A study of the arrangements for pulp discharge on pebble mills, and their influence on mill performance

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SYNOPSIS

Although used for many years in the gold-mining industry, the conventional pan lifter at the discharge end of a pebble mill has hindered attempts at optimization of speed of rotation and pulp flow. A theoretical approach to flow patterns in pan lifters has pointed to limitations in the capacity of these lifters, with capacity decreasing as speed of rotation is increased.

Complete elimination of the pan lifter by the adoption of peripheral or end discharge has resulted in a marked increase in pulp flow and an improvement in power utilization.

Assays of the gold content of pulp discharged after a mill stoppage from mills fitted with pan lifters show values well in excess of the current milled grade, which indicates an accumulation of gold particles in the mill and, therefore, a longer retention period. The same could apply to other heavy minerals and metals present in the ore feed or worn off mill linings and metallic grinding media. Examples are osmiridium, fine iron and iron alloys, and copper and sulphide minerals. This phenomenon is attributed to the higher specific gravities of these metals and minerals, causing them to migrate towards the mill shell, from which position they are not readily removed by the lifters at the discharge end of the mill. The close association of gold particles with iron and sulphide minerals in the crevices of mill liners might be conducive to the formation of coatings on the gold.

Assays of samples of milled pulp taken from a mill designed for end discharge show a lower accumulation of gold. The accumulation and nature of these gold particles are being studied at present.

With increasing diameters of pebble (or autogenous) mills, peripheral or end discharge appears to be the preferred method of pulp removal.

SAMEVATTING

Hoewel dit al jare lank in die goudmynbedryf gebruik word, het die konvensionele panligter aan die uitlaatkant van 'n klippiesmeule pogings om die draaispoed en pulpvloei te optimaliseer belemmer. 'n Teoretiese benadering van vloeipatrone in panligters het gewys op die beperking van die inhoudsmaat van hierdie ligters deurdat die inhoudsmaat afneem namate die draaispoed verhoog word.

Die algehele uitskakeling van die panligter deur die gebruikmaking van die rand- en entuitlaatmetode het gelei tot 'n merkbare verhoging van die pulpvloei en 'n verbetering in die kragbenutting.

Essaiëring van die goudinhoud van pulp uit meule met panligters wat uitgelaat is nadat die meule tot stilstand gebring is, toon waardes wat heelwat hoër as die huidige gemaalde graad is wat dui op 'n ophoping van gouddeeltjies in die meul en gevolglik 'n langer retensieperiode. Dieselfde kan geld vir ander swaar minerale en metale wat aanwesig is in die erts wat ingevoer word, of afgeslyt word van die meulvoerings en metaalvergruisingsmedia. Voorbeelde hiervan is osmiridium, fyn yster en ysterlegerings, en koper- en sulfiedminerale. Hierdie verskynsel word toegeskryf aan die hoër digtheid van hierdie metale en minerale wat hulle na die meulromp laat beweeg uit welke posisie hulle nie maklik deur die ligters aan die uitlaatkant van die meul verwyder word nie. Die noue assossiasie van gouddeeltjies met yster- en sulfiedminerale in die splete van meulvoerings is moontlik bevorderlik vir die vorming van lae op die goud.

Die essaiëring van monsters gemaalde pulp geneem uit 'n meul wat vir entuitlaat ontwerp is, toon 'n laer ophoping van goud. Die ophoping en aard van hierdie gouddeeltjies word op die oomblik bestudeer.

Waar die diameter van klippies- (of outogene) meule al hoe groter word, word die rand- of entuitlaatmetode blykbaar vir die pulpverwydering verkies.

INTRODUCTION

The study of pulp-discharge arrangements on pebble mills, which has been, and still is, of particular interest to the principal author of this paper, has extended over many years. Commencing at the now defunct Geduld Proprietary Mines, where one of his first duties was the periodic clearing of blocked screen holes in the 1,7 m (5ft 6in) diameter secondary mills while lying in the hot, confined space above the pebble load, and continuing through studies of screen and pan-lifter design, the work is now centred on peripheral (or end) discharge. This system eliminates the conventional pan lifter, and the study has the following main aims:

(1) optimization of pulp flow through mills for maximum utilization of input energy,
(2) determination of optimum speeds of rotation,
(3) comparison between the effects of peripheral and conventional discharge on the retention of heavy minerals in mills, and therefore on overgrinding, and the possible formation of coatings on gold particles,
(4) effects of change of rotational direction on liner wear,
(5) assessment of the merit of peripheral discharge versus conventional discharge for large-diameter mills, and
(6) design of large-diameter mills with special reference to discharge arrangements.

SECONDARY MILLS AT GEDULD

The need, at weekly intervals, to clear the blocked holes in the screens of the secondary pebble mills arose from the smaller diameter of these holes in comparison with those in the screens of the primary mills. Choking was aggravated by badly cast holes, which required even

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more frequent clearing. However, as the screens wore thinner and the holes became larger, choking diminished, this stage of the screen's life normally being accompanied by improved grinding performance.

During periods of successive choking of the screens, the flow of pulp was retarded, often leading to backwashing. The remedy to this situation lay in the reduction of feed to the mills by a decrease in the dilution water to the rake classifiers in closed circuit with the mills. This procedure reduced the underflow tonnage and sent a coarser product to the cyanidation plant. Although this action solved the problem for the operator, the cost to the company of deficiencies in design and manufacture of the screen were reflected in an unknown loss of gold, which was encased in coarsely ground pulp.

In addition to this loss, the high temperatures inside the mills pointed to a large energy loss, which was related to the diminishing flow of pulp through the mill.

These observations became the bases of the studies that were continued on other mines of the Group.

MILL OPERATION AT VAN DYK CONSOLIDATED MINES

After the commissioning of the mills at Van Dyk, similar problems were experienced with choking of the smaller holes in the screens of the secondary mills. Pebble eliminators were not in use, and one reason for the smaller screen holes was the prevention of a pebble-circulating load that would adversely affect the operation of the pumps and classifiers.

Attention was therefore paid to proper tapering of the holes in the screens and to casting techniques that would give a smooth internal surface. One solution that appeared to have promise was the prior casting of ferrules that were later to be embodied in the casting of the main screen. This procedure proved effective but failed when ferrules broke away, especially after the screen had worn to a thin section. The resulting larger holes in the screens allowed large pebbles to escape.

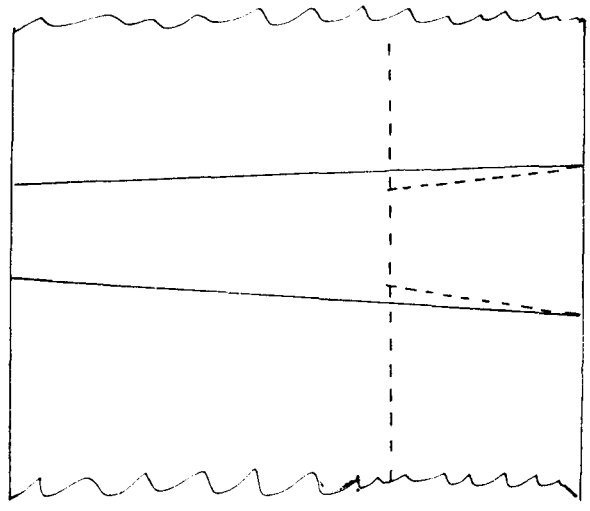


Fig. 1—Section of screen showing improved taper

Thin Plate Screen Protected by a Grid

It was generally noticed that choking problems disappeared after a screen had become well worn. By that time, the roughness of the casting had vanished, the length of the screen holes had diminished, and their diameter had increased. This observation led to the thought that a screen of about one inch in thickness would be desirable to prevent choking from the time of its installation. One important advantage would be an improved taper without the sacrifice of any of the open area (see Fig. 1). A major problem however, lay in maintaining the original thickness of the screen over a long period of operation. This could be achieved either by using a superior wear-resisting alloy for the screen, or protecting it in such a way as to minimize relative motion between the screen plate and the mill charge. The latter system was adopted for both primary and secondary mills, but it sacrificed open area in that some of the holes were now covered by the webs of the grids, and it proved to be not altogether free from chokes. However, operation was satisfactory, and a long plate life was obtained.

Measurements of Heat Generation in Primary and Secondary Mills

In order to obtain a measure of the heat generated during grinding in a milling unit consisting of one primary mill of 2,4 m in diameter by 4,8 m in length (8 ft by 16 ft) and two similar secondary mills, a 4-hour test was run during which measurements of tonnage, temperature, and pulp specific gravity were taken at regular intervals. Surprisingly, the results indicated a greater heat output than energy input. In the calculations it was assumed that the temperature of both the solids and the water leaving a mill were uniform throughout. This might not have been the case. The specific heat of solids was taken at 0,17 cal/g, and no allowance was made for heat losses by radiation, etc.

Results of the heat balance are shown in Table I.

TABLE I

Mill	Power input, kW		Heat generated, k W			(d)/(a)
	Gross	(a) Net	(b) Water	(c) Solids	(d) Total	
Primary	267	248	174	125	299	1,21
Secondary	370	344	319	103	422	1,23

The following measurements were obtained by Professor Kühl[1] on the energy balance of a tube mill:

Energy losses from trunnion friction	14 kW	32 kW = 7%
Energy losses from two-stage gearing	18 kW	
Heat contained in the material being ground	310 kW	408 kW = 91%
Radiation losses from the mill shell	20 kW	
Heat contained in the aspirated air	78 kW	
Energy used in the creation of noise, wear to liners and mill charge, and the actual grinding work		10 kW = 2%
		450 kW = 100%

It was obvious from these measurements that the conversion of energy for the production of new surface was of a very low order indeed, and thought was therefore given to ways of improving this.

Effect of increased Pulp Flow on Fineness of Grind and Gold Extraction

In the belief that high pulp volumes flowing through the mills would promote improved energy conversion by avoiding direct pebble-to-pebble contact and pebble-to-liner contact, and so providing a continuous layer of pulp between the grinding surfaces, attempts were made at Van Dyk to increase mill circulating loads at constant milled tonnage. A greater dilution of the classifier feed resulted in higher circulating loads, but, unfortunately, the unsteadiness and inadequacy of the water flow gave erratic operation, and attention was therefore directed at the water supply and reticulation system with the object of ensuring adequate and steady conditions. The modifications made to the system have been described in a previous paper[2].

Following the installation of a new water-supply system, plant-scale tests were conducted on the effect of higher circulating loads on the fineness of grind. It was found that, as circulating loads increased, the fineness of the final mill pulp increased and this was followed by improved gold recovery. Circulating loads were increased to the maximum raking capacity of the classifiers and even further by perfecting the raking motion.

Fig. 2 shows the relationship between fineness of grind and amount of material handled for the combined primary and secondary mills at virtually constant milled tonnage and constant mill power. Fig. 3, taken from a previous paper, shows the improvement in gold recovery with finer grind and the close correlation between these parameters[3].

It is evident from Fig. 2 that an optimum had not been reached. Potential improvements in the performance and capacity of mills and classifiers were possible by the

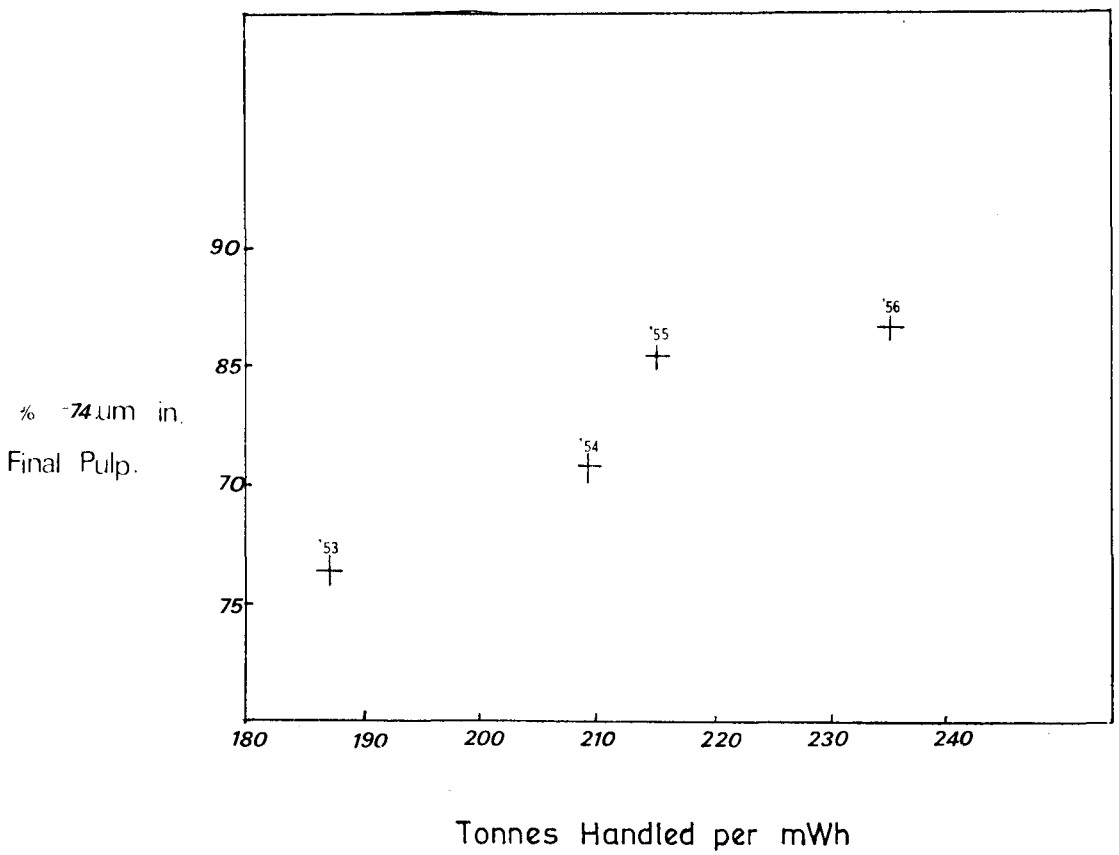


Fig. 2—Relationship between fineness of grind and amount of material handled by the combined primary and secondary mills

installation of cyclones in parallel with existing mechanical classifiers and by an increase in the speed of rotation of the secondary mills, which were running at about 71 per cent of critical speed when measured inside half-worn liners. An increase in mill speed would be accompanied by a corresponding increase in power draw, which, in turn, would require an increase in mill feed for effective utilization of this power increment.

The selection of an optimum speed of rotation was considered in the light of any possible adverse factors that could oppose the benefits expected from an increase in speed. These considerations drew attention to the pan lifter, and to its design and possible behaviour under conditions of increasing speed of rotation.

The secondary mills at Van Dyk were centrally driven, with motor, gearbox, and mill in line. An increase in mill speed could readily have been effected by fitting a V-belt drive between the motor and the gearbox. However, the choice of a suitable speed, and the chance that an optimum could lie close to 100 per cent critical, underlined the necessity for a detailed examination of the design of the pan lifter.

The Pan Lifter

An outline of the pan lifter in use at Van Dyk is shown in Fig. 4, together with an indication of the radial forces acting on a fluid element at the periphery of the pan lifter and constrained to move at critical speed.

From an analysis of the magnitude and direction of forces acting at different radii, it was realized that the pan lifter was not an efficient pulp eliminator and that its effectiveness decreased as its speed of rotation increased. It was therefore felt that any search for optimum speeds of rotation and pulp flow would be opposed by the increasing inability of the pan lifter to remove pulp from the mill as its speed increased.

Examination of worn pan lifters from the primary and secondary mills showed wear on the back of the lifter arms, being more pronounced on the primary lifters, which revolved at higher speeds, than on the secondaries. This observation showed

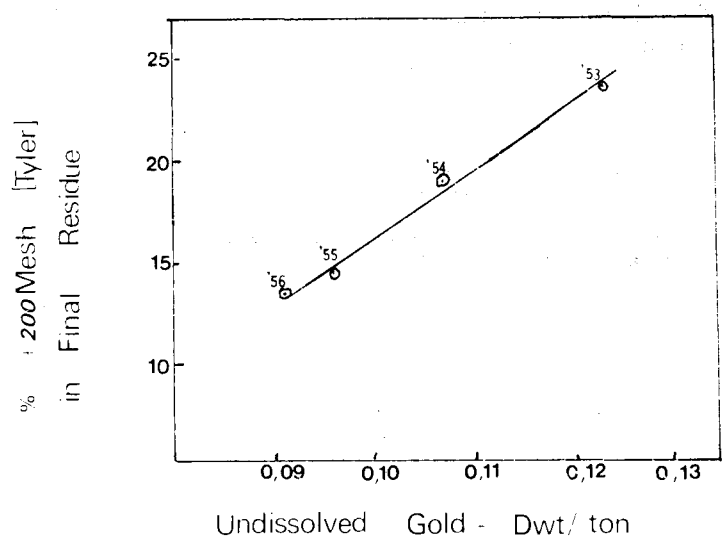


Fig. 3—Improvement of gold recovery with finer grind

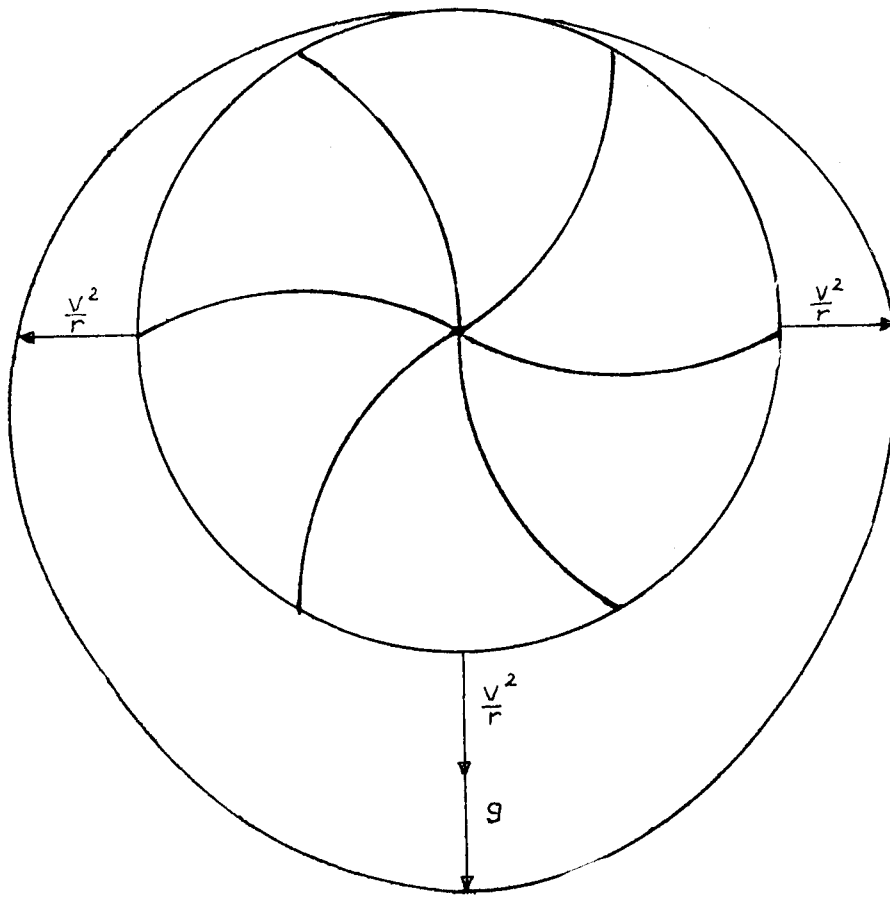


Fig. 4—The pan lifter in use at Van Dyk Consolidated Mines

that the removal of pulp from a mill is a combination of a complex number of flow patterns.

