specific gravity balance. If the base specific gravity of the ore is accepted as 2,7—which is generally the case—the specific gravity figure may also be read as percentage moisture by weight. Primary pebble mill effluent specific gravities are normally between 1,85 and 1,79, i.e. 27% and 30% moisture. The corresponding figures for ball-mills are 2,07 to 1,92 or 18% to 25%, while for secondary pebble mills they are 1,79 to 1,69 or 30% to 35%. Rod-mills operate in a similar range to ball-mills, viz. 2,01 to 1,92 or 20% to 25%.

CLASSIFICATION

Interlocked with the operation and efficiency of ball and pebble mills is the performance of the hydraulic classifiers which are installed to provide closed milling circuits. The function of the classifier is to separate the acceptable size of milled product from the oversize material. The former is passed on to the next stage, either finer milling or treatment, while the latter is returned to the mill for further grinding. This return constitutes the circulating load and is usually measured as a ratio: tonnage returned divided by the tonnage of new feed to the mill. Circulating loads vary from 0,5:1 to 4:1 and their careful control is necessary for efficient milling. This leads, therefore, to a consideration of classifying operations.

Development

When stamp milling was practised on the Witwatersrand during the period 1886 to 1890, the fineness of the crushed particles of ore produced by the stamps was determined by the apertures of the rectangular screens set vertically in the front of each mortar box. These screens were woven steel or brass wire cloth put in frames approximately 1,30 m long and 0,25 m high, with the screen openings varying from 3,0 mm down to 0,4 mm square mesh, 0,75 mm being a common size. The pulp passed through the screens on to amalgamation plates and then to the tailings dump. With the introduction of the cyanide process, it became necessary to separate the amalgam tailings into sand and slime. Initially, the sand was leached with cyanide solutions in vats or tanks for gold recovery while the slime was discarded because these very fine particles could not be percolated by the solution. Processes were soon developed, however, to treat the slime by cyanidation, but as these were not suited to sand

treatment it was necessary to effect an efficient sand and slime separation.

Screening did not lend itself to this operation owing to the fine particle sizes involved and, therefore, hydraulic classifiers were introduced in the shape of spitzkasten which had been developed by Professor von Rittinger in 1866. These comprised wooden boxes having the shape of inverted pyramids with a narrow diameter nozzle at the apex. (Figure 10.) The stream of unsized pulp entered from one end and while the finer ore particles overflowed at the opposite end of the box the coarser fragments settled to the apex of the pyramid and were discharged through the nozzle. A refinement of the operation was the introduction of a controllable ascending current of clean water injected at or near the apex of the pyramid to wash the sand clear of adhering slime and also to displace muddy water from the down-flow. In this design the classifier was termed a spitzlutte. A considerable amount of attention had to be paid to the rate of flow, siting of baffles, size of nozzle, volume and pressure of water added.

