sump.
J. Centrifugal densifier sump.
K. Magnetic separator feed pump.
L. Circulating medium pump.
M. Centrifugal densifier feed pump.
N. Magnetic separators (primary and secondary).

Scope is still offered in reducing operating costs by installing other types of separatory vessels such as rotating drums in place of hydrocyclones. The cost of pumping −13 mm +3 mm quartzitic ore through hydrocyclones constitutes about 30% of the heavy media treatment costs. As a first step to reduce this figure the incoming ore can be elevated by conveyor belts to a feed tank above the hydrocyclone. Here it is combined with the return medium which is pumped to the tank. The mixture then gravitates to the cyclone entry point under sufficient head to provide an adequate pressure for efficient separation.

Table 6

HEAVY MEDIA SEPARATION

Capacity:		4 200 tons per day	
Cyclone dimensions:	Diameter	356 mm	
	Angle	20	
	Spigot	50 mm	
Specific gravities:	Medium	2,51	
	Cyclone overflow	2,46	
	Cyclone underflow	2,69	
Separation:		Sink	Float
	Mass	31%	69%
	Gold	81%	19%
	U_3O_8	81%	19%
	Pyrite	82%	18%
Pyrite in reef band:		3,0%	
Medium consumption:		0,35 kg/t	

FLOTATION

Introduction

With the high degree of extraction efficiency developed by the cyanide process and the absence of refractoriness in nearly all the gold bearing ores in South Africa, virtually no scope has been offered for the application of flotation methods to gold recovery. From the time flotation was recognised as a major mineral dressing process, periodic attempts were made to adapt it to gold extraction in South Africa, but although reasonably satisfactory recovery percentages were attained, more economic results were achieved by direct cyanidation in the case of all the major ore bodies. The only area where flotation has been applied with good effect is on small mines currently producing in the Eastern Transvaal. Here ores containing pyrrhotite, arsenopyrite and other refractory sulphides are not amenable to direct cyanidation and consequently flotation was introduced to produce a sulphide concentrate which could be roasted to provide a more suitable material for cyaniding.

As far as the main reef on the Witwatersrand is concerned, apart from large scale but fruitless trials in 1924 on the East Rand, the only applications of flotation on a production scale occurred in two cases of a similar type, both in 1935 and both also on the East Rand. Flotation was adopted as a less costly installation than extensions to the sand treatment plant to meet demands for higher treatment

capacity. The increases in the milling rate resulted in the production of greater tonnages of coarser particle size and a corresponding proportion of coarser grains of gold and pyrite. This was particularly the case with regard to the sand fraction of the mill pulp. In the ordinary course of cyanide treatment the extraction efficiency would have decreased, but by treating the higher sand output by flotation a pyritic gold concentrate was obtained from which, after intensive grinding, a high recovery by cyaniding could be achieved. Flotation was conducted in banks of Fagergren cells in an acid circuit (pH 6) with the classical reagents of sodium butyl xanthate and pine oil. A concentrate of approximately 10% of the original feed was obtained and this material was milled by a ball mill in closed circuit with a Dorr bowl classifier. The classifier overflow was cyanided either in Crosse or pachuca tanks, followed by filtration in one case through a separate rotary filter and in the other in the main slime filtration plant. The flotation tailings still contained sufficient residual gold to warrant further treatment and so, after a water wash to remove any xanthate still present, the sand was given a curtailed period of treatment with cyanide solution in the leaching tanks.

Such an installation was less costly than the provision of additional mills, classifiers and sand tanks and also had the advantage of requiring considerably less space – an important aspect in both the two cases concerned. It should be appreciated however, that the key to the successful result was the presence of a relatively high content of pyrite with which the gold was intimately associated. On one plant the pyrite grade was so high that the flotation concentrate was cyanided and filtered separately to produce a pyrite tailing that was sold without further treatment for sulphuric acid production. The two mines under review have ceased mining operations and thus flotation is no longer an appreciable factor in the extraction of gold from the producing mines of the Witwatersrand, Evander area, Western Transvaal or Orange Free State.