A single-scoop enclosed lifter was designed and fabricated as a possible improvement on existing designs[4], but no obvious benefit could be detected from the use of these scoops except that they imparted some steadiness of operation to the primary mills. It was also realized that, at speeds approaching critical, the efficiency of the scoops would drop.

Although the shrouds of the scoop covered nearly one-third of the area of the mill screen, no problem was experienced with pulp flow. This indicated that the existing screen had adequate open area for the particular flow.

The programme of increasing mill speeds was therefore left in abeyance pending a solution to the mill-discharge problem. The only satisfactory way out seemed to lie in peripheral discharge, where centrifugal force, instead of opposing the force of gravity in drawing pulp along the lifter arms towards the mill outlet trunnion, would, in fact, assist this force over a large area of the mill discharge end.

TESTS ON MILL SPEED AT SOUTH AFRICAN LAND AND EXPLORATION COMPANY, LIMITED[5]

Milling tests were initiated at S.A. Lands in August 1959 to confirm the work of Professor Hukki in Finland on the use of supercritical speeds in rod and ball mills. A primary pebble mill was run at 136, 123, 99, and 91 per cent of critical speed and compared with a standard mill running at 81 per cent of critical. Similar tests were carried out with a secondary pebble mill.

General observations pointed to the difficulty of preventing centrifuging of the mill charge at the higher speeds, and the need for close supervision to keep high-speed mills on the grind. The best results were obtained at 91 per cent of critical, although good results were obtained in certain instances at higher speeds, in spite of the difficulty experienced with centrifuging.

A close examination of Fig. 7 of Kebble's paper[5] shows that the test mill was fitted with a screen of large oval holes, behind which was fitted a six-arm lifter. The curved arms of the lifter are clearly visible through the holes of the screen. With this arrangement, it can be accepted that lifted pulp in the peripheral area, under the influence of high centrifugal forces, was forced back into the mill through the large screen holes in the upper ranges of rotation, thus restricting flow through the mill and promoting centrifuging.

A test at critical speed on a 3,7 m (12 ft) diameter run-of-mine mill, fitted with conventional screen and pan lifter, confirmed the difficulties experienced at S.A. Lands, in that centrifuging of the load, which required high dilution of the mill feed for its prevention and frequent stopping of the mill to drop the compacted charge against the mill shell, caused the test to be abandoned.

THE TUBE MILL AS A CONCENTRATOR

Recovery of Osmiridium at Geduld Proprietary Mines

Recovery of osmiridium at Geduld was based on the collection of concentrates behind the Osborne bar liners during re-lining of a mill. Prior to 1960, the separation of gold and osmiridium in the concentrate was effected by dissolution of the gold in cyanide, followed by tabling and final clean-up. The separation process was carried out in a small, slowly rotating mill in which a few rods were placed. Aerated cyanide solution passed through the mill and joined the leaching solutions of the sand-treatment plant.

Following conversion to an all-sliming plant and elimination of sand treatment, the effluent from the concentrate mill had to be diverted to the slime plant. Because of certain difficulties that accompanied this alteration, and also because the dissolution of the gold by cyanide had proved to be inefficient, alternative methods for handling the mill concentrate were considered.

Amalgamation seemed an obvious alternative, but it was soon discovered that conventional amalgamation techniques were not effective on this particular gold.

Close examination of the gold particles showed discoloration, indicating a coating that, after treatment with various reagents, could be effectively removed only by nitric acid. After this treatment, amalgamation was rapid and complete. The most startling result of this procedure was the sevenfold to eightfold increase in recovered osmiridium, which, on the one hand, pointed to the inefficiency of the previous cyanide separation process but, on the other hand, proved the marked tendency of heavy metals and minerals, including osmiridium, gold, fine iron, and pyrites, to collect against the shell or periphery of tube mills (see Fig. 5).

In a test carried out at Geduld and described in a paper published in 1930[6], this concentrating tendency of gold in a mill was measured directly, and also indirectly by sampling of the effluent of a tube mill after a 30-minute stoppage (see Figs. 6 and 7). The observation is made in the paper that 'experience shows that mere stoppage and restarting of tubes causes a considerable discharge of gold, of which a large portion is found to be already fine enough to escape the final classifiers'. Although it is possible that this discharged gold was not necessarily dislodged from the liners it does point to the tendency of heavy minerals to concentrate in the mills, and could point to the inability of the pan lifter to remove these uniformly.

The system developed at Geduld for the recovery of osmiridium was adopted by the Group's gold mines in Evander in about the mid-sixties, and the resulting high recovery of osmiridium led to the establishment of the Group's own refinery for the separation and refining of constituent metals.

Nature and Behaviour of Coatings on Gold

After the first observations of coatings on the gold particles recovered from the mill-liner concentrate at Geduld, it was important to identify the elements or con-

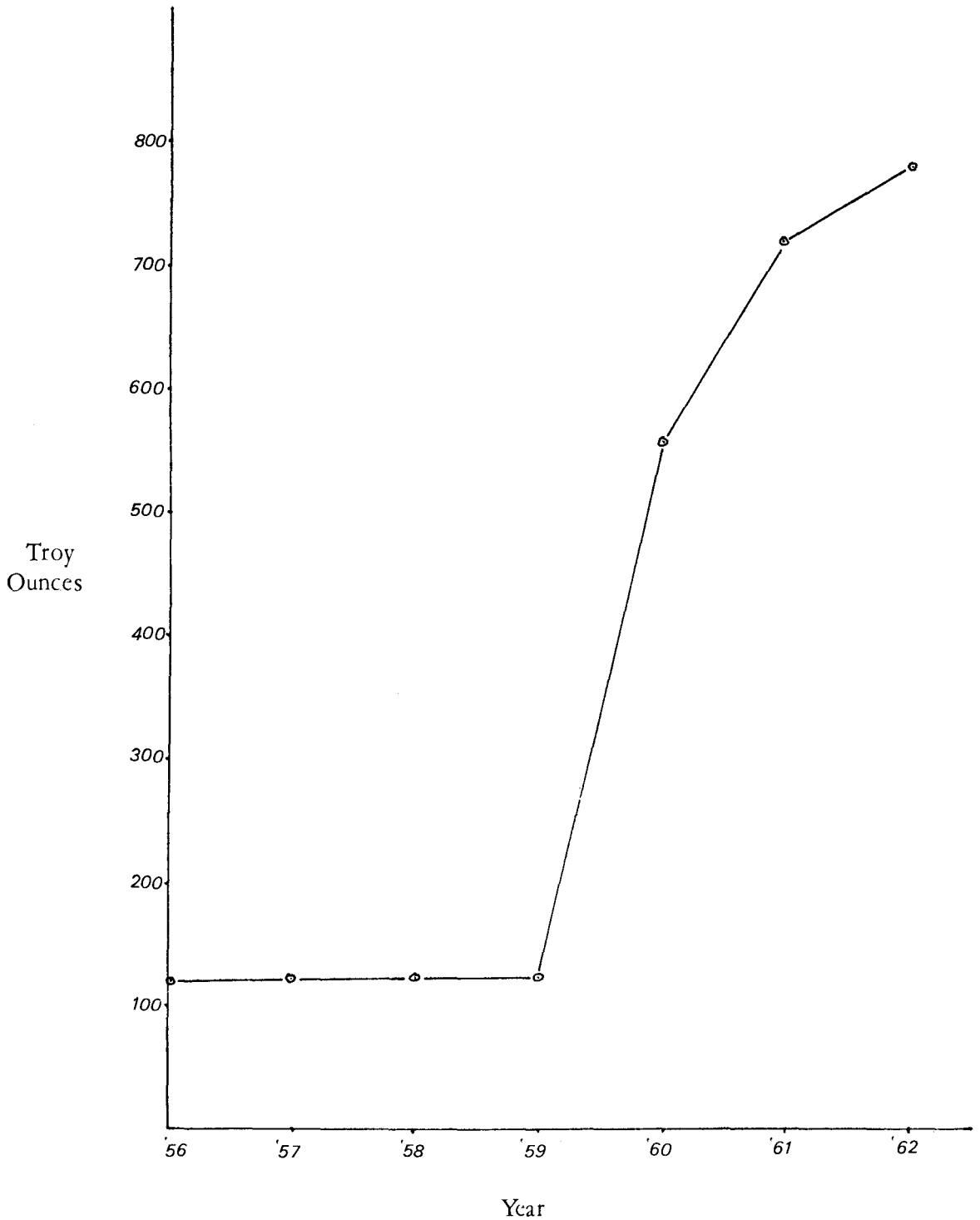


Fig. 5—Annual recovery of osmiridium at Geduld Proprietary Mines, Limited

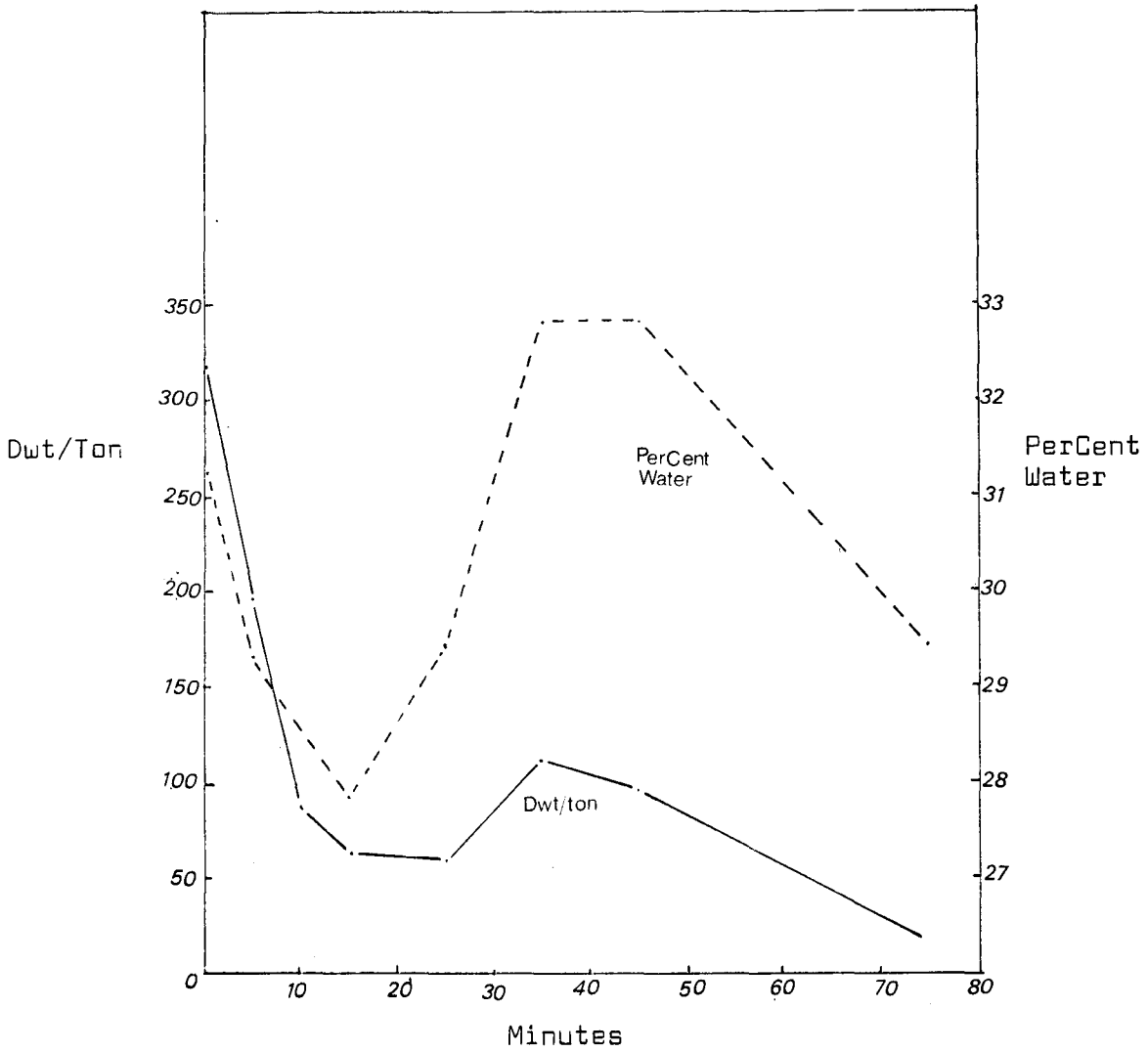


Fig. 6—Value of discharged pulp after a 30-minute stoppage of the tube mill

stituents of the coatings as a first step towards explaining the formation of the coatings and developing possible means of prevention.

To study the behaviour of coated gold in the cyanidation process, samples were taken at Winkelhaak and at Bracken[7] of cyanided pulp leaving the first agitators. It was found that the gold particles were all coated, and that uncoated gold had dissolved, practically completely, in about four hours. The presence of highly oxidized tramp iron was noted.

As a further check on the behaviour of coated gold, free coated gold was discovered in the final residue pulp from Bracken[8]. From these observations it was concluded that both coated and uncoated gold particles are present in the final pulp from the mill that, after dewatering, becomes the feed to the cyanide recovery plant. The uncoated gold is readily dissolved, while the coated gold, in the process of agitation, is to a great extent, but not completely, cleaned of its coatings. The clean gold then dissolves, and the coated gold is lost in plant residues.

A further test was carried out to identify the nature of the coatings on gold particles collected behind the liners of a mill at Kinross[9]. Electron-microprobe analysis showed the presence of silver, copper, iron, manganese, cobalt, nickel, sulphur, silica, and aluminium (see Plates I to III).

Elements in Solid Solution

It will be noted that the X-ray-distribution images of certain elements conform closely to that of gold, indicating that these elements are either in solid solution or form an even coating over the gold surfaces, or both. Elements in this category are silver and copper in all the grains; nickel and cobalt in the heavily coated grain; cobalt and sulphur in the medium-coated grain; and cobalt, nickel, sulphur, and manganese in the lightly coated grain.

Elements Present as Coatings on the Surface

The distribution of iron corresponds closely to that of a reddish-brown coating that was visible under the microscope. In the heavily and medium-coated grains, manganese and cobalt largely followed the distribution of iron. Other elements that could be present as coatings include nickel and portions of the sulphur and silicon in the heavily coated grain; sulphur in the medium-coated grain; and manganese, cobalt,

