The South African Institute of Mining and Metallurgy

SYMPOSIUM

MODERN PRACTICES IN DIAMOND MINING IN SOUTHERN AFRICA

Johannesburg

March 1961

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Modern Practices in Diamond Mining in Southern Africa

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THE SOUTH AFRICAN INSTITUTE OF MINING AND METALLURGY

OFFICERS AND COUNCIL FOR THE 1960-1961 SESSION

FOREWORD

by

MR. H. F. OPPENHEIMER
Chairman
De Beers Consolidated Mines
Limited

The publication in collected form of the papers presented to the recent symposium on modern practices in diamond mining in Southern Africa is both timely and welcome. Not only does the volume provide an authoritative and comprehensive survey of the mining techniques and recovery processes now used in the diamond industry, long one of the significant factors in our economy, but it also records the considerable technical advances made during the past ten years in particular, and is thus a tribute to the initiative, ability and skill of our scientists and technical experts.

Greater efficiency and economy in diamond mining and recovery methods have resulted from co-ordinated effort. Coupled with experiments carried out by the various mines comprising the De Beers Consolidated Mines Group has been the very successful pure and applied research conducted by the Diamond Research Laboratory in Johannesburg. The only laboratory in the world devoted exclusively to research investigations into all aspects of the diamond, the Diamond Research Laboratory has originated and evolved more efficient methods of diamond recovery and has perfected the use of diamond tools for various industrial purposes.

I am gratified and honoured to offer this foreword to a work recording the many achievements that have placed and maintain the diamond industry on a sound and stable basis.

March, 1961.

H F Oppenheimer

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OPENING ADDRESS TO THE SYMPOSIUM MODERN PRACTICES IN DIAMOND MINING IN SOUTHERN AFRICA

By **G. S. GILES*** B.Sc.(Eng.), M.I.M.M., (Member)

"Mr President, gentlemen, it gives me great pleasure to deliver the opening address in the symposium on modern practices in diamond mining in Southern Africa.

As you are aware, there will be a number of specialized technical contributions to this symposium, including papers on mining and recovery systems at both underground and alluvial mines, native labour administration, diamond drilling and research. These cover the technical aspects, so I propose to talk in a general way about the diamond mining industry, with a view to filling in the background.

As this symposium is confined to Southern Africa i.e., the Union of South Africa and South West Africa, and in view of the close ties between them, the term 'South Africa' will be used to include both countries.

WORLD PRODUCTION

The most recent statistics available are those in respect of 1958 when production was of the order of 28 million carats, excluding an estimated production in Russia of some 650,000 carats. The largest producer, by weight, was the Belgian Congo with 16,700,000 carats, followed by the Union of South Africa (2,700,000) and South West Africa (900,000) combined with 3,600,000 carats, and Ghana with 3,200,000 carats. Other large producers were Sierra Leone 1,500,000; Angola 1,000,000; Liberia 800,000; and Tanganyika 500,000 carats.

In passing, it is worthy of mention, that, including the estimated Russian production, only about 1,000,000 carats came from outside the African Continent. Based on sales by the Central Selling Organization, the value of the world production was of the order of £110 - £120 million.

Although South African production amounted to only 12 per cent of world production, it included nearly one-third of the world gem production of some six million carats, making it by far the most important producer. In value South Africa's production was almost 45 per cent of world production and approximately three times the value of Belgian Congo production.

OCCURRENCES

All known occurrences of diamonds may be divided into two main categories, namely, pipes or fissures of basic volcanic rock, commonly called 'kimberlite,' and alluvial or eluvial deposits derived originally from kimberlite pipes or fissures.

Of several hundred known kimberlite pipes in South Africa no more than a dozen of any size contain diamonds in economic quantities. The picture is the same, so far, for the rest of Africa. Less than 10 per cent of world production by weight is from underground mining in kimberlite deposits — most of it in the Union of South Africa.

* Consulting Engineer, Diamond Section, Anglo American Corporation of South Africa, Limited.

MINING AND RECOVERY

Prior to 1950, both mining and recovery of diamonds in South Africa had remained more or less static for many years. Since 1950, however, there have been radical changes in mining systems, particularly at the larger underground mines, and in concentration and recovery processes at all the larger mines.

As these methods are fully covered in the papers to be presented in this symposium, it is sufficient to say here that mining and recovery efficiencies have improved very considerably and maintained South Africa's pre-eminence in these matters.

SPECIAL PROBLEMS

The rarity of diamonds and their great value create problems peculiar to the industry.

(*a*) *Rarity*

For comparison, six dwts. of gold per short ton represents a concentration of one part in less than 100,000, whereas the average for diamonds in the pipes at Kimberley is one part in 30,000,000 and in the alluvial fields of South West Africa one in more than 100,000,000.

As assaying is obviously impossible bulk sampling is necessary. In underground sampling it is usual to put all kimberlite from a grid of advance development on a selected horizon through the sampling plant in samples, usually of 25-30 tons. This is both expensive and slow as the sampling plant, usually a small-scale production unit, has to be very thoroughly cleaned for each sample.

A further complication in the evaluation of a deposit is that grade depends not only on the weight of diamonds recovered but also on the size and quality of individual stones.

In your presidential address, Mr President, you discussed fully the problems of control in large-scale diamond recovery and it is unnecessary for me to elaborate. But I would like to assure you, Gentlemen, that in checking plant efficiency, finding the proverbial needle in a haystack is simple compared with finding the infinitesimal quantity of diamonds which, of course, should *not* be present in the tailings.

(*b*) *Value*

The great value within a small volume, especially in the case of gem diamonds, presents many problems. These, quite literally are the bane of a mine manager's life because they can never be solved absolutely and sometimes result in distressing situations.

Although mechanization is progressively reducing the risk, the possibility of diamonds being 'picked up' is present in almost all phases of mining operations. Most mines have a system of cash rewards to encourage handing in of 'pick-ups.' This, unfortunately, is not always completely effective.

As diamondiferous material passes through the concentrating and recovery processes it becomes ever more vulnerable and requires increasing protection and supervision. It is impossible to eliminate all occasions when persons are exposed to temptation and there must inevitably be a large measure of trust. It is, therefore, vitally important to select personnel as carefully as possible.

EXPLORATION

Known ore reserves of gem diamonds in South Africa are dwindling and the larger companies are spending considerable sums on exploration in endeavours to replace them. De Beers, for instance, is spending of the order of £1,000,000 per annum on prospecting either through producing subsidiaries or exploration companies.

The methods employed include aerial photographic and geophysical surveys together with extensive and detailed investigation of promising localities on the ground.

RESEARCH

In 1946, at the instance of the late Sir Ernest Oppenheimer, the Diamond Research Laboratory was established in Johannesburg, but its activities extend beyond the continent of Africa. It is financed by most of the major diamond producers in Africa.

Since its inception the success of the laboratory's work has been remarkable in every sphere.

I doubt if there is any diamond mine of consequence, especially in South Africa, the Belgian Congo, Angola, Tanganyika or West Africa, that has not derived considerable benefit from improvements in recovery processes emanating from the Diamond Research Laboratory.

Remarkable strides have also been made in extending the use of diamonds in industry, including the development of oil-drilling crowns used as far afield as Canada and the United States of America. Mention should also be made of the enormous amount of work done at the laboratory to ensure that natural industrial diamonds compete successfully with synthetic diamonds.

Last, but by no means least, it should be recorded that it was the fundamental study of the properties of diamonds at the Diamond Research Laboratory which led to the successful development last year of a commercial process for the production of synthetic diamonds at De Beers' Adamant Research Laboratory, Johannesburg.

I need hardly add that the establishment of the Adamant Laboratory, with the specific purpose of producing synthetic diamonds, was the brainchild of Sir Ernest Oppenheimer.

MARKETING

Marketing and distribution through London and, to a lesser extent, Johannesburg of more than 85 per cent of the world's production of rough diamonds is carried out by the Central Selling Organization. Contrary to popular belief, there is no real mystery about this organization. It is a complex of financially interrelated companies, each with a specific purpose and all owned very largely by the major producers.

Very briefly, the Diamond Corporation has long-term agreements for buying run-of-mine production from the major producers of Africa on a quota and co-operative sales basis. The Diamond Trading Company arranges the marketing and distribution of gem and Industrial Distributors (Sales) of industrial diamonds.

It is interesting to know that arrangements were made towards the end of 1959 for the organization to sell all diamonds the Russian authorities wish to export.

The monopolistic nature of the Central Selling Organization is frequently criticized, most vigorously, as a rule by those who understand little of its purpose and operation. But there are none, including those governments involved, who will deny the tremendous benefits derived by the producers, cutters, merchants and, mark you, the consumers of diamonds throughout the world from the stability given to the entire diamond industry by the Central Selling Organization.

Here again, tribute must be paid to Sir Ernest Oppenheimer. His foresight led to the creation of the Central Selling Organization and his courage, at the risk of his entire personal fortune, maintained it through the vicissitudes of the early 1930's.

Here, too, it should be emphasized that the remarkable developments of the last ten years in diamond research, mining and recovery could not have been undertaken without the stability and the confidence in the diamond industry engendered by the 'Sales through a single channel' policy devised and built up by Sir Ernest.

IMPORTANCE OF DIAMOND INDUSTRY TO SOUTH AFRICA

It is regrettable how few South Africans appreciate the importance of the diamond industry to this country and the vital role played by diamonds in the industrial development of South Africa. In point of fact, industrial development in this country started with the discovery of diamonds. Early on, money from the diamond fields played a major part in development of the Witwatersrand, later in opening up the Northern Rhodesian copper mines and, more recently, in financing several gold mines in the Western Transvaal and Orange Free State.

Nor is it generally known that De Beers pioneered commercial fruit-growing as well as the importation of pedigree cattle and sheep, and financed the explosives factory at Somerset West.

To date, more than £700 million worth of diamonds have been produced in South Africa. Apart from providing valuable foreign exchange, the production of diamonds here has led to the establishment of a major diamond-cutting industry.

In recent years, largely as the result of the work of the Diamond Research Laboratory, a considerable and rapidly expanding industrial diamond industry has been established, whose products in both processed and fabricated form are exported far beyond the borders of South Africa.

So much for the importance of the diamond industry to South Africa. What of the importance of South Africa in the diamond industry? Perhaps, the most conclusive answer is the fact that the congress of the World Federation of Diamond Bourses and the International Diamond Manufacturers' Association is at this moment in progress in this city of Johannesburg.

CONCLUSION

Those of us intimately connected with the production of diamonds were delighted when this symposium was proposed. We trust everyone will find it interesting and useful.

Finally, Mr President, I wish to thank you for this opportunity of addressing the Institute and for the great privilege of declaring the symposium open, which I now formally do."

MINING PROCEDURE AND METHODS AT C.D.M.*

(CONSOLIDATED DIAMOND MINES OF SOUTH-WEST AFRICA, LIMITED)

By **S. W. Devlin**† B.Sc.(Eng.), (Member)

SYNOPSIS

The methods of development sampling, overburden removal and excavation of diamondiferous gravels are outlined and photographically illustrated, whilst the primary treatment of the gravels through field screening plants is dealt with in more detail.

During and immediately subsequent to the Second World War, suitable machinery and material were almost unobtainable, and planned mechanisation and expansion on these fields were retarded until about 1947.

The effect of improved methods and extensive mechanisation on production over the last decade is described, with brief references to previous practice, and the resultant improved efficiency and increased productivity is illustrated by means of graphs.

The Consolidated Diamond Mines of South-West Africa, Limited, own a long term mining lease over the area known as Diamond Area No. 1, or the *Sperrgebiet*, which stretches from the north bank of the Orange River to latitude 26°S—a distance of approximately 240 miles. This area is bounded on the west by the Atlantic ocean. The eastern boundary is roughly parallel to the coast line and some 50 miles inland.

All known diamondiferous deposits in Diamond Area No. 1 lie along the sea coast and within one or two miles inland.

Although diamonds are known to occur all along this coast line, the main deposits are within an area extending over the first 55 miles northwards from the Orange River. These deposits are alluvial and are mined as opencast workings. Mining methods are therefore comparatively simple in layout and procedure.

Mining procedure may be classified under four main headings:

1. Development sampling—or prospecting—for the purpose of outlining and calculating reserves.
2. Overburden removal.
3. Excavation and tramming of diamondiferous gravels.
4. Field screening, or preliminary treatment of gravels.

1. DEVELOPMENT SAMPLING OR PROSPECTING

Ahead of the established reserves a systematic development sampling programme has been carried out over the past ten years.

Geological survey, mapping and the compilation of results from sampling are supervised by a resident geologist and his staff. The operations of sampling, including drilling, are controlled by the mine management through a prospecting superintendent and staff.

Preliminary investigation

Areas to be sampled are first investigated by a series of boreholes sunk to bedrock using a conventional jumper-type well drill. Lines of holes are drilled at 500 metre intervals on strike. These lines of holes are laid out normal to the coast line and holes are sunk 25 metres apart. A section on dip is therefore available from each line of holes showing:—

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* Reprinted from November 1958 issue.

† General Manager, C.D.M.

(i) Sand overburden depth;
(ii) Gravel bed thickness;
(iii) Bedrock contours;
(iv) Eastern and western limits for subsequent trenching.

Trench overburden removal

The first step in the sampling programme is the removal of overburden. This work is undertaken by the earthmoving department and will be dealt with under heading No. 2.

The quantity of overburden removed is controlled by the required trench length and depth of overburden, allowing for a 40° angle of repose of sand and for a width of 10 to 12 metres at the point of contact between overburden and gravel beds. See Fig. 1 showing portion of stripped trench line.

Trenching

The necessary overburden having been removed, a trench is cut, on deposit, one metre wide from the upper limits of the diamondiferous gravels to bedrock. Bedrock in most cases is a schistose rock which formed the original sea bed and which is usually eroded into gullies and potholes.

Fig. 2 shows the eroded nature of this bedrock.

Sampling trenches may range from a few hundred to 3,000 or more metres in length. Each trench is divided into sections 5 metres long and each section is zoned in depth at 0·5 metre intervals to bedrock. The trench being 1 metre wide, each zone is thus $2\frac{1}{2}$ cubic metres in volume and is separately excavated and treated.

Fig. 3 shows a typical trench and sampling equipment.

Sampling procedure

Excavation of the trench is done by hand, using African labour, and each separate zone is loaded directly into a wooden tray from which the material is fed

Fig. 1—Sampling trench and sand overburden

into a hand-operated rotating trommel fitted with a cone at the feed end for throwing out +25 mm. material. The trommel is covered with stainless steel screencloth of 1·3 mm. aperture for elimination of sand.

The resultant gravel (—25 mm. +1·3 mm. material) so recovered is sized on a classifying trommel and then jigged. Concentrates are hand-sorted for diamonds.

Fig. 4 shows the type of jig and trommel used in prospecting.

From the results obtained, sections of each trench are drawn up and ore reserves are calculated and extended.

2. OVERBURDEN REMOVAL

The removal of overburden from diamondiferous deposits for both prospecting and mining is undertaken by the earthmoving department under the supervision of the earthmoving superintendent.

During the years 1947-1958 there has been a progressive improvement in the types of machinery used for earthmoving and in the methods adopted.

Fig. 5 shows a graph of the average volume stripped per month for prospecting and mining combined over these years and the average volumes of terrace mined.

For several years prior to 1947 practically no development or prospecting had been undertaken and overburden removal for mining, owing to the limited equipment available, was somewhat primitive.

In order to open up a new mining cut, or face, a narrow trench was first opened up west to east across the reserves—usually about 200 metres long. The overburden removal was performed using a small bucket-wheel excavator of German origin which had, at best, a capacity of 80 to 100 tons per hour. This excavator discharged

Fig. 2—Bedrock cleaning

Fig. 3—Prospecting trench and sampling equipment

Fig. 4—Classifying trommel and jig

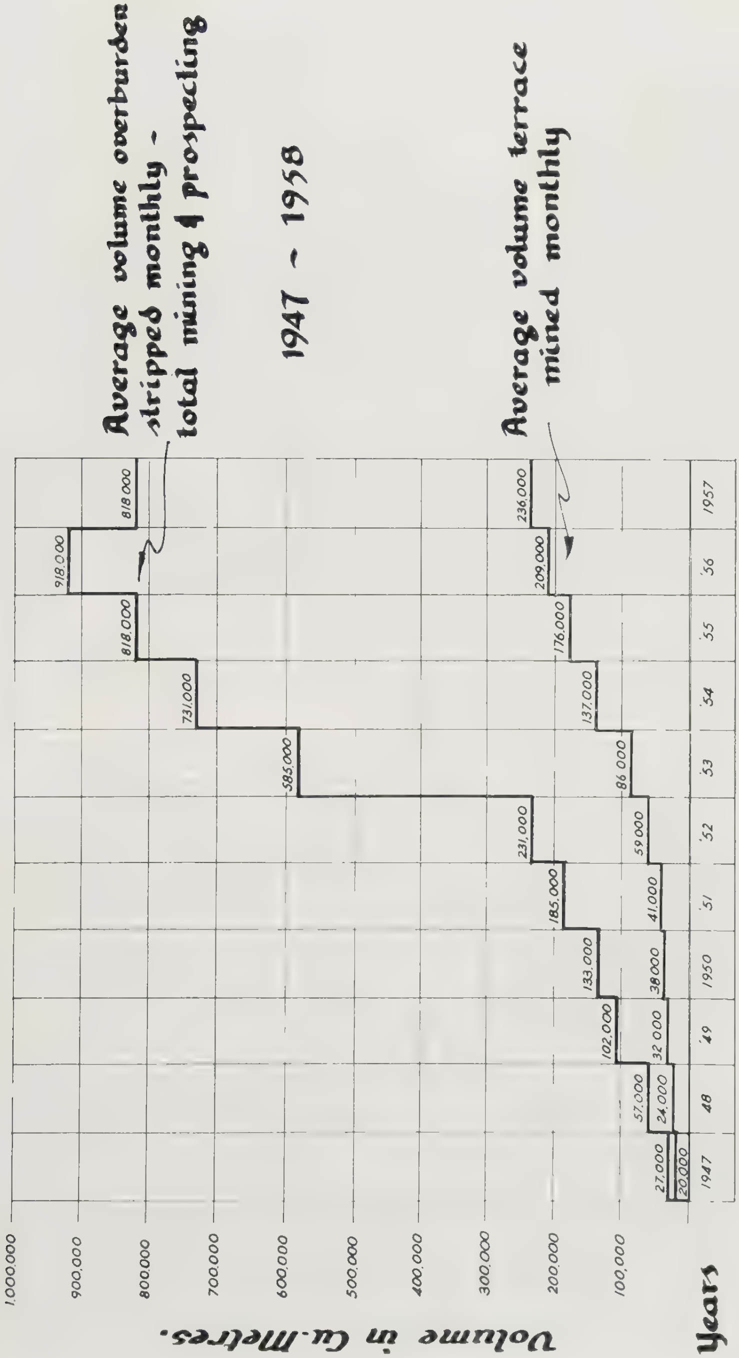

Average volume overburden stripped monthly - total mining & prospecting
1947 - 1958
Average volume terrace mined monthly
Volume in Cu.-Metres.
1,000,000
900,000
800,000
700,000
600,000
500,000
400,000
300,000
200,000
100,000
Years
1947
48
'49
1950
'51
'52
'53
'54
'55
'56
1957
27,000
20,000
57,000
24,000
102,000
32 000
133,000
38000
185,000
41,000
231,000
59 000
585,000
86 000
731,000
137,000
818,000
176,000
918,000
209,000
818 000
236,000

Fig. 5

into 36 cu. ft. side tipping trucks on a 24 in. gauge track. The trucks, in spans of two, were hauled by mules. Obviously, areas with very shallow overburden depths had to be chosen for this initial opening up process.

Having opened the "cut" to the required depth, mining commenced and all gravels were excavated to bedrock.

Progressive stripping of overburden ahead of mining faces was undertaken by three larger bucket-wheel excavators, also of German origin, purchased about 1936, which had an excavating capacity of some 300 cu. yards per hour. Overburden was thrown back into the worked out area.

Generally speaking, this method of stripping ahead of mining is still used, as are the original three large bucket-wheel excavators which, however, have been modified and improved. See Fig. 6.

Year by year higher recovery of diamonds was called for, reduction in mining grade was required and greater overburden depths had to be overcome. Also, an increased programme for development sampling and prospecting had to be organised.

To cope with the extensive volume of sand overburden which had to be removed in these operations, various new types of earthmoving machines were introduced and improvements made to existing methods.

Bucket-wheel excavators and stacking conveyors

The three crawler-mounted diesel-electric bucket-wheel excavators were completely overhauled and fitted with modern gearboxes and motors, and track shoes were redesigned. The first increases to the earthmoving fleet were four-wheeled pneumatic tyred 12 cu. yd. scrapers designed to be drawn by crawler tractors (Fig. 7). Such prime movers in sufficient quantity were not available at the time, but the mine was able to purchase a number of Sherman tanks which, stripped of their gun turrets, but otherwise in the same condition in which they had been used for military operations, were used in place of tractors.