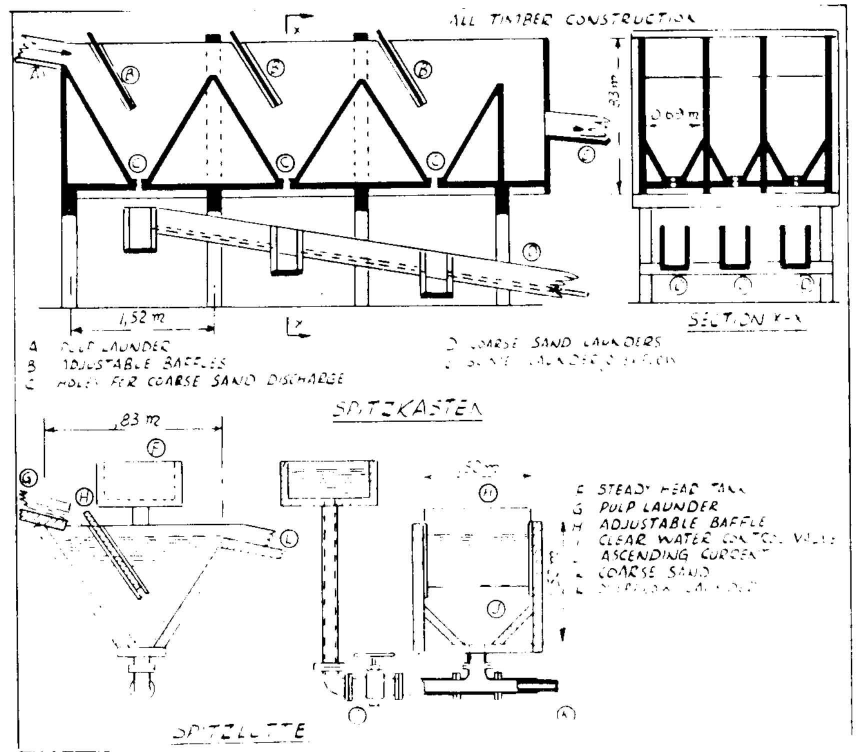

Figure 10. Spitzkasten and spitzlutte.

With the advent of tube-milling and the concept of a closed circuit the large area required for these classifiers created difficulties both in design and operation and consequently they were replaced by 60° cones made of steel plate. These were originally entitled Callow cones, in honour of their inventor, but were subsequently referred to as Caldecott cones in recognition of the improvements introduced by W. A. Caldecott. The cones were usually 2,44 m in diameter at the top and 3,05 m deep and could handle up to 1 000 tons of solids per 24 hours of the circulating pulp. Essential features were the control diaphragm near the apex and the cut-off gate for regulating the underflow.

The cone classifier was superseded by the Dorr rake classifier when the all-sliming tube milling circuits were introduced in 1921 and later the Dorr bowl classifier proved the most satisfactory secondary separator when stage milling was adopted in 1928. Other mechanically operated hydraulic classifiers were tried in place of the Dorr types, but the only one to find general favour was the Akins spiral classifier, with either single or twin helices. The spiral classifier returned a sand or grit with a slightly lower percentage moisture than the rake classifier and, therefore, was preferred where this feature was particularly desirable. A further advantage subsequently appreciated was that in a closed primary milling circuit the spiral shaft and tank could be lengthened and steepened thus permitting the mill discharge to gravitate to the classifier tank while the return product was elevated sufficiently by the spiral to report directly to the mill inlet. This procedure eliminated the need for a pulp pump to elevate the mill discharge to the classifier.

A major break-through in the mechanics of hydraulic classification was achieved at Rand Leases reduction plant in 1951 when a 27 inch diameter, 20° hydrocyclone was substituted for an 18 foot diameter bowl classifier in the secondary milling circuit. While hydrocyclones had previously been utilized in water clarification and coal cleaning, nowhere in the world had its application as a classifier in a grinding circuit been considered until the experiment at Rand Leases was initiated. The results were so spectacularly successful that hydrocyclones were rapidly adopted as classifiers in most South African gold milling plants, firstly in secondary circuits and then in primary circuits. Thereafter they readily received world wide acceptance in many types of milling installations for closed circuit classification.

The low cost of these units, the small space occupied and the ease of installation resulted in the discarding of many mechanical classifiers, particularly in secondary and tertiary circuits. Amongst the advantages gained by the use of hydrocyclones were reduced maintenance costs, less lock-up of gold in the milling circuits, minimum volume of pulp in circulation, thus facilitating the stopping and starting of mills, smaller surge sumps and a greater degree of selective grinding due to a higher mineral concentration in the cyclone underflow. In the design of new plants, a further capital saving resulted from the reduced area and housing required, a feature which is particularly important when large diameter mills are installed—in fact it is probable that limitations might have been put to the size and resultant throughput of cylindrical mills if hydrocyclone classifiers had not been available to replace mechanical types.

