MINING AND TREATMENT PLANT PRACTICE AT THE FINSCH MINE, DE BEERS CONSOLIDATED MINES, LIMITED

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SYNOPSIS

Planning for the Finsch opencast mine is described with particular reference to the proposed stages of mining, the method used to determine the limits of surface waste rock stripping and the derivation of the waste rock mining rate. Actual mining practice is described and the possibility of installing a skipway into the open mine at a later date is discussed. In addition, the waste dumping policy is referred to: this includes the cutting of slots through the hill on the west side of the pipe to increase tramming speeds.

The crushing, washing and recovery sections of the treatment plant are described together with the various changes being made to the circuit. The most important of these are the installation of X-ray sorters for the final recovery of +7 mesh diamonds, and the vibrating grease belt for the recovery of −7 mesh diamonds.

INTRODUCTION

The Finsch Mine, a subsidiary of De Beers Consolidated Mines, Limited, is situated approximately 90 miles west of Kimberley, and some two miles from the village of Lime Acres, in the Northern Cape. Lime Acres is a company village shared with the Northern Lime Company which operates a limestone quarry and works near the township. The mine is near the southern end of a well mineralized belt producing limestone, asbestos, diamonds, manganese and iron ore, and the whole area has developed rapidly in the last 20 years.

Power for mining operations is drawn from the Electricity Supply Commission. The mine is conveniently close to the main Kimberley-Postmasburg railway line. Water is drawn from the Vaal River through a 57 mile State-owned pipeline at present temporarily operated by mine staff.

DISCOVERY AND PROSPECTING OPERATIONS

The diamondiferous orebody is situated on the farm Brits, which is State-owned ground that has been leased to the Company for the period of mining operations. There are indications of interest in the area by prospectors (Richter, de Bruin and others) over a period of some 30 years and the discovery would possibly have been made years earlier but for a law preventing prospecting for precious stones on State-owned land. Although the pipe formed a clear depression on the hill where it was situated, it was overlain by some 5 to 40 ft of banded ironstone rubble which was washed into the depression from the hillside above the pipe and obscured the deposit.

In 1958, Mr Allister Thornton Fincham applied for, and was granted, the right to prospect for base minerals on the farm Brits. Asbestos is known to occur in the area and, initially, was Mr Fincham's primary interest. However, during the prospecting operations, garnets were discovered and it was realised that diamonds might be present. In 1960, when the Precious Stones Amendment Act was passed, a company

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was formed in which Messrs Fincham and Schwabel were the main shareholders, and this company was granted a permit to prospect for precious stones. The first recovery operations took place in November, 1961, and the ground proved diamondiferous. Prospecting continued but Mr Fincham came to an agreement with De Beers Consolidated Mines, Limited, to check the prospecting results. This work commenced in November, 1962, with a view to confirming results to a depth of 200 ft.

At this stage it should be mentioned that the sampling of a diamond pipe requires large samples for accurate results. The ratio of diamonds to kimberlite is of the order of 1 part in 8 million and, with a highly irregular distribution, small samples can give erroneous results.

The De Beers prospecting programme consisted of:

(*a*) Drilling a systematic pattern of seventy-eight 54 in. diameter holes to a depth of 120 ft.

(*b*) Sinking 5 vertical winzes ($\pm$ 6 ft square) to 200 ft.

All ground from the above operations was treated to determine diamond content. Except for isolated intrusions, all samples indicated diamondiferous ground at payable values.

(*c*) Four diamond drill holes were drilled to a depth of 1,000 ft at 100 ft inside the estimated position of the pipe contact. Only one hole intersected country rock, at a depth of 620 ft.

The results of the prospecting were such that in May, 1963, the share capital of Finsch Diamonds was purchased by De Beers for R4,500,000. Two months later the property was proclaimed a mine.

GEOLOGY OF THE DEPOSIT

In common with other major primary diamond deposits, the Finsch orebody is a near-vertical volcanic intrusion. It is roughly circular in shape, with a diameter of from 1,500 to 1,700 ft and covers an area of approximately 45 acres. Although all diamond pipes tend to decrease in area with depth, available evidence indicates that at Finsch this decrease is not significant and that the walls of the pipe are near vertical down to 1,000 ft.

The pipe is situated in the Lower Griquatown ironstones of the Campbell-Rand Series. These ironstones form a shallow syncline trending north-south, overlaying the Campbell-Rand Dolomites. The latter extend from the Vaal River to the Asbestos Mountains, and appear on the other side of the syncline at Postmasburg; they are estimated to be some 10,000 ft thick in the area. This series is closely allied to the Transvaal System both in sequence and in rock types.

The country rock surrounding the pipe therefore consists of hard abrasive banded ironstone down to approximately 450 ft, followed by dolomite. It is of interest to note that both crocidolite asbestos and silicified asbestos (Tiger's Eye) occur on Brits although not in economic quantities.

The kimberlite forming the orebody is a relatively soft ultra-basic rock containing numerous inclusions of siltstone, dolerite, ironstone, shales and lava. Apart from the siltstone, these inclusions are generally too small to be mined selectively. In the upper 100 to 200 ft the kimberlite has been weathered to 'yellow ground'. This gradually becomes less decomposed with depth until the unaltered 'blue ground' is reached.

MINE PLANNING AND OPERATIONS

The definition of a load:

A load is the traditional volume measure used in diamond mining and, at Finsch, is defined as 10 cu ft of ground *in situ.* On the upper levels, in yellow ground, the load weighs approximately 1,200 lb but this increases with depth. It reaches a maximum of approximately 1,600 lb at 400 ft below surface in the unweathered kimberlite. The banded ironstone weighs 1,900 lb per load while the corresponding figure for dolomite is of the order of 1,800.

The use of a volume measure in ore and waste rock of varying densities has advantages over a tonnage measure as all open pit calculations are essentially based on volume.

Mining operations to date:

(A) *Overburden:*

The upper portion of the pipe, to an average depth of 29 ft below surface, was heavily contaminated with ironstone rubble from the surrounding country rock. This ground is known to contain diamonds, but cannot be treated effectively in the plant, which is designed to handle ground with a low content of heavy minerals.

Stripping of the overburden had therefore to be carried out during the construction of the main plant in order to expose sufficient ore for plant feed. The kimberlite content of the overburden increases with depth and the overburden is roughly classified into two zones based on this content and termed:

(*a*) 'Overburden'—extending to an average depth of 16 ft.

(*b*) 'Contact Zone'—extending from an average depth of 16 ft to 29 ft.

(i) *Overburden Mining:*

Full-scale mining operations commencted in October, 1964. The overburden was passed through a screening plant, the $-\frac{1}{2}$ in. fraction being stockpiled for future re-treatment, and the $+\frac{1}{2}$ in. fraction being discarded as waste.

It was desirable to mine the overburden as rapidly as possible in order to:

(*a*) Minimise the requirements of earthmoving machinery, as equipment was only purchased to match ore requirements with the treatment plant at full output. Stripping had therefore to be completed before full production was reached.

(*b*) The screening plant utilized equipment destined for the main treatment plant.

Overburden stripping fell behind schedule and was not completed until January, 1966. Sufficient ore was, however, exposed for mining, but a separate, unscreened, overburden dump had to be established. A total of 3·3 million loads were mined in this programme.

(ii) *Contact zone mining:*

This ground contained both a high proportion of clay from weathered kimberlite and unbroken kimberlite lumps. Consequently it could not be screened efficiently and has, therefore, been stockpiled for future treatment. The rate of mining was determined by the spare capacity of equipment largely committed to ore loading.

The extent of the Contact Zone was initially underestimated due to the irregular footwall contact of the material and, in certain areas, lenses persist to below 50 ft. Due to the irregular nature of the footwall, mining was carried out selectively using front-end loaders handling ripped ground.

Loading started in December, 1964, and 2·8 million loads have been mined and dumped. A limited amount of Contact Zone is likely to be exposed on lower ore levels.

(B) *Ore mining:*

Yellow mining, as it is termed, started in December, 1964, to provide feed for the pilot plant. In October, 1965, the main plant was commissioned, and since then ore production has steadily increased and is now at a daily rate of approximately 19,000 loads.

Three ore levels, 40 ft apart, have been established. The initial (30 ft) level, is mined out and has been replaced by the 150 ft level.

(C) *Waste rock stripping:*

Equipment purchases for this work were made in 1967 and the monthly tonnage handled was gradually increased as equipment was delivered and faces established. Output is now running at 26,000 loads per day.

Proposed stages of open cast mining (see Fig. 1):

At an early stage, the decision was taken to mine the pipe in four stages as follows:

Stage 1 (see Fig. 2)—This consisted of mining entirely within the orebody at a fairly flat slope angle formed by 100 to 200 ft wide benches and 40 ft faces. This had the following advantages:

(i) Mining during this stage was fairly simple and allowed supervision to concentrate on the more important phase of plant construction.
(ii) It allowed operators and supervisors to become familiar with the operation and maintenance of earthmoving equipment.
(iii) The potential of the equipment was tested. On the basis of this experience the equipment for waste rock stripping was purchased.

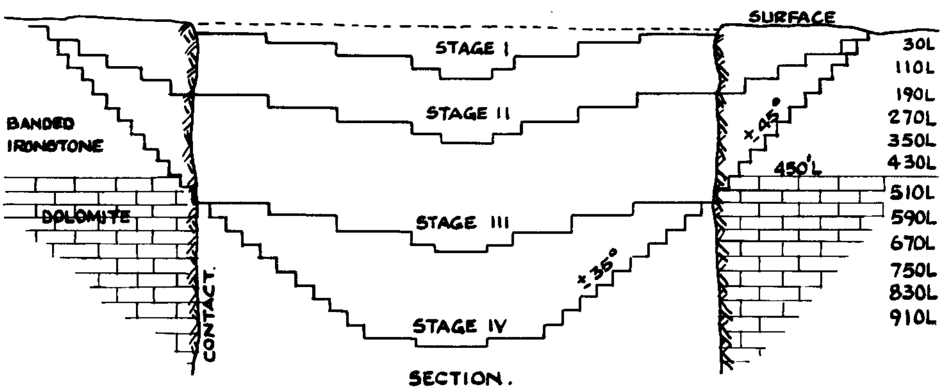


Fig. 1—The stages of mining the Finsch pit as an opencast

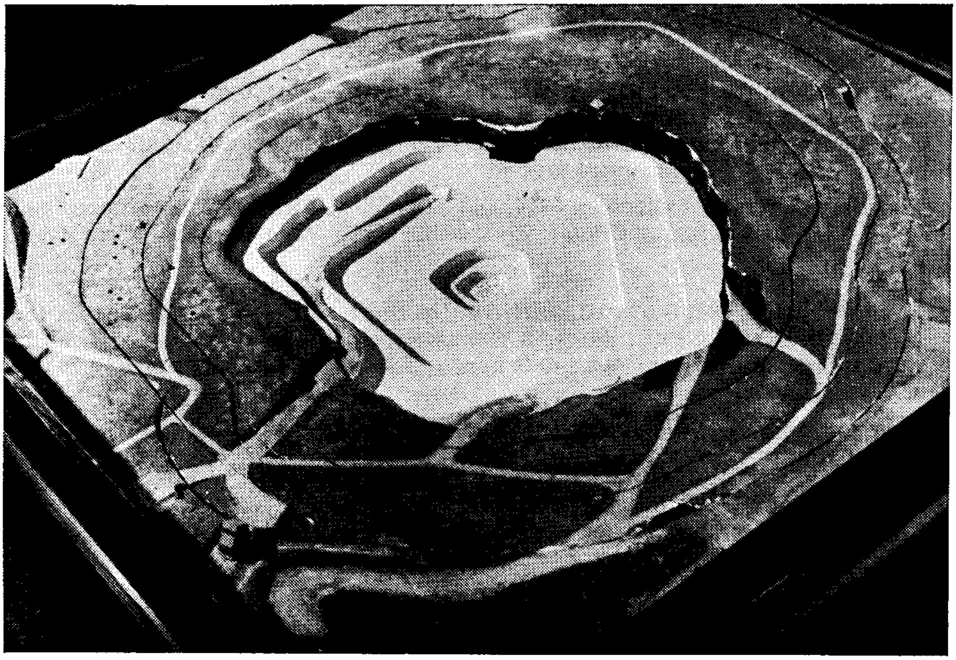


Fig. 2—Model of the mine showing the completion of Stage I mining

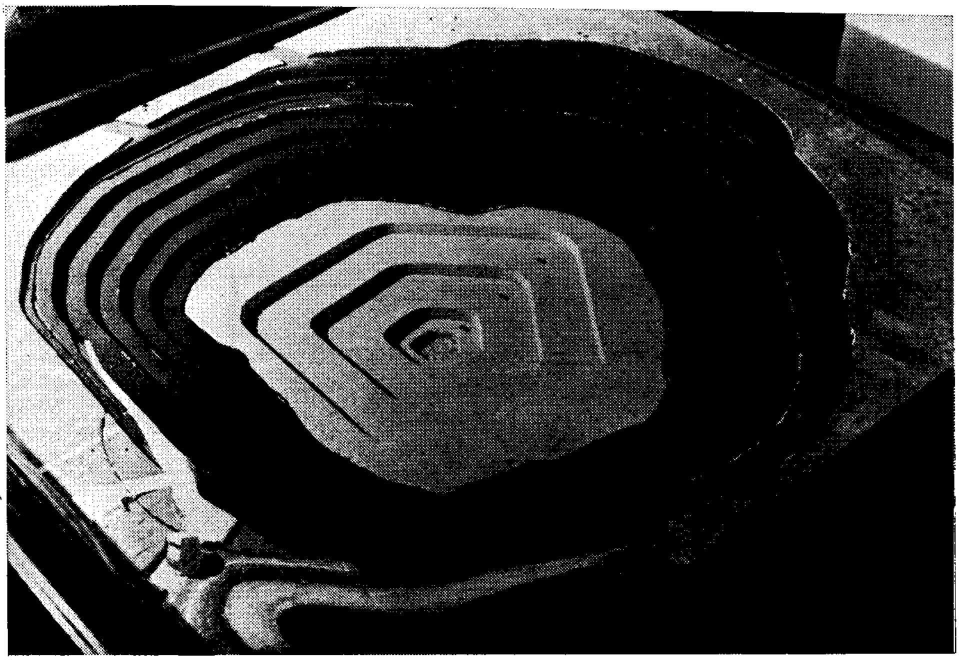


Fig. 3—Completion of Stage II mining. The bench width in waste rock is 100 ft while that in ore is 200 ft. Waste stripping has reached the final surface limits

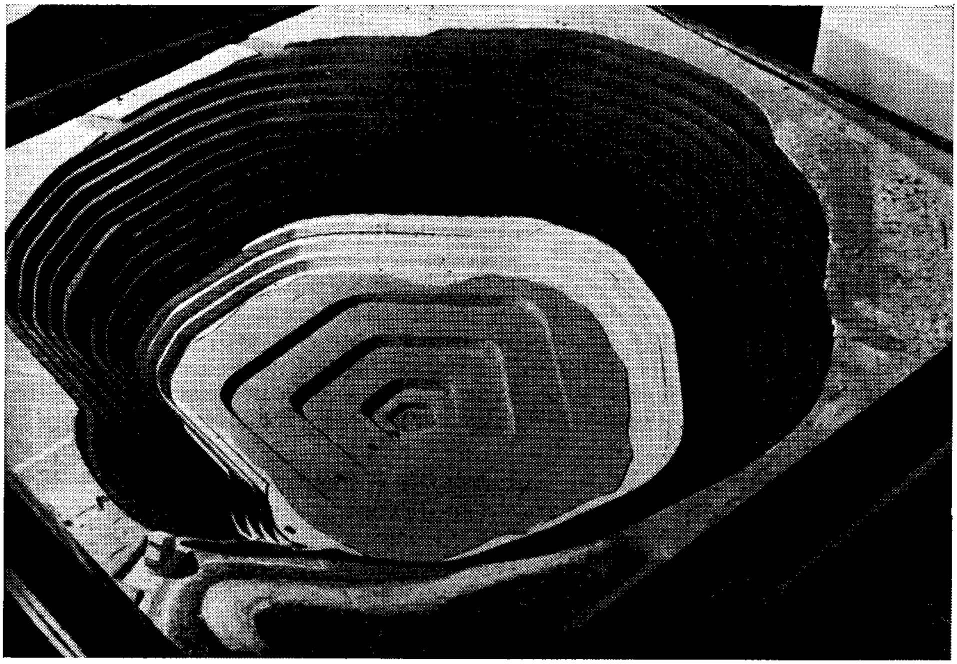


Fig. 4—Completion of Stage III mining. The waste rock benches have now been steepened to their final angle. The primary crusher and skipway can be seen on the bottom left-hand corner of the photograph. For simplicity the roads are not shown. The change from banded ironstone to dolomite wall rocks can also be seen.