However, in the Eastern Transvaal, around the town of Barberton, scene of the early gold strikes in the 1880's, four small producers, with a total output of about 500 kg of gold per month, follow the practice initiated by the Transvaal Gold Mining Estates in 1935 of floating the refractory auriferous sulphides from the mill pulp and roasting the flotation concentrates to produce a calcine amenable to cyanidation. In keeping with the history of gold discoveries in the Barberton area, the reefs or lodes vary tremendously in their mineralogical composition and, therefore, as distinct from the deposits in the rest of South Africa, no uniform type of treatment can be regularly adopted. The gold in this area is intimately associated with complex mineral assemblages of sulphides, dominantly pyrite, but including arsenopyrite, pyrrhotite, chalcopyrite, stibnite, galena, tetrahedrite and arsenides. Graphite is also fairly common. Consequently the ores are invariably refractory, but to different degrees even in the same mining locality. Direct cyanidation is obviously not a suitable process and so economic recovery of the gold is based on the flotation of sulphidic concentrates which can be roasted and the resultant oxides leached in cyanide solution.

Although gravity concentration in the shape of plane and shaking tables is included in the extraction process, flotation is the essential requirement to ensure that the recovery method is viable. Conventional types of reagents are employed and their respective consumptions are not unduly excessive. Flotation is conducted around a pH of 8,5 and concentration ratios are approximately 16:1. Flotation concentrates are roasted in suitable furnaces and the calcines treated by normal cyanidation methods to recover the gold. Although the four Barberton producers do not rank highly in their contribution to the South African gold output they do provide extensive and varied methods of gold extraction and are thus of greater technical interest to gold metallurgists than the main producers in the rest of the Transvaal and in the Orange Free State.

Pioneer work in this regard was undertaken by the Transvaal Gold Mining Estates when a flotation plant of 2 250 tons per day capacity was erected in 1935. This was followed by a second plant of 2 000 tons per day in 1937. Founded in 1896, Transvaal Gold Mining Estates represented an amalgamation of several mining companies in the Lydenburg area and by 1970 was the longest operating gold producer in South Africa. In its earlier days the Company dealt with free-milling ores which were amenable to amalgamation and cyanidation for gold recovery. However, in the early 1930's extraction fell away markedly as refractory ores were encountered in the deeper mining zones. Copper sulphides, graphite and siderite in particular were the causes of poor recoveries. Flotation proved the best means of concentrating the gold values and consequently the mining programme was arranged so that the remaining

oxidised ore was treated at the current cyanidation plant and the increasing quantities of sulphidic and carbonaceous ores were treated in two flotation plants. In the first instance flotation concentrates were dispatched overseas for sale to custom smelters. Subsequently the concentrates were subjected to a sulphating roast in Edwards furnaces. Copper sulphate was dissolved from the calcine and precipitated on scrap iron, the resultant cement copper being exported. The residual calcine was cyanided to recover the gold. The complexity of the complete gold recovery procedure was in striking contrast to the simple all-sliming, all-cyaniding circuits that were being adopted on contemporary installations on the Far East Rand. A further interesting development at T.G.M.E. was the replacement of the Edwards roaster by a fluosolids reactor to calcine the sulphidic concentrates. The sulphur dioxide was used to produce sulphuric acid and thus eliminate a noxious gaseous effluent. The acid was utilised in the production of super-phosphate fertilizer by treating phosphate from a nearby deposit. In a minor way this process duplicated the innovation by Cominco, at Trail, Canada, in establishing the production of Bear Brand fertilizer.

A later application of flotation for gold recovery in South Africa was the installation of a plant on the New Machavie Mine in the Western Transvaal. The plant was erected in 1936 and comprised a 40 000 ton per month flotation plant and an Edwards roaster for treating the concentrates. The roasted oxide was then subjected to cyanidation. Operations ceased at the end of 1943, however, as high mining and treatment costs made the venture uneconomic at the prevailing price of gold.