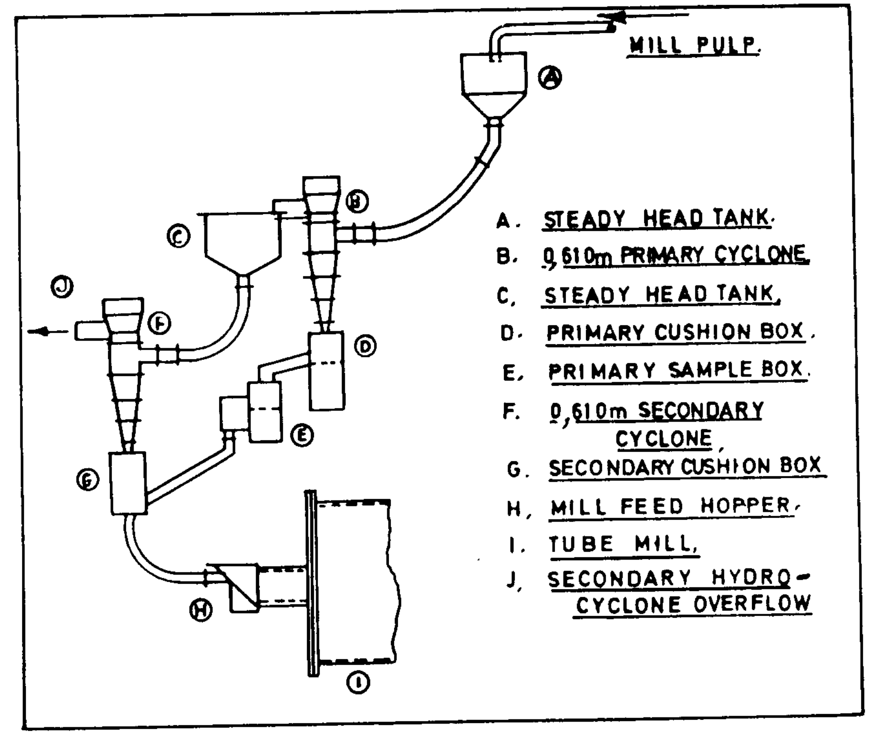

Figure 11. Duplex hydrocyclone classification circuit

Hydrocyclones have therefore been accepted as the standard type of classifier in all modern milling circuits on South African gold reduction plants.

The decision regarding the size of the classifiers to be installed for an extension or in a new plant is usually based on information available from existing plants. Should, however, a special case arise, then theoretical calculations or manufacturer's tabulations can be used as a base for the design of a pilot installation. If a new mill is under consideration, the pilot plant can be set up on an adjacent producing mine and the design sizes verified or amended to ensure that the correct hydrocyclones are provided for the new installation. With gold recovery enjoying the rare distinction of being a non-competitive business, no difficulties arise in obtaining the co-operation of neighbouring producers with regard to experimental operations. The application of classifier efficiency formulae is rarely required on producing mills as the through-put of the various milling and classification units is constant within very narrow limits. Any untoward feature in classification is promptly detected by means of the moisture and grading samples that are taken over each shift.

The formula for classifier efficiency usually applied is:

$$E = \frac{10\,000\,(c-f)\,(f-t)}{f\,(100-f)\,(c-t)}$$

Where E is the percentage efficiency;

c is the % undersize in the overflow;
f is the % undersize in the feed;
t is the % undersize in the underflow.

Calculations on classification efficiency are discussed in detail in Section 19 of the Handbook of Mineral Dressing by A. F. Taggart.

CLASSIFIER PERFORMANCE

1. *Hydrocyclones*

These are used for dewatering crusher station washings, as primary classifiers for ball and pebble mills, as secondary classifiers with pebble mills, as tertiary classifiers, as dewatering classifiers for concentrator tailings and occasionally for partial dewatering or thickening units ahead of the slime treatment plant. From the time they were generally adopted in the nineteen-fifties, a 20° cone angle has proved most suited to all the above services as far as South African gold ores are concerned. Cyclone sizes are stated in terms of the diameter of the cylindrical sections. Sizes commonly used vary from 0,30 metre to 1,14 metres, the most frequent