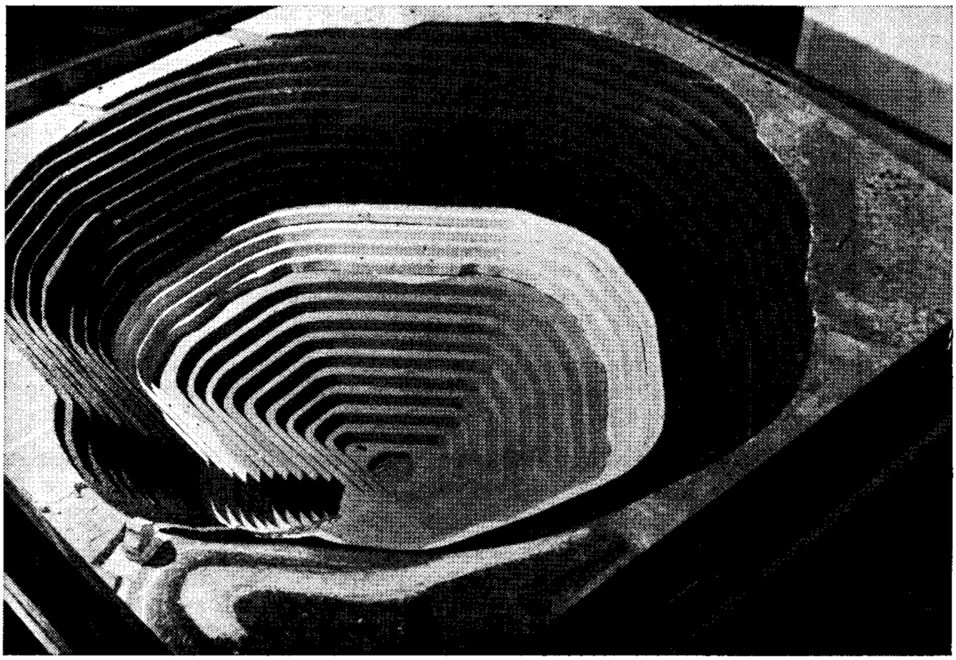


Fig. 5—Completion of Stage IV mining

Stage II (see Fig. 3)—This is the start of waste rock stripping, with the waste being mined back to pre-determined pit limits. In this stripping, the minimum bench width is again 100 ft to allow for the easy operation of heavy equipment. Face heights are 40 ft as in ore, and the same slope angle is thus maintained as in Stage I. The final slope angle that can be achieved in waste rock stripping is not known and work has to be carried out during this stage to determine the likely value. The mine is at present in the early part of Stage II.

Stage III (see Fig. 4)—The waste rock benches are now mined back to the limit set by the maximum slope angle. Most of this stripping will be carried out in banded ironstone where a final angle of approximately 45 degrees is anticipated. The profile of the working faces in this stripping will be maintained at the same flat angle as Stage II, mining being bounded by the 45 degrees line and the ore contact. It should be noted that a flatter angle is maintained in ore during Stage III due to the weaker nature of the ground.

Stage IV (see Fig. 5)—Waste rock stripping is completed in Stage III, and the ore benches are now mined back to the maximum safe slope angle. It is expected that an angle of at least 35 degrees will be achieved.

Notes on graphs used in planning:

Considerable use has been made of graphs for the determination of the waste rock stripping policy. It is possible to plot curves for both the maximum and minimum rates of waste stripping for any opencast operation.

In the case of the maximum rate (see Fig. 6), waste rock stripping is done to the final pit boundaries before the ore mining on the next level can be completed. As indicated in the diagram, when the ore of level 1 is mined out, the waste area A must be removed before the ore on level 2 can be fully mined. Similarly, level 3 can only be mined out when the waste area B has been stripped. This develops the maximum curve shown in Fig. 7.

The case for minimum waste stripping rate is also shown (Figs. 7 and 8). At the completion of mining the ore on level 1, waste blocks A must be removed before ore level 2 can be fully mined. Similarly waste blocks B must be removed before ore level 3 can be fully mined out.

The actual graph of the waste rock stripping planned for Finsch Mine is given in Fig. 9. The minimum curve for Stage II is shown, then the slope of the curve reverses for Stage III due to the proposed sequence of mining operations. In effect, the latter is a maximum curve. It should be noted that:

(*a*) The slope of the minimum curve at any point represents the instantaneous stripping ratio, or the additional number of units of waste rock to be stripped to release one additional unit of ore.

(*b*) The average stripping ratio required, at any particular ore tonnage, can be read off the graph by the intersection of that tonnage with the minimum curve.

(*c*) With any stripping policy it is necessary to remain between the maximum and minimum limits. If the maximum curve is intersected, 'box cutting' into waste rock would have to be resorted to. If the minimum curve is intersected, the overall slope in ore would have to be increased.

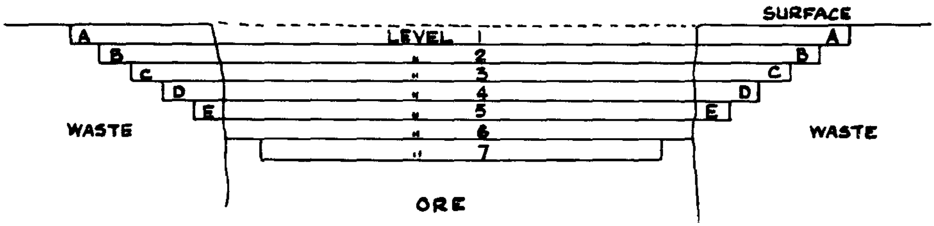


Fig. 6—The maximum rate of waste stripping

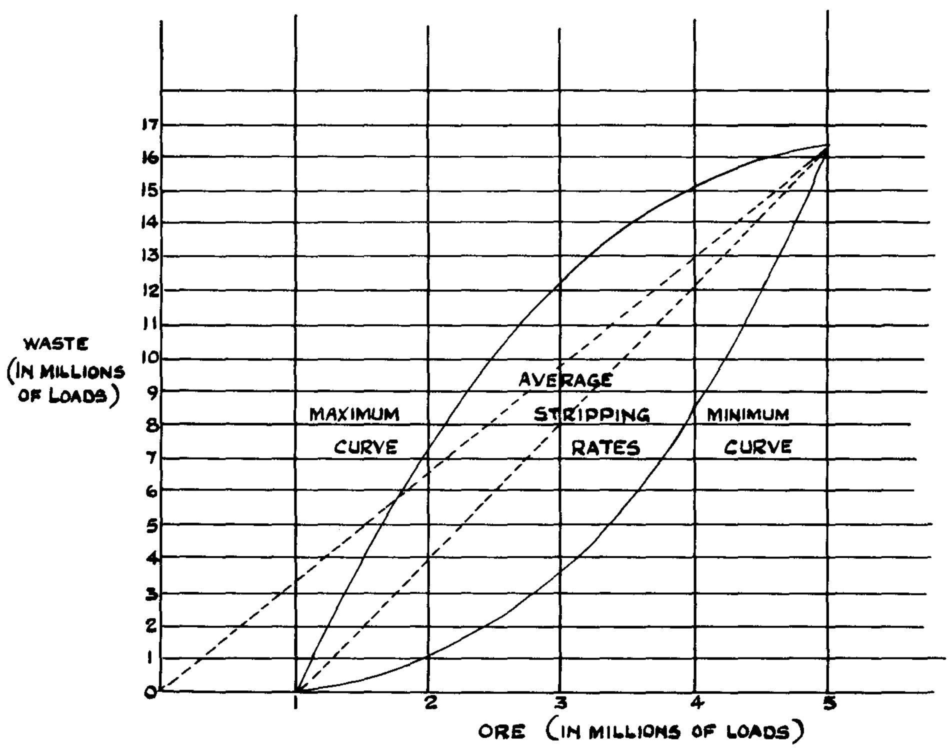


Fig. 7—Curves of maximum and minimum stripping rates

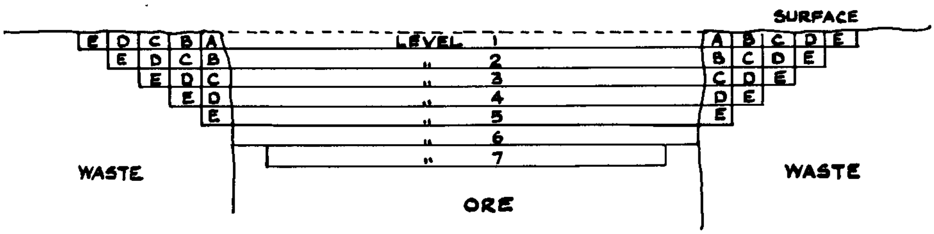


Fig. 8—The minimum rate of waste stripping

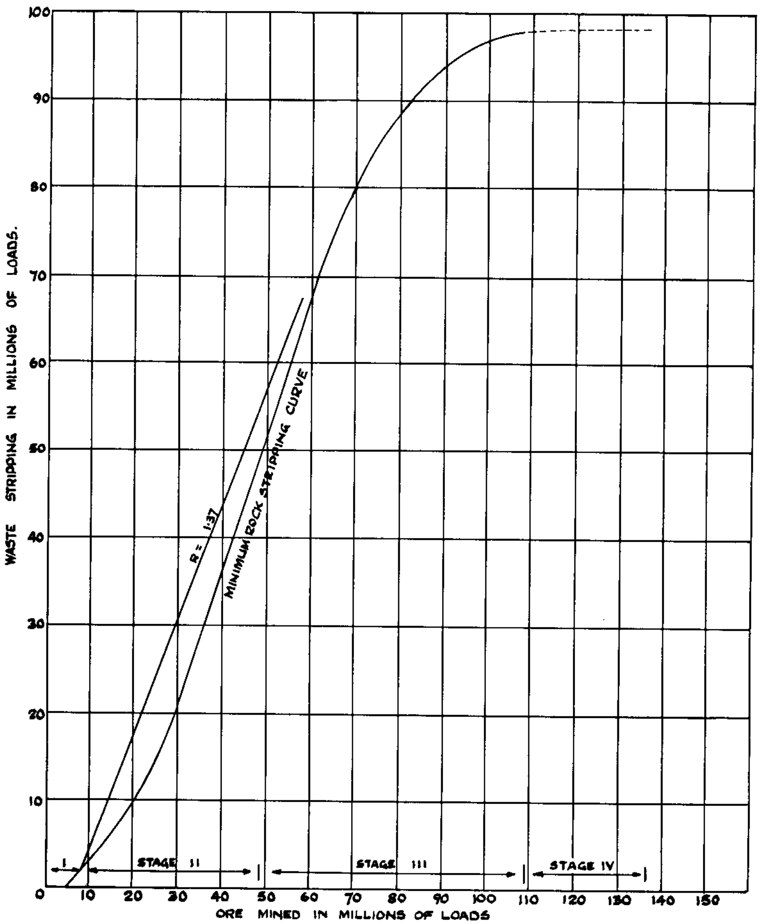


Fig. 9—Waste Rock stripping rate at Finsch Mine

Defining the surface limits of waste rock stripping:

In all opencast mining operations it is essential for future planning to have some indication of the final pit limits. The method adopted for this calculation at Finsch Mine is outlined below.

In the mining of the Finsch pipe, several factors have to be taken into account:

(*a*) The pipe is roughly circular in shape and equivalent in area to a circle with a radius of approximately 800 ft. All planning has been based on near vertical wall rocks, as previously mentioned.

(*b*) Although still to be confirmed by sampling, the trend in values with depth is such that it is anticipated that payable ore will persist to depths well below opencast limits.

The problem is therefore not merely to continue with opencast mining until the production cost is equal to the value of the product, but rather to continue opencast operations until the cost of such mining is equal to the cost of underground mining.

(A) *The critical stripping ratio, R:*

All mining costs increase with depth. However, due to the inverted cone shape of an opencast mine, which is constantly being enlarged, these costs rise with increasing depth far more rapidly than underground mining costs. A critical depth is reached where the two costs are equal and a changeover has to be made from opencast to underground methods.

In forecasting the depth of this change from surface to underground methods, the relevant cost figures are those that will pertain when this change takes place. To forecast these accurately is clearly impossible and present day costs must therefore be used with the following reservations:

(*a*) Inflation is unlikely to affect the result significantly as it will tend to have a similar effect on all items.

(*b*) Differential cost changes will be more significant in view of the lower man-power requirements of opencast mining and the trend of wages to rise more steeply than prices.

The economic stripping ratio (R_E) is defined as the value of the instantaneous stripping ratio when the cost of opencast mining is equal to the cost of underground mining. In other words, the ratio of waste to ore for the thin slice represented by *LABL* in Fig. 10 is equal to R_E, if *AB* is the 'critical' level at which the two mining costs are equal.

Then, if S = cost per load, of opencast ore mining,
U = cost per load, of underground mining,
W = cost per load, of waste rock stripping,
$\bar{R}$ = instantaneous stripping ratio,
R_E = economic stripping ratio.

the cost per load of opencast mining $= S + \bar{R}W$.

When the cost per load of opencast mining equals the cost per load of underground mining, i.e. $\bar{R} = R_E$ then

$$U = S + R_E W$$

and
$$R_E = \frac{U - S}{W}$$

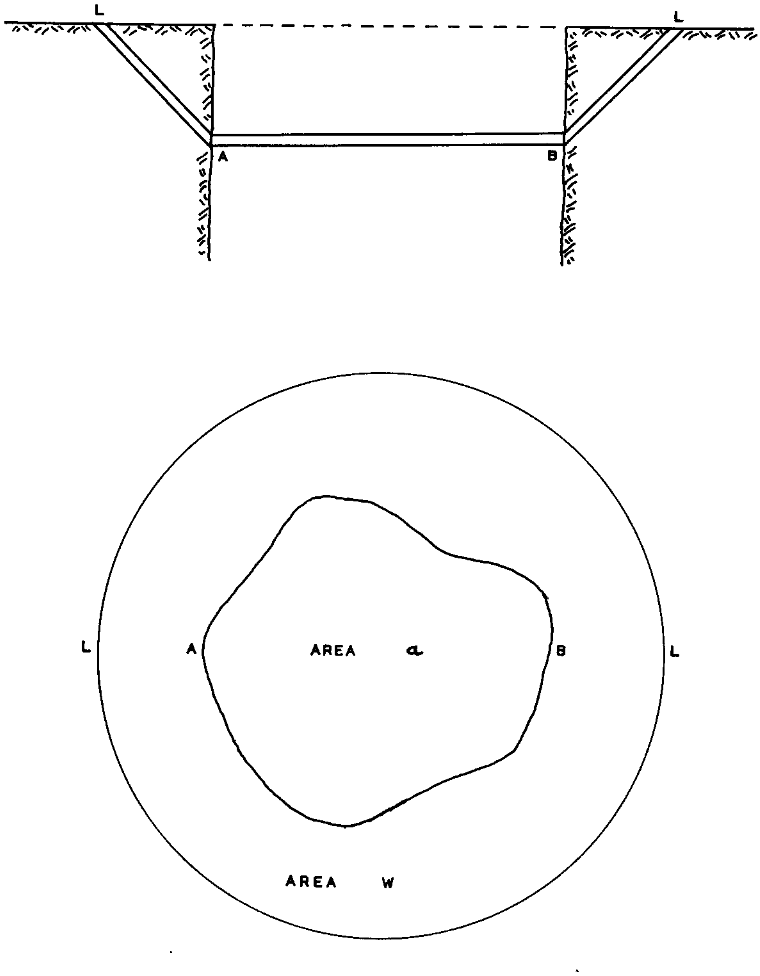


Fig. 10—Indicating the instantaneous stripping ratio

This value for R_E has been calculated for Finsch Mine to be 1·79. Using this figure it is possible to obtain the surface limits of stripping operations.