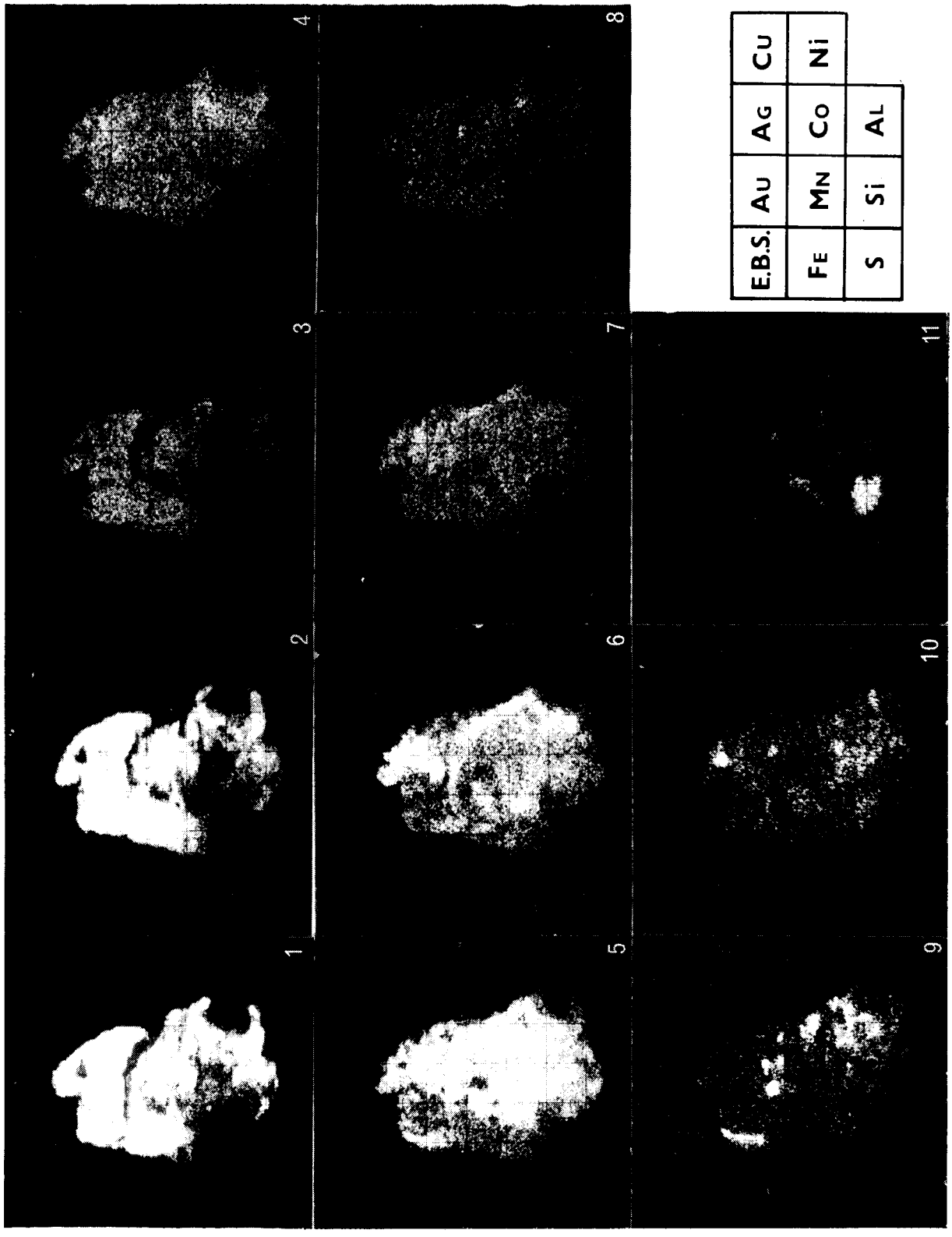


Plate I—Electron-microprobe analyses of the elements listed above

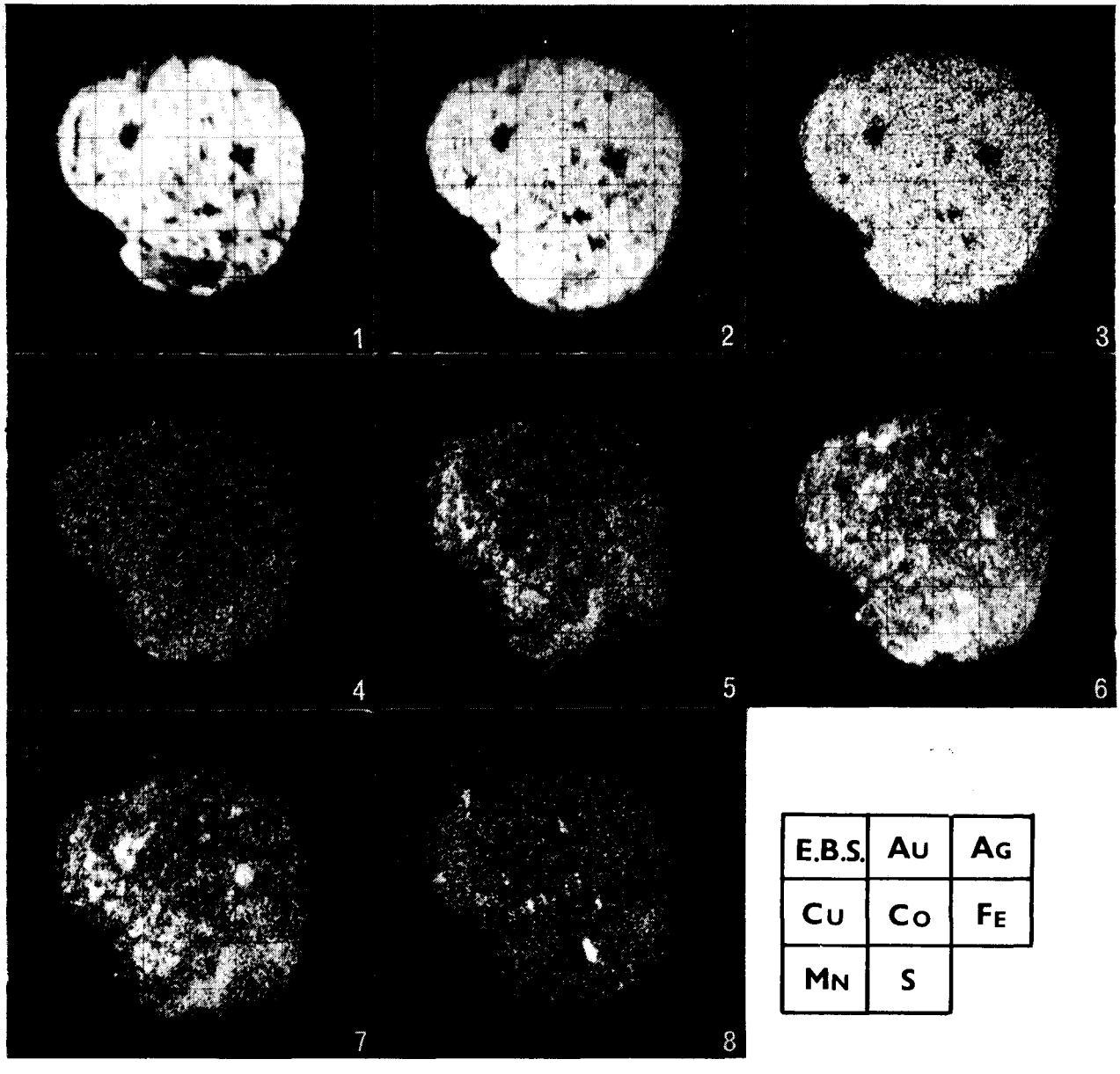
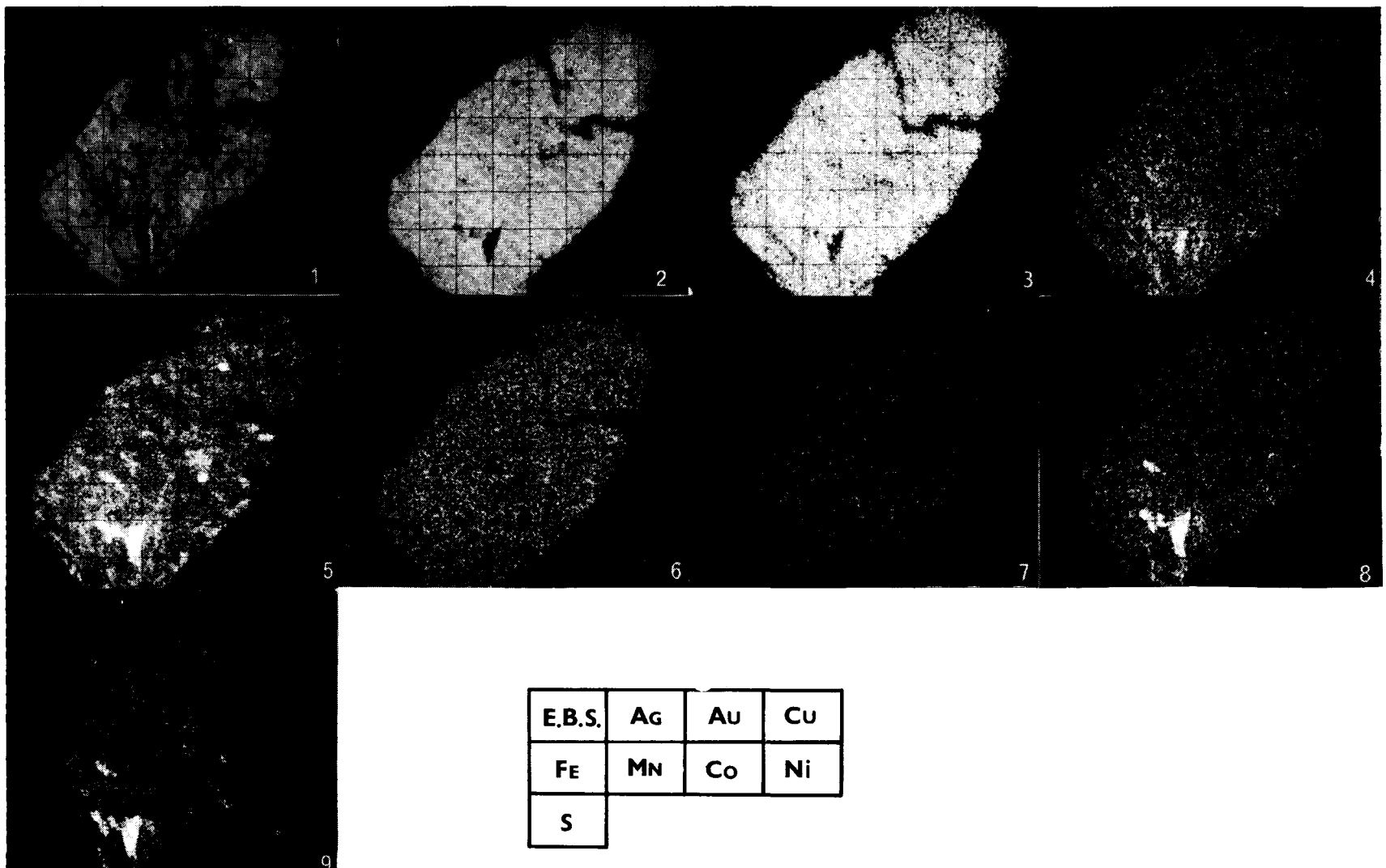


Plate II—Electron-microprobe analyses of the elements listed above



E.B.S.	Ag	Au	Cu
Fe	Mn	Co	Ni
S			

Plate III—Electron-microprobe analyses of the elements listed above

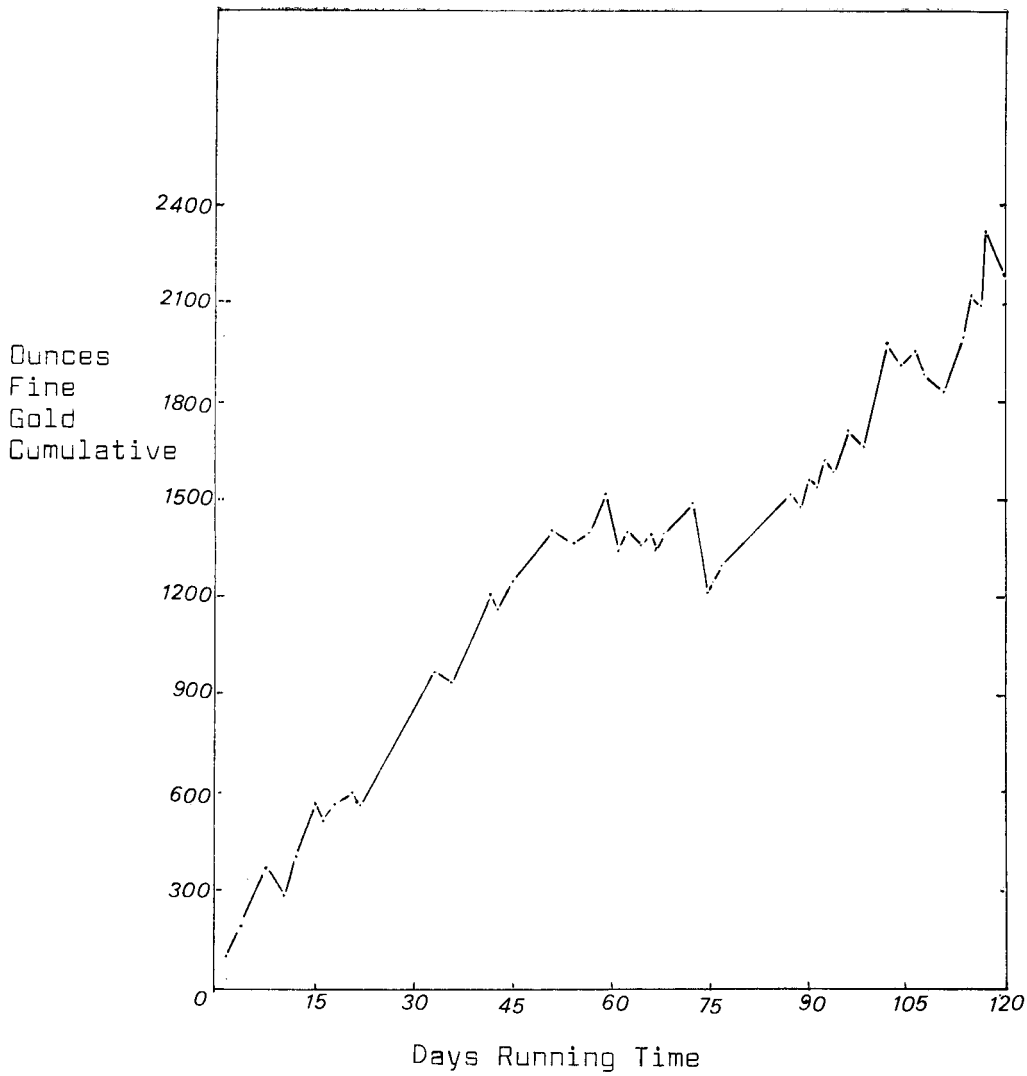


Fig. 7—Accumulation of gold in a tube mill

and portions of the nickel and sulphur in the lightly coated grain.

Concentration of Gold in a Run-of-mine Mill

To check the effect on the gold content of mill pulp discharged after a 15 minute stoppage of a mill at Kinross, samples were taken over a period of 110 minutes.

The results are shown in Fig. 8 and confirm the original testwork done at Geduld in 1930.

ALTERNATIVE SYSTEM FOR THE REMOVAL OF PULP FROM MILLS

The adverse conditions associated with the flow of mineralized pulp through mills pointed to a need for an alternative means of pulp removal that would permit uninterrupted flow, promote rapid passage of heavy minerals with minimum overgrinding, reduce gold lock-up and perhaps coating formation, and facilitate the selection of optimum speeds of rotation. Attention was therefore directed at peripheral discharge.

Tests on Peripheral Discharge at Grootvlei Proprietary Mines

Modification to an Existing Conventional Mill

Following a recommendation to the Research Organization of the Chamber of Mines of South Africa, a grant was made towards the cost of converting and equipping a 2,4 m diameter by 4,8 m long primary mill (8 ft × 16 ft) from conventional to peripheral discharge.

The mill was centrally driven through the outlet trunnion, and it was therefore an easy matter to provide a peripheral discharge at the outlet end of the mill. This was done by the cutting of oval holes in the mill shell. A suitable cover with labyrinth seals was fitted round the mill over the ports to receive ejected pulp and to lead it to the pump sump. Provision for varying the rotational speed of the mill was made by the fitting of a V-belt drive between the motor and gearbox (see Plates IV and V, and Fig. 9).

Determination of Size of Peripheral Screens

To obtain a measure of the discharge capacity of a screen on the mill periphery, a screen plate fabri-

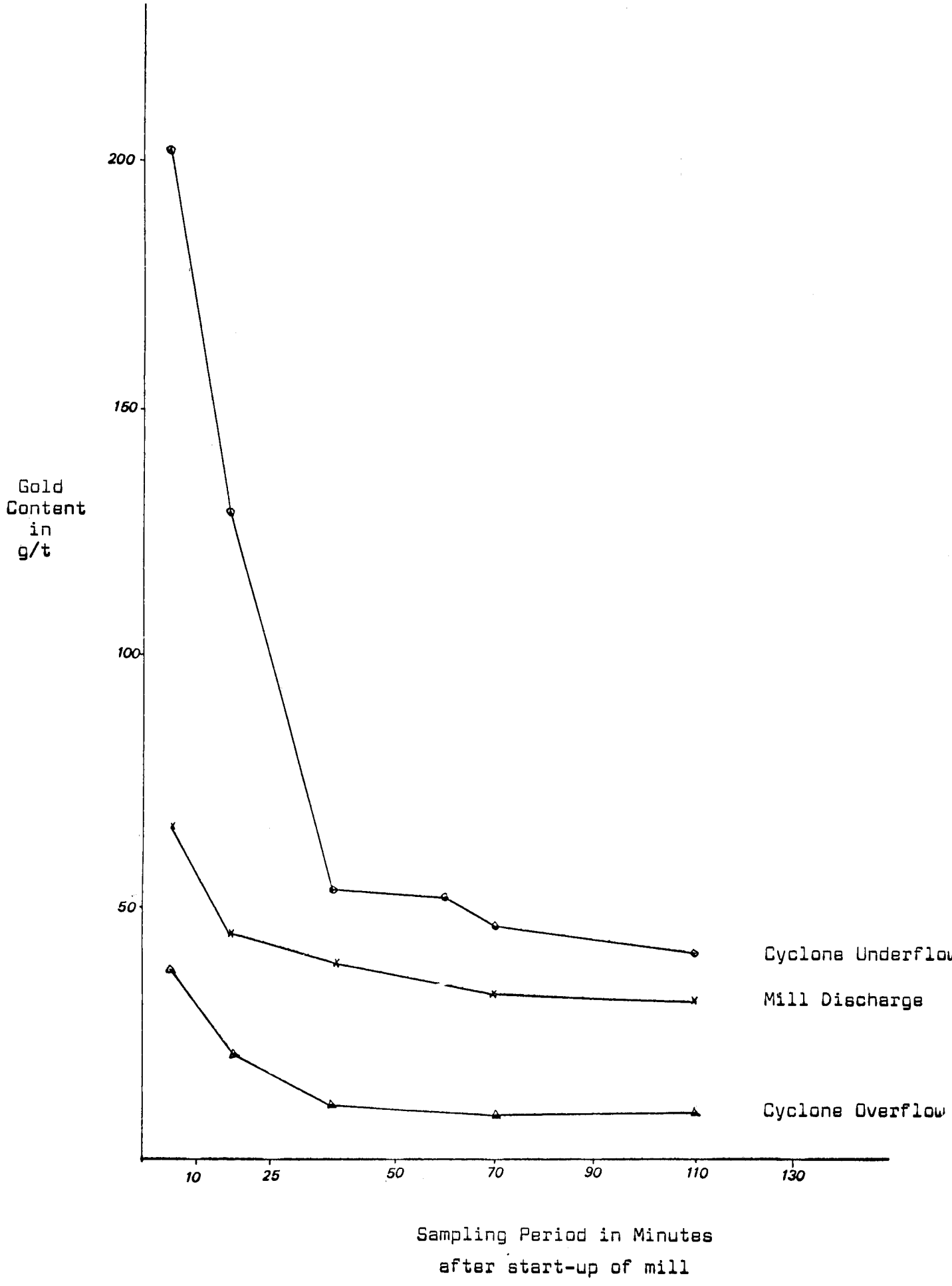


Fig. 8—Gold content of mill and cyclone products following a stoppage of 15 minutes (Kinross Mines Ltd, mill fitted with a screen and pan lifter)

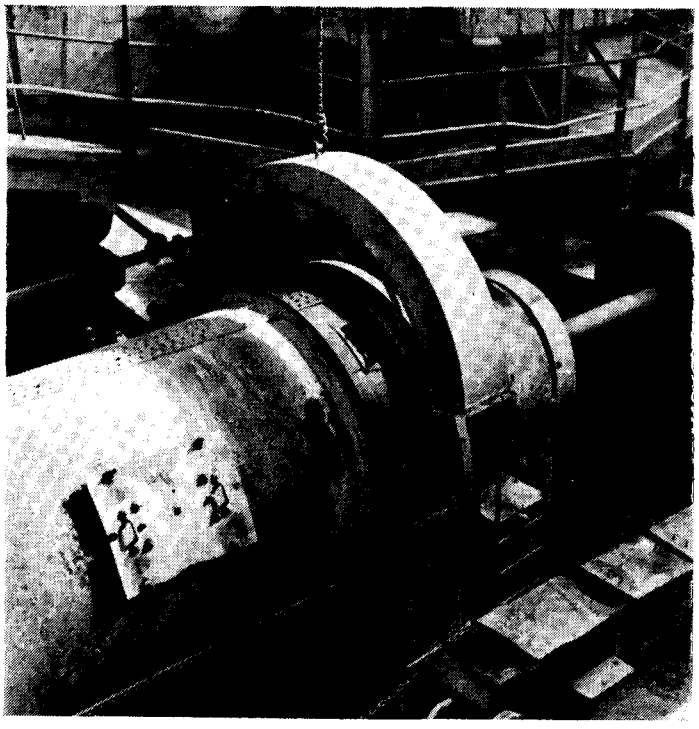


Plate IV—Grootvlei mill showing the peripheral ports and cowl

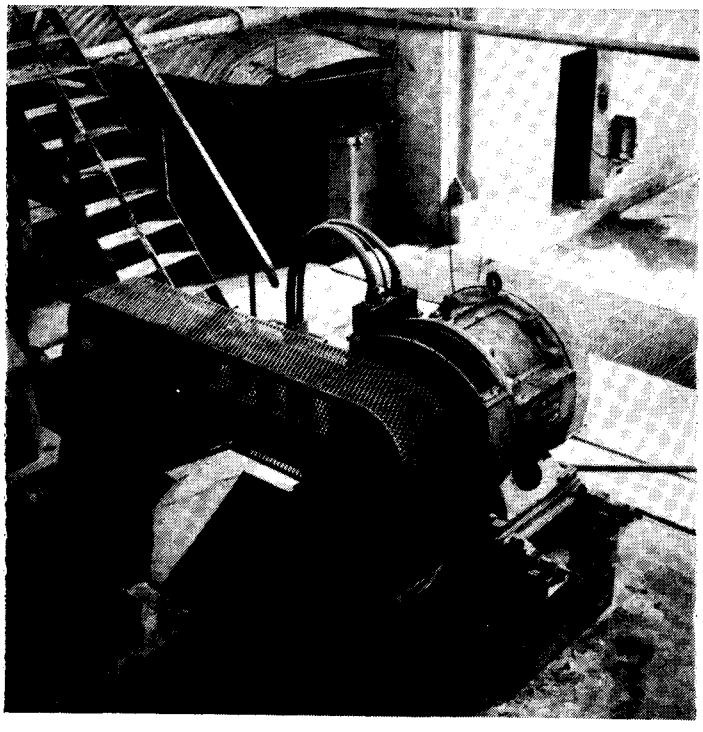


Plate V—V-belt drive between the motor and gearbox of the Grootvlei mill

cated from mild steel was fitted to the mill door (Plate VI). The discharged pulp was collected from this screen during normal rotation of the mill and gave a measure of capacity, which formed the basis for screen design.

The system adopted is shown in Fig. 10. Ground pulp, after passing through the screen holes, converged at the two circular discharge ports, through which it was ejected from the mill.

Although the screen was effective at the outset, the accumulation of tramp steel in the screen holes not directly above the circular opening in the discharge ports reduced the discharge capacity and necessitated periodic cleaning. The problem was alleviated by the provision of a third discharge port.

Ancillary Equipment

The mill circuit was closed with a 152 mm pump and a Krebs D20 B cyclone. New feed to the mill was weighed on a Philips massmeter, which was calibrated at regular intervals.

A separate conveyor, feeding pebbles direct to the mill hopper, was installed so that pebbles would be prevented from falling onto the mill-feed conveyors and thus damaging the weighing mechanism.

Mill power was measured by a kWh-meter, previously calibrated against a sub-standard meter. Automatic samplers were installed for mill discharge (sampled at the pump delivery), cyclone overflow, and cyclone underflow.