In 1954 the Sherman tanks and scrapers were replaced by more of the pneumatic tyred scraper units mentioned above.

The Sherman tank hulls, with tracks, gave rise to new ideas and local engineering talent put them to good use. The hulls, stripped down to bare essentials and with their diesel engines replaced by electric motors, were used to carry conveyor belt gear as mobile stacker conveyors (see Fig. 8). These stacking conveyors, used in series with the bucket-wheel excavators, considerably improved sand removal technique. Later, Sherman tank hulls were similarly used as undercarriages for smaller bucket-wheel excavators designed to a pattern similar to that of the German models. These small scoops, as they are locally named, are capable of excavating an average of about 113 cu. yd. per hour at a direct cost of 8d. per cu. yd.

Diesel driven dragline excavators

For overburden stripping close to the coast where the inflow of seawater through the sand at depth obviates the use of track type or pneumatic tyred machinery, diesel powered dragline excavators, fitted with 55 ft. boom and $\frac{5}{8}$ cu. yd. bucket, are used.

During the years 1954—1957 up to 19 of these machines have been in use. The average excavating rate proved to be 37·5 cu. yd. per hour at 12·7d. per cu. yd. (see Fig. 9).

Pneumatic tyred, diesel driven scrapers

The removal of overburden for development sampling and also for the initial opening of mining faces is best performed by the four-wheel scraper fitted with giant pneumatic tyres.

Fig. 6—Bucket-wheel excavator with stacker

Fig. 7—Pneumatic tyred scraper and bulldozer in action

Fig. 8—Mobile stacker conveyor mounted on Sherman tank hull

Fig. 9—Dragline excavator in action

These machines are fast and extremely mobile and are convenient for use in remote areas. Sand, both loose and consolidated, is removed to the depth of the gravel beds which may lie anything from a few feet to 40 feet or more below surface. Fig. 7 shows these scrapers in action.

From 1954, when these scrapers were first introduced, to 1957, a volume of some $12\frac{1}{2}$ million cu. yd. of overburden was removed by thirty 10-ton scrapers at an average of 57·5 cu. yd. per hour and a direct overall cost of 10d. per cu. yd.

These scrapers work in teams of three scrapers to one pusher tractor. The latter originally were pneumatic tyred four-wheel drive tractors, but these are now gradually being replaced by track type pushers.

During 1956 six 22-ton scrapers were added to the fleet. These machines average 105 cu. yd. per hour and, so far, the direct cost including pusher tractor is 5·5d. per cu. yd.

The round trip for these scraper teams is usually about a half to one mile.

3. EXCAVATION OF DIAMONDIFEROUS GRAVELS

The diamondiferous gravels, or "terrace" as this material is locally named, are in fact raised marine beaches which have over the centuries become covered by wind-blown sand overburden.

The terrace itself is a heterogeneous mixture of material, as thrown up by the sea, and consists of fine sand, gravels and boulders. All gravels and boulders have been water-worn to well rounded shapes.

Contained within the terraces in many areas is a cemented material, known locally as conglomerate, which frequently has to be blasted. Mainly this conglomerate is collected at the mining face and hauled away for separate treatment through a central crushing plant.

Mining faces are opened on deposit and advance north and south from the primary cut. Depending upon the width of the reserves, mining faces may be anything from about 200 metres to 1,000 metres in length east to west.

Overburden is cleared ahead of each mining face for a distance of ± 50 metres, the loose sand being deposited back into the worked out area. Where consolidated sand overburden, which is too "tough" for the bucket-wheel excavators to handle, is encountered below the loose sand, this is removed by the 22-ton scrapers.

The mining procedure cycle is:—

(*a*) Overburden removal.

(*b*) Excavation of free gravels.

(*c*) Bedrock cleaning.

(*d*) Overburden removal, when mining face reaches *in situ* overburden bank.

Each mining area has several sets of faces available so that all stages of the mining rotation are in progress at all times.

As stated earlier, prior to 1947 all loading from mining faces was done by hand directly into 36 cu. ft. side tipping trucks running on a 24 in. gauge track. Haulage was with mules, each animal coupled to two trucks.

As soon as they became available, diesel powered 19 cu. ft. capacity bucket excavators were introduced for loading and $2\frac{3}{4}$-ton diesel locomotives for haulage (see Fig. 10).

During 1954 a further advance was made with the introduction of 10-ton capacity, pneumatic-tyred diesel-powered rear dump trucks. Eighty-two of these machines are now in operation on the mine. With the introduction of these large haulage machines, 38 cu. ft. capacity excavators were introduced for loading (see Fig. 11).

Further improvements in mining methods were:—

(*a*) Conversion of diesel power to electric power by trailing cable for the operation of bucket excavators, wherever power supply permitted.

(*b*) The introduction of overhead line electric locomotives for main line tramming.

(*c*) The introduction of heavy diesel-hydraulic locos for main line tramming over long distances.

The present complement of locomotives in use on the mine is:—

Fifty $2\frac{3}{4}$-ton diesel locos.
Six 5-ton electric locos.
Nine 10-ton electric locos.
One 20-ton electric loco.
Two 20-ton diesel-hydraulic locos.
Three 20-ton diesel-electric locos.

These latter three locos date back to the early 1930's and were recommissioned during the rapid expansion period of the mine.

All main line haulage is on 60 lb. per yard track at 24 in. gauge (see Fig. 12).

At present, with the exception of two areas where conditions do not favour a change, all haulage from mining faces is by the pneumatic-tyred rear dump trucks.

Compressed air at mining faces is supplied, where required, by mobile electric or diesel powered compressors of 105 c.f.m. output.

Terrace depths vary over the mine from about 12 inches up to 12 feet, although the average depth throughout is approximately $3\frac{1}{2}$ ft. Straight loading by excavator is resorted to wherever a depth of approximately 3 ft. or more prevails. With lesser depths the terrace is bulldozed into convenient stockpiles using conventional crawler type 70 h.p. bulldozers.

The footwall throughout most of the mining area is a relatively soft schistose rock which has been eroded by sea action into gullies and potholes, sometimes 10 ft. or more in depth. Diamonds naturally tend to concentrate on or near the footwall and particularly in these gullies and potholes.

Owing to the very uneven nature of the footwall, no suitable machinery has been found for the final cleaning and loading from these gullies and the major portion of the African labour used on the mine is employed in bedrock cleaning. See Fig. 2.

Previously, bedrock cleanings were hand-lashed forward to the face and there loaded with the main terrace. Several types of machines were tested to assist this slow and arduous work, including a mobile scraper fitted with a grizzly and mounted on crawler tracks, and also a crawler-mounted front end loader. Neither of these machines was really successful. Conventional type light bulldozers assist quite appreciably and are used where bedrock is not too uneven.

The most successful method so far evolved is a combination of hand labour, using compressed-air-driven rock drills and paving breakers, and troughed conveyor belts, called "face conveyors," running parallel to the mining face. These face conveyors (Fig. 2) are 24 in. wide running at 250 ft. per minute. They are made up of light steel channel sections 15 ft. long which are articulated to follow the uneven rock contours. The sections rest on 60 lb. tracks laid at 90° to the line of the conveyor which can thus be "barred" forward as cleaning advances.

At the loading points conveyors are fitted with "lashing shields," made up of old conveyor belting, to prevent overthrow. The face conveyors each feed on to a

Fig. 10—2·75 ton locomotive with 36 cu. ft. trucks

Fig. 11—Ten-ton rear dump truck and 38 cu. ft. excavator

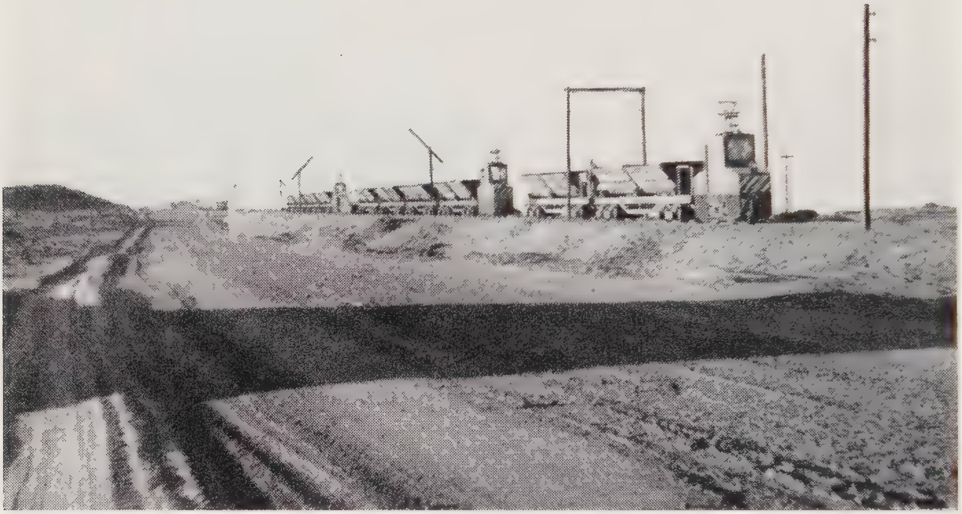


Fig. 12—Main haulage. Ten-ton electric locomotive with 5 cu. metre trucks

mobile stacking conveyor, similar to those used in overburden stripping, which piles the excavated material into a stockpile just clear of the mining area. Lashing gangs engaged in bedrock cleaning are paid on an "incentive bonus" scheme based upon area cleaned. Bedrock cleaning must follow closely behind the face mining otherwise overburden removal ahead of the face is held up and the sequence of mining procedure disrupted.

4. FIELD SCREENING OR PRELIMINARY TREATMENT OF GRAVELS

Normally the work of a mining section terminates with the haulage of mined material. In The Consolidated Diamond Mines of South-West Africa, Limited, however, the mining department's duties are carried one stage further, i.e. to the preliminary treatment, or classification, of the diamondiferous gravels.

Owing to the fact that these deposits are being worked over a long narrow strip, extending at present for about 30 miles, it would be uneconomic to haul the large volume mined over long distances to a central plant when some 80 per cent or more of the material mined can be eliminated in the field.

For this reason field screening plants are sited at convenient intervals along the strike of the deposits and usually on the western, or seaward, side thereof. In the best interests of mining and tramming, the field screening plants should be located not more than 4 miles apart.

Fig. 13 shows a sketch plan of that portion of the reserves at present being mined. The positions of the field screening plants and the central treatment plant are marked and it will be noted that three field screening plants are located well inland from the coast. This is due to the disposition of the terraces in the southern section of the mine and the plants are so situated to obviate long tramming distances for the unwanted "tailings" portion of the deposits.

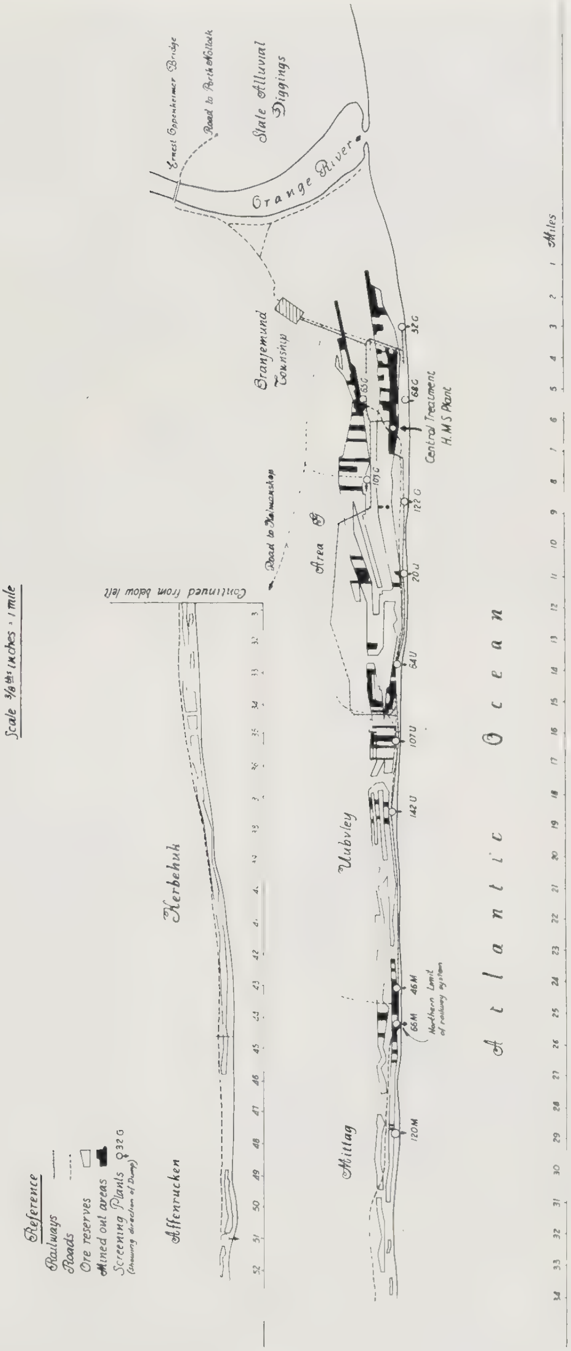


Fig. 13—General plan, Consolidated Diamond Mines.

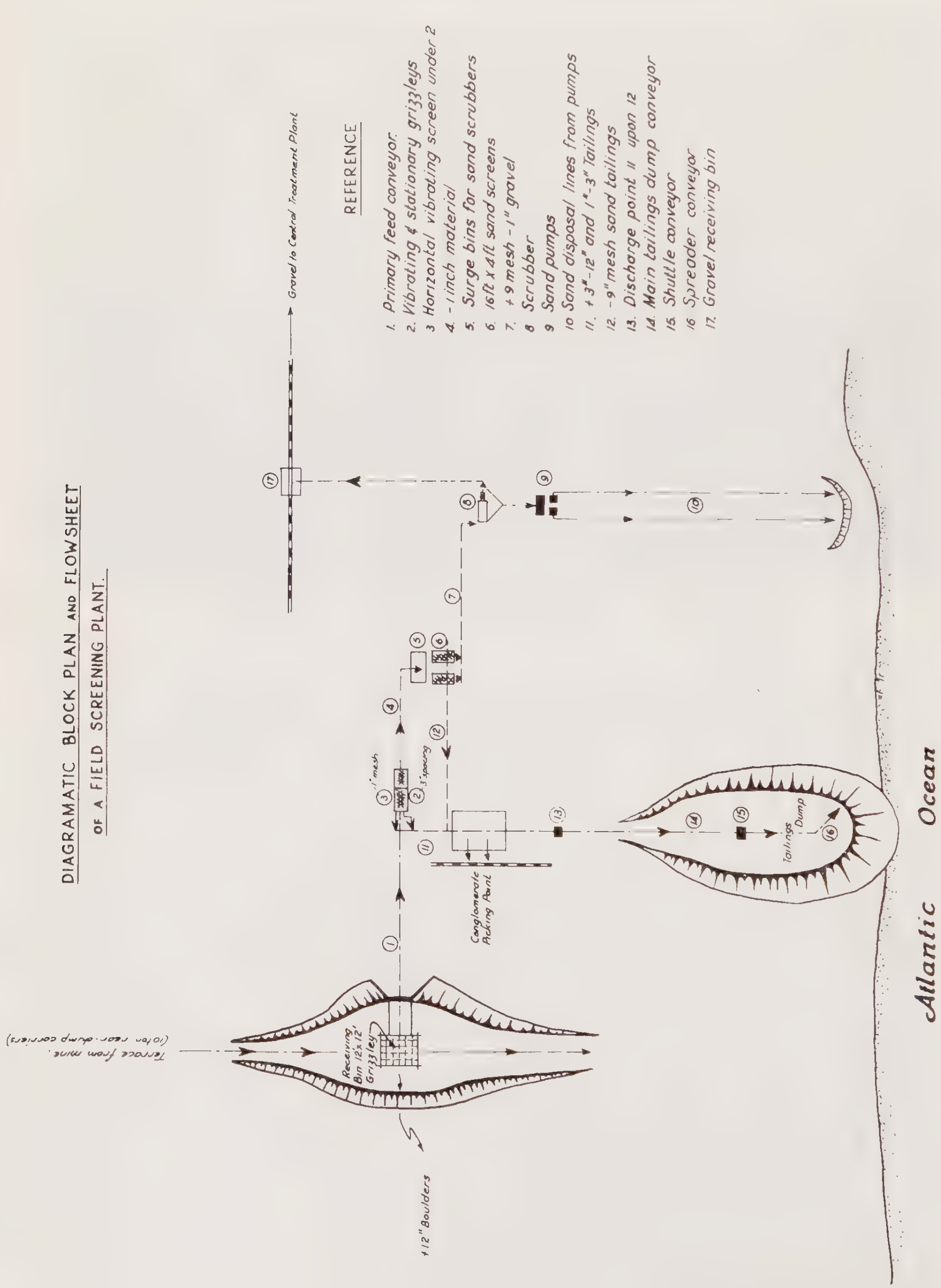

DIAGRAMATIC BLOCK PLAN AND FLOWSHEET
OF A FIELD SCREENING PLANT.
REFERENCE
1. Primary feed conveyor.
2. Vibrating & stationary grizzleys
3 Horizontal vibrating screen under 2
4. -1 inch material
5. Surge bins for sand scrubbers
6. 16ft x 4ft sand screens
7. +9 mesh -1" gravel
8 Scrubber
9 Sand pumps
10 Sand disposal lines from pumps
11. +3"-12" and 1"-3" Tailings
12. -9" mesh sand tailings
13. Discharge point 11 upon 12
14. Main tailings dump conveyor
15. Shuttle conveyor
16 Spreader conveyor
17. Gravel receiving bin
Gravel to Central Treatment Plant
Terrace from mine.
(10 ton rear-dump carriers)
Receiving Bin 12'x12' Grizzley
+12" Boulders
1" mesh
3" spacing
Conglomerate Picking Point
Tailings Dump
Atlantic Ocean

Fig. 14

Fig. 15—Field screening plant

Where possible the best position for field screening plants is to the west of the deposits where water is easily available and so that waste material (tailings) can be deposited in to the sea.

The field screening plant is the first step in the cycle of treatment of the mined terraces and the plants in use today, of which 12 are at present operating, have been designed to suit the particular needs of the area.

Thirteen years ago no such plants existed on these fields. Terrace was screened near each mining face through hand-rotated trommels similar to those still in use in prospecting. The trommels were hand-fed and gravels recovered were hand-loaded into 36 cu. ft. trucks, which were hauled by mules to a central jigging plant.

During 1946 experiments were conducted using an electrically operated, though somewhat antiquated, vibrating screen of German origin. These experiments were sufficiently successful to warrant further investigation and a screening plant was eventually designed and built on the mine which incorporated two 10 ft. by 3 ft. vibrating screens fed from a 2 cubic metre bin by an inclined conveyor belt. Water sprays were used on the screens to assist separation.

The object of screening, or classification, at this point is to eliminate material unlikely to contain diamonds and which therefore is not required for further treatment. The terrace mined is classified into three products, viz.:—

1. *Sand:* any material less than 9 mesh, or approximately 1/10th inch in size.
2. *Gravel:* material of +9 mesh —25 mm. in size.
3. *Boulder:* material greater than 25 mm. in size.

Sand plus boulders make up on the average some 85 per cent of the total terrace and constitute the waste, or tailings product, eliminated by the field screening plant.

The field screening plant is, therefore, essentially a scalping unit and its general design and operation are illustrated in Figs. 14 and 15. As shown, these field screening plants include a receiving bin of 70 tons capacity from which the ore is discharged by a vibrating feeder and a 30 in. conveyor belt to a grizzly system where the +3 in. boulder is removed. The —3 in. product passes to horizontal vibrating screens equipped with 25 mm. mesh screen cloth. As diamonds of +25 mm. screen size are not found in the gravels, the oversize is discarded. The —25 mm. fraction is carried by conveyor belts to surge bins ahead of sand removal screens where the undersize (—9 mesh) gravitates to a tailings belt which also collects the +25 mm. —3 in. and +3 in. sizings and conveys them to the tailings dump. The resultant +9 mesh —25 mm. gravel flowing from the screen decks is carried forward (*a*) in the case of inland plants (dry screening types) directly to a bin for loading and transport to the central treatment plant or (*b*) in the case of plants positioned near the sea (wet screening plants) is given a further cleaning in a rotating cylindrical scrubber. The scrubber is also fitted with 9 mesh screening and sea water is used for washing. This washing stage is necessary in the case of the more westerly terraces as these deposits are damp from seepage and heavy sea mists and in consequence sand cannot be completely cleared from the gravel merely by screening. After the scrubbing process the gravels are binned and finally transported to the central treatment plant.