(B) *The mining limits associated with* R_E:

It can be proved that the following approximation holds good for orebodies having a regular shape.

(i) The instantaneous stripping ratio is equal to the ratio of the respective plan areas of waste stripping and ore mining.

(ii) Hence, if the surface waste stripping limits are fixed, the instantaneous stripping ratio is the same whatever pit slope is taken.

Therefore if L = Total plan area of the excavation (ore plus waste) at the limits LL (Fig. 10),

a = Plan area of ore,

and W = Plan area of proposed waste stripping at surface,

Then, $L = a + W$

and $\bar{R} = W/a = \dfrac{L - a}{a}$

Therefore $L = a(\bar{R} + 1)$

The economic stripping ratio R_E, at which point the costs of surface and underground mining are equal, is known and in the Finsch case is 1·79. In addition the value of a, the area of the orebody, is known. It is therefore possible to calculate the surface limits of waste rock stripping by solving for L in the equation above. For Finsch Mine, the result showed that waste rock stripping should be carried to 500 ft (as shown in Fig. 1) from the North pipe contact with a corresponding increased distance to allow for the hill on the South side of the mine.

Using these limits, and assuming that a final angle of 45 degrees is achieved in waste and a corresponding angle of 35 degrees in ore, then the changeover to underground mining will take place when the open pit has reached a depth of approximately 900 ft (Fig. 1).

The derivation of the waste rock mining rate:

As mentioned above, opencast mines can adopt one of three basic waste stripping policies.

(*a*) Mine only as much waste at any time as is required to release the ore called for. That is, mine at the instantaneous stripping rate, following the minimum curve on the graph. Theoretically this method tends to give the greatest overall profits when calculated on a present value basis, as the more intensive period of waste stripping is deferred into the future. This method, however, would be impractical at Finsch for the following reasons:

(i) The ultimate fleet size is dependent on the maximum value of the instantaneous stripping ratio.

(ii) Any delays in the waste rock programme would immediately affect ore mining.

(*b*) Follow the maximum curve of the graph, that is, mine out each waste bench to the final limits before mining of the next waste bench is commenced. This will normally result in a high proportion of waste to begin with, gradually decreasing as the pit deepens. This is clearly unsuited to most operations except where the waste rock fleet is eventually required for ore mining.

(*c*) Mine at a fixed stripping rate above the minimum curve, which results in a fixed waste rock call which may extend for the life of the mine. This is not possible at Finsch Mine, since the overall stripping ratio is unknown, being dependent on the final slope angle. Hence the decision to separate Stages II and III and to mine at a constant rate during Stage II. The stripping rate for Stage III can, however, be estimated using assumed values for the final slope angle.

As can be seen from the graph in Fig. 9, the minimum curve for Stage II has been used to determine the waste rock stripping rate. The rate chosen had to be such that the minimum curve was not intersected and, for safety, at $R = 1 \cdot 37$, is at a higher rate than required. This rate, based on 26,000 loads per day of waste, is shown on Fig. 9. The mining rate in Stage III will probably follow the curve with a gradually decreasing rate of rock stripping.

The graph also shows the effect of raising the ore call without a corresponding increase in the waste rock call. This decreases the slope of the waste stripping line bringing it closer to the minimum curve.

The mining layout (see Figs. 11 and 12)

The initial ore level, the 30 ft level, had to be laid out to contend with a non-uniform rise between the north and south contacts. A face was established, near the north contact, across the pipe on an east-west axis. This face was then advanced south at a positive gradient of 1 per cent (a drainage requirement) with a maximum face height of 60 ft being reached at the south contact. The footwall of this level thus allowed future levels to be mined at a relatively constant face height of 40 ft.

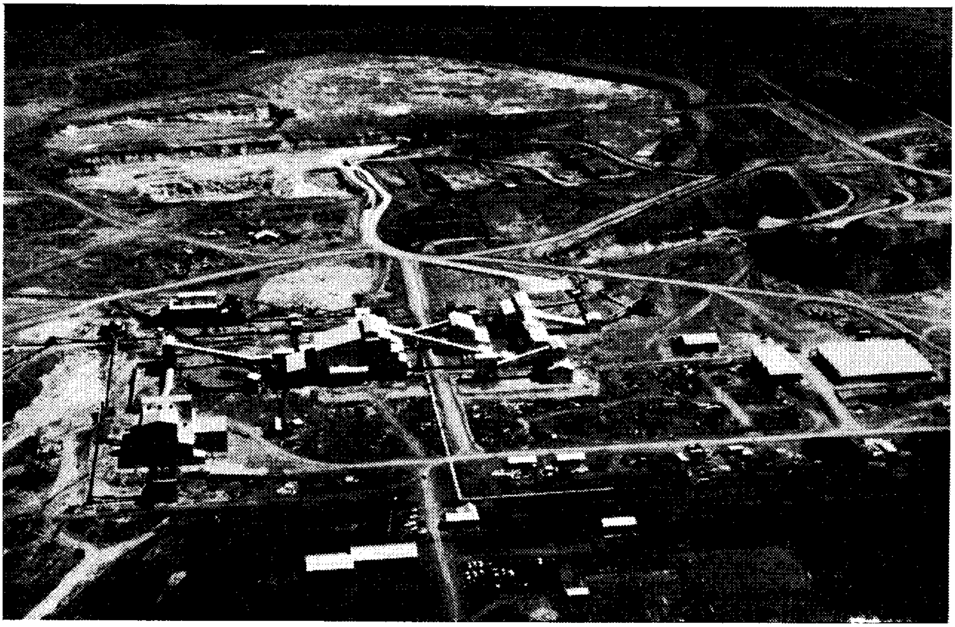


Fig. 11—A view of the mine taken early in 1966. The orebody has been fully exposed and the E-W face being established across the pipe can clearly be seen.

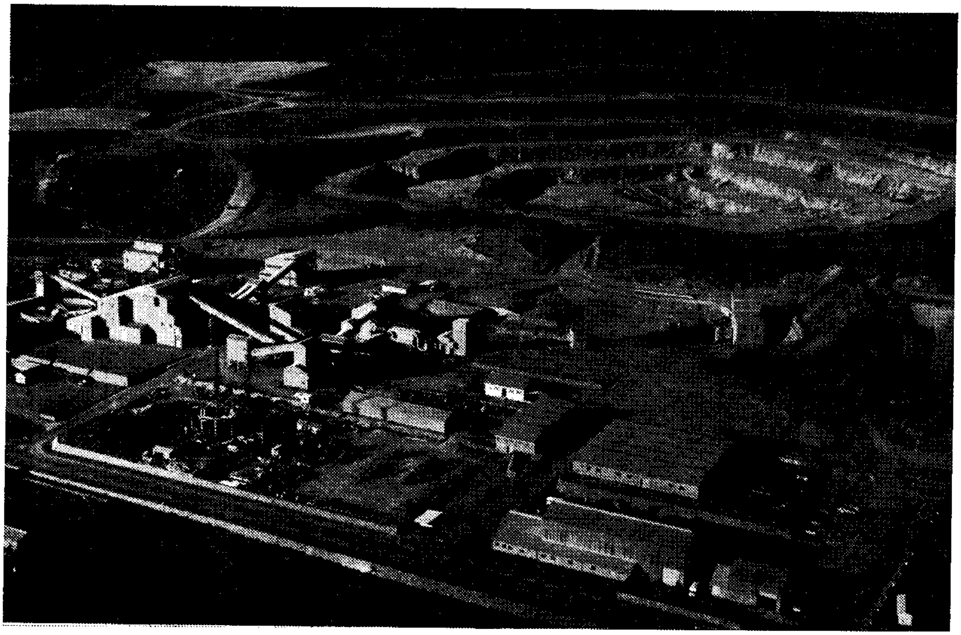


Fig. 12—A recent view (mid 1968) of the mine, looking south, showing the workshops and plant area in the foreground and the open pit in the background

Initially, pit planning was limited to the mining of Stage I only. Each level was to be mined out to its Stage I limits before the following level was fully established from the spiral road into the pit. In addition, each level was to be established with a single main face perpendicular to the exit of that level to the spiral road system. Two factors, however, necessitated modifications to the planning.

(*a*) To mine the deposit to an average grade meant that two or three levels had to be available. Low grade ore from the contact area could then be up-graded with ground from more central areas.

(*b*) With the start of waste rock stripping the mining method took the form suited to Stage II.

The sequence of mining operations is determined by the need to open additional ore levels as required, and is similar to the sequence used when mining to the instantaneous stripping ratio. Initially, however, waste rock stripping is at a higher rate and the waste rock benches are therefore mined beyond the positions required to expose additional ore.

Each level is opened up from the centre of the pipe in a polygonal shape, the sides of the polygon roughly conforming to the shape of the pipe (Fig. 13). The opening up of each new level is carried out using front-end loaders until approximately 100,000 loads have been removed, by which time the level is established for production. This, the lowest level, also serves as a sump during the rainy season; in this period work is suspended.

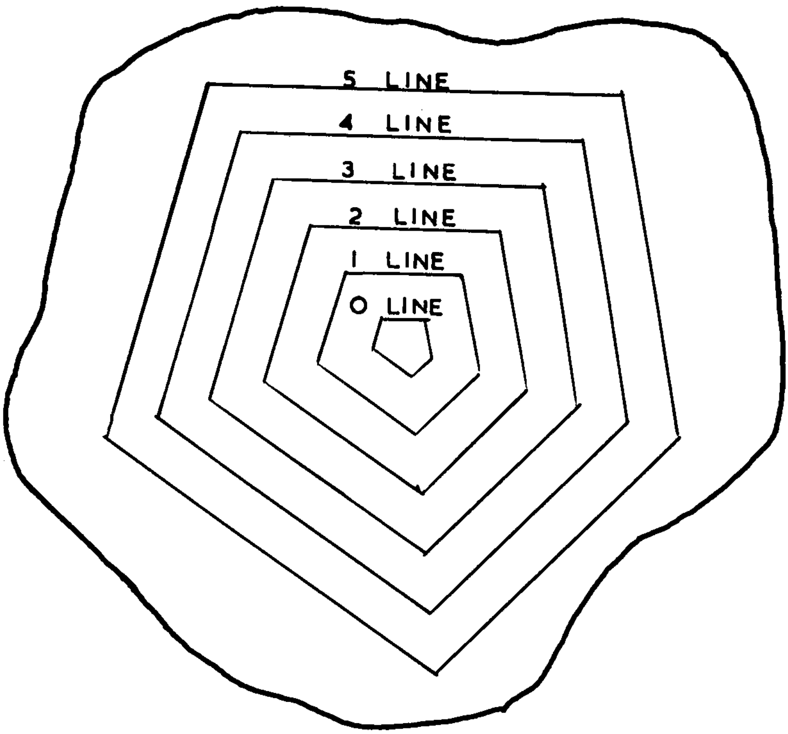


Fig. 13—Showing the block lines mined to on each ore level

Thereafter mining proceeds in a planned sequence by mining out concentric 100 ft strips from two or three production levels. After each strip is completed, equipment is moved to the level above or below to repeat the cycle.

When the contact is reached, the level becomes available for further widening out under the waste stripping programme to the planned limits.

Other features of this layout are:

(i) The opening of each new level in the centre of the pipe and its transition from an ore to a waste level simplifies planning.

(ii) Should waste rock stripping fall behind schedule, a reserve is available by steepening the angle of the ore benches.

(iii) Each level is mined on an upgrade of 1 per cent until the contact is reached, to allow adequate drainage. Waste benches are, however, carried level as water from these drains rapidly and has little effect on operations.

(iv) The spiral roadways into the pit have to be continually re-established as the levels are mined out.

(v) Bench widths are maintained at between 100 ft and 200 ft in ore. In waste this width is maintained at a minimum of 100 ft until the final limits are reached. Loading efficiencies are reduced if bench widths are reduced below 100 ft.

Skipway hoisting out of the mine:

It is apparent that the hauling of ore, in the later stages of the life of the mine, may become an expensive operation with a large number of trucks required to maintain production. Even with larger trucks costs will still be high and such units have the disadvantage of requiring wider haul roads.

It therefore appears that at a certain depth it may become economic to install an inclined skipway into the open mine. A smaller fleet of trucks will then haul ground from the mining faces to the skip loading bins, from where the ground will be hoisted up the side of the pit and tipped into the primary crusher reception bin. It is envisaged that:

(i) The skipway will be progressively lengthened as the pit deepens.

(ii) The skip size will match the truck size, i.e., 35 ton skips will be installed.

(iii) The skipway will be installed at the estimated final slope angle of the pit.

The primary crusher has, however, been installed to allow for the stripping of an additional 100 ft of waste around the pipe beyond the present planned limits. Should the present limits be retained therefore, a slot will have to be cut back into the wall rocks on the line of the proposed skipway.

The depth at which it is expected that the change from truck haulage to truck plus skipway haulage was calculated, using provisional cost figures, as follows:

(*a*) The depth at which the costs of skipway plus truck haulage in the pit equalled truck haulage throughout was estimated.

(*b*) The present value of the saving in haulage cost by skips compared to trucks for all ground below this level was then calculated to check whether the skipway installation was likely to be economic.

On present data it has been estimated that the installation of a skipway may be economic at a pit depth of between 300 and 350 ft below surface.

Consideration was also given to the installation of a skipway for the handling of waste rock. This was rejected on the following grounds:

(i) The favourable topography allows level tramming routes down to the 110 ft level (see next paragraph below).

(ii) Waste mining is at a higher rate than ore mining.

(iii) On reaching surface, waste must be hauled to a rock dump, whereas ore skips discharge at the primary crushers.

(iv) Most of the waste comes from the upper benches.

(v) The waste comes from a larger plan area, and haul distances are consequently longer even if a skipway is used; hence this long haul may just as well be used for climbing out of the pit as for travelling horizontally to a skipway.

Generally there appears to be a move away from skipways in opencast mines and Finsch is not committed to the concept, particularly in view of the continuous development of truck haulage.

Slots for waste dumping:

As can be seen below (Figs. 14 and 15), most of the waste rock mined is to be dumped on the west side of the pipe, where there is a pronounced valley. Slots are therefore to be cut through the hill to enable tramming to be carried out on level tramming routes from the mining faces to the dumps. It will be economic to carry these slots down to the 70 ft level, and possibly even to the 110 ft level. Below this, roads will spiral down to the waste stripping benches.

Slots will effect the following economies:

(i) Due to the higher speeds of horizontal tramming, a given quantity of rock can be hauled with a smaller fleet.

(ii) The cost of continually re-mining the inclined roadways, is eliminated by the slots.

(iii) The vertical lift, in moving the ground out of the pit, is reduced.

However, slots do increase certain costs as follows:

(*a*) Dumps start further from the orebody and advance more rapidly, thus increasing the tramming distance.

(*b*) Certain ground has to be mined beyond the planned waste rock stripping limits.

It has, however, been calculated that these slots will result in a considerable saving. Roughly one-third of the total waste to be mined will be moved out on horizontal routes while the balance will benefit by the reduced lift.