Operating Results

Features of the operation of the mill were a high pebble consumption, and circulating load ratios averaging about 6 to 1.

The performance of the mill in terms of minus 74 μm material produced per megawatt-hour against tons handled per megawatt-hour, being an average of eight tests, is shown in Fig. 11.

Modified Arrangements for Discharge

To overcome the choking of screen holes with tramp steel and to reduce pebble consumption, the peripheral screens were replaced by a conventional screen, 25,4 mm thick, made from Benox steel plate and placed immediately ahead of the discharge ports. Holes in the screen

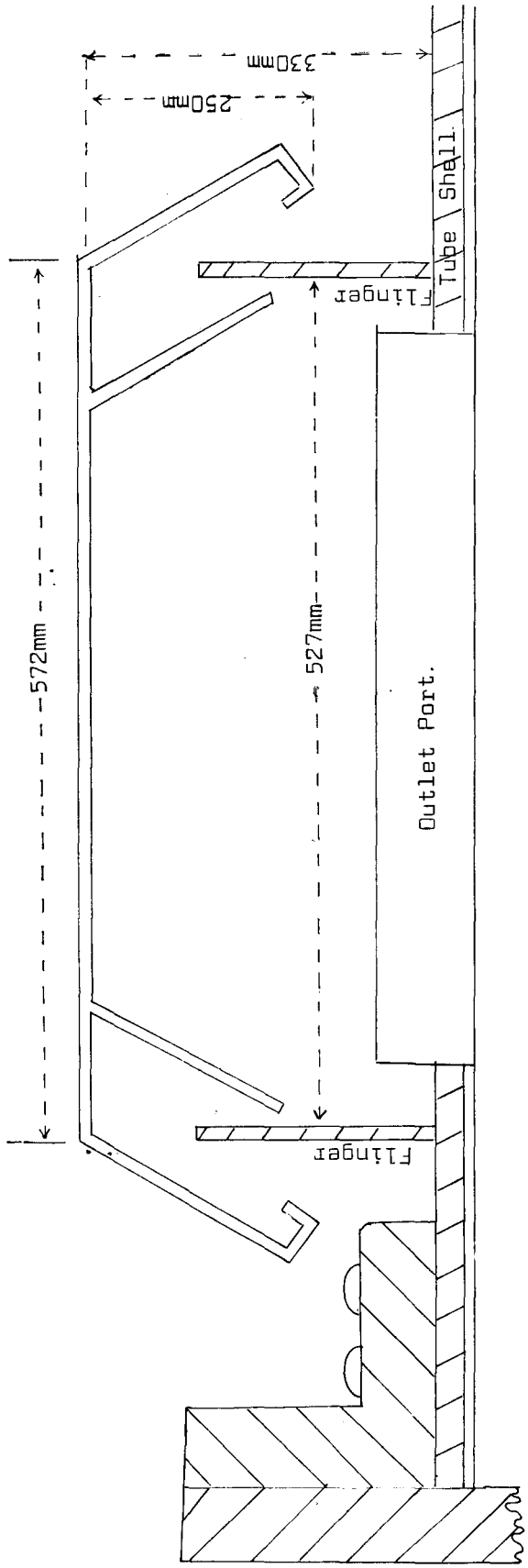


Fig. 9—Sectional view of the mill shell and cowl

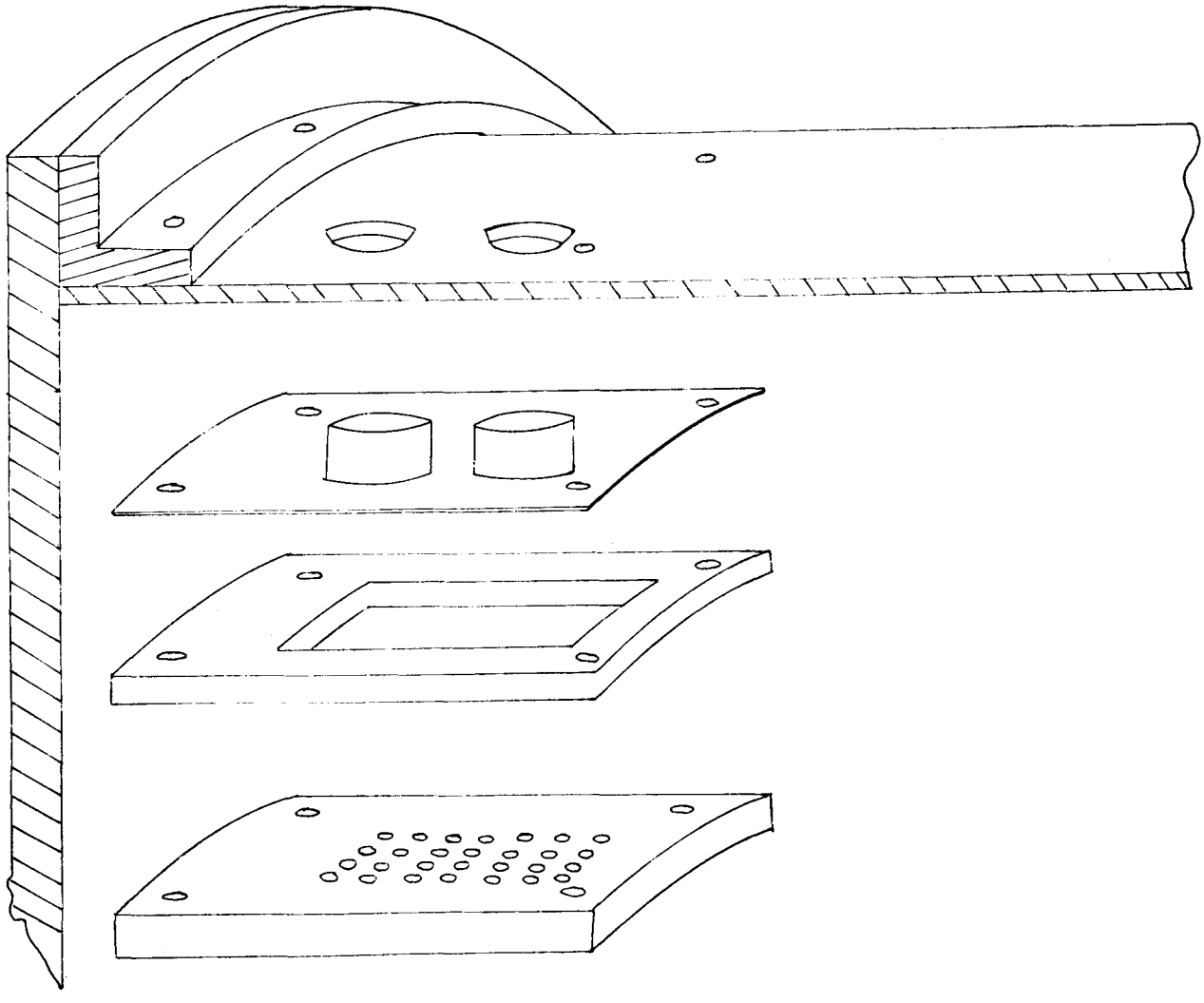


Fig. 10—Screen assembly for peripheral discharge

were tapered from 16 to 19 mm, and were drilled inside an annular ring 560 mm wide. A blank ring 152 mm wide at the periphery of the screen was left to allow a shallow pool of pulp to remain in the mill. The total number of holes drilled in the screen was 2670, giving an open area of 0,53 m^2. The discharge area per average mill kilowatt unit was 0,53/181 = 0,00293 m^2. An ejector with radial arms was designed and fitted to support the screen against the end of the mill, being positioned in such a manner that the peripheral ends of the arms were flush with the lagging ends of the peripheral ports in the mill shell.

The above design reduced the net effective length of the mill by 356 mm when compared with a similar mill fitted with a conventional screen and pan lifter.

The mill was run as a normal production unit under control of the mill operating staff, the tests on the unit being done under the supervision of a qualified metallurgical engineer. Most of the tests were done at a speed of rotation equal to 87 per cent of critical when measured at the surface of half-worn shell liners. This arrangement allowed direct comparison of results with those at a number of mills operating on various mines of the Union Corporation Group.

Further tests were run at speeds equal to 92 and 102 per cent of critical. The tests at 92 per cent were adversely affected by a spate of pump troubles. The tests at 102 per cent of critical speed were interesting in that they showed that

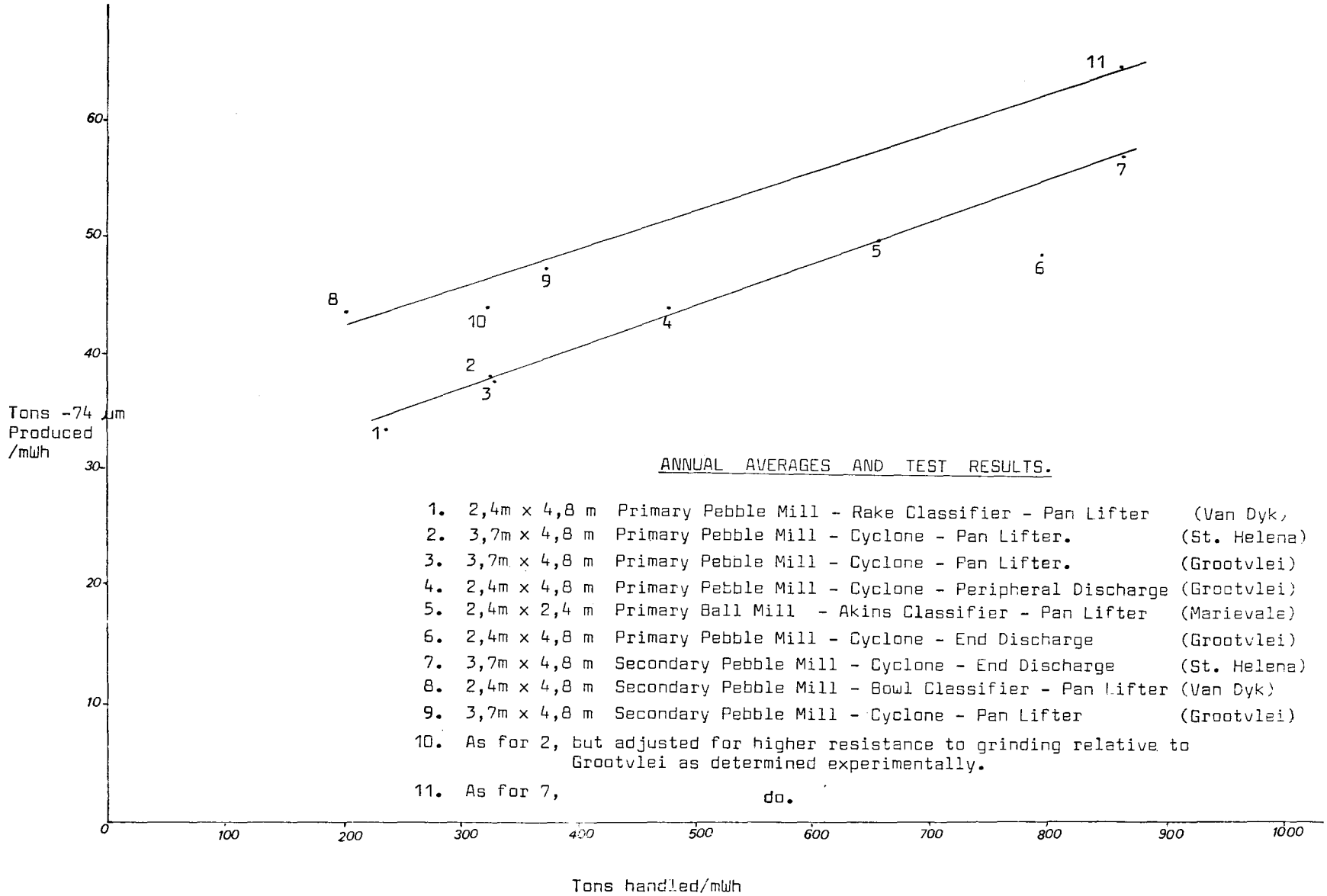


Fig. 11—The performance of the mill with peripheral discharge in relation to that of other mills

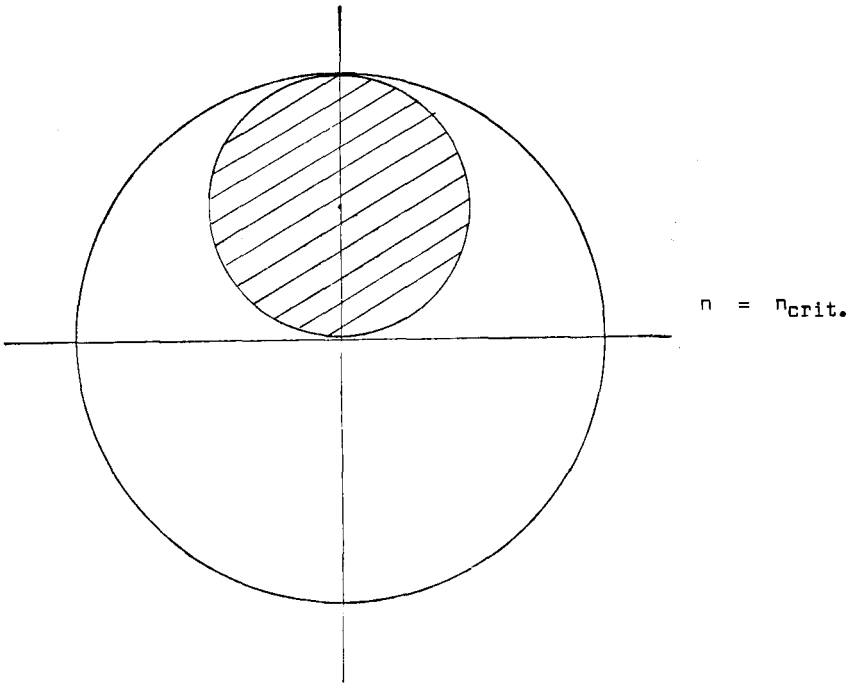


Fig. 12—The flow pattern for a mill running at critical speed

continuous milling at this particular speed is possible, provided that the specific gravity of the discharged pulp does not exceed 1,930 (WS= 0,31/3). At higher specific gravities, centrifuging of the load occurred.

A common feature of the tests was the very high tonnage of pulp that the mill could handle. These high tonnages caused occasional blockage of the pump delivery column, and necessitated changes to the launders handling the cyclone underflow to give a more direct path to the mill-feed hopper. As a result, automatic sampling of the underflow was no longer possible and hand sampling had to be adopted.

A further improvement made to the system was the provision of a separate water supply, independent of the main water system for the mill. This provision improved cyclone operation by ensuring an adequate supply of water at constant pressure.

Detailed Results

Calculation of the tonnage was based on the reading of the mass meter and on the grading analyses of the mill products.

The averaged results of ten tests at 87 per cent critical speed are shown in Fig. 11. It will be noted that this point on the graph falls below the straight line drawn through the points of annual average performance for primary pebble and primary ball mills and the averaged results of the same mill fitted with peripheral screens. However, the point is higher than that achieved by the pebble mills fitted with pan lifters. Pumping problems affected the tests on the peripheral-discharge mill, and it was felt that better and more consistent results would have been obtained with a larger pump.

It is interesting to note that the tonnage handled in conventional pebble mills never reached that attained in the peripheral-discharge mill. If it is accepted that some limiting influence is operative in these mills, which prevents or opposes the passage of higher tonnages, these limitations could result only from the screen or the pan lifter. A comparison was therefore made between the discharge area of the holes in the screen of the peripheral discharge mill and that in the screens of operating mills on the basis of power input:

Mine no.	mm^2 per mill kW
1	2025
2	3435
3	1073
4	1402
End-discharge mill	2924

These results indicated that, in spite of screen-discharge areas greater and smaller than that of the end-discharge mill, the latter can pass about three times more tonnage than that handled by conventional mills. It was therefore concluded that the screens in conventional mills are not the limiting factor, and that the obstacle to higher

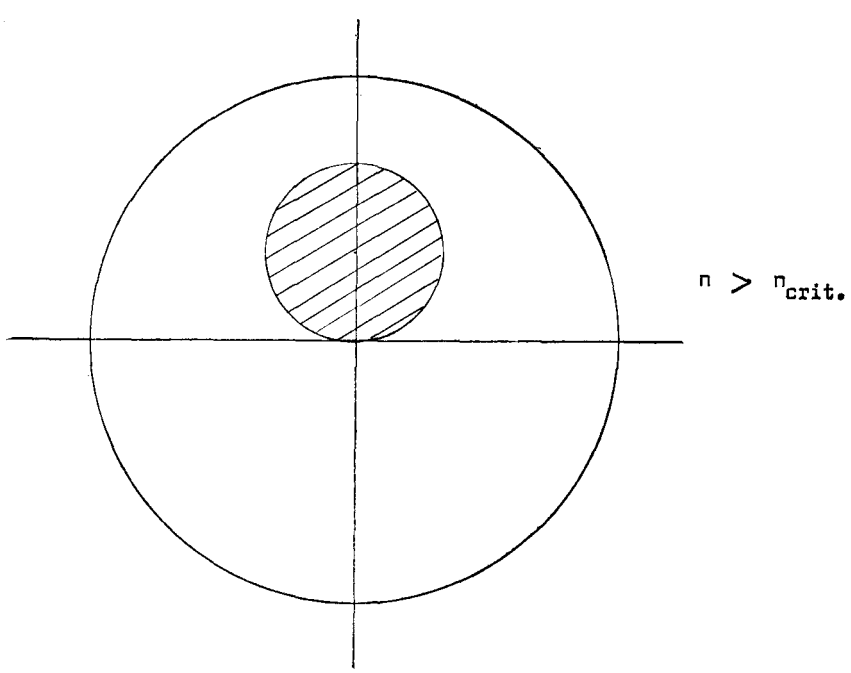


Fig. 13—The flow pattern for a mill running at super-critical speed

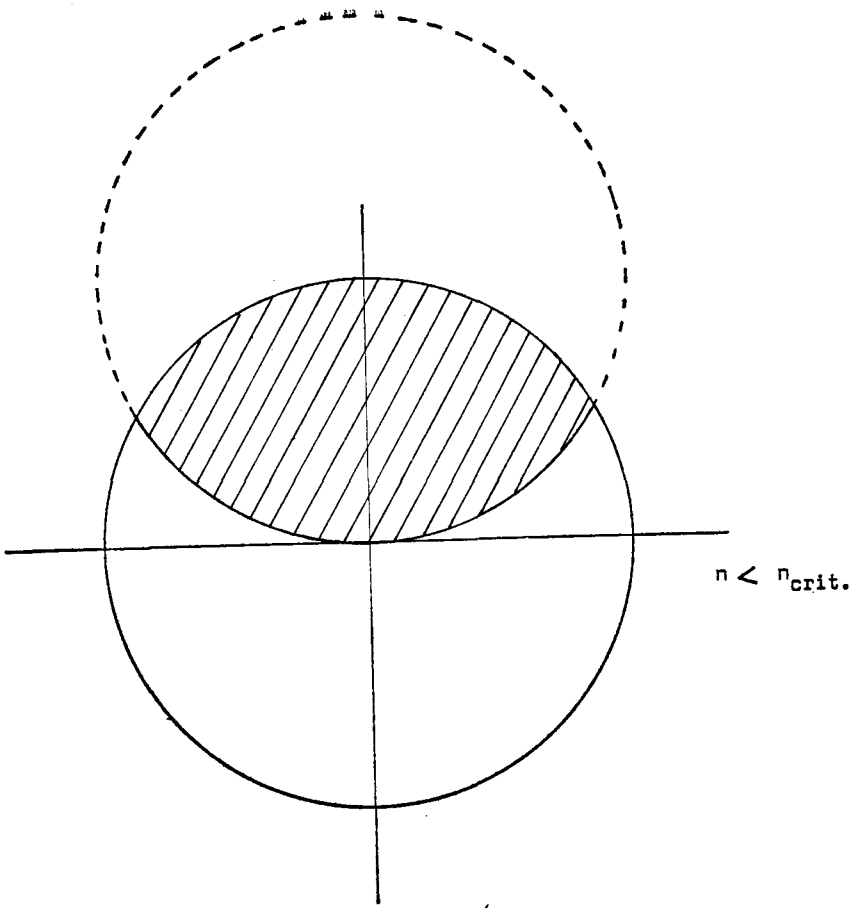


Fig. 14—The flow pattern for a mill running at sub-critical speed

mill throughput must be sought in the pan lifter.