Pyrite Flotation

The recovery of uranium oxide as a by-product from South African gold ores from 1952 onwards had a marked effect on flotation practice in South Africa. The extraction of uranium oxides from the gold plant residues was accomplished by leaching the slime with sulphuric acid and consequently large quantities of acid had to be provided. By 1956 requirements were of the order of 1 500 tons of 98% acid concentration per day. The sulphur for the acid plants was derived from iron pyrites floated from the gold plant residues and sufficient pyrite was available amongst the gold producers to meet the demand. In many cases a viable grade of pyrites was present in the slime treated for uranium extraction and in this event flotation costs and percentage extractions were highly satisfactory as the slime residue from the uranium plant had been acidified, was completely free of cyanide and was readily amenable to flotation. A similar situation applied in the case of slime recovered from old tailings dams, as a slight oxidation of the pyrite in the dam had resulted in an acid pulp being formed. However, not all gold mines had a payable uranium grade nor did uranium bearing slime residues necessarily contain a pyrite content worthy of extraction. Thus demands for pyrite were made on some mines where the ore had a reasonably high pyrite grade, but an uneconomic uranium value. In such cases flotation costs were markedly increased due to the necessity of acidifying the alkaline pulp received from the gold plant in order to reduce the pH value to 6,0 and to eliminate cyanide from the pulp solution. An alkaline pulp of pH 10,5 was an essential requirement for gold dissolution by cyanide and to attain this alkalinity about a kilogram of unslaked lime was added to each ton of ore milled. Consequently, an amount of 2 to 3 kilograms of sulphuric acid was subsequently needed to neutralize the alkalinity in the flotation plant feed and the expenditure on this acid formed a considerable part of the flotation costs. Fine iron oxide derived from mill liner wear and in many cases from rod or ball attrition was also a source of acid consumption. Therefore where this situation arose considerable attention was paid to the possibilities of floating iron pyrites from the ore before cyanidation and thus forestall the depressant effect of cyanide. Successful results were obtained in floating primary classifier overflow pulp in a few milling plants at a pH of 10,5 with gradings between 35% and 40% −200 mesh Tyler and a pulp moisture of approximately 45%, i.e. a specific gravity of 1,58. Copper sulphate, xanthates or amines, Aerofloats 25 and 65 and cresylic acid were the reagents generally favoured, and these produced satisfactory pyrite recoveries and concentrate grades. The flotation concentrate, containing about 75% of iron pyrites, was reground in a ball mill before cyaniding and filtering to extract the gold content which was generally of the order of 300 grams per ton. The flotation tailings passed to the secondary milling circuit and thence to the cyanide plant. The presence of xanthates and, to a lesser degree, amines, in the cyanided slime, however, had an adverse effect on gold recovery due to adsorption on the surface of the gold particles and measures had to be found to overcome this drawback. The most satisfactory method was to keep the xanthate or amine concentration in the cyanide solution to a minimum, the cyanide concentration as high as economically possible and the pH value of the pulp between 9 and 10 before and during cyanidation. In a few cases when certain types of clay such as pyrophyllite were present in the ore the presence of amines had no adverse effect, presumably due to a desorbing action by the clay particles.

Whatever procedure is adopted to extract iron pyrites from gold bearing ores, a small but appreciable gold content, from 3 to 5 grams per ton, is contained in the concentrate delivered to the acid plants, despite the fine grinding and intensive cyanide treatment afforded beforehand. After roasting to obtain sulphur dioxide for acid production, the residual calcine, mainly iron oxide in the ferrous state, is amenable to cyanidation for gold extraction and approximately 80% of the gold content of the calcine can be recovered. Thus gold extraction has become a by-product of the uranium oxide recovery process, although the output only extends to a few kilograms per month on the mines concerned. Nevertheless, the small cost of leaching and filtering the iron oxide makes its treatment a profitable venture. An additional source of revenue can be the extraction of uranium oxide from the flotation concentrate as some of the uraninite in the normally unpayable uranium ore reports in the pyrite concentrate.

With new gold mines entering production and an enhanced demand for sulphuric acid both in the fertiliser and industrial fields, flotation of pyrite from gold bearing ores is expanding to an even greater degree than when the uranium programme initiated a new phase in extraction metallurgy in South Africa.