Waste dumping policy:

With the main tailings disposal area on the east side of the mine, approximately 90 per cent of all waste mined will have to be dumped on the west (Fig. 15). The valley on this side provides adequate tipping space, and dumps are therefore to be carried flat.

The use of the slots will mean that dumping will take place on 3 to 4 different elevations, each 40 ft vertically apart. Dumps will extend 2,500 ft north and south from the slots along the strike of the hill and will advance to the west across the valley. A typical section through the dumps would be as shown in Fig. 14.

The length of the dumps is controlled by:

(i) The available space.

(ii) Each dump serves roughly a similar length of mining face. The tramming distance can therefore be kept approximately constant by relating the loading position to the tipping point.

Also, in the event of emergency truck shortages, both the shovels and the dumping sites can be moved closer to the slots so that high production rates can be maintained with a minimum number of trucks.

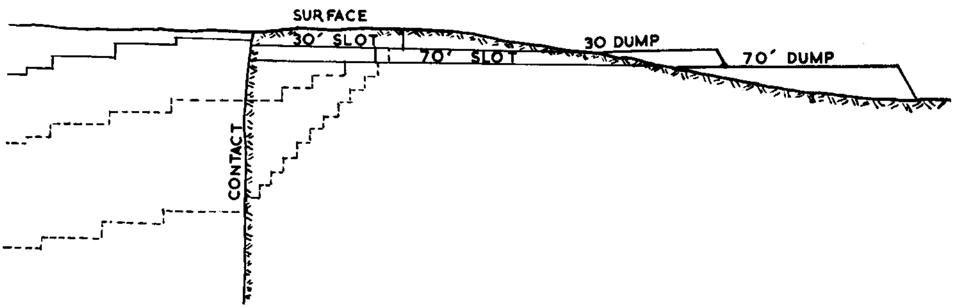


Fig. 14—Section through slots and dumps

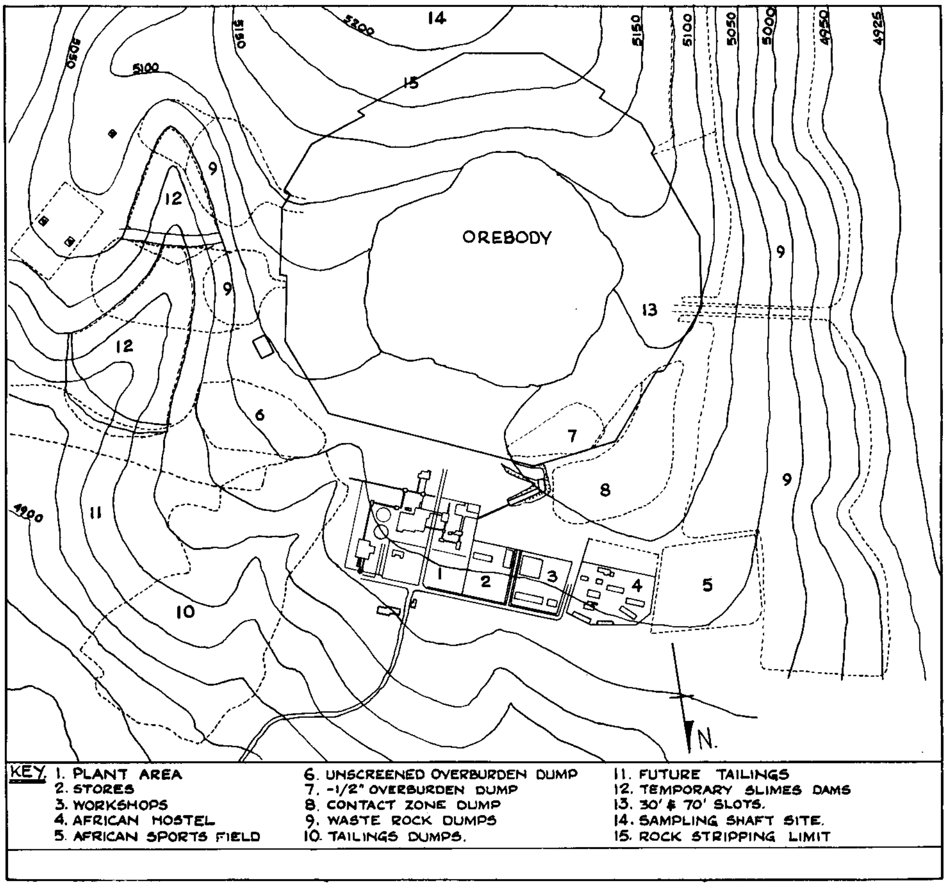


Fig. 15—General site layout

In addition to the main dumping site on the west, three sites have been established to the east. These are of limited area and, to compensate for this, two of the three are being carried at an upgrade of 8 degrees. The third dump has been established in an area too low for tailings disposal and will be used essentially for disposing of ground from the north-east corner of the mine, to prevent cross-tramming with the ore fleet.

MINING EQUIPMENT

The following equipment is in use at Finsch Mine:

(A) *Loaders:*

(*a*) Three $4\frac{1}{2}$ cu yd electric shovels of which two are allocated to waste rock stripping and one to ore.

(*b*) Two front-end loaders with 6 cu yd buckets. These are used for:

(i) Standby units for minor breakdowns and servicing of the ore shovel. Although these loaders can not individually match the output of a $4\frac{1}{2}$ yard shovel consistently, the surge capacity in the plant is such that plant output can normally be maintained for limited periods. The loaders are not used as standby for the waste rock shovels, as maintenance and tyre costs in the more arduous conditions of the latter have proved to be excessive.

(ii) Contact Zone loading—ore contaminated with ironstone is being stockpiled separately for later re-treatment.

(iii) Initial development of new levels. A front-end loader requires far less room in which to work and can be moved clear of blasts in the confined area rapidly.

(iv) Stockpile loading. A 100,000 load surface ore stockpile is used to ensure that plant feed is maintained when wet weather makes the pit inaccessible.

(v) Blending plant feed when the plant has difficulty in accepting ground from any particular area.

(vi) Miscellaneous work in the mining area.

The extreme mobility of the machines makes them ideal for the diversity of work being undertaken.

(B) *Rear dump trucks:*

Initially the ore fleet consisted of six 20-ton, and one 30-ton, trucks. The former have now been replaced by four 35-ton trucks to obtain uniformity with the nine 35-ton trucks which make up the waste fleet. It is of note that:

(i) The ore trucks have been equipped with specially enlarged bowls to handle bigger loads of low density ore.

(ii) A comprehensive range of sub-assemblies was purchased with these trucks to reduce downtime.

(C) *Service vehicles:*

These consist of:

2 large Crawler Dozers,
2 large Rubber-tyred Dozers,
1 small Rubber-tyred Dozer,
2 Road Graders,
1 Water Cart.

The dozers are essentially used for:

(i) Clean up of spillage at the shovel loading points.

(ii) Dozing on the waste rock dumps.

(iii) Maintaining the footwall in the pit at the correct elevation. Although experienced shovel operators can maintain a level footwall, it is often easier to rip an area back onto grade with dozers, although this can only be done in ore. In the hard waste rock any serious loss in grade has to be rectified by drilling and blasting. Minor and temporary loss in grade in the waste stripping is normally neglected.

Selection of sizes:

The reasons for the selection of the various sizes of vehicles were as follows:

(i) 4½ *Yard shovel (ore)*—The ore shovel was selected at an early stage to match likely peak plant requirements. It is capable of consistently loading 1,000 loads per hour.

(ii) 4½ *Yard shovels (rock)*—These were selected to match the ore shovel. This has the following advantages:

(*a*) The three shovels are interchangeable. A shovel approaching a major overhaul can therefore be moved to the less arduous ore loading conditions.

(*b*) Long breakdowns on the ore shovel can be countered by moving a rock shovel to ore and putting the other rock shovel onto three, as opposed to two, shifts.

(*c*) There is considerable, although not complete, interchangeability of spare parts among the machines. As a result the stores stock held for these machines can be kept to a minimum.

These shovels have been acquired to produce 26,000 loads per day on two shifts and present performance indicates that this level of output can be achieved.

A feature of the Finsch operation when compared to other opencast mines is the small number of shovels. Allied to this is the high loading rate of $\pm$ 800 tons per hour per shovel on waste rock stripping, this being mainly due to the excellent fragmentation achieved. The high loading rate is also due to the policy of operating two, instead of three shifts, on waste stripping. Except in truly continuous and smooth-running operations the advantage of the third shift is limited as no time is available for blasting, maintenance and emergencies.

In addition to the 4½ yard shovels, a 6-yard machine is now on order. This conflicts with the policy of standardization but will allow increased flexibility in mining operations.

(iii) *Front-end Loaders:* As previously mentioned, it would be desirable for the standby front-end loaders to fully match the output of the ore shovel. This policy has not been adopted for the following reasons:

(*a*) Major breakdowns on the ore shovel are handled by the waste rock shovels.

(*b*) No major loss in plant tonnage has occurred due to the size of these standby units.

(iv) *Trucks:* The truck size was based on the following criteria:

(*a*) The general rule that the shovel should fill the truck in between 3 and 6 passes was used for the selection of the trucks for waste rock stripping.

(*b*) 35-ton trucks would probably be the largest that could be matched with the skipway, particularly as it is desirable to keep the skip loading bins as small as possible to simplify bridge design.

(*c*) Standardization was achieved between the ore and waste rock fleets.

(v) *Dozers:* As conditions in waste rock stripping were expected to be arduous due to the flinty, abrasive nature of the product handled, it was decided to purchase crawler dozers for this duty. However, experience has indicated that with the blade ahead of the machine, and with proper road and dump surfacing, tyre life is acceptable and a gradual changeover is being made to rubber-tyred vehicles.

General criteria for the selection of earthmoving equipment:

Experience at Finsch Mine suggests the following general criteria for the selection of equipment.

(i) For an operation having a potentially long life, electric power shovels are preferred to front-end loaders, due to the low running costs and consistently high availability. Front-end loaders have, however, proved to be highly successful as back-up units due to their low capital cost and manoeuvreability.

(ii) In the selection of shovels it is wise to purchase a size larger than actually required for the following reasons:

(*a*) The possibility of a higher plant call, and a corresponding increase in the waste rock stripping rate, is covered.

(*b*) The shovel will handle the required output more easily and correspondingly be less prone to breakdown.

(*c*) Breakdowns can be handled by using the spare capacity of the remaining units.

(iii) It is considered that the general rule of matching trucks to shovels by the number of buckets to fill the truck has application, but preference should be given to the larger truck sizes depending on the haul conditions, shovel loading cycle, etc.

(iv) Equipment purchased should be in fairly general use in the country where it is to be operated, to:

(*a*) Prevent downtime due to the non-availability of spares.

(*b*) Prevent teething troubles that the local agents are unable to handle.

(v) Standardization is of major importance for the following reasons:

(*a*) The technical staff become used to a limited range of vehicles.

(*b*) Stores stock levels can be kept to a minimum.

(vi) Rubber-tyred dozers should be used in preference to crawler dozers wherever possible, as maintenance costs on the latter are high.

DRILLING AND BLASTING

Drilling: The following drilling equipment is in use at the mine:

(i) Three Ingersoll-Rand Crawlmasters, each with a 900 cfm compresor. Two machines are used for waste rock drilling on a two shift basis, and 30 ft/hr is called for from each drill. The third unit serves as a standby for both waste rock and ore drilling.

(ii) One crawler-mounted Schramm Rotadrill (Model C-64H-B) for rotary drilling (see Fig. 16), using a tri-cone bit, in ore. The unit is equipped with a 250 lb/in.^{2} compressor mounted on the same platform as the engine and drill mast. The unit drills a hole from $5\frac{1}{2}$ in. to 8 in. in diameter and is, due to the high penetration rates in the soft ore, able to obtain full plant call on two-shift drilling. In emergencies the unit can also be used for hammer drilling in waste.

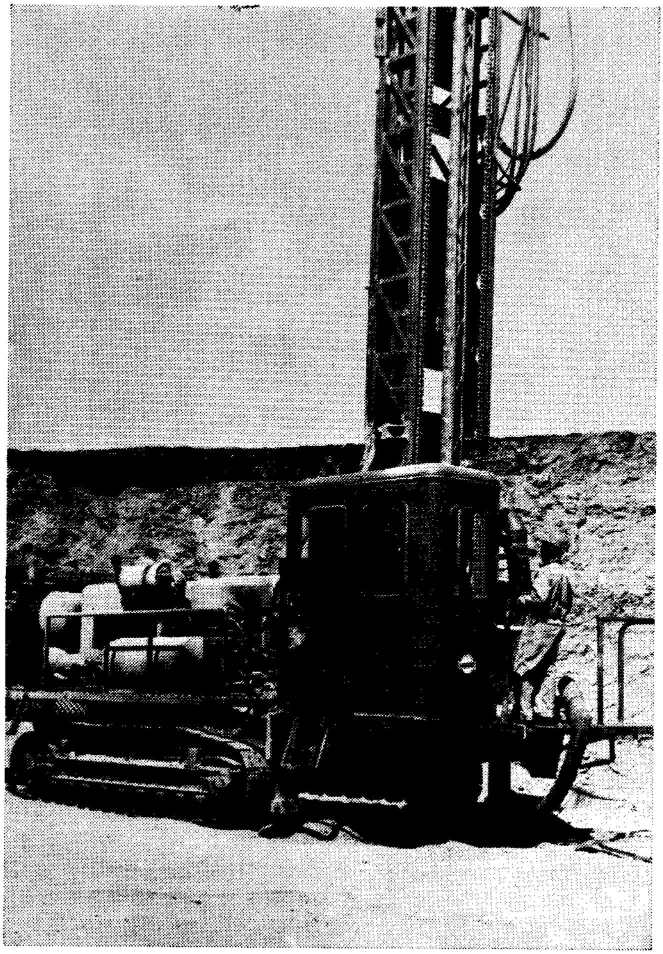


Fig. 16—Rotary drill in ore

(iii) Additional smaller units are used for secondary drilling.

Blasting: All blasting is done using a mixture of dieselene and Ammonium Nitrate, the latter in a porous prill form. The mixing is done on-site, with the mixer being permanently mounted on the rear of a 7-ton truck, the mixer being operated by a power take-off from the truck engine (Fig. 17). This method of charging up is very rapid and each hole normally takes less than a minute to complete. Single lines of Cordtex detonating fuse are used down the hole and as trunk lines, and the charge is boosted with pentolite primers. Approximately 5 loads are produced per pound of explosives in the waste rock stripping. Due to numerous bedding planes and fractures, fragmentation in the banded ironstone is good.

Fig. 17—Charging up operations using on-site mixing techniques

In ore, however, fragmentation varies widely due to the nature of the ground. Increasing experience among the pit personnel has reduced the variation in fragmentation, but this still remains a problem. Explosive efficiencies vary between 4 and 7 loads per pound of explosives. The use of a drop ball for secondary breaking is under investigation; initial results have been encouraging.

Normally only single row blasting is used. Holes are drilled at 20 degrees off the vertical in both ore and waste. In waste this results in a better breakout of the toe, while in ore, loading, particularly with front-end loaders, is safer.

LABOUR

As with all opencast mines, labour efficiencies are high. The direct operating staff (excluding maintenance personnel) is as follows:

Supervisory:	Pit Superintendent	1
	Pit Foremen	3
First Line Supervisors:	Drillers and Blasters	4
Operators:	4½ yard Shovels	7
	Trucks (Rock)	15
	Trucks (Ore)	10
	Bulldozers	5
	Front-end Loaders	3
	Grader and Water cart	4
	Drills (Rock)	8
	Drills (Ore)	4
	Charging and Blasting	3
	Bit Sharpening	1
	Clerk	1
	Dump Attendants	4
	Shovel Attendants	7

Drivers of all heavy equipment are Coloureds and the balance of the operators are Africans. A small village of 61 houses was built for the Coloureds, who are preferred to Africans, as the migrant labour system necessitates excessive re-training.