The Pan Lifter

In an attempt to establish the flow characteristics of pulp in pan lifters, it was decided, as a first step, to define the area in the upper half of the discharge area where flow towards the centre of the mill is theoretically possible, at various speeds of rotation. It can be shown that this area is limited to a circle, the radius of which is inversely proportional to the square of the speed of rotation.

For a mill running at critical speed, the only area within which a particle could move towards the centre of the mill is shown by the hatched area in Fig. 12. All other areas in the circle representing the mill favour peripheral discharge.

For a mill running at super-critical speed, the corresponding area is shown in Fig. 13, and for a mill running at sub-critical speed the corresponding area is shown in Fig. 14.

Theoretical Approach

The object of this analysis is the definition of the locus of point P so that the radial force relative to the mill centre is zero. At these points, the centrifugal force acting on a particle must be balanced by the component of the gravitational force acting towards the mill centre. Referring to Fig. 15,

$$\frac{v^2}{r} = g \sin \alpha$$

$$\therefore\ r = \frac{v^2}{g \sin \alpha}, \quad \ldots\ldots \quad (1)$$

but $v = \dfrac{2\pi r\, n}{60}$, where n = rev/min.

$\therefore$ Substituting in equation (1),

$$r = \frac{4\pi^2 r^2 n^2}{60^2\, g \sin \alpha},$$

$$\text{i.e., } r = \frac{60^2\, g \sin \alpha}{4\pi^2 n^2}.$$

$$\therefore\ r = \frac{900\, g}{\pi^2 n^2} \sin \alpha,$$

or $r = C \sin \alpha$, where C is a constant at a specific speed of rotation. . (2)

This equation represents a circle with its centre on the y-axis and having the x-axis as a tangent.

This can be shown as follows.

The equation of a circle with its centre at $x=0$ and $y=R$ is given by (see Fig. 15):

$$x^2 + (y-R)^2 = R^2$$
$$x^2 + y^2 - 2Ry + R^2 = R^2$$
$$x^2 + y^2 - 2Ry = 0.$$

Converting to polar co-ordinates ($x = r \cos \theta$, $y = r \sin \theta$), we have $r^2 \cos^2 \theta + r^2 \sin^2 \theta - 2R\, r \sin \theta = 0$,

$$\text{or } r^2 (\cos^2 \theta + \sin^2 \theta) = 2R\, r \sin \theta$$
$$r^2 = 2R\, r \sin \theta$$

$$\boxed{r = 2R \sin \theta.} \quad \ldots \quad (3)$$

Combining equations (2) and (3),

$$2R = C = \frac{900\, g}{\pi^2 n^2}, \quad \ldots \quad (4)$$

At critical speed, $\dfrac{v^2}{r_m} = g$, . (5)

where r_m represents the internal mill radius.

$$v = \frac{2\pi\, r_m\, n}{60}$$ (n = rev/min. of mill at critical speed).

Substituting in equation (5),

$$\frac{4\pi^2\, r^2_m\, n^2}{60^2\, r_m} = g,$$

$$\therefore\ n^2 = \frac{60^2\, g}{4\pi^2\, r_m}, \text{ or } n^2 \frac{900\, g}{\pi^2\, r_m}.$$

Hence, at critical speed,

$$2R = \frac{900\, g\, \pi^2\, r_m}{\pi^2\, 900\, g} \text{ or } 2R = r_m$$

(compare equation (4)).

Therefore, in a mill running at critical speed, the locus of point P is a circle inscribed in the semi-circle representing the mill above the x-axis. For super-critical speeds the diameter of the circle will be smaller than the radius of the mill, and for sub-critical speeds the diameter will be larger than the mill radius.

From the above investigations, it has been realized that the flow pattern in conventional pan lifters is very complex. However, the

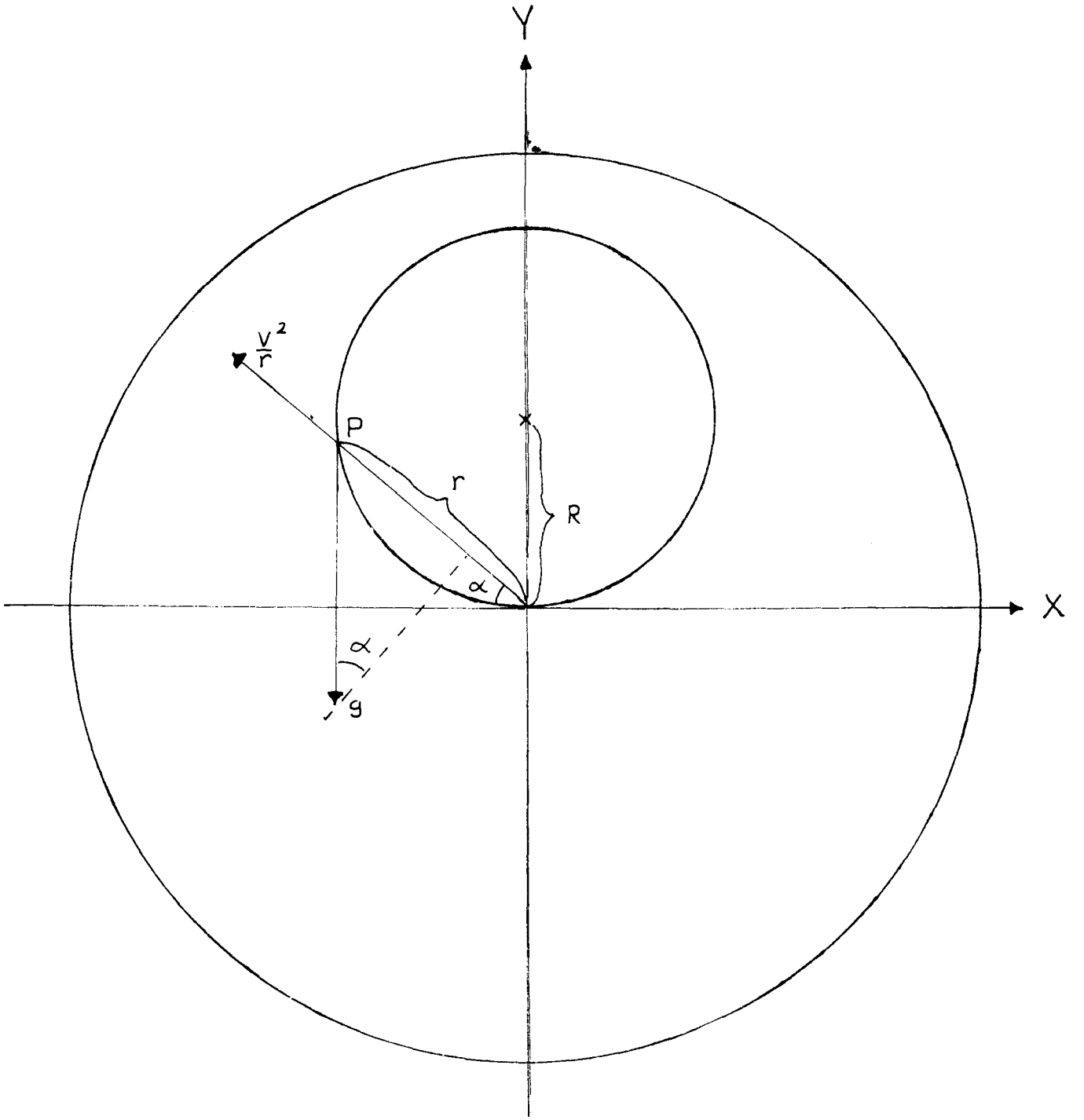


Fig. 15—Basis for the theoretical analysis

following tentative observations can be made.

1. From the rate of pulp flow through the screen in the end-discharge mill compared with that from screens in pebble mills fitted with pan lifters, it would appear that only a third of the holes in the screens of these mills are effectively utilized.
2. Pulp issuing at the periphery of screens in mills fitted with pan lifters could pass back into the mills through screen holes during the ascending period of the lifter arms and perhaps even on descent.
3. It is likely that the pulp level in mills fitted with screens or grates and pan lifters is higher than normally imagined, and that likewise the gradient of pulp, from inlet to outlet, is not steep.
4. Removal of pulp by a pan lifter, especially from a mill rotating at relatively high speed, is far from orderly.

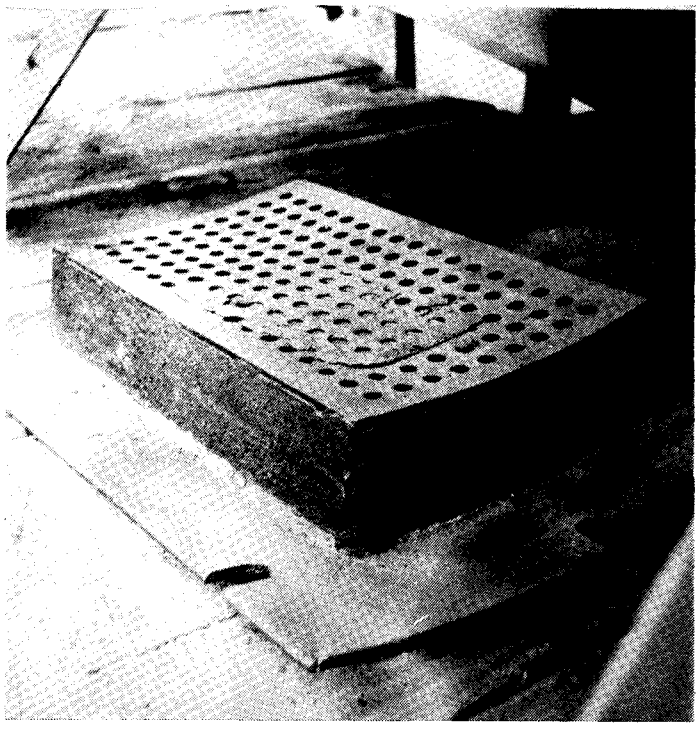


Plate VI—The mild-steel punched plate screen fitted onto the door of Grootvlei mill

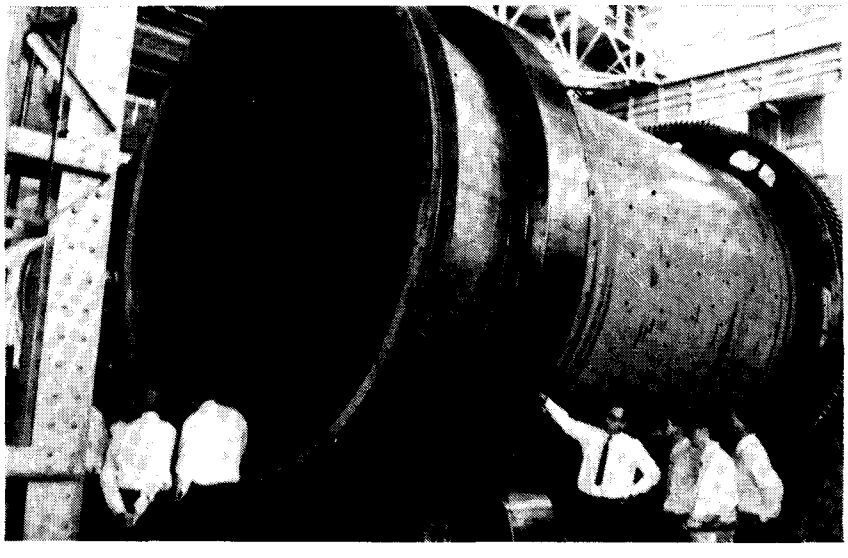


Plate VII—An end-discharge mill, showing the riding ring and roller

INSTALLATION OF A PERIPHERAL DISCHARGE MILL AT ST. HELENA GOLD MINE

The encouraging results obtained at Grootvlei led to the decision to install end discharge on a new secondary mill, 3,7 m in diameter by 4,8 m in length (12 ft by 16 ft), that formed part of a plant extension at St. Helena.

To utilize the full length of the mill, it was decided to discharge pulp through a screen fitted against a spider placed at the end of the mill as shown in Plate VII. (Choking of screen holes was the result of a design error in the taper of the holes.) This arrangement eliminated the conventional trunnion end and necessitated an alternative system for supporting the mill. This was effected by bolting a riding ring to the mill outlet flange and supporting the ring on two rollers. Provision was made in the design of the foundations of the mill for a conventional trunnion discharge (see Plates VIII to X).

A further reason for adopting the open end and riding ring was based on considerations of an acceptable design for future large-diameter mills, in which the shell would be supported on riding rings at both ends, thereby eliminating the mill trunnion ends. Operating experience gained from the St. Helena mill would therefore be helpful in future projects.

To counter lateral stresses, the inlet trunnion bearing of the mill was provided with thrust rings. Special precautions were taken to ensure that there were no stress raisers, or designs that could induce stress-raiser formation on the roller shafts.

Provision was made with the suppliers of the roller bearings to provide facilities in the bearing housings that would permit connection of their testing equipment.

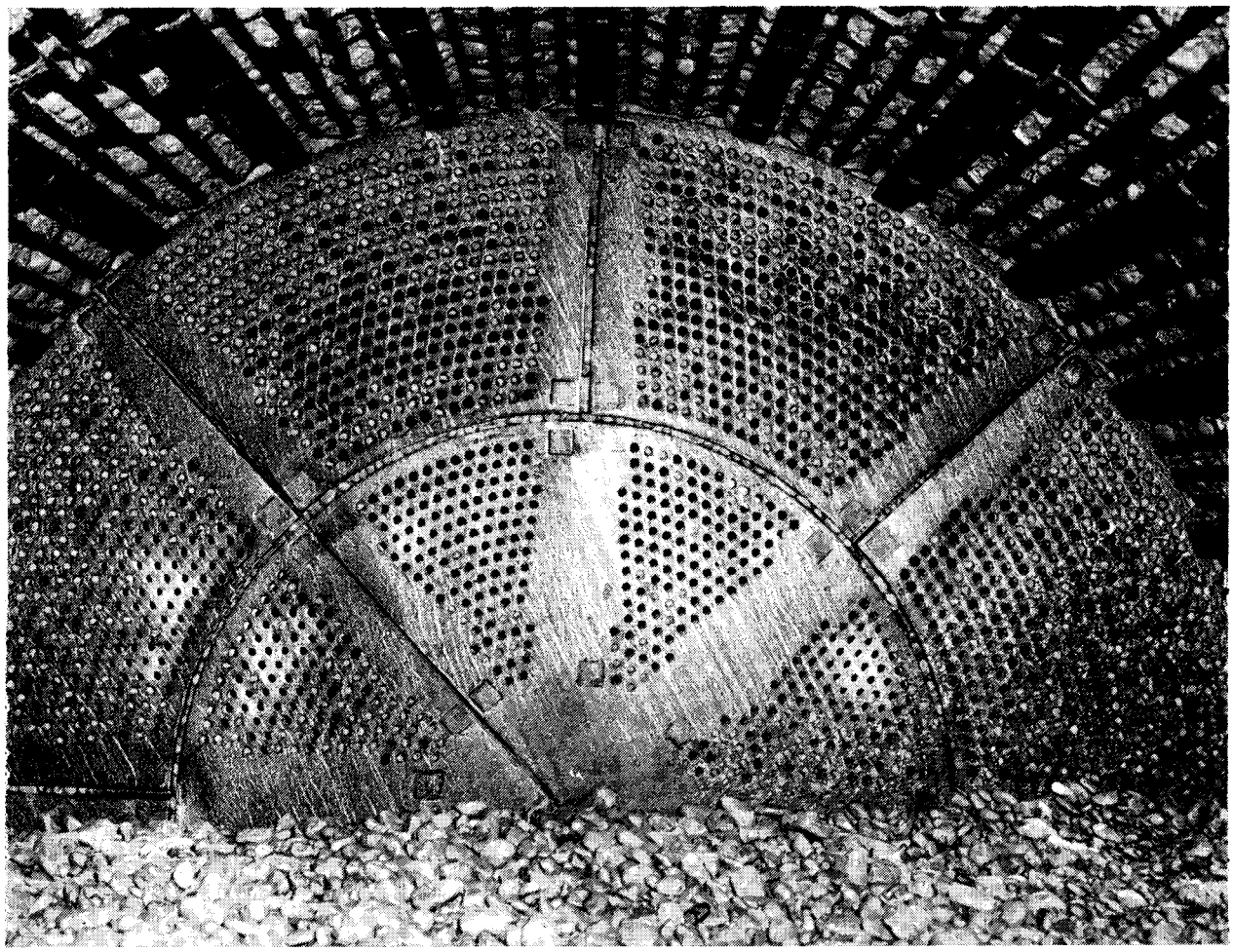


Plate VIII—The screen of the end-discharge mill at St. Helena

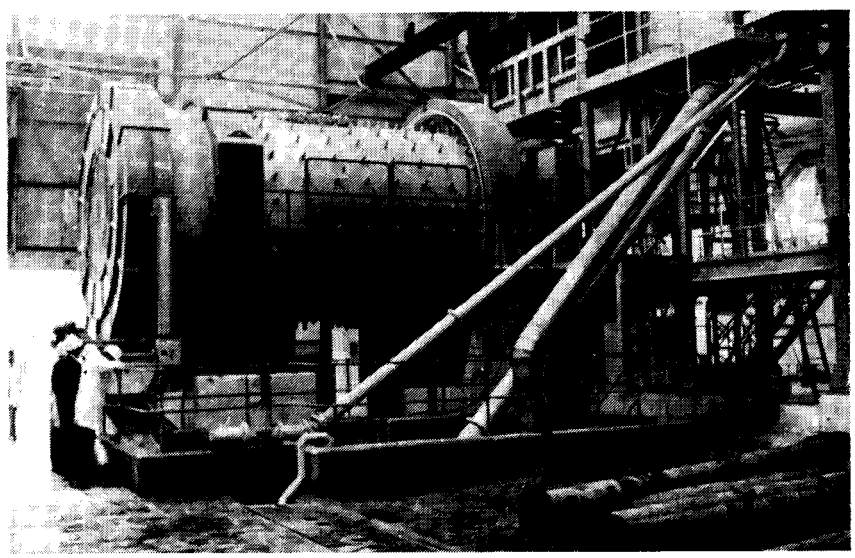


Plate IX—The end-discharge mill at St. Helena

The mill drive, consisting of a motor, gearbox, pinion, and ring gear at the inlet end of the mill, together with foundations, was designed to allow change of pinion size and reversal of direction of rotation. Mill speed for the initial installation was set at 90 per cent of critical.

A single pump was installed to handle the mill discharge and the overflow from the primary-mill cyclone. The pump delivers to 4 Krebs D20 B cyclones, and pump speed is automatically controlled from the level of pulp in the pump sump. A cone, with labyrinth seals fitted to the discharge end of the mill, guides pulp to the launder feeding the pump.