Tailings disposal

From three separate points in the dry screening plants and from four points in the wet screening plants tailings or waste material have to be disposed. The additional point in the wet screening plant is from the scrubbers. Here the wet sand passes to a sump from which it is pumped by 4 in. rubber-lined pumps through flanged rubber piping out to sea.

The disposal of the other three products is similar for both types of plant. These are —12 in. +3 in. boulders, —3 in. +1 in. (25 mm.) boulders and —9 mesh sand. Both sizes of boulders are well rounded pebbles and cannot be elevated on conveyor belts if these are inclined more than 4° to 5° to the horizontal. The 9 mesh sand can, however, if necessary, be carried on belts inclined as much as 24° to the horizontal. The tailings disposal belt system, therefore, is arranged to load the sand on to the conveyor belt first and so form a bed to carry the boulders up to the tailings dump at an angle of 18°.

In the case of the inland or dry screening plants, tailings dumps are carried to a height of approximately 120 ft. after which they are continued horizontally at this elevation. Plants situated near the sea form a dump on the edge of the beach approximately 80 ft. in height. These dumps are not continued further as the erosion by sea obviates the need for any extension. On the inland plants a shuttle conveyor is used and is extended as the dump grows. A similar shuttle is frequently used in the case of the wet screening plant as, should the sea wash away part of the dump, the discharge point can be retracted away from the sea.

The shuttle conveyor discharges on to the final disposal belt, known as the spreader, which is a troughed 30 in. belt moving at a speed of 475 ft. per minute. The spreader is of lattice steel construction, 60 ft. long, operating horizontally. It pivots about the tail end and is supported at a point slightly beyond the centre on a roller mounted on a rail car chassis which runs on a semi-circular track. The discharge position on to the spreader remains unaltered no matter what radial position is chosen for the spreader as the discharge chute from the shuttle conveyor is fixed over a receiving chute attached to the spreader directly over its pivot point. This arrangement permits dumping over the periphery of a semi-circle of 60 ft. radius to a horizontal distance of 30 ft. As further extensions to the dump are required, additions are made to the shuttle conveyor (or to the main tailings conveyor) and

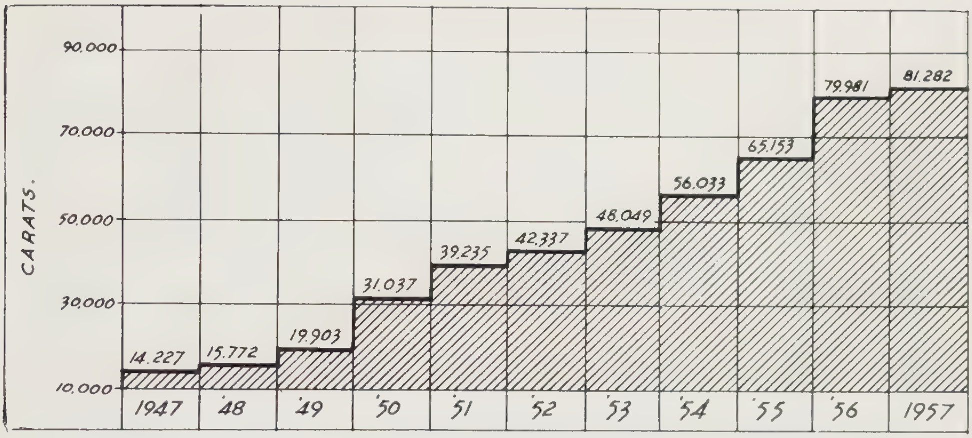
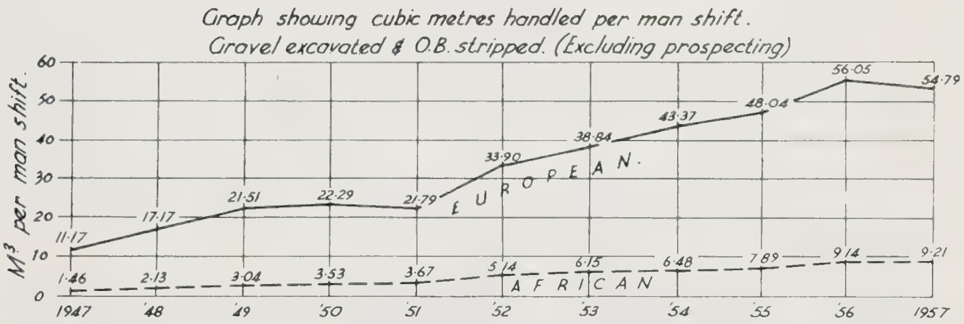


Fig. 16—Annual production of diamonds



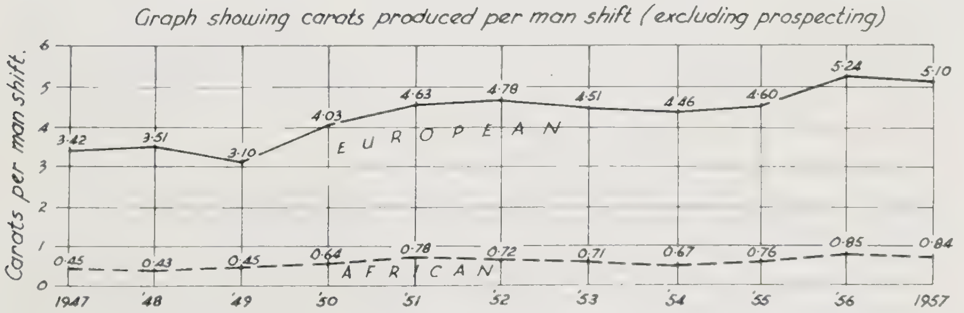


Fig. 17

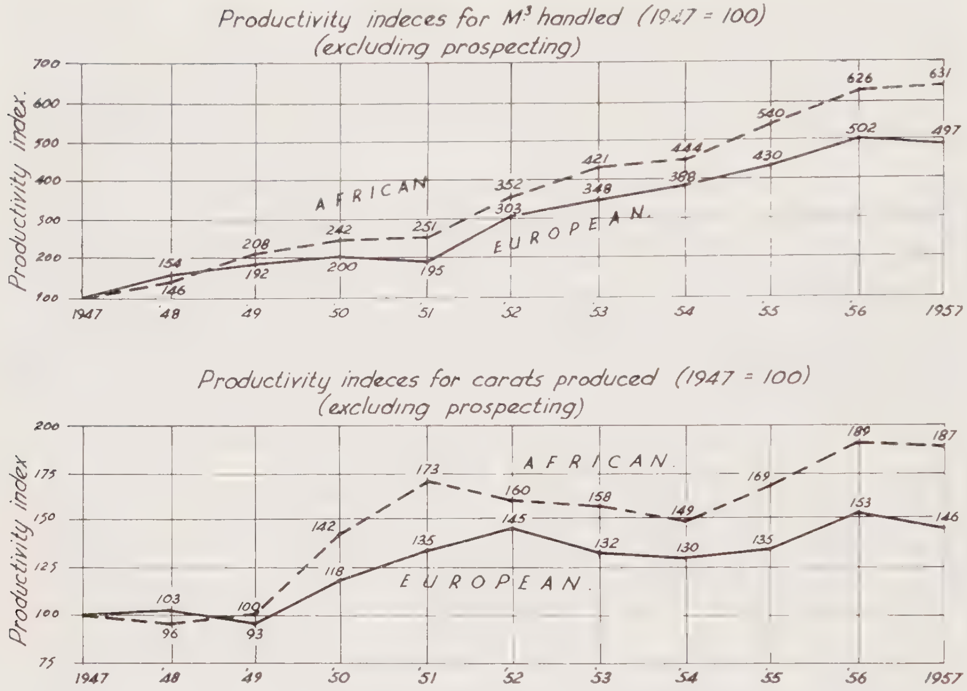


Fig. 18

the circular track is repositioned. All conveyor belt joints are vulcanised and most of the conveyor framework is of timber, 4 in. by 3 in. being used for the frames and 6 in. by 3 in. for runners.

This conveyor system for tailings disposal has, apart from other advantages, considerably reduced the African labour requirements on screening plants and has also resulted in a distinct decrease in the accident rate.

Final Transport

From the field screening plants the final product of -25 mm. $+9$ mesh gravel is trammed in 10 cubic metre capacity side tipping rail trucks to the central treatment plant. Here the gravel passes through tube mills and on to the heavy media separation plant. Concentrates from this process are passed over grease belts or through electrostatic separators, depending on size, for further concentration. Finally, the concentrates from the grease belts and electro-static separators are hand-sorted for diamonds. These final concentrates constitute 0·016 per cent of the feed to the heavy media separation cone and 0·00226 per cent of the initial terrace mined.

CONCLUSION

The 11 years 1947 to 1957 inclusive have been accepted as the period most indicative of the expansion and improvement in mining methods and procedure in the diamond fields operated by C.D.M.

The graph in fig. 5 shows the advance made in the volume of diamondiferous material mined and treated over this period. Fig. 16 shows in the same form the increase in production of carats of diamonds over the same period.

These figures, however, gain even more significance when it is realised that the mining grade has been considerably reduced to obviate the previous tendency to over-mining. During the years 1947 to 1951 inclusive the average grade mined amounted to 0·77 carats per cubic metre. From 1952 to 1957 the influence of improved methods and mechanisation began to take effect and the grade was reduced steadily from 0·72 carats per cubic metre in 1952 to 0·34 carats per cubic metre in 1957. Costs over this latter period were reduced from 29s. 7d. per cubic metre mined to 21s. 7d. per cubic metre mined.

The effect of improved methods and mechanisation are illustrated by means of graphs in Fig. 17, which show the cubic metres handled, i.e. overburden stripped plus terrace mined, and carats produced (excluding prospecting), per European and African shift from 1947–1957.

The fluctuation in carats produced per man shift results from the policy of reducing the mining grade; an overall improved efficiency is, however, indicated.

Fig. 18 illustrates the productivity indices for the same period. In the graphs the productivity per man shift of carats and of cubic metres handled (excluding prospecting) is based on the productivity in 1947 for which an index of 100 is accepted.

A further point of interest in the interpretation of the above graphs is the fact that during the period under discussion, mining operations were extended considerably. In 1947 a strip of approximately 3 miles only was being exploited along the western boundary of the deposits immediately adjacent to the township. By 1957 mining operations were in progress up to a point some 30 miles further northward, and advance development was in progress up to 50 miles to the north.

This fact, plus the very necessary reduction in mining grade, gives adequate reason for the large-scale mechanisation which has taken place over the last decade. Results, as indicated by the above graphs, show that the principle of mechanisation has been proved to be economically sound.

THE TREATMENT OF GRAVELS FOR THE RECOVERY OF DIAMONDS AT C.D.M.

By **S. W. DEVLIN**, B.Sc. (Eng.) (Rand.)* (Member)

SYNOPSIS

The diamondiferous gravels from field screening plants on the mining terraces are conveyed to the central treatment plant by a variety of methods. The subsequent extraction of the diamonds from the gravels is effected by a series of concentration stages which comprise tube milling, washing, two-stage heavy media concentration, magnetic separation, grease or electrostatic processing and finally hand-sorting.

These procedures are described and illustrated.

INTRODUCTION

The methods of prospecting and mining the marine terraces at C.D.M. have been covered in an earlier paper "Mining procedure and method at C.D.M." presented to the Institute in November, 1958. No new or modified methods of any consequence have been introduced since that presentation.

The preliminary treatment of the mined deposits in field screening plants was also discussed. The gravels are subjected to a size separation in these screening plants whereby the oversize boulder (+25 mm) and undersize sand (−1·91 mm) are discarded and the residual diamondiferous gravels despatched to the central treatment plant.

The present paper deals with the transport of the gravels and the treatment processes which result in the final recovery of the diamonds.

MINING AREA

Ore reserves at present known and exploited by The Consolidated Diamond Mines of South-West Africa, Limited, extend over a distance of some 55 miles northward from the Orange River and parallel and adjacent to the sea coast of the Atlantic Ocean.

Owing to the extended nature of the property the conveyance or tramming of gravels from the field screening plants to the central treatment plant is a major operation of organization and transport.

A plan of reserves showing the position of the various field screening plants relative to the central treatment plant is shown in Fig. 1. There are 12 field screening plants in operation and, as explained in the previous paper,[1] their function is to effect the separation and discard of oversize and undersize waste matter.

For the year 1959 the average recovery of diamond-bearing gravels from the screening plants amounted to 15·5 per cent of the total deposits screened, or some 37,000 cubic metres (64,750 tons) per month.

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* General Manager, The Consolidated Diamond Mines of South-West Africa, Limited.

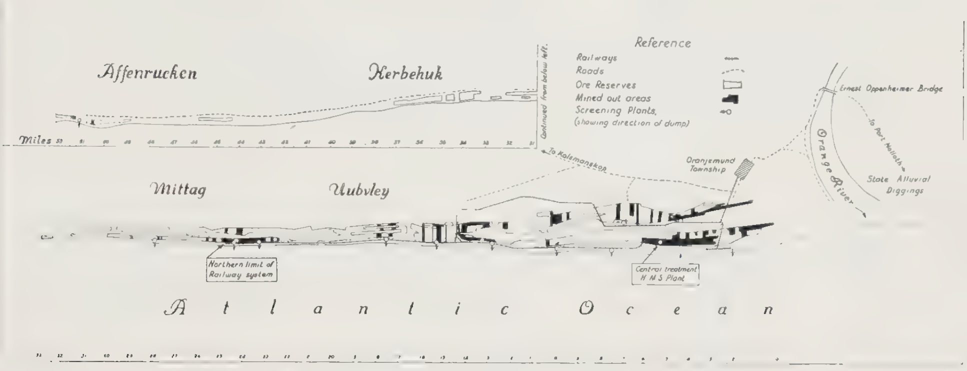

Fig. 1

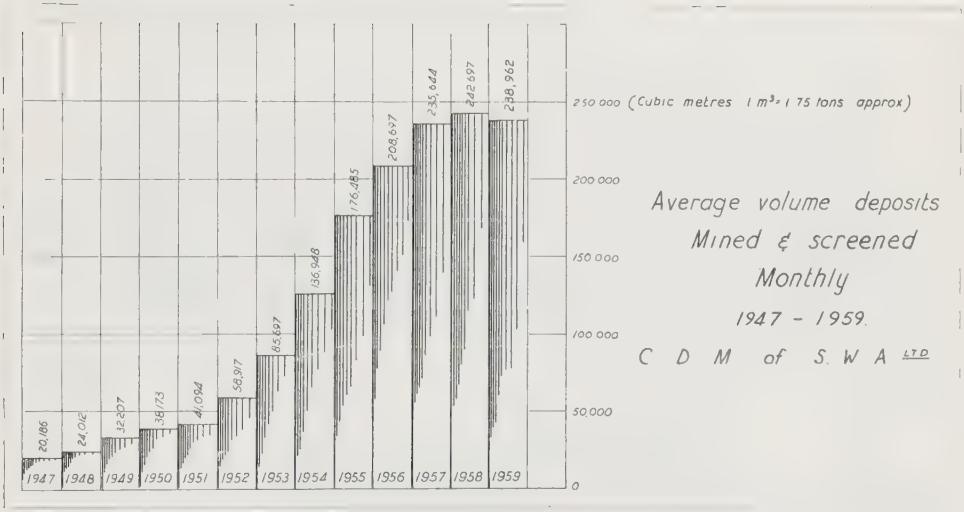

Fig. 2

Fig. 2 is a graph of the volume per month of deposits mined and screened for the period 1947-1959 inclusive.

Owing to improved scalping and screening practice the percentage of gravel recovered to terrace mined has steadily decreased from over 25 per cent in 1947 to 15·5 per cent in 1959. After the screening operation the gravel is conveyed to a central treatment plant where the recovery of the diamonds is carried out.

CONGLOMERATE CRUSHING PLANTS

The incidence of conglomerate in the deposits has been previously referred to.[1] It is collected mainly at the mining faces, though a small quantity is also collected from picking belts at the field screening plants.

Three crushing plants are at present in operation on the property—one in area H where the crushed product is treated in the local plant, one adjacent to the field screening plant north of the northern rail terminal and one immediately adjacent to the main storage bins of the central treatment plant. The crushed product from the latter two plants is all treated in the central treatment plant.

Fig. 3 shows the flow diagram of a conglomerate crushing plant.

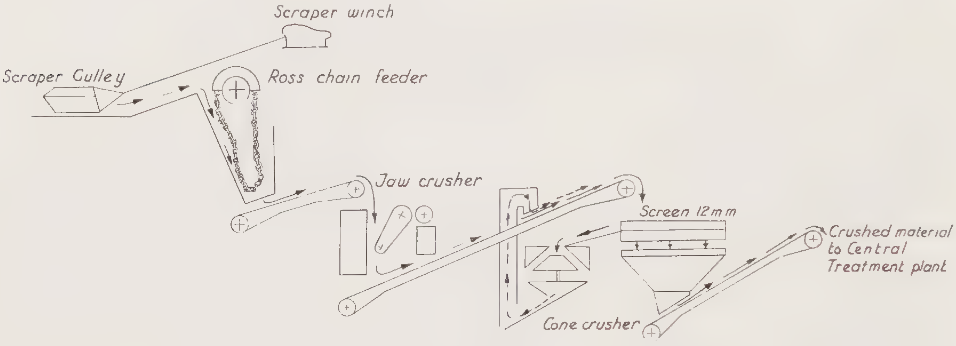

Fig. 3

The conglomerate found in these deposits consists of the usual hard, water-worn gravels and pebbles of the terraces bound together by a relatively soft calcareous cement. Lenses or patches of conglomerate occur, in no definite pattern, both in the terrace above bedrock and in the potholes and gullies of the bedrock. Conglomerate forms only a relatively small proportion of the deposits mined in any locality.

In past practice the breaking down of this material has been attempted by rolls and by pebble milling without much success. The existing practice of two-stage crushing was eventually evolved and the first small crusher plant was commissioned during 1950.

Conglomerate collected from field screening plants and mining faces is dumped either from rail trucks or rear dump trucks into a 75 ft. long by 7 ft. wide steel-lined bin (old rail track used as liners). From this bin the material is scraped by a conventional 35 h.p. double-drum scraper winch, using a crescent-shaped scoop, onto a 12 in. by 12 in. grizzley and thence via a Ross chain feeder to a 24 in. by 14 in. jaw crusher set to a 1 in. gap. The crushed product is conveyed to a 4 ft. by 10 ft. vibrating screen with 14 mm aperture screen cloth. Minus 14 mm crushings join the gravel from field screening plants into the main storage bins while the +14 mm material is recirculated through a 36 in. short head cone crusher. This circulating load seldom builds up to unmanageable proportions.

CONVEYANCE OF GRAVELS

A light railway system using 60 lb per yard rail at 24 in. gauge serves all field screening plants in the area up to a point some 20 miles north of the central treatment plant and all plants to the south and east thereof.

The main line parallel to the sea coast for the first 10 miles north of the central treatment plant is a double track and the total railway network involves some 55 miles of track.

All rail-track is laid on sand embankment and is ballasted with gravel tailings from the treatment plant. Owing to frequent high winds and heavy sea mist conditions, constant maintenance is required to keep the track in good condition.

One field screening plant is sited about four miles beyond the northern limit of the track and gravel is trammed from this plant to a railhead transfer bin by the pneumatic-tyred rear dump trucks used for tramming mined deposits to the screen plants.

At a point some 50 miles north of Oranjemund (known locally as area H) a field screening plant has been recently commissioned (see insert Fig. 1). This plant serves a self-contained area which has a small power station, workshop, compound, European single quarters and mess, and gravels are treated on the spot by mechanized jigs similar to those used in prospecting operations. These machines are described later in the treatment plant section (Figs. 4 and 5).

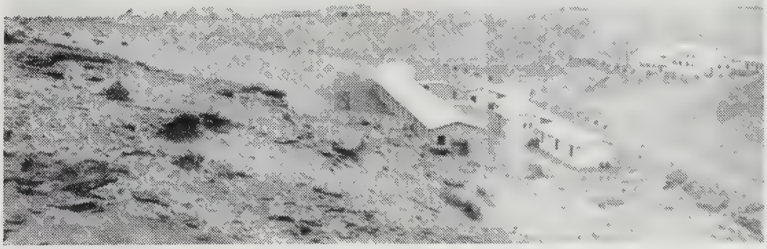

Fig. 4

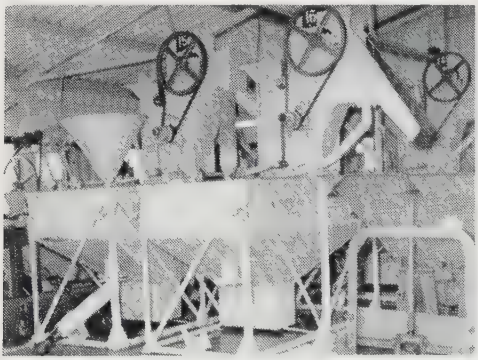

Fig. 5

LOCOMOTIVES

The rapid expansion of mining operations in the early post-World War II years laid a big strain on main line haulage facilities and it was some years before the right type of locomotives became available.