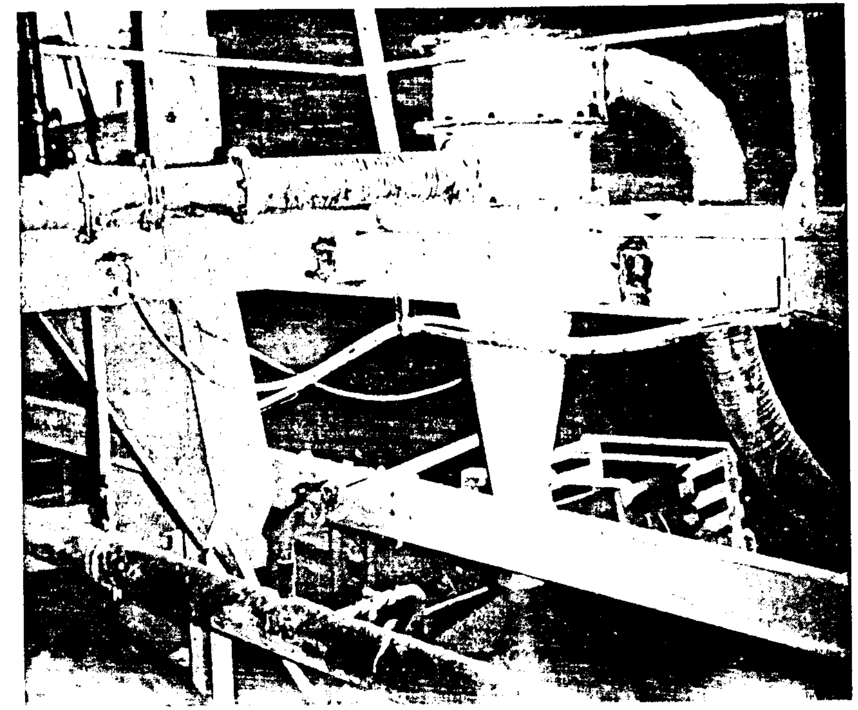

Figure 12. Pioneer hydrocyclone classification installation. Rand Leases 1951

sizes being 0,61 m, 0,76 m and 0,91 m. Spigot diameters, vortex finder dimensions, inlet and outlet areas are not necessarily standard for a particular size of hydrocyclone and vary according to the demands of each application. Hydrocyclones are usually fabricated in steel plate and lined with a suitable type of rubber to reduce wear by abrasion, but in a few cases they are cast in wear-resisting metal alloys. The pulp to be classified is either introduced into the cyclones from a pulp pump or—more satisfactory if the layout is suitable—from a steady-head tank which is maintained at constant level by an overflow back to the pump that elevates the pulp to the steady-head. Whichever system is employed, a sump level sensor which controls the mill feed rate, or the dilution to the pump sump, is advisable for fully efficient operation. By installing units of generous capacity, inlet pressures are kept low—of the order of 35 kP (5 p.s.i.)—and thus wear and tear of the lining of the spigot is greatly reduced.

The inlet pressure used is a compromise between the increased efficiency obtained with a high pressure (giving greater velocity and higher centrifugal force inside the

Table 4. HYDROCYCLONE DATA

Nominal Size		18″	21″	24″	27″	30″	36″	42″	45″
Diameter of cylinder	m	0,457	0,533	0,610	0,686	0,762	0,914	1,07	1,143
Inlet dimensions	mm	140 × 160	203	152	165 × 203	203	254	203 × 305	356
Outlet diameter	mm	127	203	762 × 381	254	254	254	300	406
Spigot diameter	mm	57	76	89	67	76	38	6,3	83
Vortex finder	d × l mm	178 × 378	178 × 451	229 × 457	197 × 331	203 × 508	304 × 406	305 × 686	355 × 710
Cone angle		20°	20°	20°	20°	20°	20°	20°	20°
Feed source		Primary mill discharge	Ball mill discharge	Primary mill discharge	Secondary milling circuit	Rod and pebble mill outlets	Tertiary milling circuit	Tertiary mill outlet	Primary cyclone overflow
Feed rate (solids)	tons/hr	75	95	130	80	190	60	50	187
Overflow tonnage	,,	25	40	30	23	70	50	27	114
Underflow tonnage	,,	50	55	100	57	120	10	23	73
Inlet moisture	%	66	32	31	75	50	91	54	77
Outlet moisture	%	83	45	46	90	66	92	65	84
Spigot moisture	%	35	18	25	37	30	40	26	36
Inlet pressure	bars*	0,69	0,69	0,52	0,52	0,24	0,34	0,48	0,55
Inlet gradings (% Tyler):†	+ 48	35	56	52	10	16	—	7	1
	+100	26	9	14	23	19	2	21	19
	+200	21	11	16	41	18	20	37	30
	−200	18	24	18	26	47	78	35	50
Outlet gradings (% Tyler):	+ 48	13	31	25	—	—	—	—	—
	+100	26	12	18	10	7	1	13	2
	+200	25	17	16	28	22	17	39	23
	−200	36	40	41	62	71	82	48	75
Spigot gradings (% Tyler):	+ 48	47	74	61	14	25	1	10	3
	+100	26	7	12	28	25	3	35	47
	+200	19	7	16	46	17	34	35	37
	−200	8	12	11	12	33	62	20	13

* 1 bar = 0,1 Mega Pascal = 14,5 psi

† Tyler mesh: 48 = 0,295 mm, 100 = 0,147 mm, 200 = 0,074 mm aperture

cyclone), and the reduced rate of wear obtained with a lower pressure. The diameter of the cyclone used at a particular feed pressure is determined by the grading required in the overflow, while the dilution of the feed determines the efficiency of the unit as a size separator. The latter parameter has the effect of giving a coarser overflow at lower dilution due to reduced efficiency.