Although flotation is being applied to the upgrading of uraninite in some gold ores, using organic phosphates as collectors, this practice will not be discussed as the operation has no bearing on gold extraction.

FLOTATION PRACTICE

The Hartebeestfontein Flotation Plant

The original object of pyrite recovery at Hartebeestfontein was to supply the nearby pyrite-burning sulphuric acid plant at Stilfontein. In addition to the revenue from the sale of the pyrite, the return of the calcine for further gold and uranium recovery seemed to be financially attractive. Jigging and plane tabling were not considered to be sufficiently efficient concentration methods.

Flotation of the cyanide circuit tailings was not attractive because of the fine grind at Hartebeestfontein, the high cost of acidifying with acid, and the well-known deleterious effects of a combination of lime and cyanide on flotation. Laboratory testwork showed, however, that good recoveries could be obtained by floating at a pH as high as 11 at the end of the second stage of the three-stage Hartebeestfontein milling circuit. Hartebeestfontein ore has a high content of sericite and it was originally considered necessary to operate a two-stage flotation process, the first stage with frother only to remove most of this mineral and up to 85 per

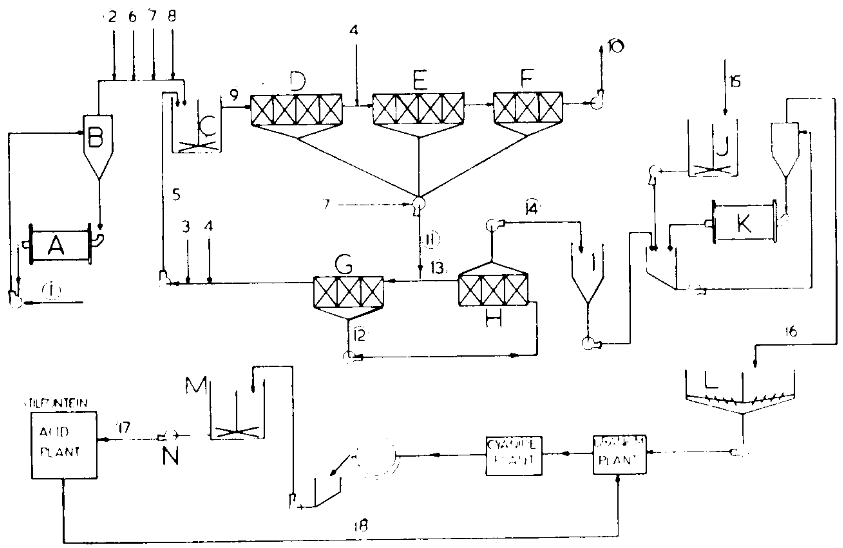


Figure 9. Pyrite flotation and treatment flow diagram (Hartebeestfontein)

FLOW DIAGRAM DATA. FIGURE 9.

A. Secondary pebble mills.
B. Secondary cyclones.
C. One 3,66 × 3,66 m mechanical agitator.
D. First stage roughers. Two banks each of 7 Fagergren cells.
E. Second stage roughers: Four banks each of 9 Fagergreen cells.
F. Third stage roughers: Four banks each of 6 × 1,13 m³ Agitair cells.
G. Cleaners: Two parallel banks each of 4 × 1,13 m³ Agitair cells.
H. Recleaners: Two parallel banks of 4 × 1,13 m³ Agitair cells.
I. Three stock tanks
J. Two mechanical agitators.
K. One 1,98 × 2,44 m ball mill.
L. Two 15,24 m diameter thickeners.
M. One mechanical agitator.
N. Five 152 mm rubber-lined pumps.
1. Rod mill effluent.
2. Frother.
3. Copper sulphate.
4. Xanthate.
5. Cleaner tails.
6. Aerofloat 25.
7. Sulphuric acid
8. Mill water.
9. Float feed.
10. Rougher tails to tertiary pebble mills.
11. Rougher concentrate.
12. Cleaner concentrate.
13. Recleaner tails.
14. Final pyrite concentrate.
15. Pyrite concentrate from Zandpan plant.
16. Combined milled pyrite.
17. Pyrite slurry pumped to Stilfontein acid plant
18. Calcine slurry returned to Hartebeestfontein reduction plant.

cent of the gold, and the second stage to recover the pyrite. The first-stage "slime" float tailings joined the second-stage tailings. It was subsequently found that this two-stage process was unnecessary and the present conventional rougher-cleaner-recleaner circuit has been found to give remarkably high recoveries of the pyrite.