The overall complement, as above, totals 80. This labour force is responsible for producing 45,000 loads per day of ore and waste. (Equivalent to 36,100 tons per day or 450 tons per man shift.)

UNDERGROUND SAMPLING

The standard method of sampling a diamond pipe is to develop a tunnel grid on selected levels and to pass all the ground so developed through a sampling plant. Sinking of a circular shaft is therefore to be undertaken in 1969 for the sampling of the 1,000 and possibly the 2,000 ft levels, to obtain the following information:

(i) The size and shape of the pipe at depth.

(ii) The diamond content and value of the pipe and the possible economic limit of mining operations.

(iii) Physical properties of the ore at depth.

The sampling shaft has been sited on the south side of the open mine, 1,100 ft from the ore contact, to allow adequately for breakback. It has been so sited that it will eventually be used as the main return airway with the main hoisting shaft being sited to the north of the pipe, adjacent to the plant area.

Preliminary thoughts on the mine as an underground producer indicate that the Premier Mine open slot method may be adopted. With the perimeter waste rock having been mined back during the opencast phase, and with strong dolomite wall rock at depth, conditions appear to be favourable for the open slot method which has a high output potential.

TREATMENT PLANT

General

The treatment of diamond-bearing ground is in three stages, namely, Crushing, Washing and Recovery. The primary concentration of the ore is carried out in the Washing Plant in the traditional washing pan, while the Recovery Plant treats concentrates for final diamond recovery. Two physical properties of the diamond are used to separate the diamond from the gangue material. The diamond, with a specific gravity of 3·5, is heavier than most ore minerals. In addition, the diamond normally has a non-wettable surface and will therefore preferentially stick to a greased surface.

The design of the Finsch plant was complicated by several factors, these being mainly associated with the transition from weathered ground in the upper section of the pipe to unweathered ground at depth. This has meant modifications in plant design to cater for the changing physical characteristics of the ore.

The basic decision was to install a pan plant in preference to heavy media for primary concentration. In the upper, weathered, zone of the pipe a high proportion of fines is present in the ore. This forms a suitable material for the suspension media, or puddle, used in a pan plant. Tests on ore samples at depth have indicated that sufficient fines will still be produced when mining from the lower levels. Under suitable conditions pan plants have low operating costs due to the use of a natural medium. Efficiency, however, declines in the recovery of smaller diamonds, particularly in the −14 mesh size range, but monetary losses are low.

The grade of the Finsch ore is, at 36 carats per 100 loads, high, but the revenue per carat is comparatively low. This is mainly due to the high proportion of fine diamonds produced, two-thirds of the production being −7 mesh (Tyler) in size. The predominance of this size range has increased recovery problems.

Crushing plant (see Fig. 18):

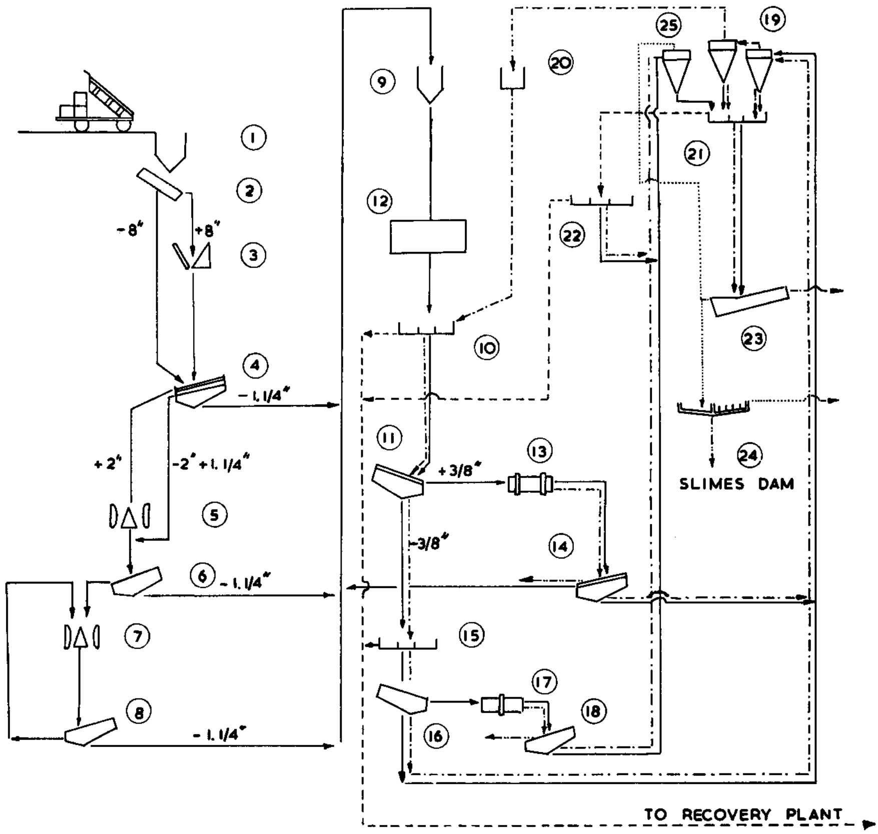
Initially, due to the friable nature of the yellow ground, ore was tipped through a grizzley, with 13 in. spacing, into the main reception bin. From the bottom of the bin ground is fed onto a conveyor by a plough feeder. This layout served adequately for ground from the upper levels, but construction of the main crusher station was phased in before this ore was exhausted and the harder ground encountered. The primary crusher station contains two 48 in. × 42 in. Allis Chalmers jaw crushers each fed from a 350 ton reception bin, by a 5 ft × 20 ft Eliptex feeder which also serves to scalp off fines. Crushing is to a nominal −8 in., the ground then being delivered, with the by-passed fines, to a 100-ton control bin. Although one crusher can handle plant requirements, two were installed for the following reasons:

(*a*) Both crushers were available from other mines in the De Beers group.

(*b*) Major overhauls can be undertaken during normal working hours.

(*c*) Hang-ups at the Eliptex feeders, or in the crusher jaws, can be handled without loss in plant throughput.

Ore is drawn off from the 100 ton control bin by two vibrating feeders onto conveyors each feeding a double-deck Ripl-flo screen. Oversize passes to two Allis Chalmers 16–50 Superior Gyratory crushers. Middlings report to two Allis Chalmers 6 in. × 51 in. Hydrocone tertiary crushers while undersize forms part of the main plant feed. Crushing is in closed circuit and all material is reduced to −1¼ in. A 1¼ in. screen mesh will just pass a 350 carat diamond of regular shape—the largest recovered at the mine to date weighed 168 carats.



CRUSHING / WASHING PLANTS

KEY

————	ORE	–·–·–·	TAILINGS
–·–·–·	PUDDLE	···········	WATER
– – – – –	CONCENTRATES		

1. 350 TON RECEPTION BIN
2. 2 - ELIPTEX FEEDER / GRIZZLYS
3. 2 - 42"x48" JAW CRUSHERS
4. 2 - 5'x14' SCREENS - APERTURES 2" & 1.1/4"
5. 2 - 16-50 A.C. GYRATORY CRUSHERS
6. 2 - 5'x12' SCREENS - APERTURE 1.1/4"
7. 2 - 6 - 51 A.C. CONE CRUSHERS
8. 2 - 5'x12' SCREENS - APERTURE 1.1/4"
9. 3 - 3000 TON STORAGE SILOS
10. 12 - 14' DIAM. PRIMARY PANS
11. 6 - 5'x10.5' RODDECK SCREENS - 3/8" SPACING
12. FEED CONTROL BIN
13. 2 - 6' x 10.5' SCRUBBERS & 1 HAZEMAG IMPACT CRUSHER
14. 3 - 5'x14' SCREENS - APERTURES 5 1/2 MESH & 3/8"
15. 12 - 14' DIAM. SECONDARY PANS
16. 6 - 6'x12' SCREENS - APERTURE 5.1/2 MESH
17. 1 - 6.5'x12' ROD MILL
18. 1 - 5'x14' SCREEN - APERTURE 5 1/2 MESH
19. 6 - 42" CYCLONES - 3 PAIRS IN SERIES
20. 1 - 15,000 GAL. PUDDLE TANK
21. 8 - 14' DIAM. TERTIARY PANS
22. 2 - 14' DIAM. QUATERNARY PANS
23. 6 - 54" SPIRAL CLASSIFIERS
24. 2 - 100' DIAM. THICKENERS
25. 1 - 24" DIAM. CYCLONE

Fig. 18

The $-1\frac{1}{4}$ in. material is conveyed to a 300 load feed control bin from where it is drawn off, by vibrating feeders, onto conveyors leading to a 100 ton bin at the top of the Washing Plant.

Construction is nearing completion of three 3,000 ton concrete silos, each 42 ft in diameter, for the storage of crushed ore (see Fig. 19). Feed from the crushing plant will be conveyed direct to the silos with provision, however, to by-pass damp ground to prevent hang-ups in the silos. Ground is recovered from the silos by variable speed 5 ft $\times$ 14 ft Eliptex feeders and conveyed to the feed control bin. Arrangements have been made for remote control silo selection and also for the rate of withdrawal from the silos to be controlled by a weightometer on the main conveyor into the Washing Plant.

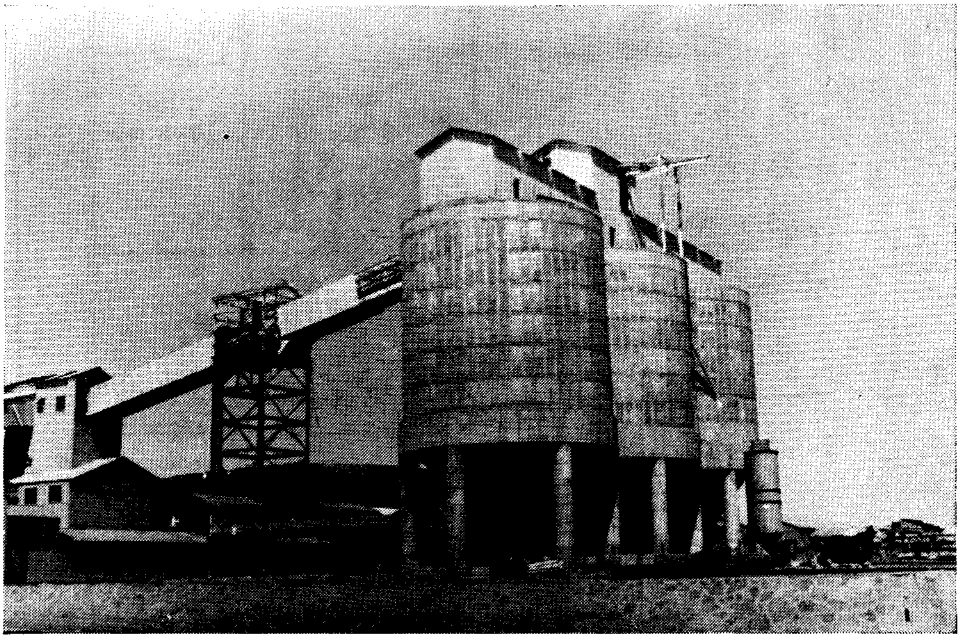


Fig. 19—Stockpile silos

The reasons for the installation of a stockpile, and the particular design chosen, were:

(*a*) The stockpile formed part of the original plant design, but construction was delayed until the full effects of wet weather on mine operation could be assessed. Rain normally necessitates stopping pit operations and although this is alleviated by an uncrushed surface ore stockpile, delays still occur.

(*b*) The crushing section, which is largely single line flow, can be completely divorced from the washing section. Even major delays in the former will have little effect.

(*c*) Some areas of the orebody have a low fines content. It will therefore be possible to store ground with good puddle making properties to blend with ground from these areas.

(*d*) Three independent silos were chosen for ease of cleaning and effective dust control. In addition, preliminary estimates indicated that it would not be more expensive than conventional designs.

Washing plant (see Fig. 18):

(*a*) *General:* The washing plant serves to reduce the daily 11,000 tons of feed to a concentrate of between 200 and 300 tons. This is done in a series of washing pans, a concentrator unique to the diamond industry. It ranks, along with the development of grease tables, as one of the most significant developments in the early history of the Kimberley diamond fields. As far as can be established, pans were brought into use in 1874. Since that date, mechanical improvements have been made, and the size has been increased to 14 ft, but the basic principles of operation have been retained. As this forms the basis of primary concentration, a detailed description of the construction and operation of the pan is given[1].

(i) *Construction* (see Fig. 20):

The washing pan is an annular-shaped vessel formed by two concentric sides of steel plate. The outer ring is 14 ft in diameter and 20 in. high while the inner ring is 6 ft in diameter and 12 in. high. Between these the annular space has a flat cast iron bottom with renewable manganese-steel liner plates. A sliding door, fitted in this bottom, permits the complete cleaning out of the pan when required.

A vertical shaft passes through the centre of the pan and rests on a footstep bearing. The shaft has 10 horizontal arms attached to it which are curved back from the direction of rotation. To these arms are bolted 52 pan teeth of triangular cross-section ($1\frac{3}{4}$ in. $\times$ $1\frac{1}{2}$ in. $\times$ 1 in.) and one tooth of circular cross-section. The pan teeth are bolted to the arms in such a manner that the thinnest edge points in the direction of rotation. One face of the tooth is tangential to the circle of motion while the outer surface of the tooth is at an angle to the tangent and causes an outward thrust towards the periphery on the material through which it moves. In addition, the teeth are attached to the arms in a spiral curve which ensures that the whole area is swept by the ends of the teeth which are suspended $\frac{1}{4}$ in. above the pan bottom. The single circular tooth is fitted at the end of the arm where the spiral ends and is there for the purpose of moving the settled concentrates to the discharge opening at the outer periphery of the pan. This discharge opening is at 270 degrees from the inlet, and concentrates are removed by a screw feeder.

A scraper gear on two horizontal shafts, each fitted with a scraper blade covers the width of the annular space. A screw-down wheel lowers the blades to the floor of the pan when it is required to clean out the contents through the sliding door in the floor, usually once per week. The feed inlet is tangential to the outer periphery. At a point 270 degrees from the inlet an overflow weir is cut in the inner circle to a height of $4\frac{1}{2}$ in. from the pan bottom and with a chord length of 36 in. The speed of rotation of the rakes is 8 rev/min for the primary and tertiary pans, and 6 rev/min for the secondary pans.

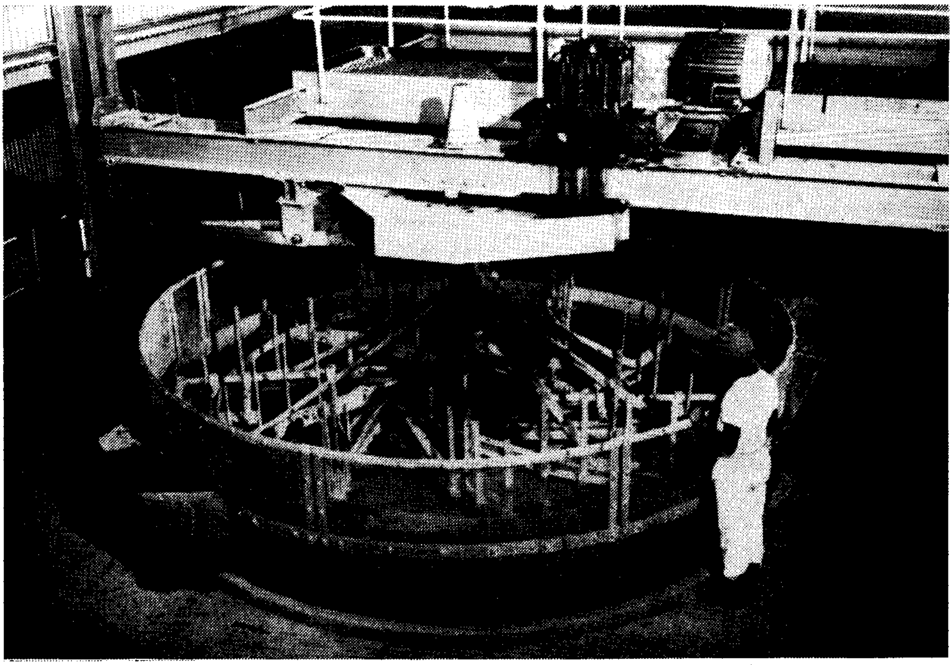


Fig. 20—The rotary washing pan

A pan is normally rated at 35 to 70 loads per hour, depending on the size of material treated.