Table 4 gives operating data for typical hydrocyclone performances.

2. *Akins Spiral Classifiers*

Both the simplex and duplex types are used in some milling circuits and, as already mentioned, can be installed alongside a mill with the mill discharge gravitating into the classifier tank and the elevated spiral return gravitating into the feed hopper at the mill inlet, thus eliminating the need for a pulp pump between the mill and classifier. Operating results currently obtained are given in Table 5.

Table 5

AKINS SPIRAL CLASSIFIERS

Type and size		Duplex 1,22 m	Simplex 1,37 m	Simplex 1,98 m
R.p.m.		4	6	1,5
Motor kW		11,19	4,47	18,64
Feed Source		Secondary mill outlet	Ball mill outlet	Ball mill outlet
Feed rate solids t.p.h..		65	75	74
Overflow t.p.h..		25	30	47
Spiral return t.p.h.		40	45	27
Entering moisture %		49	46	60
Overflow moisture %		66	63	70
Spiral return moisture		27	26	24
Entering gradings (% Tyler):	+ 48	17	41	53
	+100	24	27	17
	+200	29	12	11
	−200	30	20	19
Overflow gradings (% Tyler):	+ 48	1	15	42
	+100	6	27	20
	+200	25	20	13
	−200	68	38	25
Return gradings (% Tyler):	+ 48	31	60	71
	+100	30	24	15
	+200	31	8	8
	−200	8	8	6

Table 6

DORR RAKE CLASSIFIERS

		D Style	HX	D.S.F.
Type		D Style	HX	D.S.F.
Rake width—metres		1,22	1,12	2,44
Strokes per minute		16	16	28
Power—kW		5,60	14,91	5,60
Feed source		Primary mill discharge	Crusher washings	Crusher washings
Feed rate t/h		23	22	43
Rake return t/h		4	7	35
Overflow t/h		19	15	8
Entering moisture %		36	70	55
Overflow moisture %		37	80	84
Return moisture %		30	15	18
Entering gradings	+ 48	38	55	60
(% Tyler):	+100	22	23	15
	+200	15	12	11
	—200	25	10	14
Overflow gradings	+ 48	29	9	1
(% Tyler):	+100	23	24	20
	+200	15	23	36
	—200	33	44	43
Return gradings	+ 48	71	76	74
(% Tyler):	+100	12	15	14
	+200	7	5	6
	—200	10	4	6

3. *Dorr Rake and Bowl Classifiers*

In the past these hydraulically controlled mechanical classifiers found considerable application in gold milling circuits in South Africa, but since 1954 have been almost completely displaced by hydrocyclones. Both the straight rake type and the rake cum bowl design had two operating advantages—surge capacity with regard to circulating loads and clear visual evidence of the size and sizing of the circulating load. However, they occupied a considerable floor space, required some degree of maintenance and absorbed a considerable amount of gold, particularly in the case of bowl classifiers. This gold lock-up was not only the source of a loss of interest on unrealized gold, but presented a security hazard with regard to gold theft, especially during maintenance periods. Rake classifiers were used in primary milling circuits and bowl classifiers in secondary and tertiary

circuits. The most common types used were duplex rakes, either 0,91 m or 1,22 m wide and bowl classifiers with 1,22 m duplex rakes and 6,10 m diameter bowls.

Rake classifiers still have application in crusher station washing plants to effect an initial classification of the grit and slime washed from the ore. Operating data for Dorr rake classifiers currently in use are given in Table 6.