The plant was designed to treat 131 500 tons per month but after changing to conventional one-stage pyrite flotation and with some alteration, the plant is presently handling up to 150 000 tons per month. It was commissioned in October, 1966.

Figure 9 is a flow diagram of the plant and Table 7 gives operating data.

Cyclone overflow with an average S.G. of 1,4 from the second stage milling is adjusted automatically to S.G. 1,3 and the pH maintained at 9,9 by an automated sulphuric acid feed. Copper sulphate is essential for successful flotation and this is added together with a frother. Aerofloat 25 and xanthate before a conditioning period of 4,1 minutes. Rougher flotation takes place in three stages in series with a total residence time of 10,6 minutes. There is no particular design significance to the three-stage roughers and a single-stage would no doubt be equally effective. The first stage was in fact the original "slime float" rougher bank. Additional xanthate is also added after the first rougher bank.

The rougher tailings return to the milling circuit via the tertiary pebble mills. Sulphuric acid is added to the rougher concentrate to control the pH of the feed to the cleaner cells at 9,0. A low pH produces an excessively high sulphur grade and a high pH decreases froth mobility during cleaning and recleaning operations.

Features in the design of the plant are:—

1. Flotation reagents are fed by positive displacement metering pumps and these have been found to be very accurate and reliable.

2. All plant control samples are taken automatically and each rougher bank is provided with a continuously operating shaking table for a visual indication of the pyrite content of the tailings.

3. Due to the high slime content of the pulp and the stable frothers used, the froth is difficult to handle. Extra large froth launders up to 0,46 m wide and 1,42 m deep and with 0,25 m diameter outlets have been installed. All pumps pumping froth have been generously designed.

Table 7

AVERAGE FLOTATION PLANT OPERATING DATA

Flotation feed:	Tons/month	150 000
	Specific gravity	1,30
	−200 mesh Tyler	38,9%
	pH	9,9
	Sulphur	1,8%
Reagent consumption kg/t:	Sulphuric acid	0,101
	Secondary butyl xanthate	0,034
	Dow froth 250	0,005 5
	Aerofloat 25	0,001 5
	Copper sulphate	0,042 5
Concentrate:	Sulphur	40,0%
	−200 mesh Tyler	65%
	−325 mesh Tyler	50%
Recoveries:	Mass	3,5%
	Pyrite	90,0%
	Gold	79,4%
	Uranium	28,9%

The Fairview Flotation Plant

The procedures followed at the Fairview Mine in the Eastern Transvaal are briefly described and should be read in conjunction with Figure 10.

About half the required quantities of flotation reagents are added to the ball mill discharge sump before the milled product is pumped to the 0,30 m diameter hydrocyclone depicted under the milling circuit. This primary cyclone overflow, at 50% minus 200 mesh, gravitates to two No. 500 Denver unit cells. The unit cells concentrates are cleaned in one No. 5 Denver flotation cell and the cleaned concentrates gravitate into a 9,14 m diameter concentrates thickener.

Tailings from the two unit cells are treated in parallel through two rougher and scavenger circuits each consisting of a 1,83 m × 1,83 m conditioner, (where the rest of the flotation reagents are added), four No. 18 Denver rougher cells and then four 1,12 m. Fagergren cells in one case and four No. 18 Denver cells in the other case for scavenging. The scavenger tailings are pumped through a 0,30 m hydro-cyclone where the cyclone underflow is returned to the milling circuit and the cyclone overflow is disposed of to the tailings dam at 65% minus 200 mesh Tyler. The combined rougher concentrates from the two circuits are cleaned in two No. 18 Denver cells, and these concentrates gravitate to the 9,14 m diameter thickener. The flotation middlings, consisting of scavenger concentrates, cleaner tailings and unit cell cleaner tailings are returned for re-