At present in use are:

(*a*) Three 20 ton diesel electric locomotives,

(*b*) Six 5 ton electric locomotives,

(*c*) One 20 ton electric locomotive,

(*d*) Nine 10 ton electric locomotives,

(*e*) Three 20 ton diesel hydraulic locomotives.

Item (*a*): These three locomotives date back to the mid-1930's when this company was operating mainly in Kolmanskop and Elizabeth Bay areas near Luderitzbucht. They are of German origin and though virtually "written-off" they were in fair condition. They were stripped and transported to Oranjemund about 1947-48, overhauled and commissioned, using the original M.A.N. engines and Siemens generators.

Later G.M. diesel engines were substituted, but the original generators are still in use (Fig. 6).

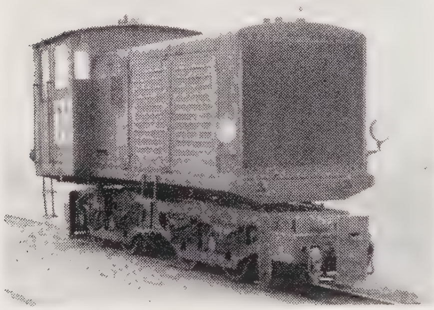

Fig. 6

Fig. 7

Item (*b*): These locomotives have an interesting history. With locomotives being unobtainable for the steadily increasing haulage requirements, attention was drawn to a number of electric motor bogies which had formed the driving units of large double-bogie diesel electric locomotives in use at Kolmanskop in pre-war days. These units had long since been stripped and scrapped. The electric motors, wheels and chassis were, however, still in usable condition, and the mine engineering staff designed and built an overhead trolley line type locomotive on one bogie (Fig. 7).

After some teething troubles with pantograph pick-up and control gear the first 5 ton electric locomotive was commissioned and proved so successful that all available bogies were later built up into similar units and for some years formed the mainstay of the haulage fleet.

Item (*c*): Later it was found possible to purchase a second-hand double-bogie 20 ton electric locomotive (Fig. 8) and subsequently nine new 10 ton Greenbat electric locomotives were purchased.

Item (*d*): These 10 ton locomotives are illustrated in Fig. 9.

Item (*e*): Jung diesel hydraulic locomotives are shown in Fig. 10.

Owing to the strong winds on this coast, considerable erosion from "sand blasting" of metal structures is experienced. If not kept fully protected, the metal also

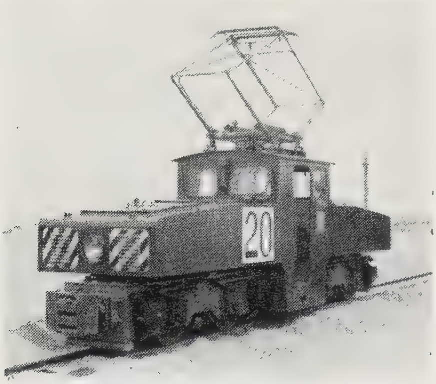

Fig. 8

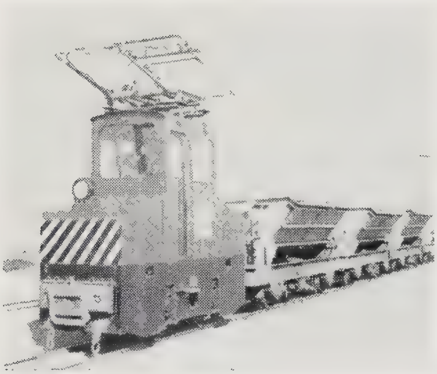

Fig. 9

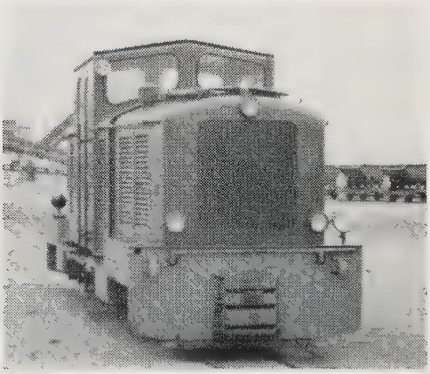

Fig. 10

rapidly corrodes from the action of sea mists. For this reason wooden poles are used for the support of overhead trolley lines. However, these are expensive to install because of the high cost of transporting the poles to this isolated area and because of the high initial and transport cost of the trolley wire, insulators, etc.

When the mining operations were extended beyond area U (a point some 10 miles north of the central treatment plant) it was decided to extend only the railway track and not the power line, replacing electric locomotives with diesel electric units over this 10 mile extension.

Further expansion of mining operations necessitated an increase in the locomotive fleet and the three diesel hydraulic locomotives were recently purchased. These are faster and more easily operated than the diesel electric machines.

RAIL TRUCKS

In the conveyance of the -25 mm $+1{\cdot}9$ mm gravels from the field screening plants to the central treatment plant it is of great importance that no spillage whatsoever should occur. Bottom or side discharge cars, therefore, cannot be used as the rounded, waterworn pebbles tend to jam in doors and allow the smaller gravel to escape. Where any gravel particle can escape a diamond could also be lost! For this reason also cars are never completely filled.

The trucks in use are therefore all of a side-tipping type, with 5 cubic metre capacity pans (Fig. 11).

Fig. 11

CENTRAL TREATMENT PLANT

GENERAL

Up to and including the year 1950 all gravel recovered from the field screening plants was treated in a central plant consisting of a battery of gravitating jigs similar to those still used on this property in prospecting operations.

These jigs, which are a local modification of the Pleitz jigs, are small individually motorized machines which depend for their operation upon the pulsation of an 18 in. diameter sieve in water. This circular sieve, which is maintained constantly horizontal throughout its movement, pulsates in a vertical plane through a distance of $\frac{1}{2}$ in. to $\frac{3}{4}$ in., depending upon the size of gravel to be concentrated. Water depth is kept just above the highest point reached by gravel particles during the pulsating movement (Fig. 12).

The action of the jig, which is exactly similar to the method of hand gravitation used by prospectors in the field, concentrates the heavier portion of the feed into the centre of the sieve and spills lighter material over the periphery. The heavy material seeps through valve-controlled holes in the centre of the sieve and is "tapped off" at intervals through a side discharge pipe, whilst tailings are continuously discharged by a small bucket elevator from a collection point at the base of the machine.

Each machine in the original plant was fitted with two sieves, each having a gravitating capacity of 1 cubic metre per hour. The plant battery consisted of a number of primary jigs from which the tailings discharged into the feed hoppers of secondary jigs in series.

Originally these machines were hand-fed and tailings were discharged into side-tipping cubic metre rail trucks which were then hoisted up a dump and discharged.

Concentrates from the machines were manhandled by bucket into the sorting house for final hand-sorting and recovery of diamonds.

During 1950 preparations were made for completely mechanizing this jigging plant by conveying the gravel from storage bins, through classifying screens for sizing, into surge bins which would continuously discharge into the feed hoppers of

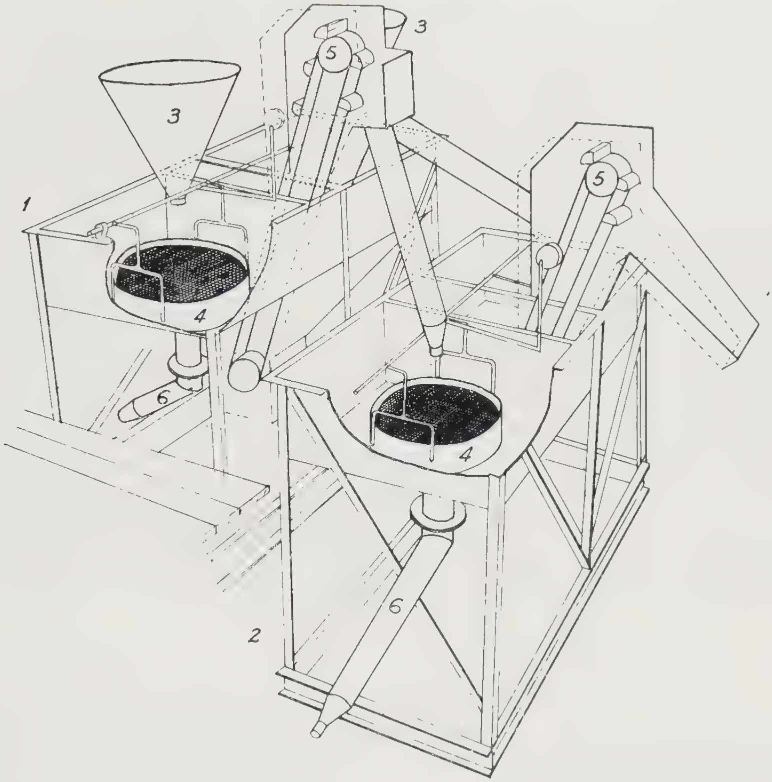

Fig. 12—Legend

1. Primary 'double jig'
2. Secondary 'double jig'
3. Feed cones
4. Jig sieves—with concentrate control valves
5. Tailings elevators
6. Concentrate 'draw-off' pipes
7. Final tailings discharge chute

the jigs. An airlift pump was developed on the mine which could continuously draw off heavy mineral concentrates and discharge them directly into the sorting house bins.

Before this new plant was completed, however, the process of concentrating diamondiferous gravels by heavy media separation (H.M.S.) was established at the Premier Diamond Mine, Cullinan, Transvaal, and the original pilot plant from Premier Mine was transferred and installed at C.D.M. This H.M.S. plant was commissioned in May, 1951.[2]

During the past nine years the central treatment plant has been considerably modified, improved and enlarged but the primary H.M.S. section is still fundamentally the same as that which was originally installed (Fig. 13).

Basically the process of diamond recovery by heavy media separation is dependent upon the principle of selecting a fluid of suitable density which will permit the

Fig. 13

bulk of the unwanted material (tailings) in the diamondiferous gravel to float and so be discharged to waste, and the remaining small proportion of heavy mineral content of approximately the same specific gravity as the diamond to sink through the fluid to a suitable collection point.

In this regard heavy media separation may be termed a "qualitative process" as opposed to the "quantitative process" of the jig whereby a predetermined quantity of the gravel is permitted to pass through the control valves of the jig sieve.

Obviously the former process is more accurate and is unaffected by a varying heavy mineral content. Particle size plays no appreciable part in the action of heavy media separation which obviates the necessity of classification or pre-sizing of gravels. Furthermore, diamonds are segregated by virtue of their specific gravity alone and no mechanical action, in the true sense, plays any part in their recovery. In consequence the H.M.S. process is by far a more definite and easily controllable process and the ultimate efficiency is therefore greater than that obtained by jigging.

The gravel concentrates leaving the H.M.S. circuit average in quantity 1·5 per cent of the feed from the main storage bins. The concentrate undergoes the following series of reconcentration for final diamond recovery:

(*a*) Magnetic separation.

(*b*) Attrition milling.

(*c*) Electrostatic separation (−6 mm+1·91 mm gravel only).

(*d*) Grease belt separation (+6 mm −25 mm gravel only).

(*e*) Hand-sorting.

The flow diagram of the central treatment plant is illustrated in Fig. 14.

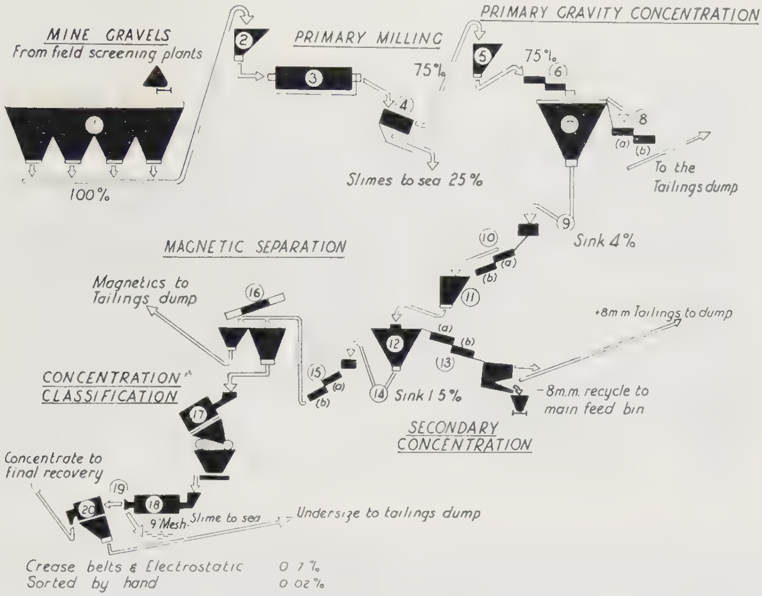

Fig. 14—Legend

1. **Receiving bins gravel from field screening plants capacity 400 tons**
2. **Tube mill bin**
3. **6 ft by 22 ft tube mills**
4. **4 ft by 10 ft vibrating wash screen**
5. **H.M.S. storage bin capacity 1,000 tons**
6. **5 ft by 10 ft vibrating wash screen, fresh water spray**
7. **Primary H.M.S. cone**
8. **(*a*) Float drain screen, (*b*) Float wash screen } Fresh water spray**
9. **Sink airlift**
10. **(*a*) Sink drain screen, (*b*) Sink wash screen } Fresh water spray**
11. **Surge bin secondary H.M.S. cone**
12. **Secondary H.M.S. cone**
13. **(*a*) Float drain screen, (*b*) Float wash screen } Fresh water spray**
14. **Sink airlift**
15. **(*a*) Sink drain screen, (*b*) Sink wash screen } Fresh water spray**
16. **Dings in-line magnet**
17. **Classifying or sizing trommels, three**
18. **Differential grinding mills**
19. **De-watering screens 1·91 aperture mesh**
20. **Clearing trommels**

PRIMARY MILLING

Diamondiferous gravel is received into the central treatment plant main storage bins from two sources:

(*a*) Gravel direct from field screening plants,

(*b*) Gravel in the form of crushed conglomerate from crushing plants.

Conglomerate comprises only approximately 5 per cent of the total gravel delivered to the central treatment plant. This gravel is tipped and stored in four 1,000 ton capacity rectangular bottom-discharge bins. The discharge from these bins is through a number of clam-shell type doors on to 30 in. troughed conveyor belts. Twelve discharge points serve each bin (Fig. 15).

Each bin discharge belt feeds separately to an inclined 30 in. conveyor system, equipped with a continuous weigher, and thence to surge bins ahead of three tube mills (Fig. 16).

Details of primary mills:

Size: 6 ft. diameter by 22 ft. long.

Discharge: peripheral.

Fig. 15

Fig. 16

Speed: 27 r.p.m., i.e. 80 per cent critical speed.
Grinding media: 5 in. diameter pebbles from mine.
Grinding load: Four tons per mill.
Average feed rate: 60 tons per hour.
Average reduction by mills: approximately 20 per cent of feed.
Mill discharge moisture: 22 per cent.

Each mill is fed from the surge bins by a locally designed water-jet feeder using approximately 4,000 gallons of sea water per hour.

Mill liners previously used were the normal white iron type but recently Ni-hard liners were fitted and are giving a much-improved liner life.

Mill discharge from each mill passes over a vibrating screen fitted with sea water sprays. Screen cloth used is of stainless steel with 1·91 mm aperture.

Clean gravel is conveyed to a 1,000 ton capacity H.M.S. surge bin similar in design and situated adjacent to the plant storage bins. The bin is filled by a shuttle conveyor operating on rails across the top of the bin.

Minus 1·91 mm sand and slimes from milling is pumped direct to the sea coast some 2,800 ft. away.

Initially sand slimes disposal was effected by 4 in. rubber-lined pumps in three stages each comprising three pumps discharging through 6 in. rubber piping. A more recent installation consists of three sets of two 6 in. Hydroseal pumps connected in series and installed in the mill building. A considerable saving in working costs and maintenance has been effected by this new arrangement.

Sand slimes disposal data

Distance pumped	2,800 ft.
Vertical lift	13 ft.
Delivery pressure at pumps	45 p.s.i.
Volume pumped	25,000 galls. per hour per set of pumps
Solids disposal	36 tons per hour

The primary function of the mills is to break down nodules of hard sand, containing fine-grained magnetite, which pass over the field screening plant screens, and to remove any sand left after dry screening.

The magnetite is present in the deposits in sufficient quantity to interfere with the operation of the H.M.S. cone as it becomes entrained in the ferro-silicon circuit.

A further function of the mills is to break up sea shells and any other friable material in the gravels, thus reducing the quantity of feed to the H.M.S. cone. Sea shells tend to collect ferro-silicon in the cone and sink with the heavy minerals where they have a distinct nuisance value in later recovery processes.

HEAVY MEDIA SEPARATION PROCESS (H.M.S.)

The process of extracting the heavy mineral content, including diamonds, from the milled gravels is done in two stages, referred to as the "primary" and "secondary" cones respectively.

Primary cone

Clean, milled -25 mm $+1{\cdot}91$ mm gravel is conveyed from the H.M.S. storage bin to the top floor of the H.M.S. building where it passes over a double-decked vibrating screen. The upper deck of this screen is fitted with $1\frac{1}{2}$ in. aperture screening, for removal to waste of any large pebbles which may have come through from the primary mills. The lower deck is equipped with 1·91 mm aperture screen cloth and is fitted with fresh water sprays which wash off salt and any accumulated dust. Washed gravel thereafter feeds directly into the primary cone through a "diving" ring near the cone centre (Fig. 17).

The cone, as its name implies, is an inverted conical steel vessel which is 12 ft. at the maximum diameter and 11 ft. in vertical depth.

This vessel contains a mixture of ferro-silicon and fresh water which forms a suspension having a density of 2·90 at the upper limit of the cone and a density of 3·03 at the concentrate discharge end. At this density the media viscosity in the cone—measured by a modified Stormer viscosimeter—is 16 to 18 centipoise.

The cone contains a volume of 3,300 gallons of fluid in which 38 tons of ferro-silicon are in suspension.

The ferro-silicon used in the primary cone circuit is an angular milled product with the following specifications:

Specific gravity ...	6.75 minimum.
Si content ...	Maximum 15·5 per cent—minimum 14·5 per cent.
Non-magnetics ...	0·75 per cent maximum.
Tyler grading ...	+48 mesh: nil.
	+65 mesh: 5 per cent maximum.
	—325 mesh: 40 per cent minimum; 50 per cent. maximum

The rate of feed to the primary cone is 112 tons per hour, which is considered to be the maximum safe average rate of feed. Feed rate is checked by a continuous weigher installed on the feed conveyor. Readings from this weigher are indicated on dials installed at three points, viz., the Plant Superintendent's office, the H.M.S. storage bins and the primary cone control panel.

Gravel of specific gravity lower than 2·9 floats in the fluid and is discharged, as tailings, at a point on the periphery of the cone on to the 5 ft. by 10 ft. float drain screen.

Fortunately there is very little material in the gravel with a specific gravity in the range between the top and bottom cone densities, and that which does collect

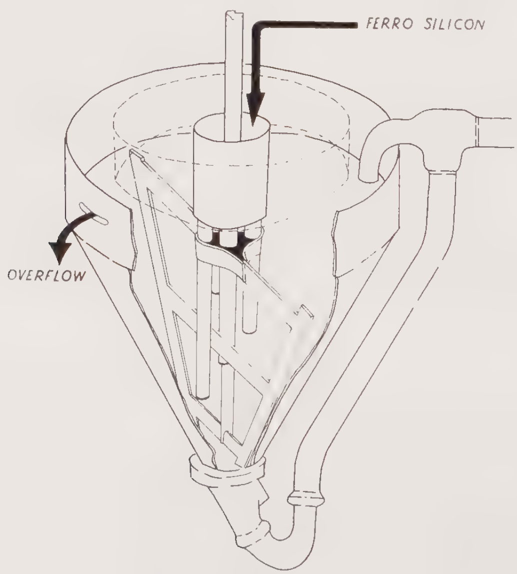

Fig. 17

in the middle of the cone is gradually removed by the down draught created by the airlift.

Both primary and secondary cones are "dropped," or emptied, once per week for maintenance and cleaning and any gravel still in circuit is then cleared before the cone is refilled.

Heavy minerals settling to the bottom of the cone are continuously withdrawn by an airlift pump through a 6 in. diameter rubber pipe which delivers the concentrates to the vibrating 3 ft. by 10 ft. sink drain screen.

With both "float" and "sink" discharges, a fairly large volume of fluid from the cone flows out with the gravel and is recovered by drainage through the 1·91 mm aperture screen cloth of the float-drain and sink-drain screen in each case. This underflow is pumped directly back to the cone (Fig. 18).

In series with the drain screen for both "float" and "sink" material, is a similar screen known as the "wash screen," which is fitted with low-pressure fresh-water sprays to wash off any ferro-silicon still adhering to the gravel. The underflow from the wash screens discharges directly to a 20 ft. diameter thickener.