MILL SIZES: METRIC EQUIVALENTS

Metric	*Imperial*
1 metre	3,281 ft. (1 ft. = 0,3048 m)
1,68 m × 6,71 m	5 ft. 6 in. × 22 ft.
1,98 m × 2,74 m	6 ft. 6 in. × 9 ft.
1,98 m × 3,35 m	6 ft. 6 in. × 11 ft.
1,98 m × 3,66 m	6 ft. 6 in. × 12 ft.
1,98 m × 6,10 m	6 ft. 6 in. × 20 ft.
2,44 m × 2,44 m	8 ft. × 8 ft.
2,44 m × 2,74 m	8 ft. × 9 ft.
2,44 m × 3,66 m	8 ft. × 12 ft.
2,44 m × 4,88 m	8 ft. × 16 ft.
2,44 m × 6,10 m	8 ft. × 20 ft.
2,74 m × 3,05 m	9 ft. × 10 ft.
2,74 m × 3,66 m	9 ft. × 12 ft.
2,74 m × 3,81 m	9 ft. × 12 ft. 6 in.
2,74 m × 6,10 m	9 ft. × 20 ft.
3,66 m × 4,27 m	12 ft. × 14 ft.
3,66 m × 4,88 m	12 ft. × 16 ft.
4,27 m × 4,88 m	14 ft. × 16 ft.
4,27 m × 6,10 m	14 ft. × 20 ft.
4,27 m × 6,71 m	14 ft. × 22 ft.

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CHAPTER 3

CONCENTRATION PROCEDURES

GRAVITY SEPARATION AND AMALGAMATION

From ancient times, as far back as 4 000 B.C., gravity concentration has played a large part in the recovery of gold particles from sand or river gravel. Gold found as nuggets could be hand picked, vein gold could be released from its matrix by hammering and sorting, but gold occurring as fine grains in alluvial deposits had to be concentrated before handling. This could be effected by washing the material in a stream of water either across sloping flat rocks or along inclined troughs or over animal hides spread along the bottom of a ditch. Most of the gold particles were retained on these surfaces as a rich concentrate while the sand flowed away with the water. The concentrates were periodically collected by hand, dried and then smelted. The great difference between the specific gravity of gold (19,3) and that of the gangue (2,6 to 2,75) resulted in a high degree of concentration and of extraction. Consequently, where gold is found in particle sizes between 600 and 30 microns, this type of concentration is still effectively employed and has application both to crushed conglomerates in South Africa and to alluvial sands elsewhere.

Amalgam Plates

However, when mining operations first commenced on the Witwatersrand, the method of extracting gold from the ore was by stamp-milling and amalgamation. This process followed the current overseas practice in treating what was known as "free-milling" ore. The pulp from the stamps was passed over mercury coated copper plates 4,57 m long by 1,52 m wide with an 18% slope. In many cases, an additional zone of amalgamation was provided by installing amalgam plates inside the stamp mill mortar boxes into which mercury was periodically added. It was claimed that this procedure ensured maximum contact of the gold particles with the mercury. However, it had the drawbacks both of reducing the capacity of the stamps owing to the enlargement of the mortar boxes in order to accommodate the internal plates, and also of accumulating gold amalgam in the bottom of the boxes which could only be recovered periodically when the worn dies under the stamps were replaced. It therefore became the general practice to place in front of each five stamp battery a single amalgam plate on which the gold particles in the crushed mill pulp amalgamated with the mercury coating. At suitable intervals, dependent on the richness of the ore, the stamps were stopped and the gold amalgam, in the shape of a stiff paste, was scraped off the plates. It was then conveyed to the "clean up" room for cleaning and subsequent retorting. The plates were then "dressed" with a fresh application of mercury and stamp milling was resumed. Despite the regular scraping of the plates, a hard layer of gold amalgam gradually accumulated on the plates and therefore in cycles of about three months, steam was applied to each of the plate surfaces to soften this hard amalgam, whereafter it was scraped from the surface without, however, exposing the copper.

Even with the introduction of cyanidation in 1890, the amalgamation of stamp mill pulp was still retained in order to recover as much gold as possible at an early stage, since both sand leaching and the subsequently introduced cyanide treatment of slime required several days to extract the gold. Once tube milling was introduced as an adjunct to the stamp mill, however, the coarser pulp emerging from the stamps was no longer suited to amalgamation and the amalgam plates were transferred to the outlet of the tube mills where a more amenable material could be treated. With the introduction of closed circuit classification, a further refinement was to install additional plates on which newly exposed particles of gold in the classifier overflow could be amalgamated prior to sand treatment.