(ii) *Operation:*

The relatively high specific gravity of the diamond (3·5), together with the comparatively small quantities of other heavy minerals, such as ilmenite, garnet, chrome diopside, permits a fairly easy separation from the low density gangue by means of these rotary pans. Ironstone, of high density, also occurs in the ground from the upper levels of the pipe but in insufficient quantities to affect pan operation.

The feed is continuously introduced tangentially to the outer circumference of the pan. This consists of the ore and puddle, the latter a suspension of fines in water. The launder at the entrance is inclined to give the feed a velocity approximately the same as the revolving mass in the pan.

The stirring action of the rakes keeps the mass in a state of semi-suspension, allowing the heavier minerals to settle and the lighter minerals to float and escape over the inner weir. The rotary motion imparted to the contents of the pan, together with the hydraulic gradient due to the incoming feed, cause the contents to take up a spiral path from inlet to outlet. The speed of rotation of the mass is about half that of the pan teeth and this speed decreases rapidly downwards due to the friction drag against the pan bottom. As the density of the mass increases towards the bottom, the thrust of the pan teeth becomes more effective thus forcing the concentrates to the outer circumference of the pan. An extract hole, at the outer edge, allows concentrates to be continually drawn off from the pan. In addition to the action described above, it is known that secondary flows play a major part in the concentration process.

The control of the process is achieved by the lowering or raising of the density of the mass in the pans by the regulation of clean make-up water to the puddle circuit. Primary, secondary and tertiary pans are run under conditions to suit the size of the diamond to be recovered in each stage.

An analysis of the circulating puddle at Finsch Mine indicates the following:

Specific gravity	1·25,	
Grading	Plus 28 mesh	4 per cent,
	-28 mesh $+325$ mesh	43 per cent,
	-325 mesh	53 per cent.

(*b*) *Flow of ground through plant:* The plant is divided into six sections, although this is to be increased to eight sections in 1969 to allow for the additional pan loading caused by re-crush, rod milling and the increased density of the ore. In the primary and secondary stages there are two pans per section with one intermediate screen after each pair of primary and secondary pans.

Six belt feeders distribute the dry ground from the 100 ton bin to the primary pans. Before entering the pan, the ground is joined by a stream of puddle. It then passes through the primary pans, the overflow reporting to 5 ft $\times$ 10 ft $7\frac{1}{2}$ in. rod-deck vibrating screens with a $\frac{3}{8}$ in. spacing. Oversize, $-1\frac{1}{4}$ in. $+\frac{3}{8}$ in., passes to the scrubber-recrush section, while the undersize from each screen is split as feed to two secondary pans. The overflow from the latter pass over 6 ft $\times$ 12 ft double-deck vibrating screens. The top deck is a relieving screen, while the bottom deck is $5\frac{1}{2}$ square mesh. All $+5\frac{1}{2}$ mesh material was previously sent to tailings, but, in future, will be rod milled (see below). The $-5\frac{1}{2}$ mesh material gravitates to the primary puddle sumps. This primary puddle is pumped to 42 in. cyclones above the tertiary pans. Two 42 in. cyclones are used in series for each primary puddle pump and the cyclones are designed to split at 28 mesh. The spigot product is fed, via pulp distributors, to the tertiary circuit, while cyclone overflow is sent to the secondary puddle sumps. This forms the circulating puddle for the plant and is pumped back to a puddle tank at the top of the plant for distribution to the primary pans. Any excess overflows and reports to two 100 ft thickeners.

The tertiary circuit consists of eight pans. Due to the size range of the feed, and hence the size of the diamonds, these pans are operated at low viscosity with clear water as make-up. In addition, a high percentage extract (10 per cent) is drawn from the pans. Tertiary pan concentrate is pumped to two quaternary pans via a dewatering cyclone and is reconcentrated in these pans. Quaternary pan overflow is pumped back to cyclones above the tertiary section.

Tertiary pan overflow is laundered via pulp distributors to six 54 in. Akins Spiral classifiers. Classifier sands are sent to tailings while classifier overflow forms the main feed to the thickeners. A guar gum flocculent is added to thickener feed; thickener overflow is returned to the plant as make-up water, while thickener underflow is returned, by 6 in. Denver Triplex diaphragm pumps, to the excess slimes sump. From here it is pumped to the main slimes disposal area.

Concentrate from the primary, secondary and quaternary pans passes over 28 mesh dewatering screens and is then conveyed to a 300 ton storage bin ahead of the Recovery Plant.

(*c*) *Scrubber—recrush section:* To prevent the breakage of large diamonds, all $-1\frac{1}{4}$ in. material is first passed through the primary pans. Primary screen oversize ($-1\frac{1}{4}$ in. $+\frac{3}{8}$ in.) still, however, contains locked diamonds and the ground has to be broken down further. This material is conveyed to the scrubber section, where the ground is passed through two 6 ft × 10 ft 6 in. Telsmith scrubbers, followed by screening on 5 ft × 14 ft Lowhead double deck screens. $-5\frac{1}{2}$ mesh material reports to a sump and is pumped, via cyclones, to the primary puddle sumps. $-\frac{3}{8}$ in. $+5\frac{1}{2}$ mesh material rejoins the head feed into the plant, while $+\frac{1}{8}$ in. material is discarded as tailings.

The decision to install scrubbers was originally taken for the following reasons:

(i) Clay is present in the plant feed from the upper, weathered, zone of the orebody.

(ii) Scrubber oversize from the upper levels contains a high proportion of ironstone.

With increased depth of mining these conditions are changing and little breakdown is being achieved in the scrubbers. Conventionally on the diamond mines the recrushing is done with Symons crushers handling a dry feed but with a high recirculating load. However, with a high proportion of clay remaining in the upper levels, it was decided to experiment with wet impact crushing. The results of test work were encouraging and one of these crushers has now been installed on the site previously occupied by one scrubber. Details are:

Type	Hazemag Impact Crusher SAP 5/S,
Speed of rotation	375 rev/min,
Machine capacity	320 tons per hour,
Anticipated circulating load	30 to 35 per cent.
Drive	Two 250 hp motors.

The above unit copes with all the scrubber feed. Tests will be continued on this unit but a parallel test will be carried out using a 3 ft Symons, as, at depth, the latter type may have advantages. These tests will be concluded before further extensions to the re-crush section.

(*d*) *Rod milling of secondary pan tailings:* As one of the main outlets to tailings, secondary screen oversize ($-\frac{3}{8}$ in. $+5\frac{1}{2}$ mesh) is a potential source of diamond loss. These losses occur as follows:

(i) As free diamonds in screen oversize. These are usually $-5\frac{1}{2}$ mesh diamonds that should report to the tertiary circuit but are lost due to inefficient screening. These diamonds are lost from the secondary pans as they are not operated under the optimum conditions for the recovery of fine diamonds.

(ii) As diamonds locked in gravel. With the preponderance of small diamonds at Finsch Mine, considerable losses can occur in this manner.

After preliminary assaying of this product, the decision was taken to erect one 6 ft 6 in. × 12 ft rod mill for test purposes. 40 to 50 per cent of the secondary pan tailings are conveyed to this section and the ground is fed to the mill via an Eliptex feeder. After milling, the ground is screened on a 5 ft $\times$ 14 ft Lowhead screen, $-5\frac{1}{2}$ mesh undersize being pumped to a cyclone above the tertiary circuit and oversize passing out to the main tailings conveyor. Details of the mill are:

Mill dimensions	6 ft 6 in. $\times$ 12 ft,
Speed	78 per cent Critical,
Load	11·2 tons (3 in., $2\frac{1}{2}$ in. and 2 in. rods),
Discharge moisture	57 per cent,
Feed rate	$\pm$ 100 tons per hour.

Assays were carried out on head feed to the mill, mill screen undersize, and mill screen oversize. Initially, results indicated that considerable diamond breakage was taking place in the mill. However, by reducing the rod charge and by increasing the pulp density, diamond breakage has been reduced to negligible proportions.

It may be queried why rod milling did not form part of the initial plant installation. As this was a new departure from standard Kimberley practice, an extended assay and test programme was required and this could not be allowed to prolong the non-earning period of the mine.

Results of the assay and test programme were such that planning is now in hand to extend the plant to treat all secondary pan tailings. Two additional mills will be installed with provision for a third.

(*e*) *Tailings and slimes disposal:* Approximately 70 per cent of the plant feed now reports to the tailings section with the balance reporting as waste slimes. This ratio will change when the recrush and rod milling sections have been commissioned.

The mine is conveniently situated on a hill and the area to the east of the pipe has been reserved for tailings disposal with the dumps being carried out flat. Tailings are conveyed to the dump area by single line 36 in. conveyors. The ground is then split and conveyed out on four 30 in. conveyor outlets at 180 ft centres, two of which have to be used for total tailings. Disposal is by means of flingers—tailings are conveyed to an elevated head pulley at each outlet, and then dropped down the throat of the thrower unit onto a short endless belt travelling at approximately 3,000 ft/min (see Fig. 21). The top of this endless belt is depressed between the head and tail pulleys by a pair of guide wheels which give the tailings an upward trajectory. This combined with the belt speed gives a throw of about 60 ft and it is possible to swivel the flinger head through 270 degrees. As soon as the outlet is filled, it is dozed onto grade, and stringer sections are added to extend the flinger forward to the edge of the dump.

Total waste slimes, at a maximum specific gravity of 1·3, are pumped to the disposal area situated some 7,000 ft from the plant. The area has been laid out in seven paddocks, each with an average area of some 40 acres. The walls of each paddock, which were constructed with material from the site, have a height of 0 to 25 ft, conforming to the contours of the area.

Only one paddock is used at a time, generally for a two-month period, thus giving an overall 14 month cycle. The advantage of the paddock-type layout is that each dam is completely dried out in the 14 month cycle. There is a considerable contraction of the slimes in this drying out period, thus giving additional capacity. Water is also recovered, through steel penstocks, and is returned to the plant for re-use.

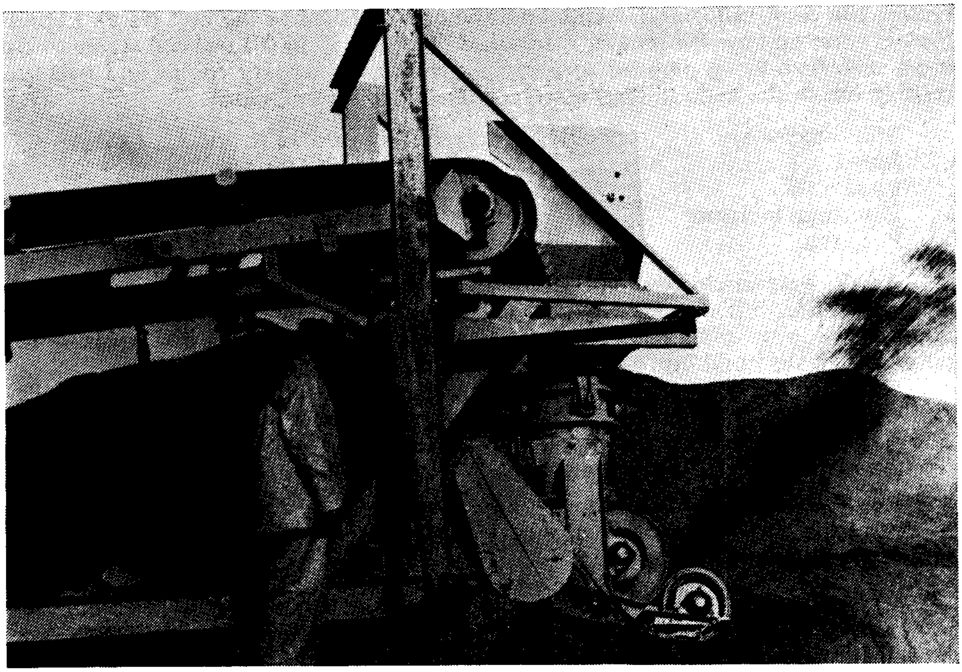


Fig. 21—A typical flinger installation

Extensive tests were carried out at the mine on filtering waste slimes both with a view to water recovery and being able to dispose of the filter cake on the tailings system. It was found that filtration in a disc filter, using flocculents, was practical but the cost of the water recovered made the project uneconomic.

(*f*) *Additional plants:* The following additional plants are under investigation or in operation.

Contact Zone treatment plant: Approximately 3½ million loads of Contact Zone have been stockpiled for treatment at a later date. This material has a high ironstone content (± 50 per cent) and, as a result, cannot be treated in the present plant which can only accept a low percentage of heavy minerals.

Some test work has been carried out but this is to be intensified in 1969. A preliminary flowsheet for this material, which will be treated at the rate of 1,000 loads per day, is as follows:

(i) Grizzley and screening to −6 in. with the removal of as much ironstone as possible during this stage.

(ii) Scrubbing in closed circuit with an impact-type crusher to break up clay and weathered kimberlite.

(iii) Sizing followed by broad beam X-ray separators (see below) for the +7 mesh fraction and magnetic separation for the finer sizes. Concentrates from the former will be sent direct to the Recovery plant while −7 mesh concentrates will join rod mill undersize.

Underground sampling plant: An underground sampling plant, to treat 100 to 150 loads per day, will be erected in the old Pilot Plant to coincide with the start of underground sampling. The plant will assay free diamonds down to 28 mesh and also locked diamonds down to $5\frac{1}{2}$ mesh.

Sampling plant: A small assay plant, based on an 8 in. heavy media cyclone unit and a 3 ft $\times$ 8 ft ball mill, exists for continuous sampling of tailings products from the Washing Plant. Sampling of Recovery Plant tailings products is mainly based on hand sorting of statistically valid amounts of ground.

Recovery plant (see Fig. 22)

The Recovery Plant, which is designed to treat 250-300 tons of material a day, carries out the final concentration followed by diamond recovery. For security reasons the plant is only operated on a single shift basis.

Major changes are to be made to this plant, some of which have already been implemented. The plant is therefore described as originally designed, followed by notes on the proposed changes.

(*a*) *Magnetic and heavy media separation:* Three vibrating feeders draw the ground from the 300 ton concentrates storage bin at the rate of approximately 40 tons per hour. The ground is conveyed to the top of the Recovery Plant where it passes over a 5 ft $\times$ 16 ft double-deck primary wash screen, with a $\frac{3}{16}$ in. top deck and a 28 mesh bottom deck. The coarse fraction passes, via a cone protection screen, to a belt magnetic separator. Tramp iron is removed and also a small proportion of the more highly magnetic ironstone.