The mill was designed to carry a composite load and was fitted with standard liners. Automatic samplers were installed for mill discharge, and cyclone underflow and

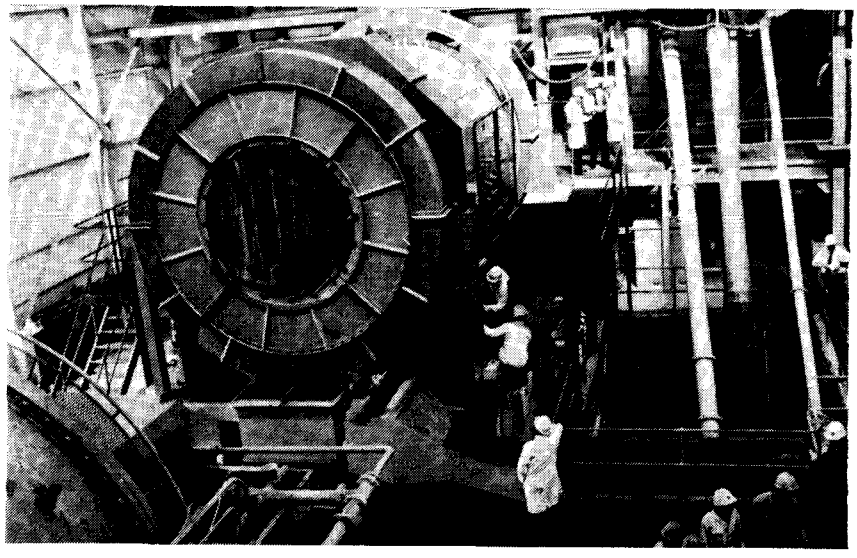


Plate X—Another view of the end-discharge mill at St. Helena

overflow. Consideration was given to the installation of a massflow meter on the pump delivery, but the problem of calibration and the additional cost, left the decision in abeyance.

Operating Performance

The mill is preceded in the grinding unit by a primary mill of similar size, and has run as an operating unit for over 18 months. Distinguishing features have been a high circulating load, ease of keeping the mill on the grind, a lower power draw compared with a similar secondary mill, and a lower power consumption per tonne of minus 74 μm material produced. The consumption of liners and pebbles is similar to that of an adjoining secondary mill with conventional screen and pan lifter and of similar size.

To give an indication of the performance of the mill, a plant-scale test was arranged over two weeks, during which special care was taken with the handling and screening of the samples of mill products on which the tonnage calculations were based. New feed to the primary mill was weighed on a mass meter, and pebble consumption was measured by rate of drop in the pebble bin.

Results of the tests:

Pebble feed, t/24h	118
Cyclone underflow, t/24h	11 423
Mill power draw, kW	556
Material handled per mWh, t	865
Minus 74 μm material produced per mWh, t	57

Reference to Fig. 11 shows that the operating results from this mill follow the general trend. When allowance is made for the higher resistance to grinding of the St. Helena ore, the results are well above those of conventional pebble mills based on mill power only.

Gold Lock-up

A parallel test was carried out on the peripheral-discharge mill and on an adjoining secondary mill with pan lifter to compare the gold content of discharged pulp after a stoppage of 30 minutes. The results of the tests are shown graphically in Fig. 16. This test, together with the two weeks' test on production, was carried out with a 380 mm wide peripheral strip on the screen to form a pool of pulp in the mill.

Further Tests Planned

The following tests are planned:

(1) re-opening of screen holes at the periphery for the liberation of concentrate and observation of the retention characteristics for concentrate,

(2) operating results at 95 per cent of critical speed (these tests were left in abeyance pending the satisfactory operation of the riding ring-roller support system,

(3) reversal of direction of rotation to assess its effects on liner wear, and

(4) performance of the mill on run-of-mine ore.

Mechanical Aspects of the Design

In the design of the mill, which was planned with the express intention of eliminating the outlet trunnion end to give an open-ended mill for maximum utilization of mill length, and to permit ready inspection of the outlet screen and observation of the pattern of discharge flow, much thought was given to the method of supporting the outlet end and of securing the screen sections at the mill end.

The riding-ring and roller support was adopted on the wide experience of the manufacturer, which, however, was mostly based on slowly rotating installations. The possible effect of the high speed of rotation of the mill on the surface of the riding ring and rollers remained an uncertainty, and would become known only once the mill had been put into operation. As explained before, special precautions were taken in the manufacture of the assembly, and regular running tests and inspections are carried out.

A more satisfactory design would be a slipper-bearing support for the riding ring, and this alternative is being investigated.

Advantages that can be seen for the riding ring-slipper bearing method of support, especially when this system is applied to both the inlet and outlet ends of the mill, are the elimination of heavy trunnion ends, which are required not only to support the mill but also to provide access to the mill for the ore feed and circulating load and for the discharge of the ground product. If these trunnion ends and the supporting foundations could be eliminated, the arrangements for feeding and discharge would be simplified and more flexible. Access to the mill interior for re-lining would also be better.

The design has a further advantage in that changes in mill length could be carried out at relatively low

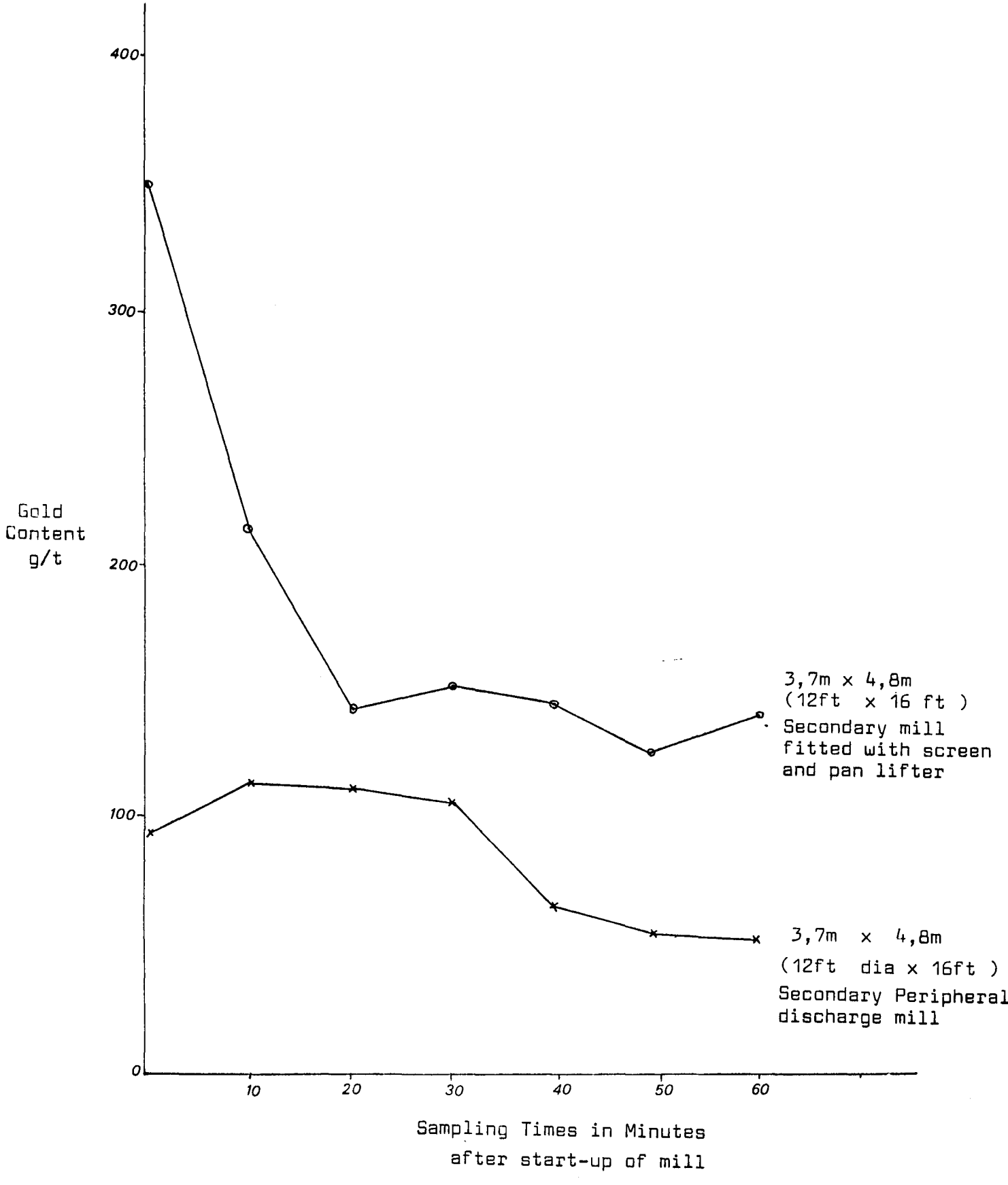


Fig. 16—The gold content of the mill discharge after a stoppage of 30 minutes

cost. This facility could contribute a measure of flexibility in design and capacity, especially where new installations are considered for the treatment of new ore-bodies. If the mills were made in sections of standard length, say 2,5m, and bolted together, flexibility of design and capacity would be coupled with ease of transportation and erection.

Following closely on the design and purchase of the St. Helena mill, a new mill was ordered for Winkelhaak Mines as an extension to their run-of-mine milling plant. Based on the experience gained from run-of-mine milling with the original 4,3 m by 4,8 m (14 ft by 16 ft) mills at Bracken and Leslie, and later with 4,3 m by 6,1 m (14 ft by 20 ft) mills at Kinross, the decision was taken to increase the length of the Winkelhaak mill to 7,3 m (24 ft). It was also decided to provide the mill with peripheral discharge facilities based on the original Grootvlei system but of improved design. These arrangements will make it possible to obtain meaningful comparisons between peripheral discharge and pan lifter-trunnion discharge.

CONCLUSION

Observations of the adverse effects on pebble-mill performance of restrictions to pulp flow focused attention on the design of the mill screen and later on the pan-lifter.

Attempts directed at improvements in screen design resulted in the development of the thin-plate screen, which has found wide application in the gold-mining industry and beyond.

A theoretical approach to the operational characteristics of the pan lifter showed that its effectiveness for the removal of pulp from mills decreases with increasing speed of rotation. This was confirmed by the behaviour of a 4,3 m-diameter mill, in which pulp flow dropped to virtually zero at critical speed. Substitution of peripheral discharge resulted in high pulp flows, which could be maintained even at critical speed.

The advantages of peripheral discharge compared with conventional pan-lifter discharge are seen in the optimization of mill speed of rotation, especially for large-diameter mills, which suffer most from pan-lifter deficiencies; maximum utilization of power input by the control of pulp flow; and a lower level of retention of heavy minerals in the mill charge, with reduced mineral overgrinding.

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Contribution to the above paper by S. K. de Kok*

This contribution to the discussion is also relevant to the other Union Corporation paper (page 249 of this issue) and to that of Mr Atkins and his colleagues[1], because it deals with an aspect of comminution that is common to the subject matter of all these papers, namely classification. It is intended to show that the variable and generally poor performance of this operation is a major reason for low milling efficiencies, and that this has a bearing on the difficulties of pulp discharge from large mills and adds to the complexities of mill-circuit control systems.

Union Corporation has been the pioneer in South Africa of the move towards larger, high-speed grate-discharge pebble mills, and they have therefore been the first to be confronted with the problem of discharging from these mills the large volumes of pulp that they demand. Their ingenuity and persistence over the years in providing solutions, culminating in improved designs of pans and grates, open-ended mills, and peripheral discharge, are highly commendable, but one wonders to what extent the problem itself has been compounded, indeed been created, by inadequate classification. The larger mills have never quite fulfilled their promise of better grinding efficiencies, which theoretical considerations have shown they should achieve, and I believe that the shortcoming is in no small measure due to poor size separation. Anticipated improvements following changes in grinding procedures cannot always be detected or measured because they are frequently swamped by classification inefficiency. In particular, attempts to improve the utilization of mill power by increasing circulation ratios have not been as successful as expected because, as is shown later, with the equipment at present available to us, the increased circulating loads invariably result in greater separation inefficiencies. The failure of the St. Helena open-end-discharge mill to rise appreciably above the performance curve of mills with much lower circulation ratios is testimony to this restraining influence.

In retrospect, the advent of cyclones as the means of imposing an upper size limit on the ore passing to the next stage of treatment must be regarded as having been a stumbling block to progress in the field of grinding: not because they are less efficient than the mechanical classifiers that they supplanted — which they are not — but because they so easily permitted the use of larger and larger grinding units with their requirements of high feed rates, which I believe mechanical classifiers would have been hard pressed to satisfy. Without the cyclone, the industry would long ago have been forced into finding a better method of size separation than that accomplished by hydraulic classification alone. Classification is undoubtedly the least efficient unit process in our plants. I am not referring to poor performance due to poor design of the pumping system, to incorrect choice of cyclone design parameters, to insufficient protection against spigot chokages by tramp material, or to lack of control of the water and ore feed rates (and these are common

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enough), but to the intrinsic inability of differential settlement to effect a sharp separation at dilutions that are practicable.

This leads to a consideration of how classification is to be appraised. All classifiers are dependent upon settling rates in a fluid medium, the separation being influenced by the shape and density of the particles as well as by their size. It is conceded that, in many cases in grinding for cyanidation, the concentrating effect of settlement (whereby the heavier minerals are enriched in the returns to the mill, resulting in selective grinding of these minerals) is beneficial, although I suspect that the case for this benefit is often overstated. Evidence of the possibly bad effects of excessive retention of the gold in the grinding circuit has been supplied by the authors, and in more than one of the papers presented at last year's colloquium on 'Developments in Gold Recovery Processes'. In many instances, such as in some flotation operations, this form of concentration is positively deleterious. Where selective grinding is considered desirable, it is not inconceivable that a system could be devised to combine accurate sizing with differential settlement of heavy and light minerals. If it is accepted that the object of classification is sizing, it has to be measured against screening, an operation in which density has no effect and in which the shape factor plays a different part.

Fundamental to all the techniques developed to evaluate classifiers as size-separators is the variously-named distribution, partition, or recovery curve that represents separation as a function of particle size (it is also often misnamed the efficiency curve). The commonest method is to plot the percentage recovery of each individual size into the two products, the curve being referred to as an instantaneous or point plot to distinguish it from the cumulative form of plotting. The point curve shown in Fig. 1 is typical, the sigmoid shape being characteristic of all hydraulic separations. It starts at the water-flow ratio point — the fraction of feed water that reports to the underflow — which marks the recovery into the underflow of the theoretical zero-sized particle. To be noted is the d_{50} size, the size of particle that is equally distributed between underflow and overflow. The curve representing perfect separation at any designated size is a straight line passing from 0 to 100 per cent on the ordinate marking this size, and the degree to which the actual curve approaches this ideal is a measure of the separation achieved. Were separation perfect and the separation size maintained constant, the impact of the classifier on the mill would be invariable, and control of the grinding circuit could be of the most rudimentary form. Only one method of size separation — screening — closely approaches the ideal.

In cyclone operation, the shape

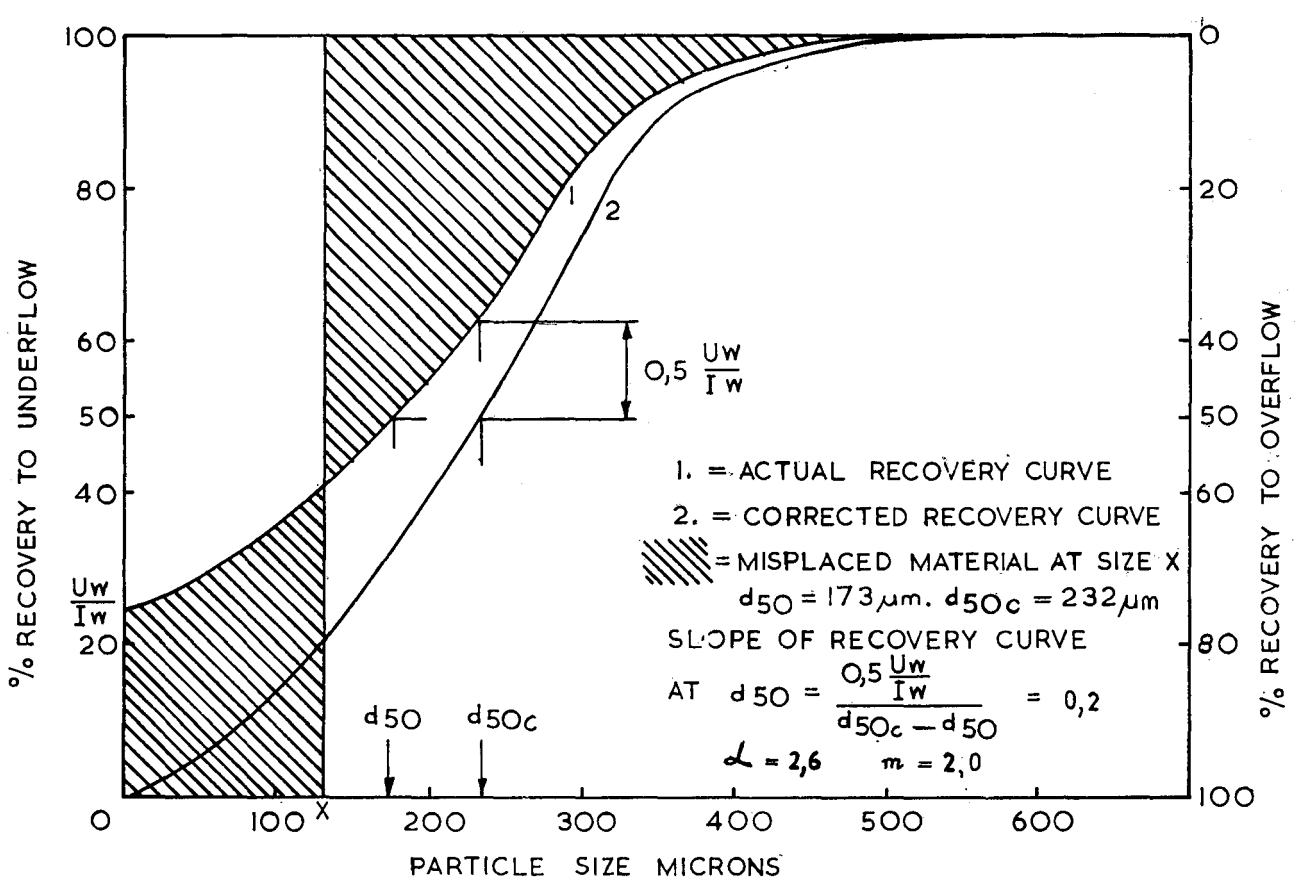

Fig. 1—Recoveries as a function of particle size

of the curve can be improved to a limited extent by modification of certain design and operating variables. In general, the greatest improvement is obtained by a reduction of the pulp viscosity with dilution water. The water-flow ratio point varies directly with the spigot diameter, and inversely with the feed water. It can therefore be reduced by the use of smaller spigots, but only to a stage safely above the 'roping' condition; the error area to the left of the cutting time is reduced, but, as often as not, at the expense of an increase of that to the right of the line. Lowering of the ratio point by an increase in feed dilution cannot be continued indefinitely. The slope of the central portion of the curve is largely a function of the cyclone inlet diameter, and restrictions are set on the minimum diameter by the pump characteristics and by considerations of wear. The flattening of the curve to form a long 'tail' at the coarser sizes can also be reduced with greater dilution and by a judicious choice of vortex finder. Our selection of cyclone variables, which decides the circulation ratio and the separation size, is aimed at containing this tail within reasonable bounds, often without regard to the slope of the curve at the smaller sizes. Attempts to improve the separation have included the injection of 'wash' water above the spigot, and series classification as is carried out in the integral two-stage unit in which the underflow of the first stage is reclassified. Neither method has been sufficiently successful to warrant general adoption. At Prieska, the overflow of the first stage is siphoned into the second stage set at a lower elevation; the first stage is fitted with a small spigot and large vortex finder, and reclassification of the overflow is effected in the second stage, which is fitted with a smaller vortex finder. Separation is superior to single-stage classification, but is still far from satisfactory. In practical terms, it may be said that a cyclone can be made to produce a clean overflow or a moderately clean underflow, but it cannot deliver these simultaneously; operation is always a compromise between the incompatible requirements of two clean products. All the control strategies so far suggested for grinding circuits have to accept the recovery curve; they may have the effect of maintaining or stabilizing it, but they cannot improve it in the slightest.