Details of the medium recovery circuit will be given later.

After washing, the tailings or "float" product discharges directly to a conveyor system which conveys it to two parallel tailings dumps.

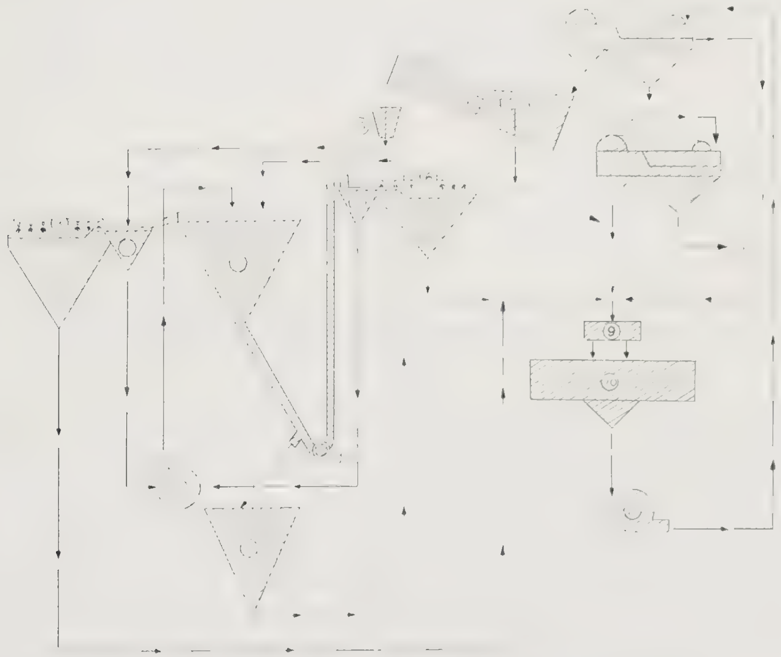

Fig. 18—Legend

1. Primary cone
2. Float drain screen
3. Float wash screen
4. Airlift pump
5. Sink drain screen
6. Sink wash screen
7. Drop sump
8. Drop sump return pump
9. Magnetising block (Electro)
10. 20 ft diameter thickener
11. Thickener underflow pump
12. Primary magnetic separator
13. Secondary magnetic separator
14. Densifier (48 in. Akins spiral classifier)
15. Demagnetising coil
16. Sand-slimes to main slimes disposal

A regular cut is taken from the tailings and tested through C.D.M. type jigs, the actual amount tested per shift being about 1 per cent of the tailings flow. The efficiency of the process is assessed mainly by the absence or presence of any heavy minerals in the float though an occasional stone has been found in the test samples. The average concentration ratio in the primary cone is at present 16 : 1.

Concentrates, or "sink" material, flow to the secondary cone feed bin.

Secondary cone

The secondary or cleaner cone circuit is similar in most respects to that of the primary cone, though the cone is smaller, being only 5 ft. diameter.

In this cone the use of a spheroidal type of ferro-silicon, instead of the angular type, permits operation at a higher specific gravity without undue increase in viscosity, which at this density averages 22 centipoise. Unfortunately this medium has not proved successful for use in larger cones.

Density in the secondary cone is maintained at 3·15 in the upper portion, and 3·38 to 3·40 at the bottom, and the concentration ratio effected by the cone is 3·3 : 1.

"Sink" and "float" materials are drained and washed as for the primary cone and the ferro-silicon recovery circuit is identical.

At the high density at which this cone is operated some of the smaller diamonds have reported with the float product. All float from the cone is therefore passed over an 8 mm aperture sizing screen, the undersize from which is recycled to the primary cone. As both cones are cleared once each week no appreciable build-up of this concentrate occurs. Projected modifications of the H.M.S. circuit indicate that the recycling of this −8 mm undersize may safely be discontinued at some later date.

Recovery efficiencies

The efficiency of both primary and secondary cones is tested regularly by the introduction into the circuit of radioactively charged diamonds of various sizes. Twenty to 50 such diamonds are introduced into the cone feed and the passage of the stones is traced by a team of three strategically placed operators armed with scintillometers.

As deduced from daily tests, the average apparent efficiency of the primary cone is 99·9 per cent and that for the secondary cone, after recycling of the −8 mm float, is 100 per cent.

Ferro-silicon recovery circuit

The ferro-silicon used in both cones is a fairly expensive material and, being extremely heavy, transport costs are high. It is therefore essential to reduce loss of this material to an economic minimum.

The average loss from the primary cone is 0·4 lb per ton of feed and that from the secondary cone is 0·6 lb per ton of feed.

The ferro-silicon recovery circuit flow diagram (Fig. 18) clearly shows the details of recovery.

Ferro-silicon having a high magnetic susceptibility is easily recoverable by conventional magnetic separation methods from the small amount of sand and slimes washed through from the sink and the float wash screens. Crockett type magnetic separators are used for this purpose.

To aid rapid settling of the ferro-silicon in the thickener, the material is coagulated by passing the pulp through a magnetizing block.

The densifier—item 14 in the flow diagram is the main holding point for ferro-silicon. Cone density is regulated by increasing or decreasing the flow from the densifier to the cone by raising or lowering the feed spiral.

Ferro-silicon particles in suspension in the cone must be completely free from magnetic attraction for one another, and prior to entering the cone from the densifier all ferro-silicon passes through a strong demagnetizing coil. This is an important feature in the control of cone density and viscosity.

The ferro-silicon recovery circuit is exactly similar for both primary and secondary cones.

MAGNETIC SEPARATION

A fairly large proportion of the heavier gravel associated with diamonds in this area is banded ironstone and similar material having a fairly high magnetic susceptibility, and over a considerable period various conventional methods of magnetic separation were experimented with at different stages in the treatment circuit. The increased concentration ratio effected by the secondary H.M.S. cone during late 1958 and 1959 led to the design of a simple magnetic separator incorporating a Dings "in-line" electro-magnet (2,000 Gauss at 1 in.). This magnetic separator was developed by the mine staff and commissioned during the latter part of 1959 (Fig. 19). An extraction of approximately 20 per cent of the feed or 0·3 tons per hour is effected by the separator.

A feature of this magnetic separator is that the field is effective on the trajectory of falling particles only, and thus the danger of diamond entrainment is negligible. To date no diamonds have been found in tailings tests.

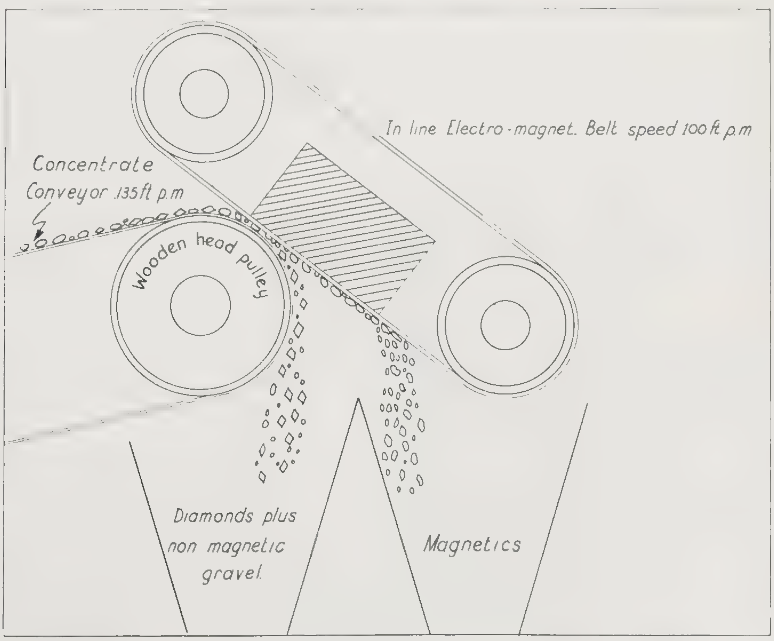

Fig. 19

ATTRITION MILLING

Concentrates from the magnetic separator are conveyed to the classifying, or sizing, section where three rotating trommels are used for screening to the four size ranges required for grease concentration or electrostatic separation.

Trommels are used here as it has been found that for the classification of low tonnages of rounded gravel they are more efficient and are self-clearing.

The four size ranges produced are:

(*a*) +1·91 mm −6 mm : referred to as "fines."
(*b*) +6 mm −12 mm : referred to as "middles."
(*c*) +12 mm −18 mm : referred to as "coarse."
(*d*) +18 mm −25 mm : referred to as "oversize."

All four products are milled separately in $3\frac{1}{4}$ ft. by 8 ft. trunnion discharge type mills fitted, at present, with manganese steel "wave" liners. Tests are being conducted to test a mild steel rubber-covered type of liner from which so far very promising results are being obtained.

Details of mills

Speed : 22 r.p.m., i.e. 50 per cent critical speed.
Grinding load: $2\frac{1}{2}$ in. steel balls, 4,000 lb.
Moisture in mill: 25 per cent.
Feed rate: 2·5 tons per hour by Hardinge belt feeders.
Overall reduction: approximately 38 per cent.

Each mill is fitted at the discharge trunnion with a desliming trommel, aperture 1·91 mm, and wash sprays.

Washed concentrate from each mill discharges into a separate cleaning trommel. Trommel screen cloth aperture for "fines" is 1·91 mm, whilst for "middles," "coarse" and "oversize" the aperture sizes are 4 mm, 10 mm, and 16 mm respectively, i.e. in each case 2 mm less than the smallest particle size in the mill feed. Undersize from the cleaning trommels is discarded as tailings.

The undersize from these mills is mostly obtained from hard cores of magnetite and cemented sand and sea shells which have passed through the primary mills but the main purpose of this milling operation is to effect a thorough scrubbing of the diamonds to remove any salts or other material adhering to them; clean diamond faces are absolutely essential for the final recovery process.

ELECTROSTATIC SEPARATION

All milled +1·91 mm −6 mm (fines) gravel is treated by this process which is, with reference to C.D.M. gravel, suitable only for the separation of particle sizes up to a maximum of about $\frac{1}{4}$ in. (6 mm).

The four electrostatic separators presently in use were developed specifically for this purpose and came into operation during 1954. The first pilot machine was introduced here during 1949 and from that time until the commissioning of the present machines a great deal of experimental work and modification has been carried out.[3]

Efficient operation of these machines requires that the gravel feed be hot, completely dry and as dust-free as possible. Strip heaters fitted under the feeders, the heat from the gravel itself and fans for circulating the air, maintain the relative humidity in the machines at between 40 and 50 per cent.

Gravel discharged from the fines attrition mill is elevated to a surge bin from which it is fed at a constant rate of about 4,000 lb per hour through a rotating cylindrical crude oil-fired dryer. Air temperature within the dryer is 200°C. Dryer discharge is elevated to the feed bins of the four electrostatic separators from which it is fed directly over small vibrating de-dusting screens. The temperature of gravel entering the machines is 95°C and the emerging tailings have a temperature of 70°C.

The separators have six stages, each of which is 4 ft. wide. The feed to each machine is maintained at a maximum of 1,000 lb per hour and is evenly distributed over the feed rolls by electrical vibrating feeders (Figs. 20, 20a and 21).

Separation of gangue material from diamonds within the machines is effected by "spraying" the hot, dry gravel feed with a high-tension charge of d.c. electricity maintained at 23 to 25 kV. The action of the separation is dependent upon the fact that diamonds are extremely poor conductors, whilst the gravel particles are relatively good conductors.

At each stage in the separator the gravel is fed in a single layer stream over the entire width of the earthed rotating feed roll. Ahead of, and somewhat above, this roll are two electrodes in positions "x" and "y" as shown in Fig. 20a. Electrode "x" is a fixed bar. Electrode "y" is a circular roll which can be turned against fixed brushes, when cleaning the machine, to remove accumulated dust. Electrodes "x" and "y" are both equally and positively charged.

The diamonds receive a strong positive charge from the pinpoint electrode. They are therefore repelled from both electrodes and are at the same time attracted to the earthed roll and tend to adhere to it. The good conductor gangue particles receive a similar positive charge but lose this immediately to the earthed roll thus assuming an earth potential and are in consequence repelled from the negative roll and attracted towards the positive electrodes. Their trajectory is therefore deflected towards the

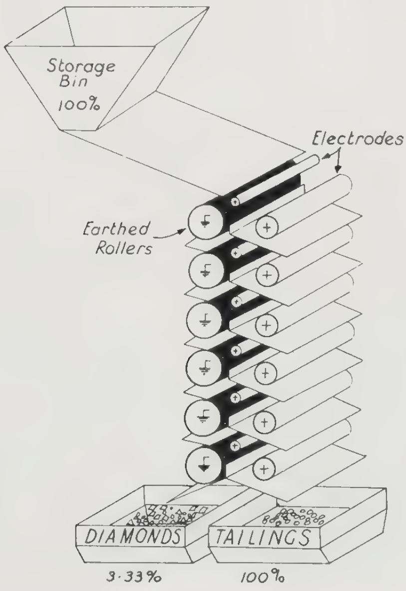

Fig. 20

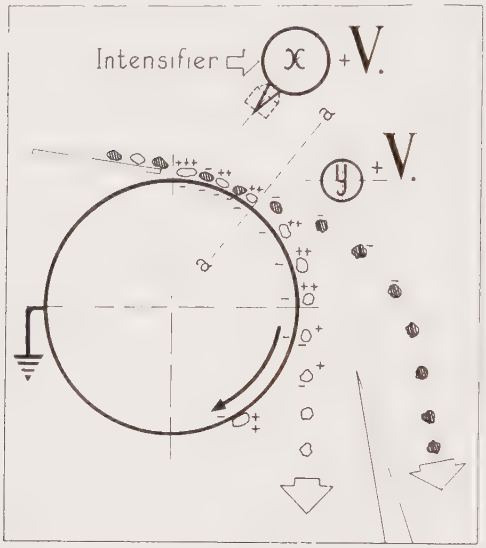

Fig. 20a

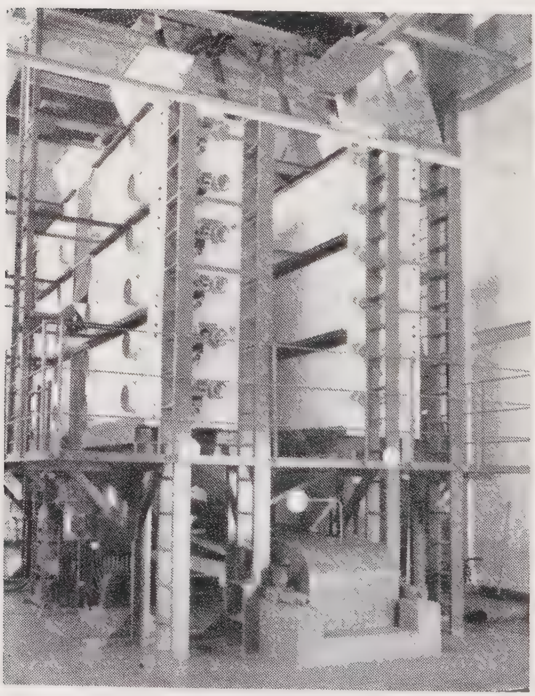

Fig. 21

charged electrodes and they fall over a divider plate on the opposite side to the path followed by the diamonds.

The reason for having two equally charged electrodes in the positions shown is that, being both positively charged, these electrodes tend to repel each other with the result that there is a strong concentration of field lines approximately along line *a—a* in Fig. 20a. In this zone the force of electrostatic deflection is at a maximum.

Concentrate from each stage falls into the feed chute of the succeeding stage from which it undergoes a similar treatment. Tailings from each stage fall directly to the tailings conveyor.

The average concentration ratio of the electrostatic separators is 30 : 1. Approximately $12\frac{1}{2}$ per cent of the tailings from each machine is tested through a four-stage, 2 ft. wide electrostatic separator. This testing system reveals an average efficiency of 99·5 per cent for the plant.

By weight 78 per cent of all diamonds recovered by the central treatment plant are recovered in the electrostatic section, the average size of these diamonds being 0·7 carat per stone.

Final concentrates from the four separators are collected in a cone-shaped bin from which they are pneumatically elevated into the sort house some 250 ft. away.

GREASE TABLE SEPARATION

Conditioning

Normally the surface of a diamond is water-resistant or non-wettable and in consequence the diamond will adhere to a greasy surface. However, many diamonds which occur in alluvial deposits will not adhere to grease due to a microscopic contamination of various salts which coat the diamond surface. This "refractory" condition of diamonds from the C.D.M. deposits is not completely removed even by scrubbing and a method of treatment has been evolved to combat this refractory state and to make the diamond surface "non-wettable." This treatment is called "conditioning."[4]

The "middle," "coarse" and "oversize" concentrates from the attrition mills are separately conveyed by locally designed shaker conveyors to storage bins from which they are fed at the rate of 1,000 lb to 1,500 lb per hour (depending upon size) by small vibrating feeders to the conditioning barrels.

These "conditioners" are rotating tubular barrels, 24 in. in diameter, fitted internally with a helix, the pitch of which is designed to pass any particle through the conditioner in $2\frac{1}{2}$ minutes.

A conditioning agent, which is a mixture of caustic soda and fish acid oil is fed into the conditioner with the concentrates. During this process the diamonds acquire a molecular coating of sodium oleate. From the conditioner the concentrates are submitted to a strong wash with fresh water over a vibrating screen. Surplus conditioning agent is thus washed off but the diamonds retain the sodium oleate coating whilst the gravels are washed clean. This molecular coating makes the diamonds non-wettable and they readily adhere to grease.

Grease tables

The C.D.M. grease tables each consist of two revolving 20 in. wide belts in series. Their speed is 14 ft. per minute. A $\frac{3}{8}$ in. thick layer of grease is constantly and smoothly spread on to the belts by a grease applicator and a thin layer is removed by electrically heated knife edges at the discharge ends of the belts. Thus a clean layer of grease is always presented at the concentrate feed point (Figs. 22 and 23).

The physical properties of the type of grease used on the belts is important. The main requirements are a reasonably high drop point (approximately 70°C) coupled with a good penetration value. The surface consistency of the grease must remain fairly constant over the normal range of temperature to be expected under working conditions and it should not easily mix or become emulsified with the water flowing over the table.

These properties have been satisfactorily embodied in a grease specially manufactured for C.D.M. by an overseas firm.

Belts are stripped and regreased every 14 shifts and most of the grease is reclaimed and re-used. The average consumption of grease in the plant is 60 gallons per month.

Washed concentrates from the conditioners are fed on to the primary grease belt at a regular controlled rate. A strong wash of fresh water flows over the table

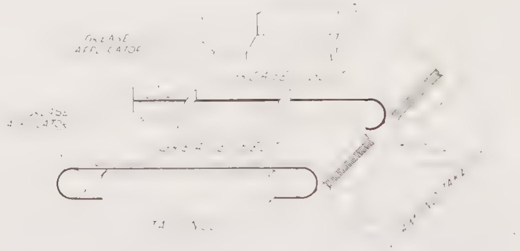

Fig. 22

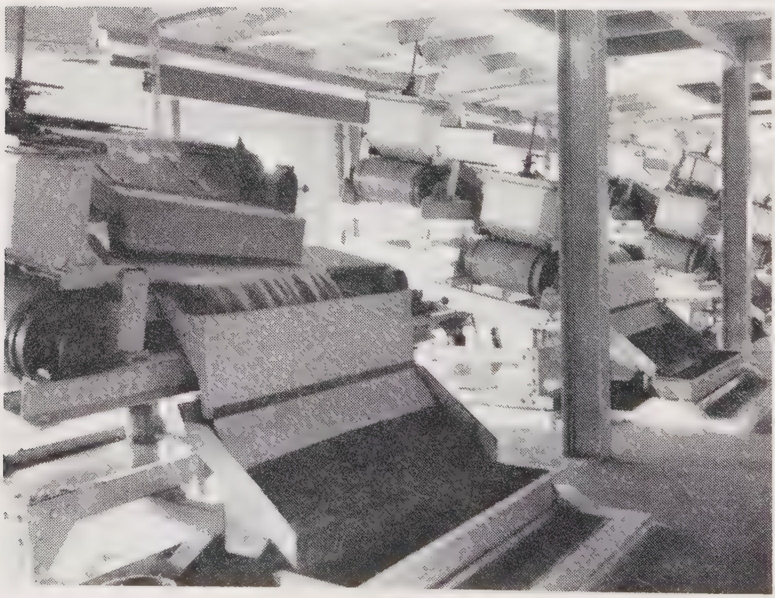

Fig. 23

and carries with it the rounded gravel particles. These roll forward with the water stream to the feed chute of the secondary belt over which they are washed to a static dewatering screen and thence to the tailing conveyor. Most of the diamonds recovered adhere to the primary belt, though occasional stones, due to peculiar shape or through not hitting the grease surface squarely, roll forward and are caught on the secondary belt. The average concentration ratio of the grease belt section is 1 : 100.