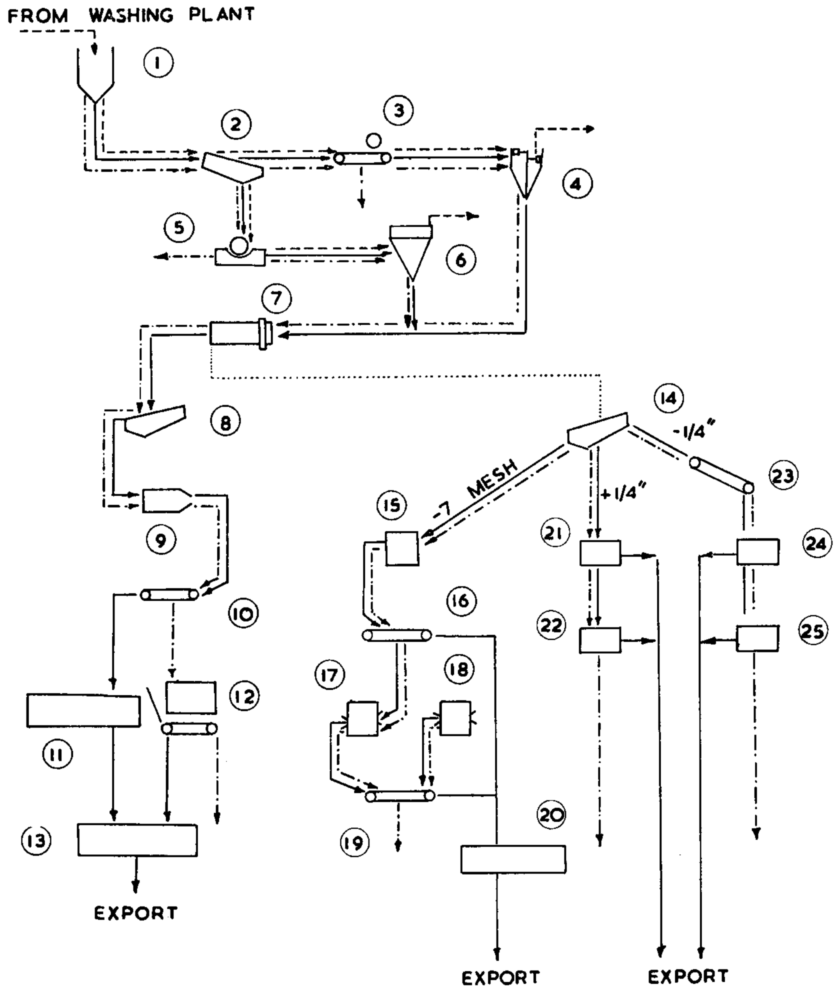
The $-\frac{3}{16}$ in. $+28$ mesh material passes to three drum-type wet magnetic separators (27 in. $\times$ 30 in. diameter) which reduce the feed by approximately 5 per cent. A higher reduction is not attempted due to the danger of trapping diamonds with the magnetics.

The $+\frac{3}{16}$ in. and $-\frac{3}{16}$ in. fractions then report to the heavy media cone and cyclone circuits respectively.

Heavy media cone circuit: Ground from the magnetic separator passes directly to a 6 ft heavy media cone. This is operated at a top density of 2·90 and a bottom density of 3·00, using spherical cyclone grade ferro-silicon. As with the cyclone circuit, a very large reduction is obtained on feed low in ironstone (that is, ground from the deeper levels), the sink fraction being approximately 5 per cent of the total.

The float product is drained and washed on a 3 ft $\times$ 16 ft double-deck vibrating screen with a $\frac{1}{4}$ in. top relieving screen and a 10 mesh bottom deck, while the sink product, which is elevated from the bottom of the cone by an air-lift, is drained and washed on a 3 ft $\times$ 10 ft vibrating screen. The first portion of each screen drains the medium from the feed, while sprays fitted on the second portion of the screens thoroughly wash the product to remove any ferro-silicon adhering to the gravels.

The medium draining from each screen is pumped back to the cone circuit. The fines from the wash section of each screen are received in a movable chute which can be positioned to take any desired cut from the drained medium and divert it to the ferro-silicon recovery circuit with the washed product. This is pumped up to a 30 in. diameter, wet-type, magnetic separator where ferro-silicon is recovered from the slimes. This is gravitated to a 3 ft Akins classifier for dewatering and is then admitted to the cone via the medium return pump, first passing through a de-magnetizing coil for dispersal.



RECOVERY PLANT

KEY

DIAMONDS ———— LIGHT WASTE- - - - -
OTHER WASTE -.-.-.-.-

1. 1-300 TON STORAGE BIN
2. 1-SCREEN-APERTURE $\frac{3}{16}$"
3. 1-DRY BELT MAGNETIC SEPARATOR
4. 1-6" DIAM. HEAVY MEDIUM CONE
5. 3-WET DRUM MAGNETIC SEPARATORS
6. 1-18" HEAVY MEDIUM CYCLONE
7. 5-3" × 8" ATTRITION MILLS
8. SIZING SCREENS + $\frac{1}{2}$" - $\frac{1}{2}$" + $\frac{1}{4}$", - $\frac{1}{4}$" + 7 MESH, - 7 MESH + 28 MESH
9. 12 CONDITIONERS IN PARALLEL
10. 21 GREASE BELTS—3 SECTIONS OF 7
11. 3 TRICHLOROETHYLENE DEGRESSING TANKS
12. 5 OPTICAL SCAVENGERS I+ 7 MESH ONLY)
13. FINAL CONCENTRATE PREPARATION
14. SIZING SCREENS
15. 2-6' × 4' FIRST PASS MILLS (PARALLEL)
16. 1-VIBRATING GREASE BELT
17. 1-6' × 4' SECOND PASS MILL
18. 1-6' × 4' SCAVENGER MILL
19. 1—VIBRATING GREASE BELT
20. TRICHLOROETHYLENE DEGREASER
21. 1-BROAD BEAM X-RAY SEPARATOR
22. 1-BROAD BEAM X-RAY SEPARATOR
23. ELECTRIC DRIERS
24. 5-FIRST STAGE X-RAY SEPARATORS
25. 5-SECOND STAGE X-RAY SEPARATORS

Fig. 22

Revolving rubber scrapers keep the sides of the cone free from settled ferro-silicon. These scrapers are carried on cross arms from a rotating central shaft. Medium return to the cone is by way of a cylindrical vessel welded around the scraper shaft at the surface level of the cone. Four pipes of different lengths pass down from the bottom of the vessel and admit medium at different depths in the cone.

Medium density is automatically controlled by a Nucleonic Density Controller which measures the specific gravity of the medium in the return line and maintains the density by the addition of water. A large drop in density is offset manually by the operator either lowering the classifier spiral or by moving the chutes onto the wash section of the screens.

Heavy media cyclone circuit: Ground from the wet magnetic separators passes over a 3 ft × 10 ft dewatering screen, with a 28 mesh deck, into a surge bin. A vibrating feeder at the bottom of the bin feeds ground into a mixing tank where it is joined by the ferro-silicon. A 6 × 4 Warman pumps the material through an 18 in. cyclone having an overflow density of 2·50 and a spigot density of 2·70. The sink product passes over a 34 in. × 22 in. static wedge wire screen (0·5 mm aperture) and then over a 3 ft × 10 ft wash screen with a 28 mesh deck. The float product passes over two static wedge wire screens followed by a 4 ft × 16 ft float wash screen. The ferro-silicon recovery circuit is similar to that used on the cone.

The sink product from both the cone and cyclone join and are then elevated to a vibrating screen. The float product passes directly to tailings.

(*b*) *Attrition mills and conditioners:* Due to the weathering of the upper few hundred feet of the orebody, the Finsch diamonds are refractory in that they do not naturally adhere to grease. This is due to a thin, invisible, layer of iron oxides which renders the surface of the diamond wettable. Diamonds may be improved in their capacity to adhere to grease by subjecting the gravels to a preliminary milling process.

The sink fraction from heavy media separation is screened on $\frac{3}{8}$ in., the $-\frac{3}{8}$ in. fraction passing to four 3 ft × 8 ft mills each charged with two tons of $1\frac{1}{4}$ in. steel balls. The $+\frac{3}{8}$ in. material passes to one 3 ft × 8 ft mill with no ball charge. The mills rotate at 34 per cent of critical speed and have a maximum feed rate of $2\frac{1}{2}$ tons per hour. A high pulp density is maintained to minimise diamond breakage. In milling some of the softer gangue minerals are ground away but a high reduction is not achieved.

From the mills, the ground passes over 2 ft × 5 ft wash screens with a $\frac{1}{4}$ in. top deck relieving screen, and a 28 mesh bottom deck. The coarse and fine fraction are re-combined and the ground is elevated to the top of the plant. It is split into three sections and screened into four size ranges in each for treatment through the conditioners and over grease belts. These size ranges are: $-1\frac{1}{4}$ in. $+\frac{1}{2}$ in., $-\frac{1}{2}$ in. $+\frac{1}{4}$ in., $-\frac{1}{4}$ in. +7 mesh and −7 mesh + 28 mesh.

After attritioning the diamonds are less refractory but a high proportion will still not be retained on grease. The ground has therefore to be conditioned. The reagent, an alkaline solution of maize acid oil, is added with the feed and mixed in the conditioner. The surface condition of the diamonds which are present in the concentrate is altered from the wettable to the non-wettable state by the action of the fatty acids present in the reagent, but the surface condition of the gravel is largely unaffected and the bulk remains in the wettable state. The conditioners are small rotating mills, 2 ft in diameter and $3\frac{1}{2}$ ft in length, with an internal spiral which ensures a retention time of ± 2 minutes. Each conditioner discharges onto a 1½ ft × 3 ft vibrating wash screen for the removal of excess reagent. The ground then passes to bins above the grease belt section.

(*c*) *Grease belts* (see Fig. 23): In designing the grease belt section, the following factors had to be considered:

(i) A large volume of final concentrates was anticipated due to ironstone contamination of the ore. It was accepted that the section had to be designed for this quantity although at a later date there would be considerable excess capacity.

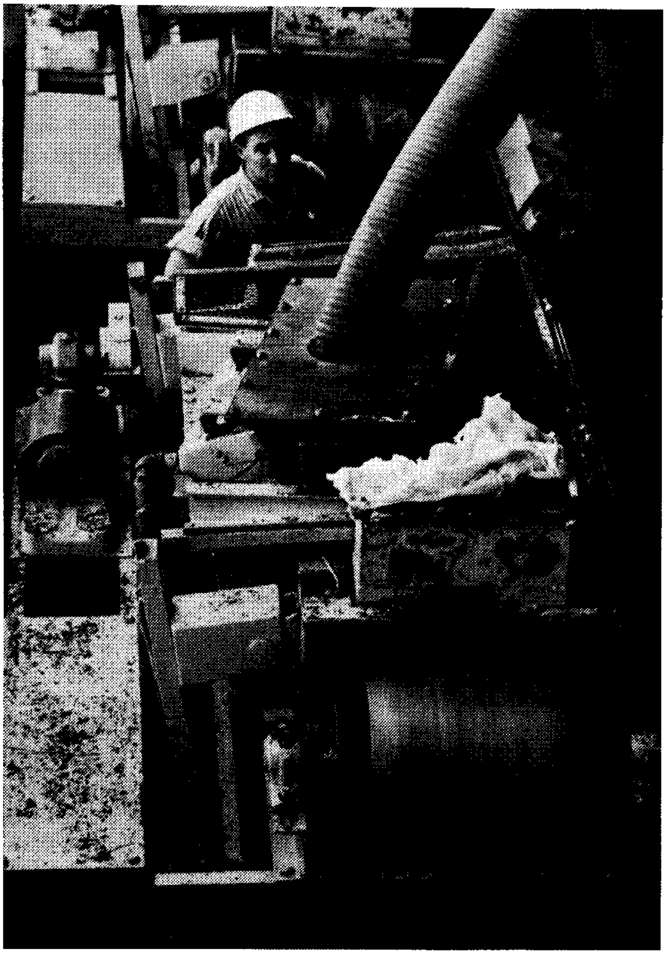


Fig. 23—View of a conventional grease belt from the tail pulley end. The grease applicator box can be clearly seen

(ii) A choice had to be made between the vibrating grease table and the grease belt. With conditioned feed the latter is a necessity as the surface of a grease table rapidly becomes contaminated with fines and conditioning reagent. The grease belt, however, has a low capacity due to the absence of a vibrating motion, but has the advantage of good security as the diamonds are mechanically skimmed off the belt.

(iii) A large volume of grease would be in circuit. It was therefore necessary to mechanize the handling of grease and concentrates.

The grease belt itself[2] consists of a 20 in. conveyor belt passing around a head and tail pulley, each 12 in. in diameter, the pulleys being at 7 ft centres. The belt is mounted on a steel framework, with the short axis inclined at 12 degrees to 15 degrees to the horizontal—this angle can be varied to suit the size range of the feed. Two vibrating feeders, with 12 in. wide trays, discharge the concentrates onto the upper edge of the belt, the lip of the feeders being some 4 in. above the surface of the belt. The latter improves diamond adherence by allowing the gravel to make a slight indentation in the grease. A water box at the back of the belt supplies a continuous stream of water to wash the gravels off the belt—this can also be varied to suit the size of material treated.

The belt is covered with an 18 in. width of grease, approximately $\frac{1}{2}$ in. thick. The grease is a mixture of two grades of petroleum jelly, the proportions of which can be varied to suit summer or winter conditions. A grease applicator box is positioned at the tail pulley end of the belt. This is designed to continuously apply a thin layer, $^1/_{64}$ in. thick, of fresh grease onto the surface, achieved by heating the grease in the applicator box. At the head pulley of the belt, a heated copper knife takes a continuous cut of the same thickness, also removing adhered gravel and diamonds.

Concentrate is continuously fed over the belt, the diamonds and a propotiron of gangue adhering to the grease. Material not adhering is washed off. The latter passes over a static dewatering screen before gravitating to tailings belts. The following is the capacity of the grease belt.

$+\frac{1}{2}$ in. feed	2,000 lb/hr
$-\frac{1}{2}$ in. $+\frac{1}{4}$ in.	1,000 to 2,000 lb/hr
$-\frac{1}{4}$ in. +7 mesh	500 to 1,000 lb/hr
−7 mesh +28 mesh	300 to 500 lb/hr

Concentrates scraped off the end of the grease belt fall into a steam-jacketed hopper and are laundered to one of four autoclaves. This, a cylindrical vessel with conical base, is steam heated, and is used for the separation of concentrates from grease. As the grease bearing concentrate enters the heated water in the autoclave, the concentrate is dropped and water and grease overflow. The latter is pumped to two grease separators, where grease and water are separated, the molten grease overflowing to drums for disposal or possible re-use.

Concentrates, amounting to 1,000-2,000 lb/day, are regularly tapped from each autoclave, through a 6 in. valve, into stainless steel mesh baskets. These are sealed and then degreased in trichloroethylene tanks.

(*d*) *Third stage recovery:* All final hand sorting of gravels is done in Kimberley but the 1,000-2,000 lb of concentrate produced daily is far in excess of what can be conveniently transported and hand sorted. A small section, the Third Stage Recovery, was therefore incorporated in the plant to achieve a further reduction as follows:

(*i*) All -14 mesh material is batch milled for 5 hours in a concrete mixer containing 1 in. steel balls in a 2 : 1 ball to ground ratio. This milling grinds away the softer minerals which are removed on a small wash screen. All oversize is exported to Kimberley.

(*ii*) $-\frac{3}{4}$ in. $+\frac{1}{4}$ in., $-\frac{1}{4}$ in. $+7$ mesh and -7 mesh $+14$ mesh material, is given several passes through three Gunson 621 electronic colour sorters (see Fig. 24). These machines, which are widely used for seed sorting, were adapted to diamond sorting some three years ago. Dry gravel is fed into a hopper at the back of the machine. Vibrating feeders draw the ground from the hopper and drop the particles, in single file, onto two short endless belts running at high speed. The particles leave the belt in a continuous stream in a trajectory towards the tailings outlet. Three photocell assemblies view each particle as it leaves the belt in a suitably illuminated chamber. The machine sorts by making use of a colour differential and can be set to send light or dark particles to concentrates. At Finsch, a high proportion of the diamonds are lighter in colour than the gravel. As each light-coloured particle is detected, a signal is sent to the ejection system. This triggers a short blast of compressed air through a jet, diverting the particle to concentrates.