The recovery curve as shown is the most satisfactory way of depicting separation over the entire range of particles, but its usefulness is limited because it does not directly give a single figure-of-merit of performance to facilitate further mathematical analysis.

Two criteria derived from the curve to characterize its deviation from the ideal are the 'error area' and the probable error (*écart probable*) methods, both originating in the coal preparation industry. The former makes use of the fact that the area subtended by the curve to the left of the vertical represents the recovery into the underflow of particles finer than the desired size, and the area to the right of the line above the curve represents recovery into the overflow of particles coarser than the desired size. The sum of the areas therefore represents the total misplaced material, and a single-figure index can be obtained by the laborious process of measuring the areas by means of a planimeter. The error area method has been criticized on the grounds that it gives undue weight to the extremes of the size range, which are the easiest to place correctly, and that it does not focus sufficient attention on the central range of the curve. The probable error criterion attempts to overcome this objection by defining lack of sharpness as the reciprocal of the slope of the curve in the range between the ordinates representing 25 and 75 per cent into the underflow. Since the slope in classification is very seldom constant over this range, the validity of this method is questionable. In addition, the water-flow ratio is often greater than 25 per cent.

Also plotted in Fig. 1 is the so-called corrected recovery or partition curve, which is the basis of most modern attempts at modelling the cyclone and studies of performance characteristics. It derives from the water-flow ratio point, the significance of which lies in the fact that the water in the underflow carries with it the same percentage of feed solids of each size as the percentage of feed water reporting to the underflow. In other words, the water-flow ratio represents the minimum recovery in the underflow of all size particles that would occur even if there was no separating action other than that due to the splitting of the feed in the same ratio as the water. Therefore, it is argued[2], only recovery over and above this minimum is due to classifying action, and the ratio point represents 'short circuiting' of feed to the underflow. Thus, if $R(d)$ is the actual point recovery of any size d in the underflow, the 'corrected' recovery

$$R_c(d) = \frac{R(d) - \frac{Uw}{Iw}}{1 - \frac{Uw}{Iw}}, \quad . \; . \; (1)$$

and the actual recovery of the 'equilibrium' particle size (corrected d_{50}) is given by $R = 0{,}5 + 0{,}5\,\frac{Uw}{Iw}$.

The slope of the actual recovery curve at the d_{50} point is given by

$$\frac{0{,}5 \quad \frac{Uw}{Iw}}{d_{50c} - d_{50}};$$

in most cyclone applications in grinding circuits, it appears to vary between 0,15 and 0,3 if recovery is expressed as a percentage and size is given in micrometres. The probable error criterion using the 25 and 75 per cent recovery figures has also been applied to the corrected recovery curve, but the objection of non-linearity still applies. The corrected recovery concept has been the foundation of all the present-day studies of cyclone performance since Lynch and Rao[3] showed that a plot of corrected recovery versus d/d_{50} (corrected) — the 'reduced efficiency' curve conceived by Yoshioka and Hotta[4] — remains constant over a wide range of design and operating conditions.

They expressed the corrected recovery curve in the form

$$R_c(d) = \frac{e^{\alpha x} - 1}{e^{\alpha x} + e^{\alpha} - 2}, \; .... \; (2)$$

where $x = d/d_{50c}$ and α, a parameter representing the sharpness of classification, is dependent only on the nature and size consist of the feed, varying between 2,0 and 5,5 for siliceous material. The actual separation and efficiency are hence functions of the feed as well as of those variables, already mentioned, that influence the water-flow ratio (equations (1) and (2)), and those that influence d_{50c}. Much of their experimental work has been concerned with investigating the last two relationships; two of the variables that affect d_{50c} are those that control the water-flow ratio. As pointed out by Plitt[5], equation (2) is cumbersome to use, in that a lengthy iterative solution is required to solve for α from experimental results. From reasoning that a classifier can be represented as a perfect mixer with some classification effects superimposed upon it, he has proposed a relation of the form

$$R_c(d) = 1 - e^{-0,6931\, x^m} \quad \ldots. \quad (3)$$

in which m is a sharpness index varying between 1,8 and 3,8. This model appears to fit the data as closely as does equation (2), and it has the advantage that it can be linearized, thereby providing a simple method of determining d_{50c} and m. Perfect separation requires that the coefficients α and m have infinite value. They are insensitive numbers without any easily visualized physical meaning and, in spite of their low values in relation to the maximum, have not brought home how bad classification really is. What has not yet been established is the effect on their value of the size distribution and relative density of the solids in the feed.

A procedure I have found useful in assessing performance is one that concentrates on the actual recovery curve and conforms to the engineering meaning of efficiency as the ratio of attained to attainable, or the degree of success with which an objective is achieved. Thus, in evaluating classification as a sizing operation, the efficiency is simply expressed as the ratio of classified to classifiable material, or the fraction of the total feed that is correctly distributed on a size basis. If I, O, and U are the feed, mass of overflow, and mass of underflow, and i, o, and u the respective decimal parts passing any designated size, then

$$E = \frac{Oo + U(1 - u)}{I}, \quad . \; . \quad (4)$$

This expression may be regarded as the 'gross efficiency' to distinguish it from the many formulae that have been proposed to account for 'short circuiting'. It is often called the Allen formula; Dahlstrom refers to it as the 'Total Separation Efficiency'.

Fig. 2 illustrates the variation of E with particle size for the separation defined by the plots of i, o, u, which together constitute yet another form of depicting the separation shown in Fig. 1.

The gross efficiency expression

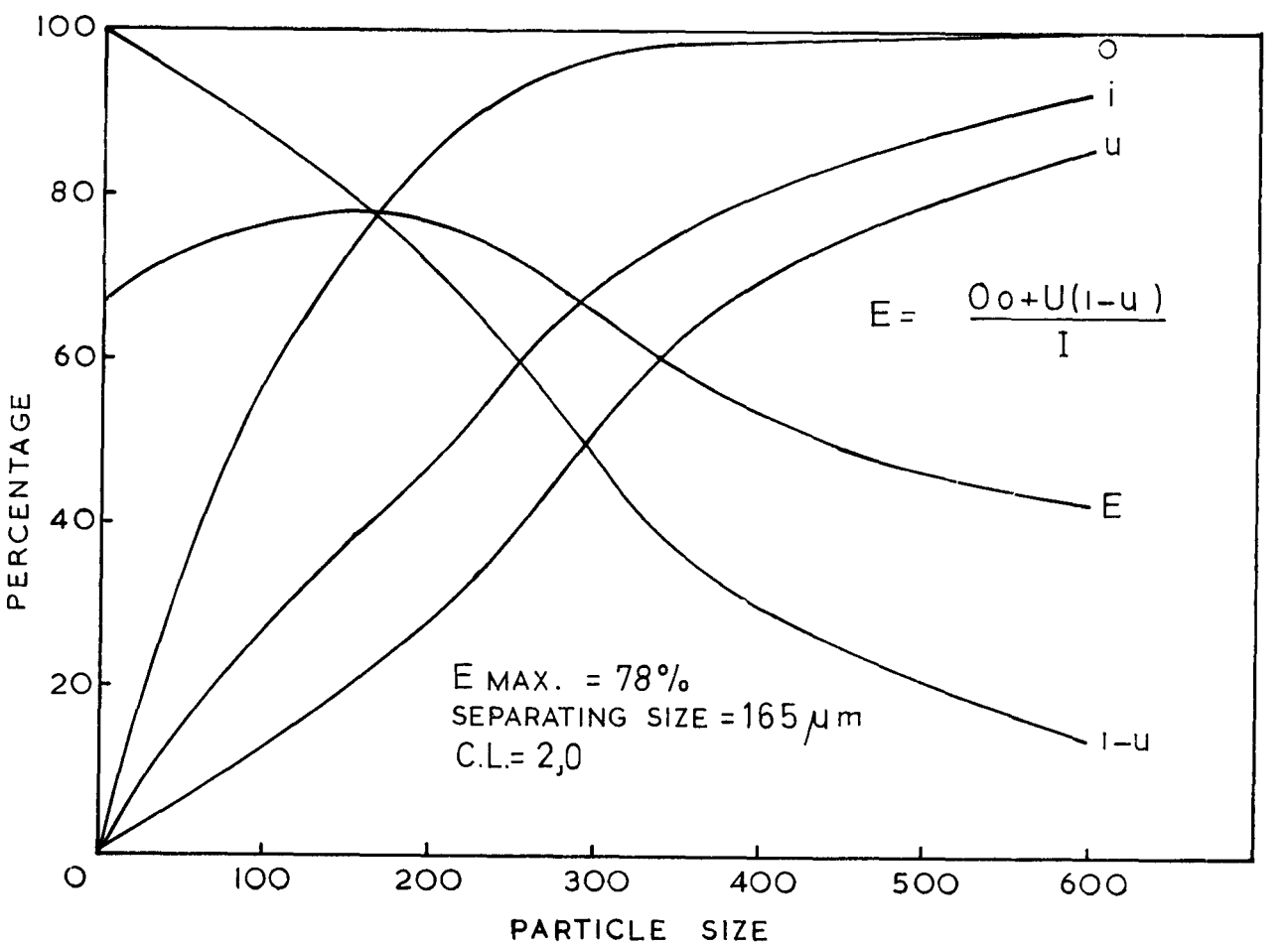

Fig. 2—Variation of efficiency with particle size

may be said to suffer from the defect that it does not make a distinction between the two sources of inefficiency, namely undersize in the underflow and oversize in the overflow. Nor does it distinguish between the separation due to classification action and that due to the mere splitting of the feed into two products. In other words, it does not have zero value when $i = o = u$, being a function of the split in mass and the size selected for calculation. The most commonly accepted method of catering for the 'sampling operation' is that embodied in the formula ascribed to Hancock, Newton, or Dean and introduced into South Africa by Dr H. A. White as the *I O U* formula. This expression is derived from the gross efficiency by subtraction, from the correctly placed material, of the mass of undersize in the overflow and of the mass of oversize in the underflow, which are correctly distributed only by virtue of their being contained in unaltered feed. Under conditions of no separation, the Hancock efficiency is therefore zero for all sizes. As far as short-circuiting to the underflow is concerned, there is an obvious analogy between the *I O U* efficiency and the corrected recovery curve.

Nevertheless, in a real situation, $o > i > u$, and the gross efficiency, which passes through a maximum before decreasing to $\frac{O}{I}$ at the maximum size present, does measure the total separation achieved at all particle sizes, regardless of whether this separation is achieved by classification action or by mere division.

The virtue of the gross efficiency is that its maximum value reveals a unique point, the size at which the classifier has best succeeded in making a clean cut. But its value goes further than this. It was Bond[6] who first stated that inspection of

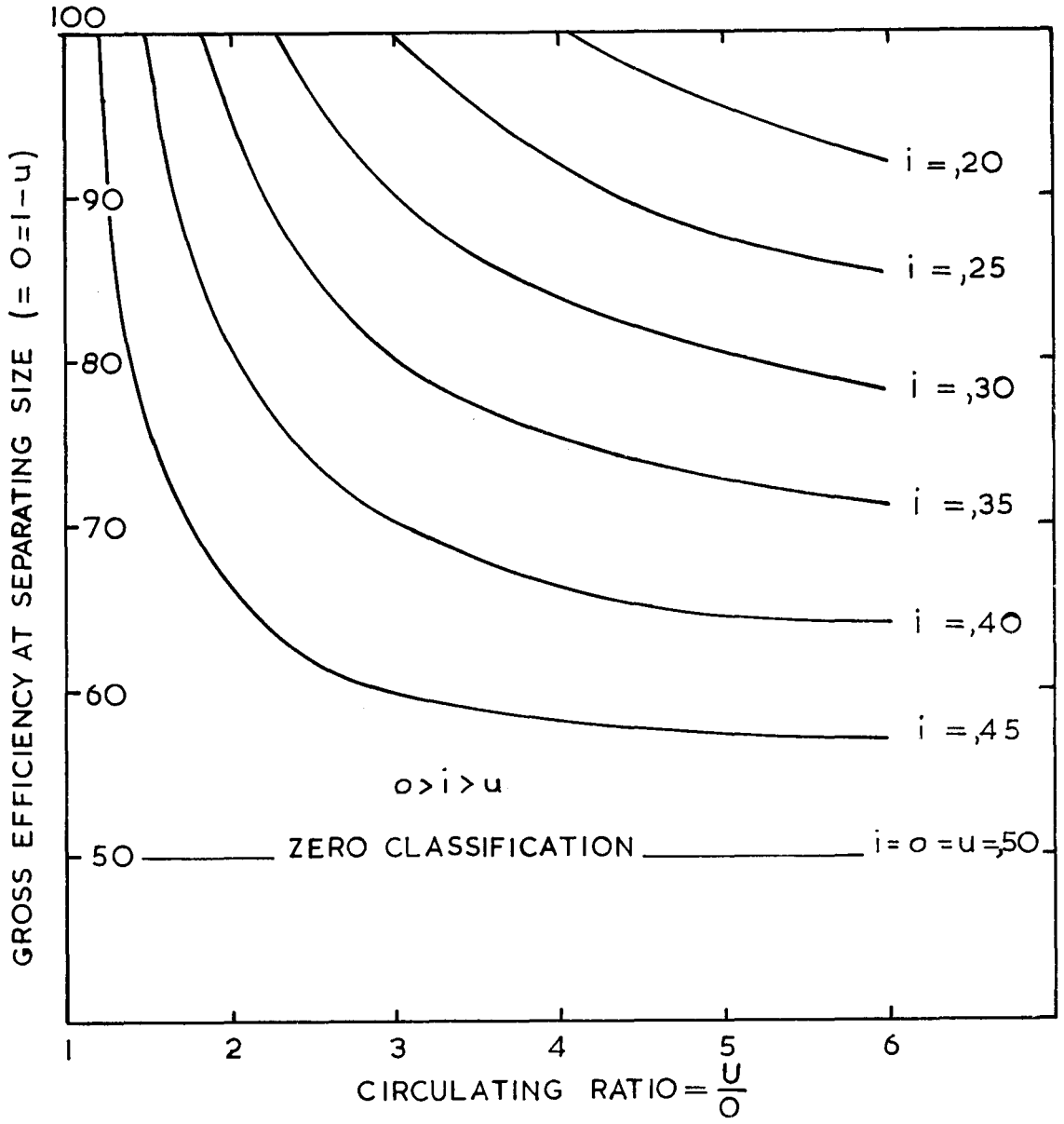

Fig. 3—Effect of circulation ratio on classification efficiency

a number of operations had shown that efficiency is a maximum at the size at which $o = 1 - u$, and he defined this size as the separating size, the efficiency being equal to 0. Analysis of scores of classifications has convinced me of the truth of Bond's finding; these observations have also revealed that the separating size as defined by Bond is very close to the d_{50} size, which can be shown by differentiation to be the size at which the gross efficiency is a maximum.

The special property of the Bond separating size provides a convenient method of spotlighting the inability of classifiers to produce clean products and of revealing the effect of circulating load on this inefficiency. Fig. 3 shows that, for a given feed, efficiency at a constant separating size inevitably decreases as the circulation ratio increases, varying, in the usual range in which we operate, between 85 per cent at low ratios and as low as about 60 per cent at the higher ratios. This effect is a corollary of the change in slope of the recovery curve in the vicinity of d_{50}, the overflow distribution curve flattening at sizes finer than the separating size, steepening at coarser sizes, and intercepting the 100 per cent passing axis at a smaller size when the underflow tonnage is increased. In closed-circuit grinding, if the ratio is increased, the feed coarsens because of the feedback; the efficiency may be kept constant, but at a different separating size. If the separating size is maintained, the efficiency will decrease. If the separating size does not coincide with the size at which the split should be made, the true efficiency will be lower than that shown; since we normally, without being conscious of the fact, fix on a separating size to limit the top size in the overflow or to obtain a fixed percentage passing 74 μm, this coincidence is probably very rare and entirely fortuitous. It is certain that, in many cases, some 50 per cent of the feed to the cyclone is misplaced.

As shown in Fig. 4, the decreased efficiency at higher circulation ratios is the net result of two effects, each of which is a function of the ratio:

(i) the decrease in the recovery of particles finer than the separating size into the overflow; in cyclone classification this is mainly due to the increase in the water-flow ratio, which cannot be countered by increased water addition.

(ii) the increase in recovery of particles coarser than the separating size into the underflow, which of course is the purpose of increasing the circulation.

The rate of decrease of (i) with circulating load is greater than the rate of increase of (ii). The key to better classification is obviously a reduction in the undersize in the underflow.

This is as far as the Bond separat-

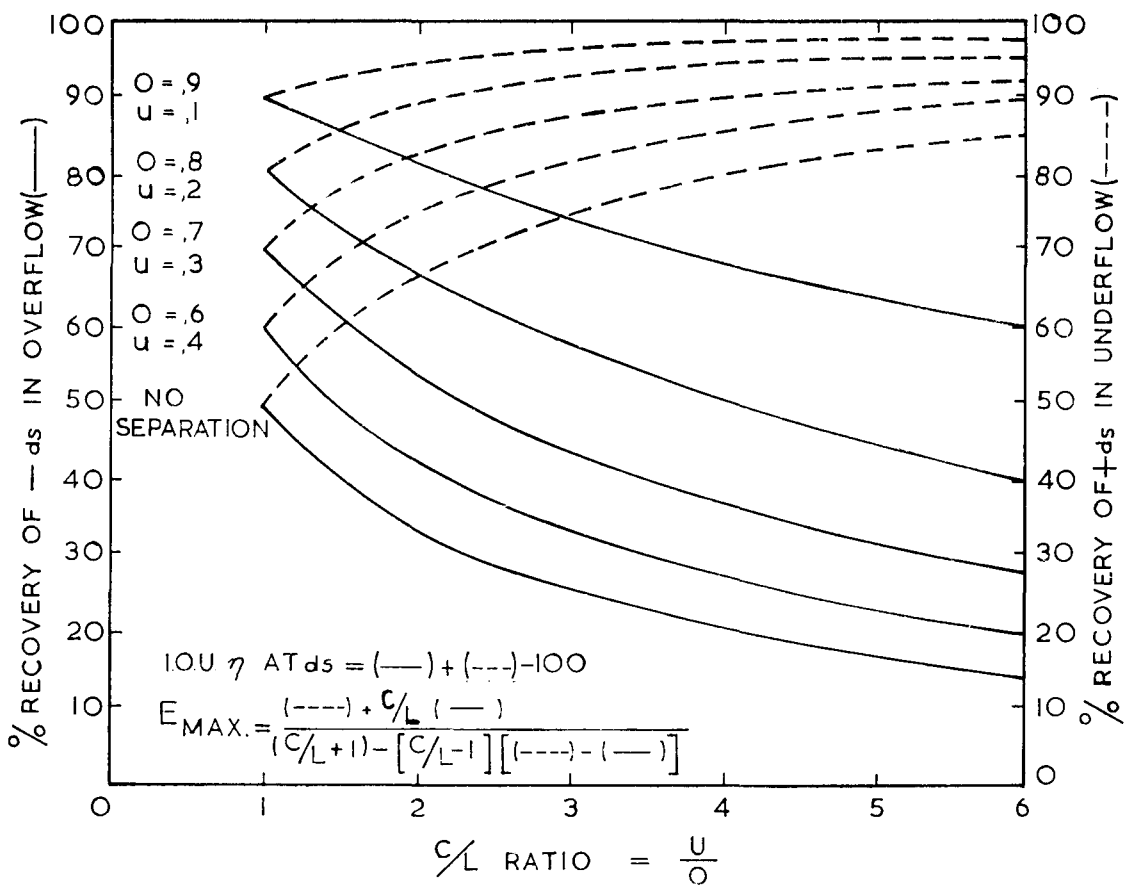

Fig. 4—Recoveries as a function of circulation ratio

ing size concept can take one in this particular exercise, but it is far enough to have demonstrated the disastrous effect of increased circulating loads on the efficiency of our present-style hydraulic classifiers, and therefore on our milling efficiencies. The classifier-mill combination is a closed circuit in which the components interact upon one another, and only a grinding model that describes the complete system can fully reveal the influence of a poorer performance of either upon the overall unit performance.