Diamonds are removed from the belts by the heated knife edges and are conveyed by shaker conveyor to a steam-heated degreasing trommel. After being thoroughly cleaned they are conveyed by a small bucket elevator directly to the sort house (Fig. 24).

All grease table tailings are regularly tested through a Harz jig and the apparent average efficiency is 99·9 per cent.

HAND SORTING

Inevitably from both the electrostatic separator and the grease belt separators a small quantity of gravel is recovered together with the diamonds in the final concentrates. For this reason hand-sorting must be resorted to for the final recovery of the diamonds (Fig. 25).

In the mining of the deposits, diamonds are picked up by African labourers cleaning the bedrock gullies and potholes. The total weight of diamonds so recovered varies from day to day in all areas, but the average for 1959 is 19·5 per cent of the total production.

The remaining 80·5 per cent is all recovered by the central treatment plant and from the mine to the final sorting table no diamonds are handled by the operators at any point in the recovery cycle.

In the sorting room two European and two African sorters are employed, one of the Europeans being the chief sorter who is responsible for the organization and security within the sorting room.

All diamonds recovered from the sorting tables are immediately dropped into sealed containers and the number found daily is recorded by hand-tally counters. At the close of each shift the sealed containers are lowered in a small elevator to the diamond room below where the plant superintendent and his assistant weigh them and deposit them in the strongroom.

At the end of each week all diamonds, including those picked up in the mine, are re-weighed and then acidized for 48 hours. In "acidizing," the diamonds are placed in plastic containers and covered with strong hydrofluoric acid which removes any siliceous particles present and cleans the diamond surfaces.

Fig. 24

Fig. 25

After acidizing, the diamonds are thoroughly washed, re-weighed and finally delivered to a responsible official in the mine office for despatch to the Central Sorting Office in Kimberley.

During any one shift the sorting room staff sort an average of 570 lb of electrostatic separator concentrates and 220 lb of grease table concentrates. This represents an average of 0·5 lb of concentrate sorted for every 100 tons of deposits mined.

CONCLUSION

Throughout the treatment of deposits mined, from field screening plants to the final sorting tables, the aim is to eliminate at each point in the cycle the maximum quantity of unwanted gravel.

Less than 12 years ago screening plants were delivering up to 25 per cent of their feed as gravel to the jigging plant. Concentrates from the jigging plant constituted 17 to 20 per cent of the initial gravel feed and, though the volume mined at that time was comparatively small, the sorting tables were constantly inundated by masses of gravel to be hand-sorted.

Fortunately C.D.M. diamonds are mostly gem stones, easily recognizable and of a size easily seen. However, it was obvious that hand-sorting is the most vulnerable process in the recovery cycle and tests of the old sorting table tailings dump showed that many valuable diamonds had been missed—and that after they had already been successfully guided through all the foregoing concentration stages. During the past two years all these tailings have been re-treated.

During the years, therefore, every effort has been made to reduce the quantity of final concentrate to reach the sorting tables by adding to and modifying existing plant, and by the introduction of new and improved methods of recovery.

This search for new and improved methods still continues and it is anticipated that the time is not far distant when the sorters' job will be to pick gravel particles out of a concentration of diamonds instead of diamonds from a heap of gravel.

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DIAMOND RECOVERY PLANT, STATE ALLUVIAL DIGGINGS, ALEXANDER BAY

By **D. B. SMIT,* N. ORSMOND,† and J. E. DE LANGE STRYDOM‡**
(Visitors)

1. HISTORY

The first discovery of diamonds on the Namaqualand coast was made by W. Carstens of Port Nolloth who, having started prospecting in August, 1925, recovered a "considerable" quantity of gem stones from a gravel occurrence $6\frac{1}{2}$ miles south of Port Nolloth. At about the same time a Kimberley syndicate prospected at Hondeklip Bay and the Buffels River, but without success.

In November, 1926, the late Dr Hans Merensky received news of the Port Nolloth discovery and immediately despatched Drs Celliers and Reuning and a prospector to the area. He followed in December and the group carried out investigations of the ground immediately south of the Orange River estuary, concentrating on the oyster-bearing marine gravels, subsequently named the Oyster Line. Their first efforts yielded some 12,500 carats within a matter of six weeks. At the same time another prospecting party, consisting of Messrs Gordon, Caplan and Loubser, was also operating at Alexander Bay, but their interests were bought by Dr Merensky as soon as he realized the potentialities of the area.

The Union Government took over control of the area shortly afterwards, the discoverer was duly compensated and in 1928 the area was proclaimed a State alluvial digging.

2. RECOVERY OF DIAMONDS 1928-1956

For many years the desired output of diamonds could be maintained by means of a small complement of Europeans, equipped with pick and shovel. Gravel was rich, very little overburden had to be removed and mechanization on a large scale was unnecessary.

Mined ground was crushed, screened and classified in a screening plant into three sizes viz. plus $\frac{5}{64}$ in. minus $\frac{3}{16}$ in., plus $\frac{3}{16}$ in. minus $\frac{7}{16}$ in., and plus $\frac{7}{16}$ in. minus $\frac{3}{4}$ in., and finally transported to a small jigging plant. The gravel was fed to duplex mechanical jigs (in series) at the rate of $2\frac{1}{2}$ tons per hour per jig.

The jig plant was expanded piecemeal over the years, until 276,000 tons per year of screened gravel was processed in 1956, working on double shift.

3. FACTORS WHICH INFLUENCED THE CHANGE FROM THE JIG PLANT TO HEAVY MEDIA SEPARATION

Tailings dump. Tests carried out on the tailings from the jig plant indicated that if these gravels were reprocessed in a heavy media separation plant sufficient

* General Manager, State Alluvial Diggings.
† Assistant Manager, State Alluvial Diggings.
‡ Resident Engineer, State Alluvial Diggings.

diamonds would be recovered to pay for the erection of the plant and still show a profit.

Recovery. The tests also proved that the old recovery plant was only about 83 per cent efficient and that most of the diamonds not recovered were the off-colour and the smaller ones, i.e. below two carats. The average size of diamonds becomes smaller as one moves south from the Orange River and as most of the future mining would have to be done in this area, it was apparent that a more efficient means of recovery was essential if undue diamond losses were to be avoided.

Labour costs. Although a heavy media separation plant must be operated by skilled labour, the labour costs would still be below that of the large unskilled labour force required to operate a jig plant of the same capacity.

Taking all these factors into consideration, it was finally decided to erect a heavy media separation plant and work was put in hand for the erection of such a plant with a capacity of 70 tons of gravel per hour.

4. DECIDING FACTORS WHICH LED TO THE ADDITION OF A SCRUBBING AND A RECOVERY SECTION TO THE HEAVY MEDIA SEPARATION SECTION

As the market for gem diamonds continued to show improvement and world production was increasing, it was considered sound policy to step-up production at Alexander Bay but, with decreasing ore reserve values, the required increased production could only be achieved by increasing the tons handled.

Scrubbing section. Tests carried out in a pilot plant in 1955 showed that if scrubbing was applied, the volume of gravel from the screening plants could be further reduced by almost 50 per cent, thus enabling the H.M.S. plant to cope with the required increased tonnage.

Recovery section. No recovery section was originally contemplated because it was thought at the time that the concentrates from the heavy media separation cone could be hand-sorted. However, the higher production target would necessitate the hand-sorting of approximately 48 tons of concentrates daily and, as this was well-nigh impossible, assistance was sought from the Diamond Research Laboratory, Johannesburg, who carried out various experiments on concentrates and concluded "that the modified grease and electrostatic process can be applied to recover the diamonds from heavy media separation concentrates at the State alluvial diggings. All gravel larger than 4 mm. can be treated by the grease-belt process and smaller sizes by electrostatic separation."

During these tests it was also proved that most of the small stones, which normally slipped by during hand-sorting, could be recovered.

It was therefore finally decided after consideration of the various factors to erect a complete treatment plant on the lines which tests had indicated to be the most suitable for conditions at Alexander Bay. The plant was completed and commissioned during August, 1957. (See Figs. 1. & 2.)

5. DESCRIPTION OF THE PLANT

SITE

The following factors influenced the decision to erect the plant about 500 ft. from the sea shore just south of the Orange River lagoon:—

(*a*) About 1·2 million gallons of water would be required per day for primary washing and scrubbing of the gravel and this could be conveniently obtained by pumping directly from a suitable size filtration dam at the nearby lagoon.

(*b*) About 1·7 million gallons per day of scrubber and differential grinding mill tailings at an average specific gravity of about 1·18 and having approximately 15 per cent of minus $\frac{3}{64}$ in. solids in suspension could be pumped directly into the sea and thus the additional costs of a slimes dam would be eliminated.

Fig. 1

Fig. 2

(*c*) By dumping the barren float product (H.M.S. tailings) at the rate of about 53 tons per hour directly into the sea, the additional expense of an ever-increasing tailings dump would be eliminated as it was expected that the front of the dump would be continuously taken away by wave action of the sea, after it had reached a certain size.

(*d*) Transport costs would be reduced since the gravel from the screening plants would be transported down grade.

PLANT

The complete plant consists of three major sections, namely:—

Storage and scrubbing section

Heavy media separation section

Diamond recovery section

Screening plants for the removal of sand and barren boulders from the mined gravels are situated in the mining areas and do not form part of the recovery plant.

Storage and scrubbing section. (See Fig. 3.) The plus $\frac{5}{64}$ in. minus 1 in. gravel from the screening plants is tipped into a 6,000 tons storage bin from where it is constantly fed into two 7 ft. by 16 ft. Fraser and Chalmers scrubbers at the rate of about 112 tons per hour, the total operation time being 24 hours per day in a five-day week. Four to six in. diameter boulders, recovered from the mining area, are used as a grinding medium and by the addition of sea water the operating viscosity is maintained at about 52 per cent solids.

The scrubbers discharge onto two 5 ft. by 14 ft. double-deck F-type Symons vibrating screens equipped with sea water sprays having a flow rate of about 300 gallons per minute. The minus $\frac{5}{64}$ in. product washed out on the screens has a specific gravity of 1·18 and is pumped to sea at the rate of about 700 g.p.m. by means of two 10 in. by 8 in. Denver D.R.L. pumps (one stand-by). The remaining 60 tons per hour scrubbed gravel is then conveyed to the 400 ton surge bin, which is part of the heavy media separation section.

Heavy media separation section. (See Fig. 4.). This section consists of the 400 ton surge bin, H.M.S. cone and the re-crush section.

At the 400 ton surge bin the gravel from the scrubbing section is again washed on two 5 ft. by 12 ft. double-deck F-type Symons screens equipped with fresh water sprays in order to remove the chlorides and any remaining minus $\frac{5}{64}$ in. product, average spray flow being about 220 g.p.m. The minus $\frac{5}{64}$ in. product which has a s.g. of 1·03 is pumped to sea at the rate of about 300 g.p.m. by means of two 5 in. by 5 in. Denver D.R.L. pumps (one stand-by). This quantity includes the minus 10 mesh tailings from the H.M.S. cleaning circuit.

Approximately 66 tons per hour of clean gravel, including the recirculation gravel from the re-crush section, is then taken over a continuous weigher to the 9 ft. diameter heavy media separation cone, which has a maximum capacity of 70 tons per hour. All tramp iron is removed before the gravel enters the cone. The cone is operated at a s.g. of 2·9, with the s.g. differential between the cone overflow and underflow media being maintained at about 0·02 to 0·04. 65D ferrosilicon is used as the medium and the cone density is controlled by means of a 36 in. Wemco dewatering type, densifier.

The float product from the cone flows onto two 4 ft. by 10 ft., double-deck, Allis Chalmers low-head vibrating screens, arranged in series, where all ferro-silicon is washed from the gravel by means of fresh water sprays, fitted to the washing screen only, spray flow being about 250 g.p.m.

The plus 10 mesh minus $\frac{3}{4}$ in. product, being tailings, is dumped into the sea and the plus $\frac{3}{4}$ in. product is taken to the re-crush section where it is crushed to minus $\frac{3}{8}$ in. in a 3 ft. Symons cone crusher, and then conveyed back to the 400 ton surge bin for recirculation.

The sink product from the cone is similarly washed on two 2 ft. by 10 ft. single-deck Allis Chalmers low-head vibrating screens to remove all ferro-silicon, spray flow being about 50 g.p.m.

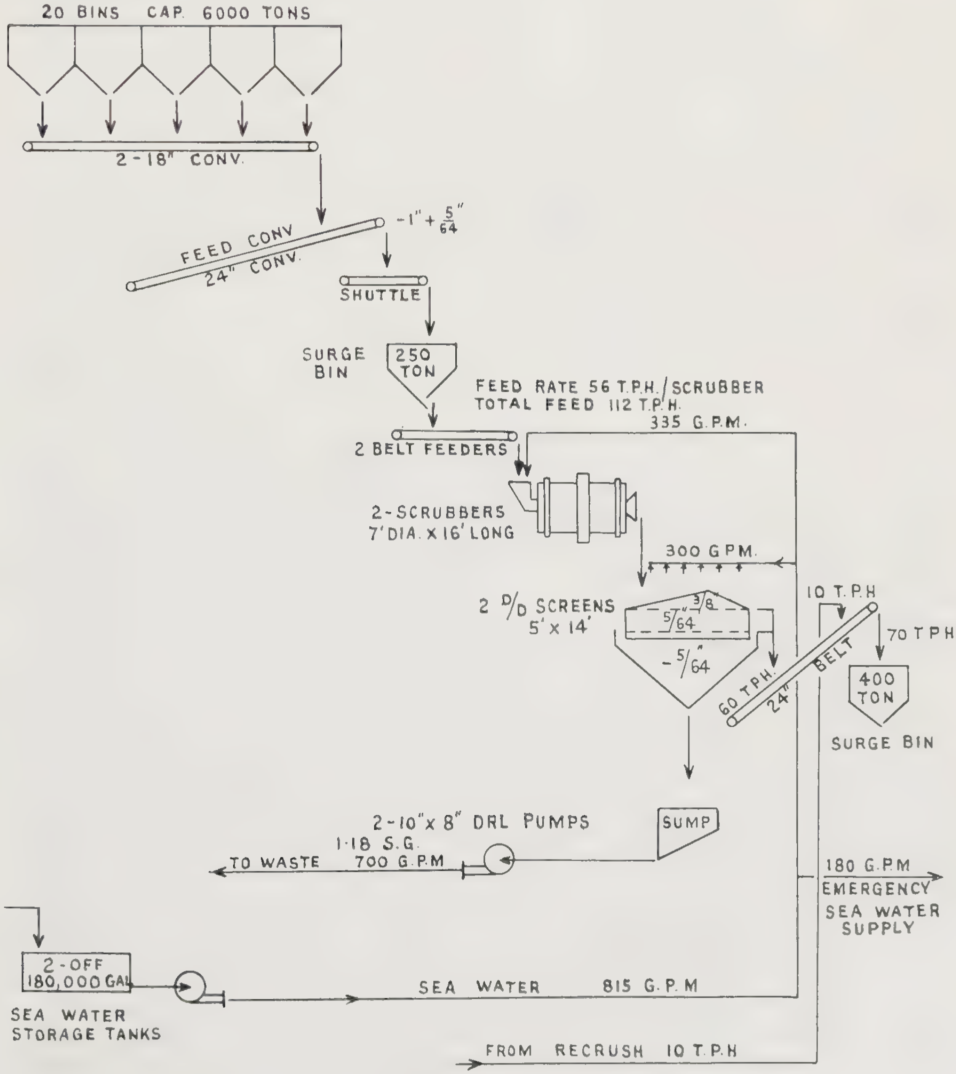

Fig. 3—Scrubbing section

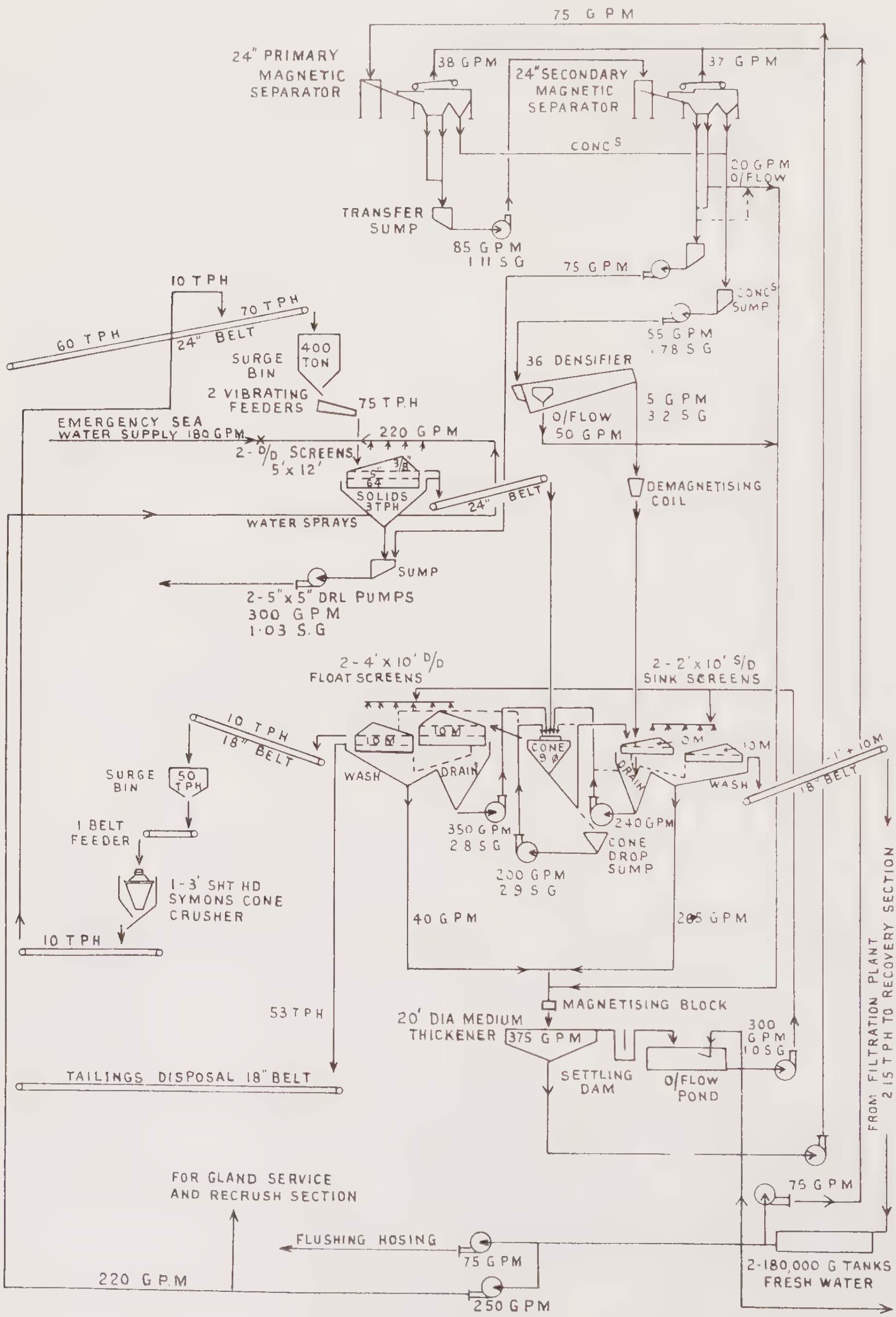

Fig. 4—H.M.S. section

The clean plus 10 mesh H.M.S. concentrate which has been reduced to between 2 and 2·5 tons per hour is then conveyed to two 60 ton storage bins in the recovery section for final treatment.

Part of the minus 10 mesh product from both sink and float drain screens is pumped back into cone circulation, the float medium return pump being a 4 in. rubber-lined K-type Wilfley and the sink medium return pump being a 4 in. rubber-lined Vacseal. The remaining minus 10 mesh product from the drain screens is diverted to the sink and float minus 10 mesh washing screen product which flows through a magnetizing block into a 20 ft. diameter Dorr thickener. The solids are flocculated and pumped over two 24 in. Wemco magnetic separators, arranged in series, where the ferrosilicon is reclaimed and the tailings gravitated to a sump and pumped to the 400 ton surge bin for disposal. A settling dam of 1,200 cu.ft. capacity which is cleaned out once a week is used as a secondary means for ferrosilicon recovery.

All pumps and pipelines handling ferrosilicon media are rubber-lined and only fresh water is used throughout the H.M.S. section, the total quantity required in the H.M.S. section being about 290 g.p.m.

Diamond recovery section. (See Fig. 5). In order to achieve maximum efficiency two independent processes for diamond recovery are used, namely (*a*) electrostatic separation for treating plus 9 mesh minus $\frac{5}{32}$ in. concentrates, and (*b*) grease belt concentration for treating plus $\frac{5}{32}$ in. minus 1 in. concentrates.