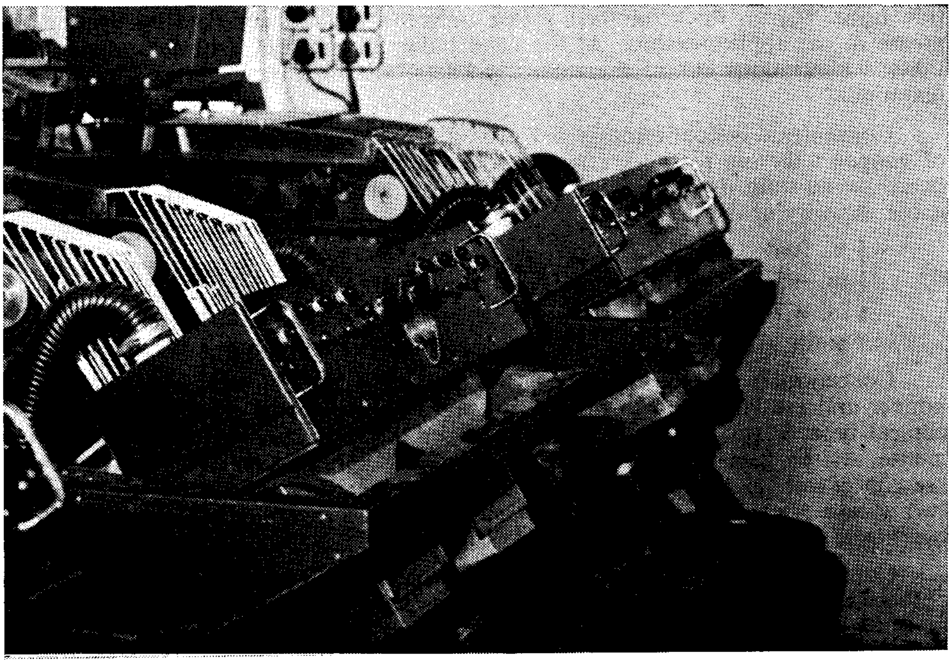


Fig. 24—The discharge end of two conventional Gunson 621 colour sorters. Tailings are discharged into the bins on the right while concentrates pass to locked receptacles

Concentrates pass to sealed cans, ready for export, but the tailings still contain dark diamonds not detected by the separator. All +7 mesh tailings are therefore hand sorted at the mine, while −7 mesh +14 mesh tailings are milled in a concrete mixer.

(*iii*) As the Gunson sorter has an upper size range of $\frac{3}{4}$ in., the $-1\frac{1}{4}$ in. $+\frac{3}{4}$ in. fraction is hand sorted.

(*e*) *Scavenging of grease belt tailings:* All +7 mesh grease belt tailings pass through one of five optical separators for scavenging diamonds lost over the grease. These units are fed by a vibrator, the material forming a monolayer on a 20 in. wide belt contained in a fully enclosed cabinet. A beam of light is thrown over the full width of the belt and reflections, particularly from diamonds, are picked up by a photomultiplier tube which triggers the ejection mechanism. This takes a cut of the feed over the width of the belt as it leaves the conveyor. Due to their low capacity on fine gravels the −7 mesh material cannot be treated by these units. The optical separators have not been a great success on Finsch gravels, as they will only recover diamonds of high quality.

Due to known inefficiencies in the recovery process, all tailings from the Recovery Plant were initially stockpiled. With increased efficiency of operation, heavy media and magnetic separator tailings are now discarded, but grease belt tailings are still stockpiled separately for later re-treatment.

(*f*) *Proposed changes to the recovery plant:* Diamond losses are known to occur in the grease belt section. The efficiency of recovery on grease of −7 mesh diamonds is between 80 and 90 per cent. Higher efficiencies are obtained on coarser gravels but this does not exceed 95 per cent. As a result, continuous research has been undertaken to improve recovery.

(i) *−7 mesh material:*

In 1966, during routine assays of pit and tailings samples for fine diamonds, it was found that high efficiencies could be obtained by milling the sample at low feed rates in a 3 ft × 8 ft ball mill followed by a vibrating screen, the end of which was covered by a layer of grease. This indicated the possibility of eliminating conditioning by more intensive attrition milling followed by recovery over a vibrating grease surface. Further work was undertaken by the Diamond Research Laboratory. Milling of −7 mesh grease belt feed was carried out in a 6 ft × 2 ft mill, followed by desliming and a vibrating grease table. Once optimum conditions had been established, high efficiencies were obtained.

It was however realised that the vibrating grease table would be unsuitable for production conditions. Work was therefore put in hand at the mine to develop a vibrating grease belt. Tests on an experimental unit indicated that, in conjunction with milling, high efficiencies could be maintained. Feed rates were high and the percentage adherence low. As a result of this test work, two 6 ft × 4 ft mills, followed by a vibrating grease belt, were installed as a scavenger circuit for −7 mesh tailings from the conventional belts. Further work showed that conditioning followed by conventional belts could be replaced by a two-pass system on the mill/vibrating belt.

−7 Mesh material passes into the first mill, the feed rate being fixed at 2,000 lb/hr being controlled by a spiral feeder. Both mills are equipped with diving rings to ensure thorough attritioning. Details of the mills are:

Size	6 ft $\times$ 4 ft,
Speed of mill	42 per cent critical,
Ball size	1 in.,
Wt. of balls to wt. of ground	1: 1,
Reduction on each pass	$\pm$ 50 per cent.

As with the attrition mills, a low moisture content is aimed at to obtain a thick pulp in the mill. The first mill discharges over a 5 ft $\times$ 3 ft vibrating screen, with 28 mesh cloth, for desliming. Oversize is discharged into a hopper, the base of which incorporates a 4 ft fluted roller feeder with variable speed drive. This feeder discharges at $\pm$ 1,000 lb/hr onto the vibrating belt. Tailings from the belt are pumped up to pass through a 12 in. cyclone. The spigot product is dewatered and passes into the second mill. A second screen, hopper and feed roller are used as for the first pass, the roller feeder discharging onto the same vibrating belt. Tailings from the second pass are laundered to a spiral classifier for dewatering and disposal to the tailings system.

The vibrating belt consists of a 42 in. conveyor passing over a head and tail pulley at 16 ft centres (see Fig. 25). The short axis of the belt is inclined at 15 degrees to the horizontal, while the belt speed is 20 ft per minute. The two 48 in. feed rollers are in

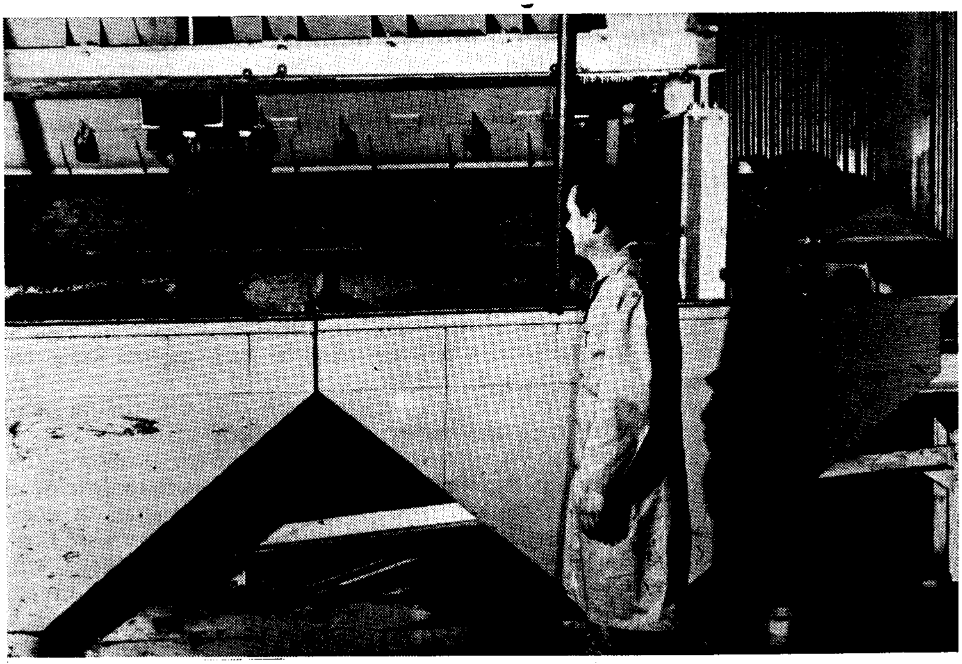


Fig. 25—The vibrating grease belt. The roller feeders are obscured by the hoppers

line but at 6 degrees to the longitudinal axis of the belt to prevent disturbance of diamonds already trapped in grease. The rollers discharge the ground 6 to 8 in. above the belt. Training of the belt is achieved by two guide rollers at both the head and tail pulleys of the belt.

The belt runs on a decking, but below each feed roller position vibrating sections are installed, these being vibrated by conventional screen mechanisms. The grease applicator box is similar to that on the conventional belt but wider, and contains additional heating elements and baffles. Two aluminium, heated, grease knives are used for the removal of concentrates adhering to the belt, these concentrates passing, as before, via a steam-heated hopper to an autoclave. Water boxes are provided at the back of the belt to wash the gravels off, the water being fed through a horizontal line of small holes over a lip plate and onto the belt.

The above arrangement has proved so successful that it will be incorporated in the final layout. Two vibrating grease belts will be installed, replacing the nine conventional belts. Each belt will be served by two mills, two for the first pass, one for the second pass, while the fourth mill will be used to scavenge Recovery Plant tailings products (see below).

(ii) *Scavenger circuit:*

Assays undertaken from 1967 onwards indicated losses of small diamond were taking place from the thickeners. In part, these reflected the inefficiency of pans in recovering -14 mesh diamonds, but it was apparent that losses were occurring elsewhere, particularly in the Recovery Plant. Assays showed that the long slot 28 mesh screens, essential to obtain adequate dewatering, were losing diamonds in the 28 mesh range. In addition, no protection was available against torn screens. As a result of the investigation all these products were diverted to a sump, pumped through a cyclone and passed through a temporary 6 ft $\times$ 2 ft mill. This was followed by a 28 mesh dewatering screen, the lower section of which was greased for diamond recovery. Efficiencies were low however, possibly due to insufficient attritioning and the product is now being fed through the second pass 6 ft $\times$ 4 ft mill handling the -7 mesh material. This product will eventually be treated through the fourth mill provided in the new -7 mesh layout.

(iii) $+7$ *Mesh material:*

With the success of both the optical Gunson sorters, and milling followed by grease recovery, initial research work was aimed at incorporating both recovery systems on $+7$ mesh material. The Gunson sorters would remove the better quality diamonds under conditions of maximum security, while milling and grease recovery would serve to scavenge the tailings. However, a significant development took place while this work was in progress.

It has long been known that a diamond will fluoresce when subjected to X-rays. Investigations at both the Anglo American Electronics Laboratory and the Diamond Research Laboratory indicated that this property could be used for effectively separating diamonds from gangue material. Few other minerals are found in diamond concentrates that will fluoresce to any degree. In addition an X-ray separation process is not affected by refractory diamonds and will also recover gem and industrial diamonds indiscriminately. An experimental unit was made based on the conventional colour sorting Gunson. The original feeding system was retained, namely a trough-shaped belt onto which particles are fed in single file. As the particles are discharged from the belt they fall past the beam of an X-ray tube. The resulting fluorescence of the diamond is detected by a photomultiplier tube which then signals an electronic circuit to trigger an air puff which ejects the diamond from the main stream. The whole process is carried out in a fully darkened cabinet.

Test work on the prototype indicated efficiencies on one pass of 98 to 99 per cent and these results were confirmed on a twin-belt machine. Capacities of the machines are the same as the colour sorters as follows:

$-\frac{1}{4}$ in. $+7$ mesh material .. .. 150 lb/belt hr,
$-\frac{1}{2}$ in. $+\frac{1}{4}$ in. material .. .. .. 750 lb/belt hr.

Ten of these machines are now on order to handle all $-\frac{1}{4}$ in. $+7$ mesh material previously fed to the grease belts. Five machines will handle ground on a first pass, while the balance will scavenge the tailings on a second pass. Concentrates, which form from 0·3 to 0·5 per cent of the total feed, will pass down pipes directly into a safe.

In $+\frac{1}{4}$ in. material, the occurrence of diamonds is low. As a result a higher percentage of concentrates can be tolerated with each ejection. The Diamond Research Laboratory have developed a prototype sorter which may form the basis of the $+\frac{1}{4}$ in. circuit. Ground is fed from a vibrator, in a darkened cabinet, onto a second vibrator. The former controls the feed rate while the latter serves to spread the feed into a mono layer. As the ground drops off the end of the second vibrator, it is subjected to a wide beam X-ray. Any resultant fluorescence is picked up by the photomultiplier tubes which operates a signal to a flapper gate which takes a cut of the feed as concentrate. An alternative to the above is a sorter locally developed by Gunson Sortex which retains the air ejection system but incorporates a 6 in. wide feed belt. Both types are now under test but either machine will handle total $+\frac{1}{4}$ in. feed—two machines will therefore be used in series.

(iv) *Proposed final circuit:*

The circuit up to, and including, the attrition mills, is retained as before. Attrition mill discharge is screeened on 7 mesh with $+7$ mesh being elevated to the top of the plant where the material is screened on $\frac{1}{4}$ in. Minus $\frac{1}{4}$ in. passes to a bin, over a dewatering screen, and then into an electrically heated drier. After drying, the product is again screened to remove any remaining -7 mesh material from the circuit. The latter reports to the mill circuit. Minus $\frac{1}{4}$ in. $+7$ mesh then passes, via a bin, to one of five first-stage Gunson XR-21 X-ray sorters. Concentrates report to the safe, while tailings pass to five second stage X-ray sorters. Concentrates again report to the safe with tailings going out to the tailings system.

Plus $\frac{1}{4}$ in. material is not dried as the particles do not adhere as with the finer gravels. The material passes to a bin and is then fed through two broad-beam, or wide belt, X-ray separators in series; the concentrates and tailings passing to the safe and tailings conveyors respectively.

Minus 7 mesh material is cycloned with the spigot product passing via a dewatering screen to a bin and then to two 6 ft $\times$ 4 ft first pass mills. After each mill, the ground is deslimed on 28 mesh deck vibrating screens followed by roller feeders and the vibrating grease belt. Concentrates report to an autoclave, while tailings are again cycloned and passed through the second pass 6 ft $\times$ 4 ft mill and, after desliming, over the second vibrating grease belt.

All scavenger products are similarly dewatered and pass through the fourth mill being discharged onto the second vibrating belt.

It is anticipated that these changes to the recovery circuit will be completed in the first half of 1969.

CONCLUSION

In conclusion, the authors wish to thank Mr R. Daniel, Consulting Engineer to the Anglo American Corporation, for permission to publish this paper. Acknowledgement is recorded of the valuable contributions made by Messrs A. N. Shand, Asst. General Manager (Dist.), M. Glover and G. Schwartz, formerly Resident Engineer and Plant Superintendent respectively, when mining and treatment started and many technical problems had to be overcome.

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BOOK REVIEW

Metal Statistics 1958-1967—Metallgesellschaft A.G., P.O. Box 2609, Frankfurt am Main, West Germany.

The 55th Annual Issue of this publication follows very closely the pattern of the previous issue.

The statistics supplied for aluminium, copper, lead, zinc and tin are very detailed and complete, covering all aspects of production, consumption, imports, exports, scrap and prices, the latter starting in 1850 and some of the other figures in 1900.

The same cannot be said for the other metals. Except for cadmium, magnesium, nickel, mercury and silver, for which price tables are given, the remaining statistics are confined to production by countries. In many cases figures for ore only are given.

This book is very well produced and invaluable to anyone who needs statistical information on metals. It has been extended and improved in recent years and it is to be hoped that the publishers will continue to extend the scope in future to give complete coverage to as many metals as possible.

D.G.M.