In Fig. 5, I have fitted the dynamic model of Professor L. G. Austin of Pennsylvania State University[7] to the results obtained in closely controlled tests at Rand Leases on a 6 ft 6 in by 20 ft composite-loaded tubemill. The upper curve illustrates the improvement in production that ideal classification could achieve with an increase in circulating load. The circulating loads of 200 to 300 per cent, which are coupled with inadequate classification, are no more productive than a ratio of less than 100 per cent would be were classification perfect. The optimum in grinding efficiency is not being achieved because imperfect separation nullifies the increased production rate that should accrue from increased circulating ratios; it is obvious that higher feed rates replacing unnecessary mill returns would raise our production rates to unprecedented levels.

The ideal may be impossible to attain, but I submit that we should be paying the same attention to closing the gap between the curves as we are paying to control systems that are aimed at preserving the *status quo*. To be more specific, I suggest that we take a closer look at the inclusion of screens in our grinding circuits as ancillaries to cyclones to overcome their deficiencies, or possibly as complete replacements. This may require a major breakthrough in screening techniques, but it must not be forgotten that high-tonnage fine screening is accepted practice in other spheres of mineral beneficiation. The manner of integrating the screens into the circuit, the choice of separation size(s), and modifications to the operation of the cyclones would depend upon the extent to which the circulation of the heavy mineral(s) should exceed that of the lighter mineral(s), and upon the degree of removal of valuable minerals from the circuit by concentration methods. Optimum results can only come from a better understanding than we now have of the real purpose of grinding a heterogeneous ore containing a variety of minerals. The sharp sizing obtainable with screens should go a long way to giving us this understanding by permitting a closer control of the size distribution of the milled product.

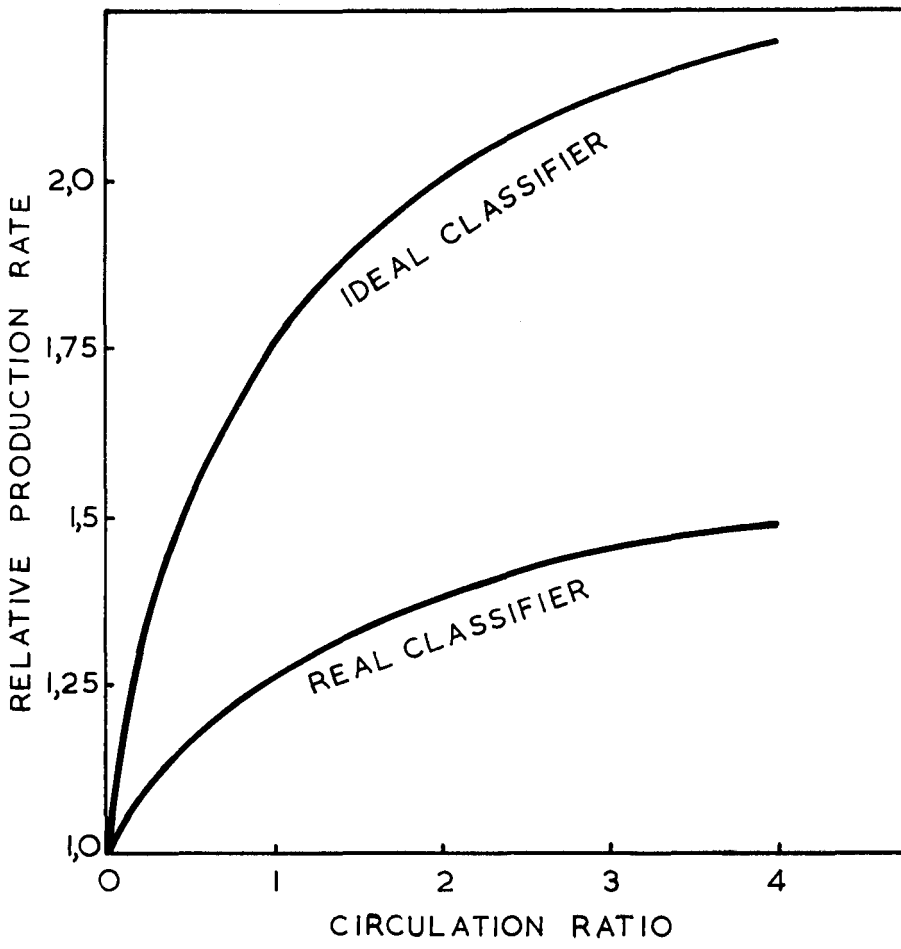

Fig. 5—Effect of classification on production (The product is 70 per cent passing minus 275μm. The ideal case assumes that the mill behaves as two equal, fully mixed reactors in series)

The benefit of improved classification does not end with increased grinding productivity. We assess grinding efficiency in terms of the percentage passing 74 μm, often losing sight of the fact that this is but one point on the size distribution curve of the final product; since the distribution does not plot as a straight line passing through the origin, no single point can characterize it. In grinding finer to increase the percentage passing 74 μm, we have to accede to a more-than-proportionate increase in the production of all the finer sizes. See Fig. 6, graphs (a) and (b). We accept this as justifiable or advantageous, even claiming that there is no such thing as overgrinding for cyanidation. If finer grinding results in better cyanidation recoveries because of an increase in minus 10 or minus 15 μm material, then 74 μm is the wrong yardstick for measuring grinding efficiency. The flattening of the curve at the coarser sizes that results from finer grinding can be deduced from Fig. 9 of the paper by Atkins *et al.*[1]; the slope of the curve between 75 and 150 μm for a grind of 80 per cent

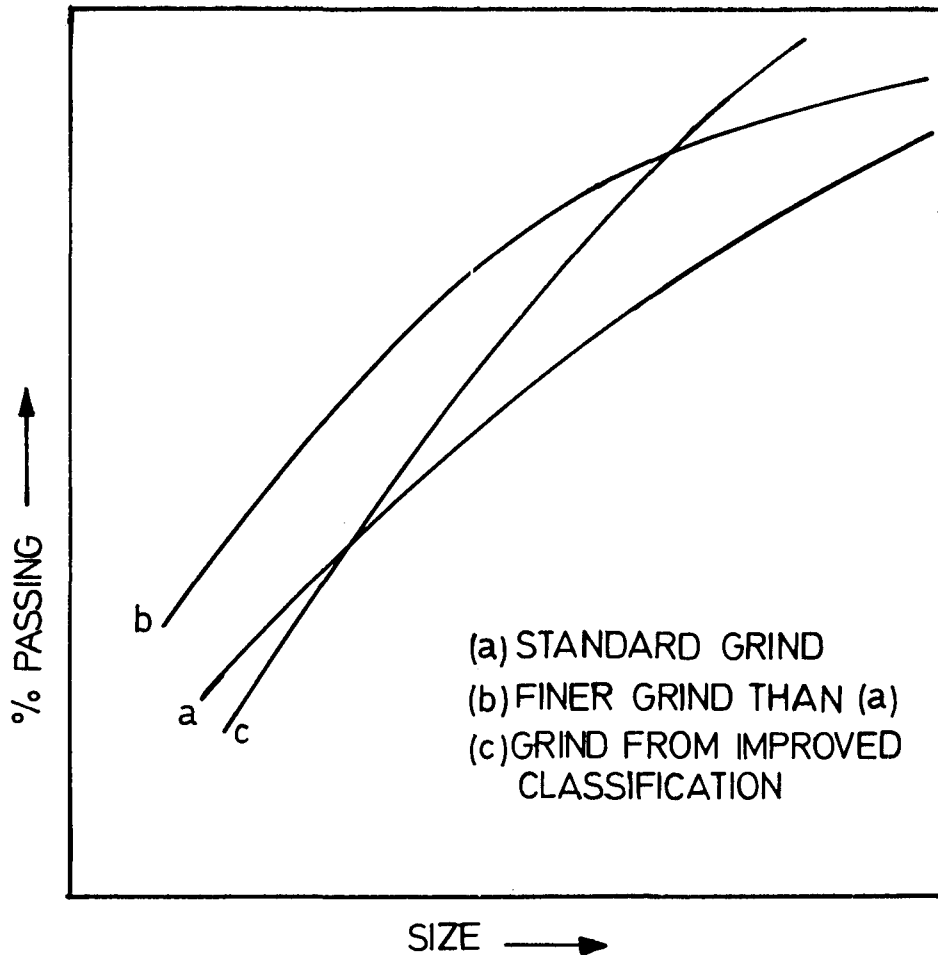

Fig. 6—Size analyses of final milled products

minus 75 μm is half that of the curve at a grind of 60 per cent minus 75 μm. Far from proving that the size distribution curve remains constant, Fig. 9 proves just the opposite; if the on-stream particle-size analyser is able to represent the entire size distribution by reference to a single size, it is not because the curve remains constant. It is quite possible for two different grinding systems to produce the identical percentage passing 74 μm, but the distribution curves may be completely different; were this not so, the manufacturers would be able to pre-calibrate the instrument, or a calibration carried out on one mine would be applicable at another mine. Better classification signifies a narrower spread of particle sizes (graph (c) of Fig. 6). Since it is the fine-sized material that predominantly influences thickening and filtration capacities, losses of dissolved gold, and consumption of flotation reagent, the case for improving separation becomes even stronger.

The main value of the particle-size analyser is its ability as a diagnostic tool to expose the variations in quality of the final product caused by disturbances to the circuit, which hydraulic classifiers cannot tolerate. Examples are supplied by Mr Atkins and by the detrimental result of intermittent ore feeding so dramatically portrayed in the other Union Corporation paper. The same effect has been noted on one of our plants in which a venturi flume is installed in the final pulp launder, a nuclear density gauge being fitted across the throat of the flume. These show an oscillation in density and mass flow rate that can be correlated with the stop-start feeding of pebbles to the secondary mills. The upsets are not as marked as in the Kinross case because only pebbles are involved, and not the total feed as in run-of-mine milling. A separating device, or combination of devices, not as sensitive to feed rate or quality as is a classifier would ride the storm, and would continue delivering a product of reasonable constancy with the minimum of stabilizing control. The analyser is the ultimate in sensing instruments for providing the means to avoid perturbations, but it must be recognized that instruments and control systems can improve a process only within its own limitations — and classification is extremely limited in its capacity to fulfil its function.

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Authors' reply

Mr De Kok's deep interest in classification and sizing is acknowledged. His major practical contribution in this field dates back to the early fifties when, at Rand Leases, he was involved, firstly, with the experimental introduction of the cyclone into milling practice and, later, with the conversion of the entire milling plant from mechanical classification to cyclones.

After proving on a plant scale that the cyclone could be accepted as a successful alternative to conventional classification, Messrs Dennehy and De Kok, in a paper published in March 1953, elaborated further on the advantages of the cyclone, and indicated that the much smaller size of the cyclone when compared with mechanical classifiers of similar capacity, and the absence of moving parts, were perhaps two of its major attractions. These advantages, considered in their wider context, pointed to far-reaching potential benefits for the mineral-processing industry as a whole, and there is no doubt that the gold-mining industry, for one, has gained a great deal from this bold step.

Mr De Kok's contribution gives the impression that, in our attempts at studying the effects of high circulating loads on mill performance, scant attention was given to classification. It should be pointed out that, in order to obtain the higher circulating loads required to test mill performance, attention was of necessity directed in the first place at the operation, condition, and control of the mechanical classifiers. In pursuing this line of action, we were tempted to adopt the cyclone, following its successful introduction at Rand Leases. In fairness to the mechanical classifiers, and to obtain meaningful comparisons between mechanical classifiers and cyclones, it was felt that the introduction of the cyclone should only follow confirmation that the mechanical classifiers were running under optimum conditions.

In pursuing this more cautious approach, we discovered, for example, that the raking motion of all the classifiers had been incorrect since their installation. The locus of the lower edge of the rear blades of the rakes was found to be as indicated in Fig. 1, whereas the correct path is shown in Fig. 2. To correct this defect, the main supporting beam of the raking mechanism had to be moved bodily by about 25mm. This modification restored the proper raking motion and resulted in increased capacity and efficiency.

The dramatic improvements in overall mill performance that had resulted from the higher circulating loads and had resulted in finer grinding, improved power utilization, and increased gold extraction, pointed to the potential advantages of even higher circulating loads. It was at this stage, therefore, that the introduction of the cyclone was being considered.

In contemplating the effect of its introduction, attention was directed at possible restricting influences that would limit pulp flow through the grinding mills. These considerations directed attention at the design and operation of the pan lifters, which resulted in modifications to the design of the pan lifter and, later, to the experimental introduction of peripheral discharge.

It is conceded that, when the parameters of ancillary equipment in the circuit of the first peripheral discharge mill were considered, it was never realized that the high pulp flows achieved in practice would be reached. As a result, both pump and cyclone were under-designed.

In the design of the end-discharge mill circuit at St. Helena, four cyclones were installed in parallel in such a manner that they could be phased in or out in accordance with changes in mill output.

With reference to Mr De Kok's observations on the use of cyclones on our larger mills and the effect of impaired separating efficiency, it should be pointed out that one major benefit achieved from the replacement of mechanical classifiers by cyclones was continuity of operation, and therefore assured production from a substantial capital investment. Cyclones, in spite of their inherent functional weaknesses, indeed came to the rescue of a production facility that had suffered heavily from repeated breakdowns.

In the planning of research that is aimed at improving the comminution process in general, it is important to remember the distinction between objectives in different fields. In the cement and iron ore industries, for example, particle size, or particle-size distribution *per se*, becomes the main objective. These industries deal with homogeneous materials. In the gold-mining industry, the obvious objective is the recovery and product-

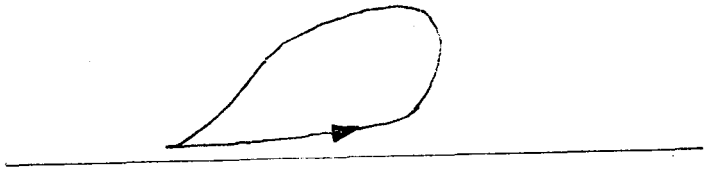

Fig. 1—Locus of lower edge of rear blades—as found

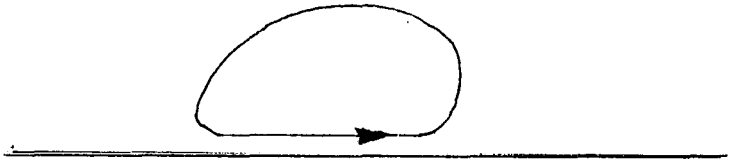

Fig. 2—Locus of lower edge of rear blades—as corrected

ion of gold. With this in mind, the efficiency of separators that form part of the milling circuit ought therefore to be determined and expressed in terms of the separation of particles of free gold and free gangue (and other free minerals) as against particles in which gold is still locked and therefore in need of further comminution.

Mr De Kok has mentioned the use of screens as sizers of potential value. When comparing screens with cyclones (as cyclones were originally compared with mechanical classifiers), we observe the re-introduction of moving parts, the possibility of mechanical failure with interruption of flow, the need for maintenance, and the provision for larger operating areas. Admittedly, fine screening has received much attention over past years, and new concepts in screen operation and design are still being proposed.

It is interesting to note in passing that most, if not all, of these developments are of overseas origin, with local talent lagging behind, in spite of the immense financial resources that the industry could muster in searching for and establishing improved techniques.

Our acceptance of the screen as a sole or partial separating device should be approached with caution. Particles passing through a screen of selected aperture would admittedly be free of oversize, but the screen cannot differentiate between free and locked particles. Herein lies the danger, and it would still be the function of the hydraulic classifier, with its accepted inefficiencies, to safeguard the operation by returning heavy particles in the screen undersize back to the mill. In very simple terms, in a gold ore in which the specific gravity of the gold and silver particles could be taken as 18,0 and that of the gangue as 2,70, any particles of whatever size with a specific gravity of 2,71 and over would be suspect.

OFS Branch

Minutes of the General Meeting Held at the President Steyn G.M. Ltd Canteen on Wednesday, 13th November, 1974, at 3.55 p.m.

Mr D. A. Smith (Chairman of the O.F.S. Branch) was in the Chair. There were also present:

Two Fellows

Messrs A. N. Shand and E. T. Wilson (Committee Members)

Four Members

Messrs D. D. Comley, A. F. Goetzsche (Committee Member), N. Mayer, and E. R. Penny.

Five Associates

Messrs D. F. Cleary, E. C. Hunter, A. J. Johansen, R. R. Perkin, and W. M. Todd

Thirty-five Visitors

Total Present

Forty-seven.

Welcome

Mr Smith declared the meeting open and extended a warm welcome to all those present. He expressed his gratitude for the good attendance, and thanked the management of President Steyn G.M. Limited for hosting the local Branch of the Institute.

A special welcome was extended to Messrs G. Bowmer and N. Bennett of C.C.L. (S.A.) (Pty) Limited.

Apologies

Apologies for non-attendance were received from Messrs J. W. Briedenhann, H. M. W. Eschenberg, H. G. Mosenthal, and Z. J. Lombard.

Minutes of Previous Meetings

The Minutes of the General Meeting held on the 12th June, 1974, were taken as read, and their adoption, proposed by D. D. Comley and seconded by E. T. Wilson, was carried. Likewise, the Minutes of the Annual General Meeting held on the 31st July, 1974, were taken as read, and their adoption, proposed by A. N. Shand and seconded by D. D. Comley, was carried.

Main Business

The Chairman introduced Mr G. Bowmer and Mr N. Bennett from C.C.L. (S.A.) (Pty) Limited, who had kindly consented to describe and demonstrate the Rauföss implosive method for the compression jointing of aluminium and steel cables and reinforcing bars with the use of detonating cords.

In his address, Mr Bowmer outlined the principle of the technique, and went on to describe the numerous applications that had proved successful. This talk and the most convincing demonstration that followed were well received.

The Chairman called upon Mr E. T. Wilson to thank the guests. In his vote of thanks, Mr Wilson said he was sure that the afternoon's instruction had been of great interest to the members and visitors present, and pointed out that it had again highlighted how particular methods or sciences were being applied on specialized bases.

Closure

The Chairman thanked members and visitors for their attendance and declared the Meeting closed at 5.30 p.m., after which refreshments were served.

Minutes of the General Meeting Held at the Welkom G.M. Co. Ltd Canteen on Wednesday, 12th February, 1975, at 4.00 p.m.

Mr D. A. Smith (Chairman of the O.F.S. Branch) was in the Chair. In addition, there was a total of 33 members and visitors present including:

Four Fellows

Messrs M. G. Cullen, Z. J. Lombard, A. N. Shand (Committee Member), and E. T. Wilson (Committee Member)

Six Members

Messrs F. J. Bayley, D. E. Couperthwaite, E. J. Dominy, B. J. Drysdale, H. G. Mosenthal (Committee Member), and E. R. Penny

One Associate Member

J. W. Briedenhann

Seven Associates

Messrs D. F. Cleary, W. F. de Lange, A. J. Johansen, R. A. R. Kebble, R. R. Perkin, C. F. K. Poulton, and C. V. van Heerden