Corduroy Concentration

This type of amalgamation continued to be practised until 1922 when as a result of satisfactory experimental investigations on several plants, the amalgam plates were replaced by corduroy strakes. In the new procedure, although the mill outlet pulp was still passed over sloping tables, three or four per mill, the particles of gold were not directly amalgamated, but were trapped in the riffles of the corduroy cloth which covered the table tops. The advantage of this method was that the corduroy cloths could be left un

attended in position for some hours, usually four. Thereafter they were removed and washed in tubs to release the concentrates that had collected in the riffles. Meanwhile fresh cloths were laid on the tables.

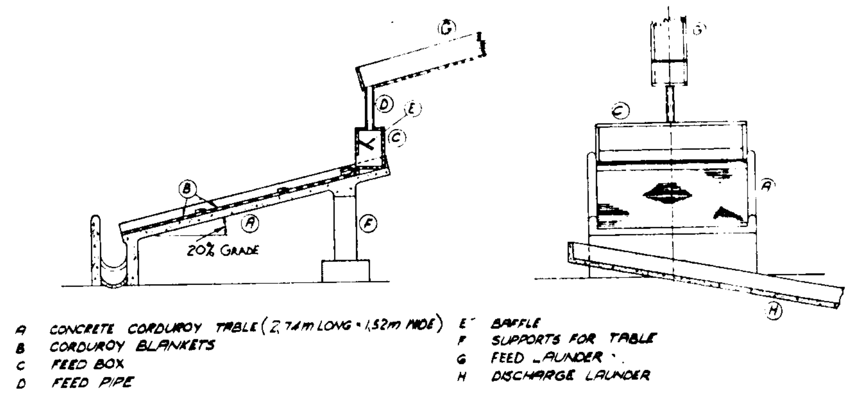
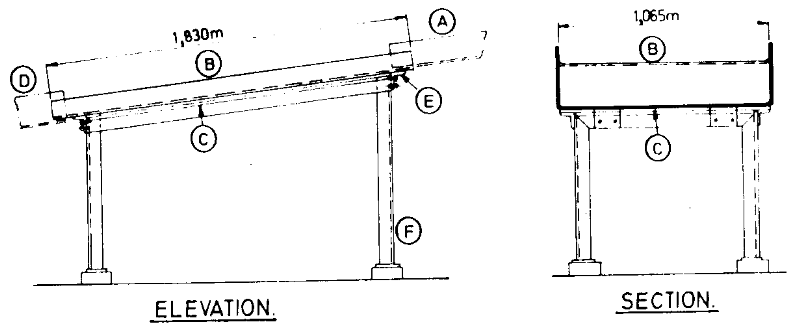


Figure 1. Corduroy strake

Initially corduroy tables were placed at the tube mill outlets but later further tables were installed in the classification circuit and as a result of the greater area provided, cloth washing at longer intervals was possible. The concentrates from the washing tubs were removed each morning and reconcentrated on a shaking table. The tailings from this table were returned to the tube milling circuit while the concentrates were introduced into a 0,61 m diameter × 0,91 m long amalgam barrel loaded with steel balls or other suitable grinding media. The barrel was rotated for a period ranging between 14 and 18 hours in order to liberate encased gold from suphides and to polish the surfaces of the free gold particles. Suitable quantities of lime, caustic soda, cyanide and detergent were usually added to the concentrates to assist in the cleaning and polishing of the surfaces of the free gold. An adequate amount of mercury, of the order of 30 kg, was then poured into the barrel and rotation continued for a few hours during which period the mercury and gold formed a soft amalgam. After amalgamation was completed the barrel was emptied of its contents and these were slowly sluiced over a single amalgam plate. The soft amalgam was deposited on the plate and after the iron had been removed by means of a magnet and pyrite particles washed away, the gold amalgam was scraped by hand from the plate surface and then cleaned, filtered and retorted.

To obviate the labour involved in scraping the plate the load in the barrel could be poured into an oscillating batea and the amalgam collected in the concave pan while the residual material was washed away. The amalgam was similarly filtered in a press to remove excess mercury prior to retorting.



A INLET LAUNDER.
B COPPER PLATE TABLE.
C MILD STEEL PLATE.
D DISCHARGE LAUNDER.
E COPPER CLEATS.
F STEEL SUPPORTS FOR TABLE.