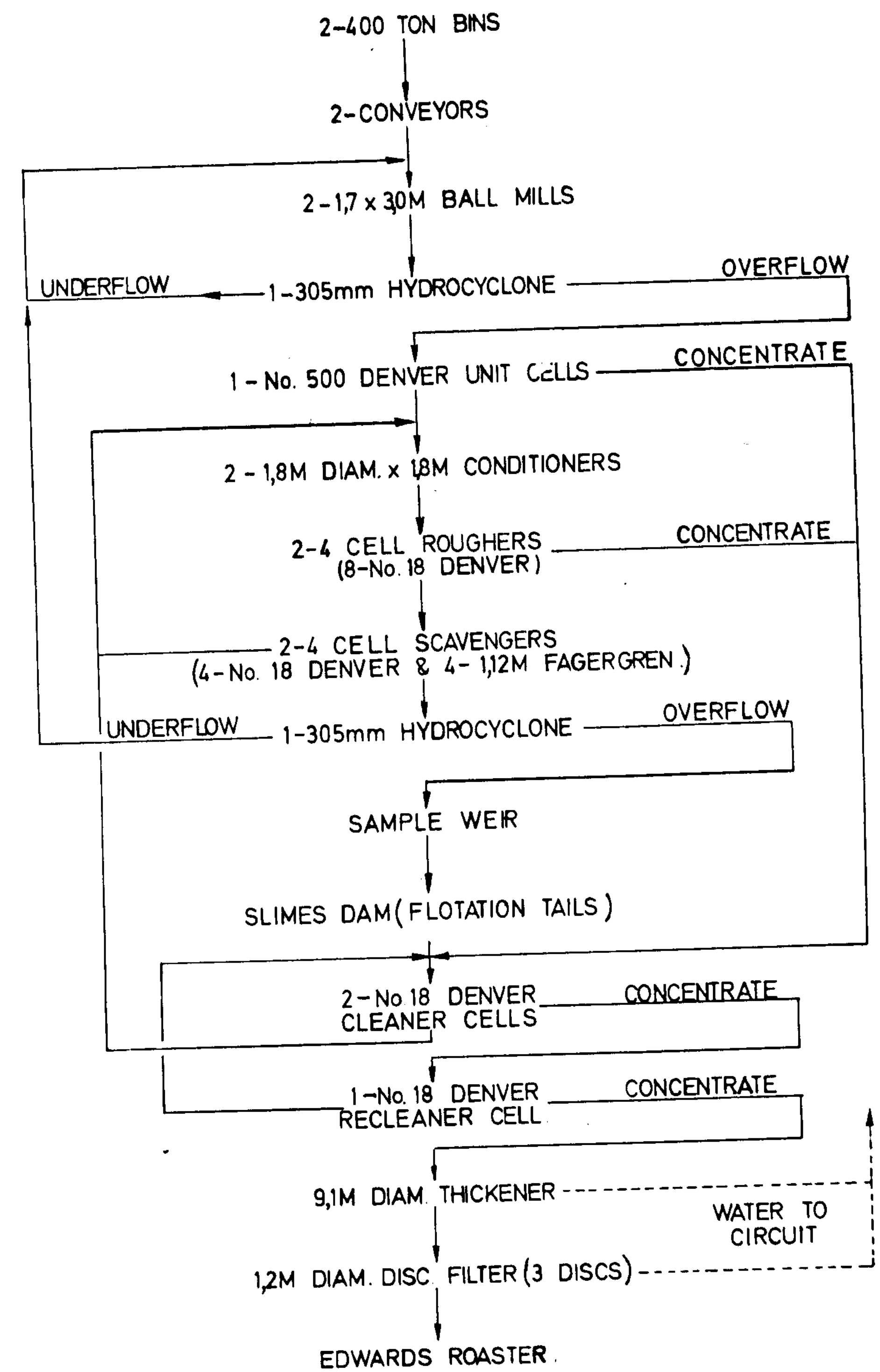


Figure 10. Flow diagram of Fairview plant