Approximately six to seven tons per hour of H.M.S. concentrates from the two 60 ton storage bins is fed at a constant rate into two 6 ft. diameter by 4 ft. Hardinge conical ball mills. $1\frac{1}{2}$ in. and 2 in. diameter steel balls are used as a grinding medium and the operating viscosity of the pulp is maintained at 80 per cent solids by the addition of fresh water.

The shell and shale carried over with the sink product are ground away and washed out of the mill discharge, together with the minus $\frac{5}{32}$ in. product, on two 3 ft. by 8 ft. single-deck, Symons F-type screens, having a spray flow of about 90 g.p.m. It is these screens that separate the concentrates into two portions—one for electrostatic separation and one for grease belt concentration.

(*a*) Electrostatic separation. The minus $\frac{5}{32}$ in. product is pumped to two 3 ft. by 8 ft. Symons F-type single-deck screens where the minus 9 mesh tailings are washed out, spray flow being about 50 g.p.m. The tailings gravitate to a common tailings sump from where they are pumped into the sea at the rate of about 410 g.p.m. by two 5 in. by 5 in. Denver D.R.L. pumps, pulp density being 1·01.

The plus 9 mesh minus $\frac{5}{32}$ in. product is discharged into a 4-ton storage bin, which allows for two days storage. Approximately 500 lb per hour is fed into a 30 in. diameter by 16 ft. differential grinding mill for final cleaning of the concentrates and elimination of any residue shell and shale. The pulp viscosity is maintained at 80 per cent solids and $1\frac{1}{2}$ in. diameter steel balls are used as a grinding medium. The minus 9 mesh product is washed out on a 2 ft. by 5 ft. single-deck Aerovibe vibrating screen equipped with sprays having a flow of about 20 g.p.m., and is pumped to the common tailings sump via the 3 ft. by 8 ft. Symons screens mentioned above. The plus 9 mesh minus $\frac{5}{32}$ in. product which has been reduced to about 250 lb per hour is accumulated in a 4-ton capacity storage bin which allows for about four days' storage.

When it is required to operate the electrostatic separator the concentrate is passed through a 1,000 lb per hour capacity Ruggles Coles rotary drier where the moisture is removed and the temperature of the concentrate raised to above 100 C. The concentrate is then passed over a 2 ft. by 5 ft. Aerovibe double-deck dedusting

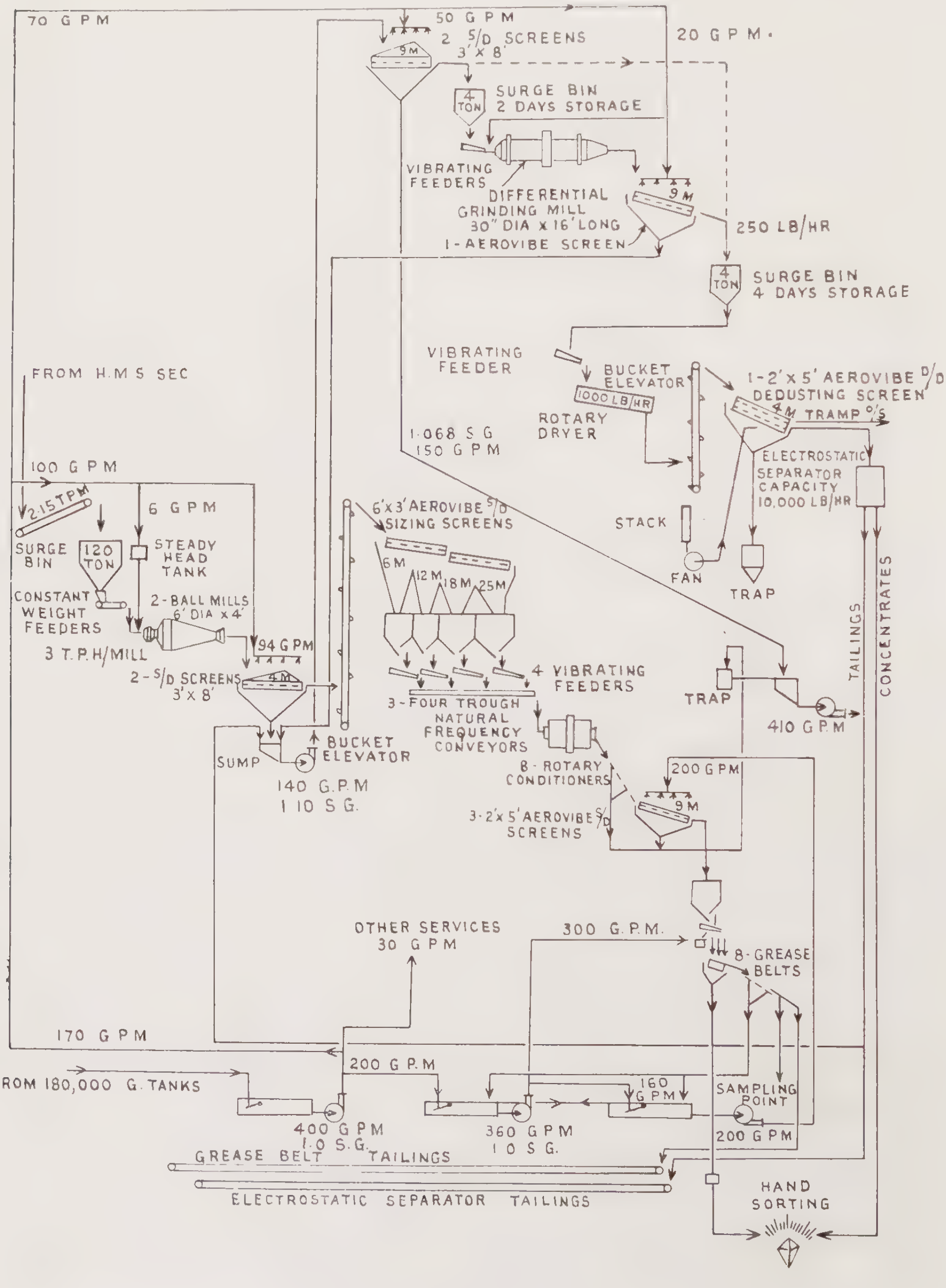

Fig. 5—Recovery section

screen, where all dust is drawn off by means of an induction fan. Any tramp oversize gravel which may disturb the even flow of the concentrate into the separator is also removed and stored in a small bin.

Approximately 800 lb per hour of minus $\frac{5}{32}$ in. concentrate is fed at a constant rate into the 4 ft. by six-stage electrostatic separator by means of vibrating feeders to ensure an even flow. Electrode potential is maintained at about 20 kV static.

An average concentration ratio of about 80 to 1 is obtained. The diamonds are handsorted from the separator concentrates.

(*b*) Grease belt concentration. The plus $\frac{5}{32}$ in. concentrate from the Hardinge mill discharge washing screens is passed over two 3 ft. by 6 ft. single-deck Aerovibe sizing screens for classification into first, second, third grade and oversize concentrate and then accumulated in four 10-ton storage bins.

APPROXIMATE SIZE OF CLASSIFIED GRAVEL

1st Grade	$+\frac{5}{32}$ in $-\frac{1}{4}$ in
2nd Grade	$+\frac{1}{4}$ in $-\frac{1}{2}$ in
3rd Grade	$+\frac{1}{2}$ in $-\frac{3}{4}$ in
Oversize	$+\frac{3}{4}$ in -1 in

As the diamonds are coated with a layer of mineral salts which would become wetted during the normal grease belt recovery process, pre-treatment with a suitable reagent or "conditioner" is necessary before the gravel is passed over the grease belts.

From the 10-ton storage bins the classified concentrate is fed into eight 30 in. diameter by 66 in. rotary conditioners at a constant rate, by means of vibrating feeders and natural frequency conveyors.

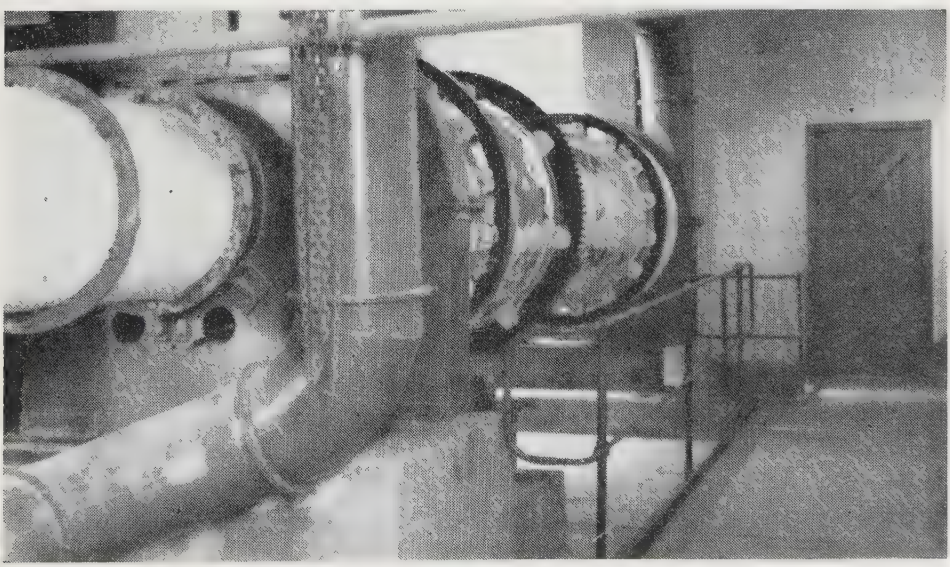

Fig. 6

FEED RATES OF CONDITIONERS AND GREASE BELT

Gravel size	Feed rate	No. of conditioners and grease belts
1st Grade	350 lb/hr	1
2nd Grade	800 lb/hr	4
3rd Grade	1000 lb/hr	2
Oversize	1000 lb/hr	1

The concentrate is brought into contact with the reagent "Aero promotor 708" in caustic soda and is retained for about two minutes in the conditioners.

The conditioned concentrate is then washed on eight 2 ft. by 5 ft. single-deck Aerovibe screens having a spray flow of about 25 g.p.m. per screen, from where the minus 9 mesh product is gravitated through a trap to the common tailings sump for disposal.

The classified plus $\frac{5}{32}$ in. concentrate is passed over eight 20 in. wide grease belts for final concentration. Flow from the water-box is approximately 35 to 40 g.p.m. per belt. (See Fig. 7)

Depending on climatic conditions, the grease used on the grease belts consists of 10 to 20 per cent wax mixed with 90 to 80 per cent petrolatum amber.

AVERAGE CONCENTRATION RATIOS OBTAINED

Size	Concentration ratio
1st Grade	20 to 1
2nd Grade	40 to 1
3rd Grade	50 to 1
Oversize	50 to 1

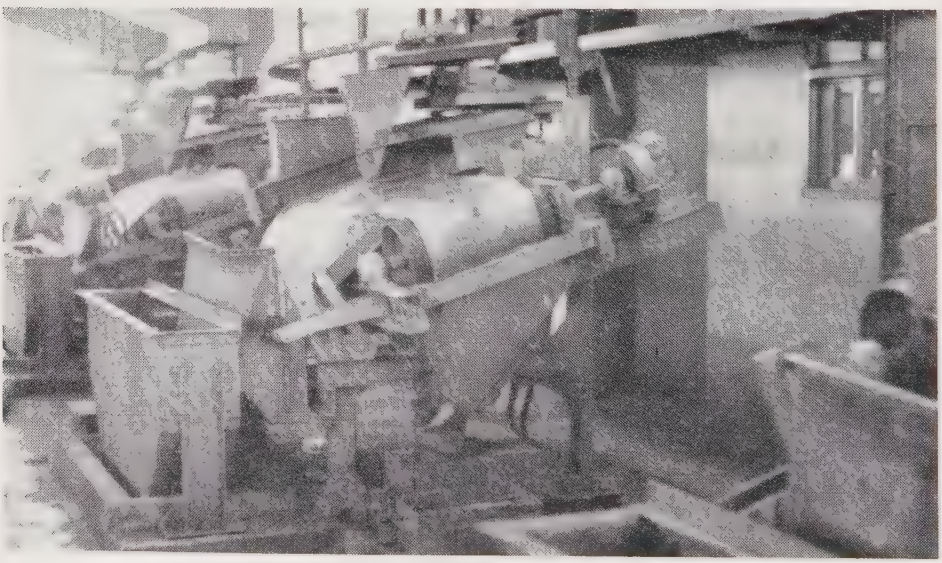

Fig. 7

The diamonds are handsorted from the grease belt concentrates after de-greasing by boiling, and the tailings dumped. The grease belt tailings are constantly checked for diamond losses.

Total fresh water consumption in the recovery section is about 400 g.p.m.

6. TEETHING TROUBLES

(*a*) As is usually the case with new plants, initial trouble was experienced with many items of machinery. Not only the plant and equipment, but also the methods and processes, were new to most of the men charged with the duty of operating this plant; but the precaution had been taken to have certain key operating personnel trained at the plant of the Consolidated Diamond Mines of S.W.A. Limited, by kind permission and co-operation of the management of that company. With these trained men in charge, the rest of the selected staff soon adapted themselves to the new operating methods, so that in a reasonably short period one of the greatest obstacles had been overcome.

(*b*) It soon became evident that a major source of trouble would be corrosion, especially at sections where sea water was used. This was prominently brought to notice when the combined factors of corrosion and abrasion played havoc with the "vibro" carbon steel screen mats, initially used on all the washing screens. The screens particularly affected were the Symons F-type screens at the scrubbing section and the 400 ton bin and the Allis Chalmers low-head screens at the H.M.S. section. Some mats did not even last one shift before failures occurred.

In one case this condition was aggravated by the fact that no means of tensioning the screen mats had been provided, with the result that the screen cloth could oscillate separately from the screen frame, causing fatigue of the carbon steel screen wire.

The trouble was solved by changing over to suitable stainless steel screening cloth and having proper tensioning decks provided to the screen sides to ensure even tension distribution along the entire screen length. To assist in the detection of ruptured screen mats, clearly visible tell-tale screens were installed at the bottom of the screen hoppers, leading to drains or tailing sumps.

(*c*) Another source of trouble was the constant breakage of the supporting coil and leaf springs of some of the screens. This may have been partly due to corrosion fatigue which had set in as a result of exposure to the corrosive air while being stored in the open prior to installation. The makers recommended replacement of the original coil springs with a lighter type of spring and this proved the answer to the problem. On the recommendation of the manufacturers, the supporting leaf springs have since been replaced with "Permali" support strips, made of a resinous bonded material, and this has improved matters greatly.

(*d*) The screening efficiency of the 9 mesh washing screens became seriously affected as a result of blinding of the screen openings. The anti-blinding devices, provided in the form of ball decks, were improved on the lines found to be the most suitable for local conditions. The best results were obtained by using 3 in. diameter hard-core rubber balls having a hardness number of about 50 and allowing a clearance of $\frac{3}{4}$ in. between the balls and the screen cloth. Ball distribution per unit area is about 2 per sq.ft.

(*e*) Excessive vibration caused by the natural frequency conveyors used for distributing and feeding the classified concentrates to the rotary conditioners was transmitted to the building structure through the reinforced concrete floor on which the conveyors were installed. Apart from inconcenience, the intensity of the vibrations caused slight damage to the wall renderings of the building. Although various attempts were made to solve this problem -by stiffening the building steelwork, adjusting the natural frequency by means of adjustable pulleys and mounting the conveyors on anti-vibration mountings—no solution could be found.

Eventually with the assistance of the South African Council for Scientific and Industrial Research the vibrations in the building were considerably reduced by section balancing of the different moving masses of the conveyors.

(*f*) When the full plant was commissioned it was found that the fresh water consumption was about 40 per cent higher than originally estimated. It was not possible to draw the extra quantity required from the caissons in the Orange River which had served up to then as the only means of supplying fresh water for the mine as well as for domestic requirements. On account of the high percentage solids in suspension, raw water from the river could not be used and a flocculation dam with a filtration plant had to be built to meet the extra demand.

(*g*) As a result of premature failures of steel pipes at the H.M.S. section due to the extremely abrasive nature of the ferrosilicon media, all pipes handling media were replaced by rubber-lined pipes.

(*h*) On one occasion when the plant was being started up after a temporary stoppage, a large quantity of ferrosilicon was lost in the H.M.S. tailings, due to washing screen blinding. The plant had to be closed down again on account of water trouble and prior to re-starting five tons of 65D ferrosilicon was added. It was found, however, that cone density could not be maintained; the specific gravity differential was so great that the sink media density was critically high and the sink product was discharged on the sink wash and drain screens almost in a solid mass. The density at the top remained about 2·7 to 2·8. All attempts to overcome the difficulties were abortive, due to lack of experience and eventually the help of technicians of the suppliers of the plant had to be called in. On their advice fine slime was added to increase the viscosity of the media and this solved the difficulty.

7. EFFICIENCY

(*a*) A factor closely related to diamond losses from grease belt recovery is the effectiveness of conditioning. Water hardness and fatty acid content of reagent are important factors controlling effective conditioning. It was found that the conditioning of gravels was adversely affected by the excessively high chloride content of the water, and to improve this a water-softening process was installed.

(*b*) The temperature gradient across the electrostatic gravel dedusting screen caused the gravel temperature to drop to below 60°C and this affected the efficiency of separation. To improve this, the gravel entering the electrostatic separator is given a second heating to a temperature of above 100°C by means of the waste gases from the rotary drier stack (about 230°C) which are passed through manifolds of a heated feed hopper.

(*c*) The $\frac{5}{64}$ in. screening cloth on the vibrating screens in the scrubbing and H.M.S. sections is subjected to the most wear. The following table gives the average life obtained from the various stainless steel screen mats:—

Screen	Duty	Section	Approximate life
5 × 14 ft Symons	Washing screen	Scrubbing	850 hours
5 × 12 ft Symons	Washing screen	400 ton bin	400 hours
4 × 10 ft Allis Chalmers	Float drain	H.M.S.	A.650 hours B.863 hours
4 × 10 ft Allis Chalmers	Float wash	H.M.S.	A.210 hours B.320 hours
2 × 10 ft Allis Chalmers	Sink drain	H.M.S.	A.668 hours B.1140 hours
2 × 10 ft Allis Chalmers	Sink wash	H.M.S.	925 hours

NOTE: Mats A are at receiving end of screens.
Mats B are at discharge end of screens.

(*d*) Daily laboratory tests revealed that the average ferrosilicon loss from H.M.S. media is limited to 0·38 lb per ton gravel treated in the H.M.S. cone.

(*e*) *Diamond losses.* Experience has proved that any diamond losses which do occur are most likely at the electrostatic and first grade grease belt tailings. As a result, all electrostatic and first grade grease belt tailings are continuously hand-sorted. About 20 per cent of second grade grease belt tailings are also continuously hand-sorted and periodical tests are carried out on third grade and oversize tailings

Average monthly diamond loss is as shown in the following table:—

Concentrates	Tons treated per month	Percentage diamond loss
Electrostatic	15 tons	0·52
Grease belts, 1st grade	23 tons	} 0·21
Grease belts, 2nd grade	203 tons	
Grease belts, 3rd grade	165 tons	0
Grease belts, oversize	8 tons	0
Total	—	0·31

8. CONCLUSION

The plant has fulfilled the greatest expectations and will have a beneficial effect on the life of the mine. It has made possible the attainment of the object to increase the tonnage treated at a time when the average ore reserve values are decreasing and has enabled a commencement to be made on the recovery of diamonds from the considerable tonnage of old tailings.

In conclusion, it is desired in particular to place on record the kind assistance received from the Diamond Research Laboratory, Messrs. Fraser and Chalmers (S.A.) (Pty.), Ltd., the Consolidated Diamond Mines of S.W.A. Limited, and the Anglo American Corporation of S.A. Limited.

This paper is presented with the kind consent of the Union Department of Mines.

BLOCK CAVING PRACTICE AT DE BEERS CONSOLIDATED MINES, LIMITED

By W. S. GALLAGHER* and W. K. B. LOFTUS† (Visitors)

1. INTRODUCTION

The producing mines directly controlled by De Beers Consolidated Mines, Limited, are the Wesselton, Dutoitspan and Bultfontein Mines in the Kimberley area and the Jagersfontein Mine, situated 100 miles from Kimberley in the Orange Free State. The Kimberley Mine, the world-famous "Big Hole," was closed in 1914, having been mined down to its economic limit. Both the De Beers and Koffiefontein Mines are dormant.

The Premier Mine near Pretoria, the Williamson Mine in Tanganyika, and the alluvial deposits of South West Africa and Namaqualand are operated by associated companies and do not fall within the scope of this paper.

2. GEOLOGY

The Kimberlite, or blue ground, occurs in the form of volcanic pipes which cut almost vertically through the surrounding country rock. It is typically an ultra-basic rock, now largely altered to serpentine, with numerous inclusions of both sedimentary and igneous origin.

On the surface the pipes are circular to oval, but they tend to assume a more elongated shape at depth. The largest pipe in the Kimberley group is Dutoitspan, which had a surface outcrop area of 1·385 million square feet. The area of the Kimberley pipe was less than a third of this figure.