Figure 2. Amalgam plate used in recovery house

The replacement of amalgam plates by corduroy strakes resulted in a considerable saving in both mercury and labour, also a reduction in the danger of theft and the elimination of salivation, a type of mercury poisoning contracted by operators engaged in steaming the plates.

Although constituting a satisfactory means of collecting the coarser gold particles, corduroy tables required a fairly extensive employment of unskilled labour to wash the cloths, thus presenting a security hazard with regard to gold theft. They also occupied a comparatively large floor space.

Automatic Concentrators

Considerable attention was therefore given to mechanising the procedure. The first practical device to be brought into production was the Johnson concentrator, introduced in 1926. It consisted of a slowly rotating cylinder, 0,91 m diameter by 3,66 m long, its axis sloping at about 10%. The discharge from the tube mill flowed through the interior of the cylinder which was lined with corduroy cloth to trap the high density constituents of the pulp. These were washed out at the apex of the revolution into a suitably placed launder and either passed to a regrinding circuit or to the recovery house.

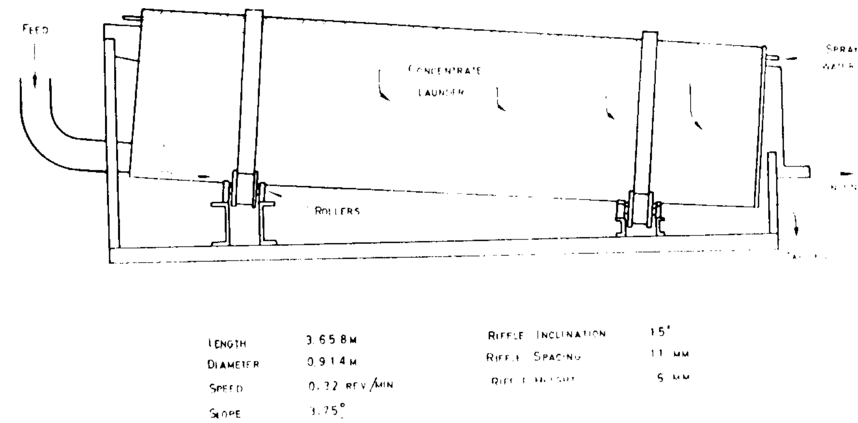


Figure 3. Johnson cylindrical concentrator

Belt Concentrator

This device was followed in 1949 by the development of a slow-moving endless rubber belt 1,52 m wide, riffled in a similar manner as the corduroy cloth. The belt sloped downwards at an angle of 12° but moved upwards at a rate up to 0,38 m per minute. The mill pulp flowed down over the riffles which collected a concentrate of gold and pyrite. The concentrate moved counter current to the flow of pulp and was washed out of the riffles from under the head pulley into a locked container which was taken to the clean-up room each morning for further treatment by amalgamation.

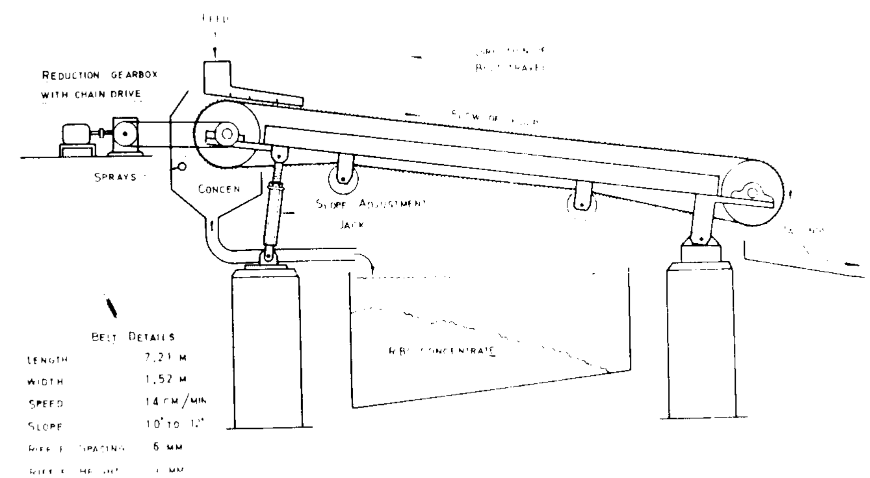


Figure 4. Riffled belt concentrator