A characteristic of the pipes is that they tend to decrease in both grade and size with depth. Bultfontein had a surface area of 1·025 million square feet, while on the 1,900 ft. level this has decreased to 314,000 square feet. At this elevation the pipe is roughly elliptical with the lengths of the major and minor axes 820 ft. and 510 ft. respectively.

The oldest geological formation encountered in the country rock surrounding the pipe is the Basement Complex, comprising granite gneiss and chlorite schist. The Ventersdorp System, composed of amygdaloidal lavas (known locally as melaphyre) and quartzites, lies unconformably on the granite gneiss and schist. These rocks are, in turn, overlain unconformably by the Karroo System of Dwyka Tillite and Upper Dwyka carbonaceous shales, with a sill of dolerite intruded into the latter. Erosion has left a present-day surface of dolerite in the vicinity of Kimberley.

3. GENERAL LAYOUT

Each mine is served by a rectangular, five-compartment, vertical shaft situated in the country rock about 1,000 ft. from the pipe perimeter. This is the main hoisting

* General Manager, De Beers Consolidated Mines Ltd.
† Assistant General Manager, De Beers Consolidated Mines. Ltd.

and downcast shaft. An upcast shaft, providing the second outlet, is situated at approximately the same distance from the pipe.

The main haulage crosscuts are driven from the shaft to the pipe at 600 ft. vertical intervals. A three-compartment, sub-vertical shaft, situated close to the pipe, provides access to the working levels.

The producing mines have been completely mined out to depths of up to 1,300 ft. below surface. Current extraction workings, including block-caving projects, extend to the 1,900 ft. level. Between the bottom of the open mine (up to 800 ft. below surface) and the unmined ground is a large mass of overburden comprising mainly caved country rock from the sides of the pipe.

The tons of blue ground sent to the treatment plants from each of the mines, for January, 1960, are given below:—

Dutoitspan	143,950
Wesselton	131,360
Bultfontein	74,620
Jagersfontein	144,810

The low figure for Bultfontein is due to the current conversion programme from chambering to block caving.

A notable feature of mining practice is that no water is used in drilling operations within the pipe due to its undesirable effects on blue ground, such as swelling and disintegration. This practice is permissible due to the ultra-basic composition of the kimberlite.

4. CHAMBERING

This mining method, illustrated in Figs. 1 and 2, was introduced into the diamond mines around 1890 and has been successfully employed, with little change, up to the present day.

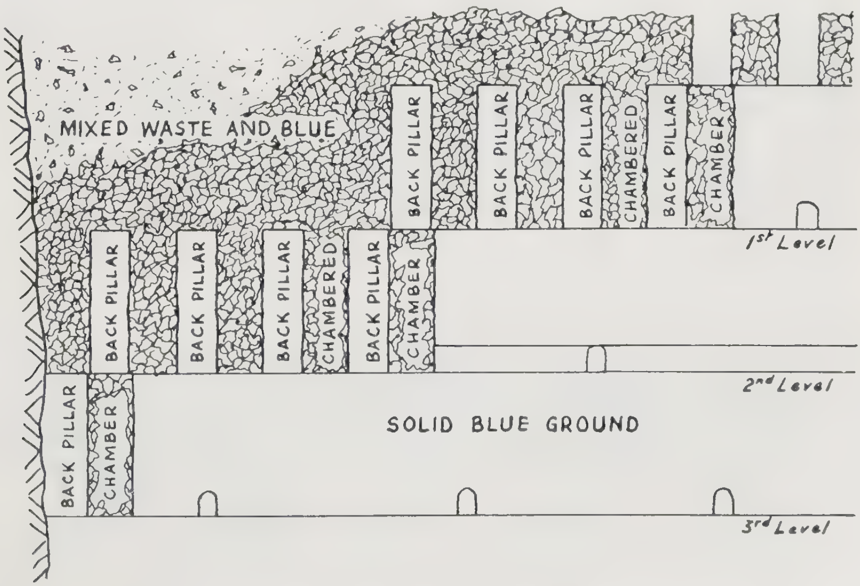

Fig. 1—Cross-section of chambering

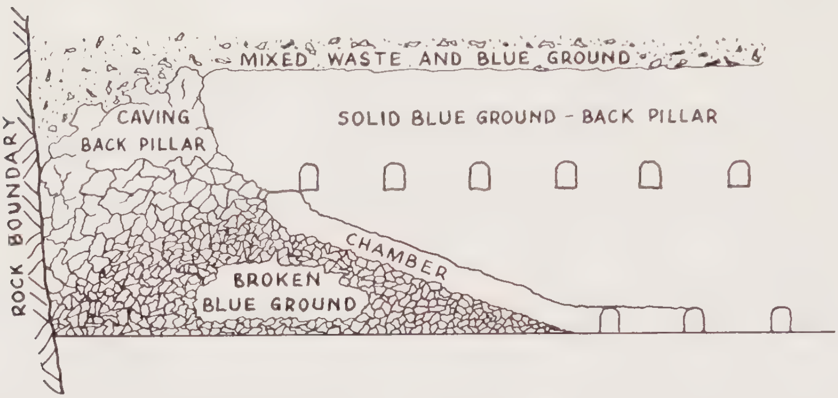

Fig. 2—Longitudinal section of chamber.

Chamber levels are established from the sub-vertical shaft at 40 ft. intervals. On each of these levels successive $22\frac{1}{2}$ ft. "cuts" are mined out across the pipe, from one rock contact to the other, and extending vertically to the footwall of the level above. Every $22\frac{1}{2}$ ft. cut is made up of a chamber and a back pillar, each approximately 11 ft. wide. The chamber is mined by overhand shrinkage stoping.

The levels are carried *en echelon*, with each chamber on a level, approximately three cuts ahead of the chamber on the level below. In addition, the chambers on alternate levels are staggered so that a chamber is always cut under a back pillar on the level above. Some six chamber levels are required to obtain a production of 4,800 tons per day.

Loading tunnels are driven at right angles to the chamber at $22\frac{1}{2}$ ft. centres. All the ground broken in the chambers and caved from the pillars is hand-loaded into cars, and trammed to ore passes in the country rock. All ground has to be broken down to -12 in. to pass through the grizzlies.

The ore passes lead to the main haulage level, where the ground is transported to the hoisting shaft by endless rope haulage or conveyors.

Efficiencies for Wesselton mine, for 1959, are:—

Tons per case of explosives	188
Tons per chamberman per shift (broken and caved)	438
Tons per loading boy per shift	20

5. EXPERIMENTAL BLOCK CAVE

Following upon a visit to a number of mines in North America where block caving was employed, it was decided to experiment with this system at one of the De Beers mines. Several of the mines visited had ground conditions similar to the diamond pipes and it was thus felt that the method might prove suitable. The great advantage of this method lies in its high labour efficiency and low operating cost; in North America efficiencies of more than 30 tons per shift per person underground have been obtained.

An easily accessible block of unmined ground, of low grade, in the north-west corner of Bultfontein Mine was selected for the experimental section. Six scraper drifts were developed on the 1,220 ft. horizon to serve the area. Results showed that block caving could be successfully applied to these mines and it was decided to introduce block caving on a large scale at the Dutoitspan, Bultfontein and Jagersfontein Mines. Due mainly to the irregular shape of the pipe at depth it has been decided to continue chambering at Wesselton.

Some remarkable efficiencies are now being obtained from the experimental 1,220 ft. block cave as the following table shows. The ground in this section is, however, softer than is usually found:—

Cost per ton mined (delivered to skips)	$12\frac{1}{2}$d.
Tons per case of explosives	400
Tons per European shift (directly employed on extraction) ...	1,200
Tons per native shift (directly employed on extraction)	86

6. BLOCK CAVING—GENERAL

The block caving method of mining is applicable to massive ore bodies which have suitable physical characteristics of both wall rock and ore. It has been successfully applied to the American porphyry copper deposits, the asbestos mines in Quebec, the iron ore deposits of the Lake Superior area and to the copper deposits of Northern Rhodesia.

Block caving as applied to diamond mining is illustrated in Figs. 3, 4 and 5. Scraper drifts are developed across the pipe from the haulage, situated in the country rock close to the pipe, the footwall elevation of the drifts corresponding approximately to the hanging wall elevation of the haulage. The drifts, which are concrete-lined, are at 45 ft. intervals and are situated up to 600 ft. below the existing workings. Inclined 4 ft. by 4 ft. openings, or draw points, are left in the side of the concrete lining at 11 ft. 3 in. intervals, staggered on opposite sides of the drift. From the draw points,

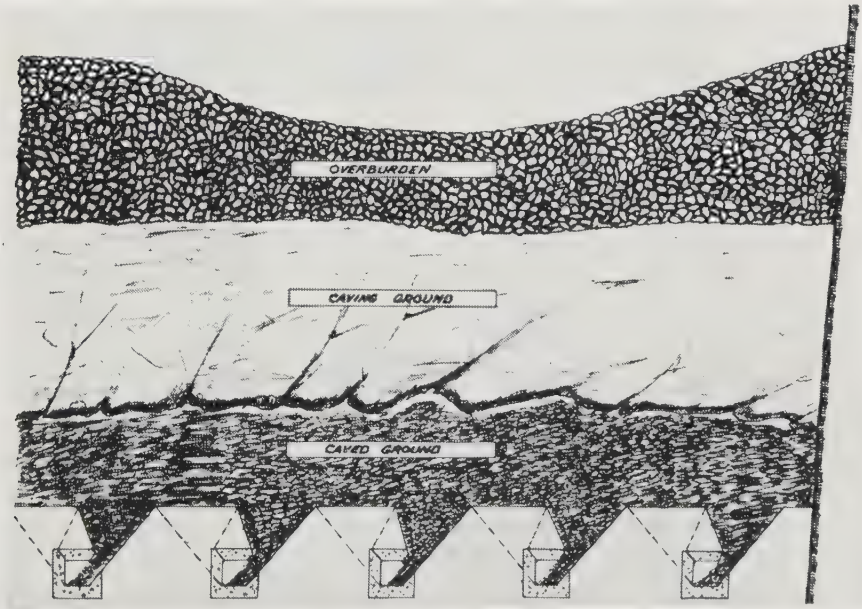

Fig. 3—Section across scraper drifts

Fig. 4—Section along scraper drift

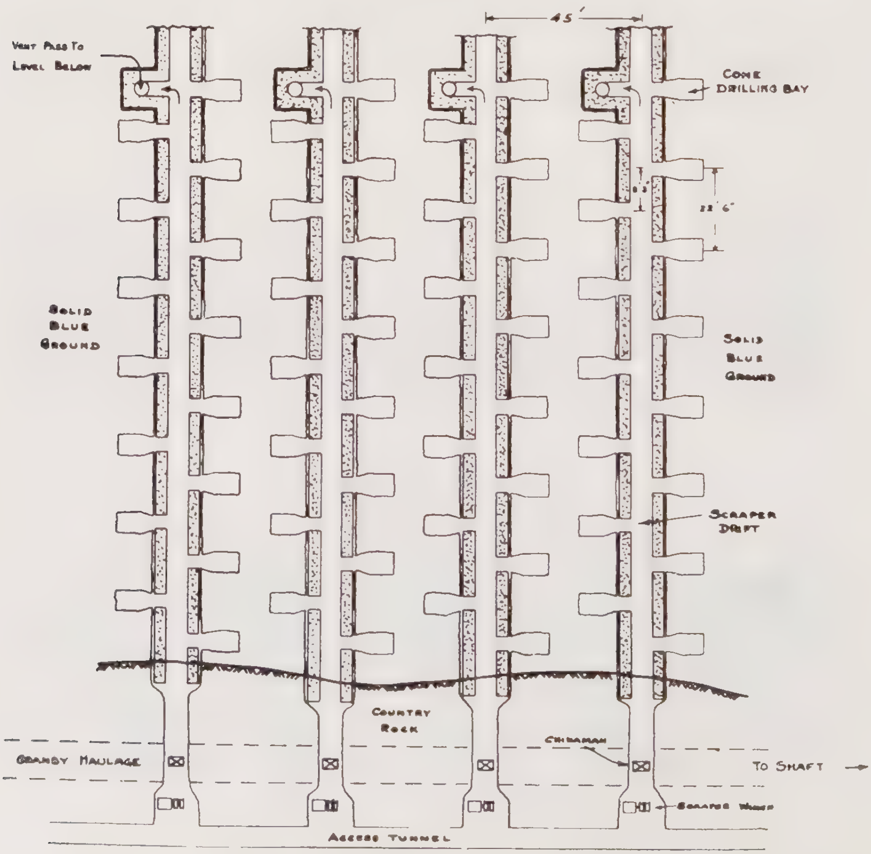

Fig. 5—Plan showing general layout of scraper drifts

raises are put up to the footwall of the undercut level. The tops of these raises are widened out to form cones which cover the footwall of the undercut level; the latter is 19 to 29 ft. above the footwall of the scraper level.

On the undercut level itself a horizontal slice, 7 ft. high, is mined out over the entire area of the pipe. The mass of unsupported ground above then starts caving, the ground being drawn off through the cones and drawpoints into the scraper drift below where it is scraped directly into Granby hoppers in the country rock haulage.

The caving and drawing-off of ore continues, only sufficient ground being drawn-off to compensate for the increased volume of the broken ore. The level of caved ore and caving ore thus gradually move up the pipe until the cave breaks through to the overburden above. At this stage some 60—70 per cent of the original ore still remains to be drawn off. Hence, drawing continues until eventually the waste overburden appears at the drawpoints.

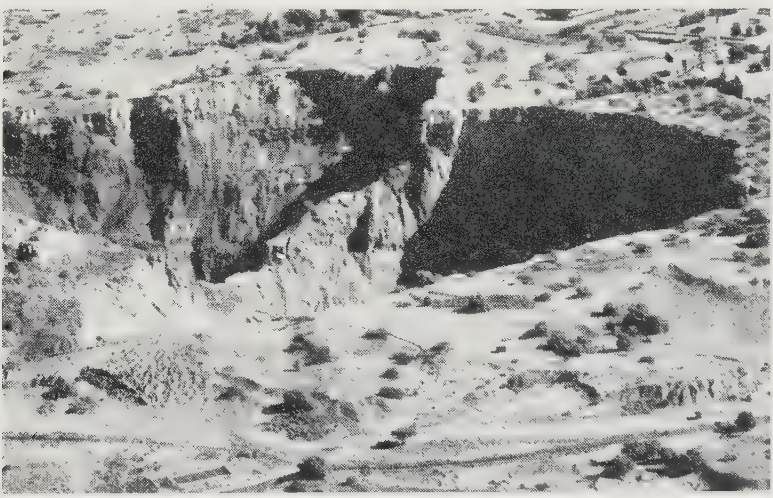

Fig. 6—View of Dutoitspan open mine. The unpayable area can be clearly seen

7. LOCATION

The location of block cave projects was largely determined by existing development which had been carried out under the chambering system.

Jagersfontein was the first mine to go fully over to block caving. The change-over from chambering to caving was started in 1955 and completed in 1958. The 1,870 ft. level was chosen as the scraper drift level, the mean vertical lift of the undercut block being just under 400 ft.

Dutoitspan Mine is divided into two sections, east and west, separated by an area of unpayable ground (Fig. 6). The west side of the mine is being block-caved from the 1,350 ft. level. Chambering continues on the eastern side but this will be superseded by block-caving, on the 1,900 ft. level, in 1962.

Chambering has ceased at Bultfontein Mine and the concreting of the scraper drifts is well advanced and scheduled for completion in July, 1960. Undercutting commenced recently and will not be completed until early in 1962.

Details of the Dutoitspan and Bultfontein projects are shown in Table I:—

TABLE I

	Dutoitspan Mine		Bultfontein Mine	
Level	1,350ft. west	1,900 ft east	1,220 ft	1,900 ft
Number of drifts	15	27	6	30
Number of drawpoints	250	550	67	610
Mean vertical lift (back)	300 ft	600 ft	600 ft	400 ft
Estimated tonnage	4 million	21 million	2 million	14½ million

8. INITIAL DEVELOPMENT

From the main loco haulage connecting the hoisting shaft to the pipe a 10½ ft. by 10½ ft. haulage is driven around the pipe in the country rock, at least 30 ft. from the contact. The haulage is equipped with 500 volt d.c. overhead trolley wire, six-ton Goodman locos (60 h.p.) and six-ton Granby hoppers running on 30 in. gauge, 60 lb track. The haulage is single track except for the necessary passing loops.

During haulage development, a short raise is put up at the position of each scraper drift, and the drift is developed to the kimberlite contact on the one side and to the winch chamber on the other. The winch chamber (11 ft. wide by 12 ft. long by 7 ft. high) is then cut, and a 4½ ft. by 7 ft. tunnel is developed behind the winch chambers for travelling from drift to drift. Chinamen and winches are installed as these excavations are completed so that they are ready for the start of drift development within the pipe.

The batching and placing chamber, and passes for cement and aggregate, are developed in the country rock.

When the above development in the country rock has been completed, development and concreting of the scraper drifts is started.

9. CONCRETING OF THE SCRAPER DRIFTS

A. Method of support

A major difficulty in the application of block-caving to blue ground is the method of support of the scraper drifts.

It was realized that conventional steel or timber support would not stand the high stresses expected nor would they prove suitable from many operational aspects. For this reason concrete lining of the drifts was decided upon. Pneumatic placing of the concrete was adopted as the most suitable and economical method of concreting.

B. Cement and aggregate handling

This section refers to Bultfontein mine, the only mine where concreting is at present being conducted. The handling methods have developed from the original method of transporting all requirements to the site underground in trucks.

(*i*) *Scope of operations.* The current programme for the conversion of Bultfontein Mine to block caving entails the concrete-lining of over 7,000 ft. of scraper drifts, plus various auxiliary concreting work, over a period of about 22 months. This will be followed by a similar programme on the 1,900 ft. level at Dutoitspan Mine.

The total estimated quantity of concrete materials required for the Bultfontein project and the maximum daily requirements are as follows:—

	Total required	Max. daily requirements
Cement	400,000 pockets	500 pockets
Stone	70,000 cubic yards	88 cubic yards
Sand	32,000 cubic yards	40 cubic yards

Fig. 7 shows the route followed by the concreting materials.

(ii) *Cement handling.* Bulk cement storage is located in an old compound adjacent to a railway siding.

From the old compound a tunnel (30 ft. below surface) leads to the main vertical shaft, and provides a convenient route for the transporting column and site for the cement transporter.

The transporter consists of an old concrete placer with modified air jets. A by-pass valve serves to keep the cement in the column in motion during filling of the transporter.

Ten pockets of cement are blown at a time, the total cycle time being seven to eight minutes. Thus the required daily quantity is obtained in one shift without difficulty. The transporting column is 3 in. in diameter and it conveys the cement 3,500 ft. to the bulk storage pass underground.

Compressed air is supplied at a pressure of 65 to 70 lb per sq.in. Average compressed air consumed per pocket is 200 cu.ft. Taking compressed air at our average generating cost of 3d. per 1,000 cu.ft., compressed air cost per pocket is thus 0·6d.

The storage pass has a capacity of more than one day's consumption. The pass is covered by a plate to the top of which are attached flannel bag filters. Twelve bags, 5 ft. high by 18 in. diameter are used, these being inflated by the delivery air. The water gauge across the flannel varies from $\frac{1}{2}$ in. to 2 in., the bags being cleaned twice per shift by manual shaking.

(iii) *Sand and rock handling.* From the surface stockpiles the rock and sand are scraped alternately onto an 18 in. conveyor delivering into a small box which feeds either the rock or sand columns, in the shaft, through a simple door.

The rock ($-1\frac{1}{2}$ in. $+\frac{3}{8}$ in.) is conveyed vertically down the shafts in 12 in. columns. To minimize pipe wear this column is kept full. Conversely, the 6 in. sand column must be kept empty, and the sand allowed to run straight through, otherwise damp sand causes blockages.

Simple dead boxes, which require little attention, are installed in the rock pipe at 300 ft. intervals to prevent undue pressure and to break the momentum.

The rock and sand are conveyed horizontally on the 1,600 ft. level by diesel loco and one-ton trucks, and by an 18 in. conveyor on the 1,790 ft. level. The conveyor discharges into a bratticed aggregate pass where it is deflected either into the rock or sand compartment. The aggregate storage pass, situated vertically above the batch plant, contains sufficient material for two days concreting.

C. Batch plant

This is situated in the country rock, convenient to the area to be served (Fig. 8).

It is equipped with two $\frac{1}{2}$ cu.yd. mixers. Cement is weighed and fed into the mixers through two 6 in. pipes which pass through the aggregate pass; each mix contains 2·2 pockets of cement. Rock and sand are fed into the weighbatcher through clam shell doors, there being one weighbatcher for each mixer. The measuring flask of the weighbatcher is tipped, when required, feeding the mixer.

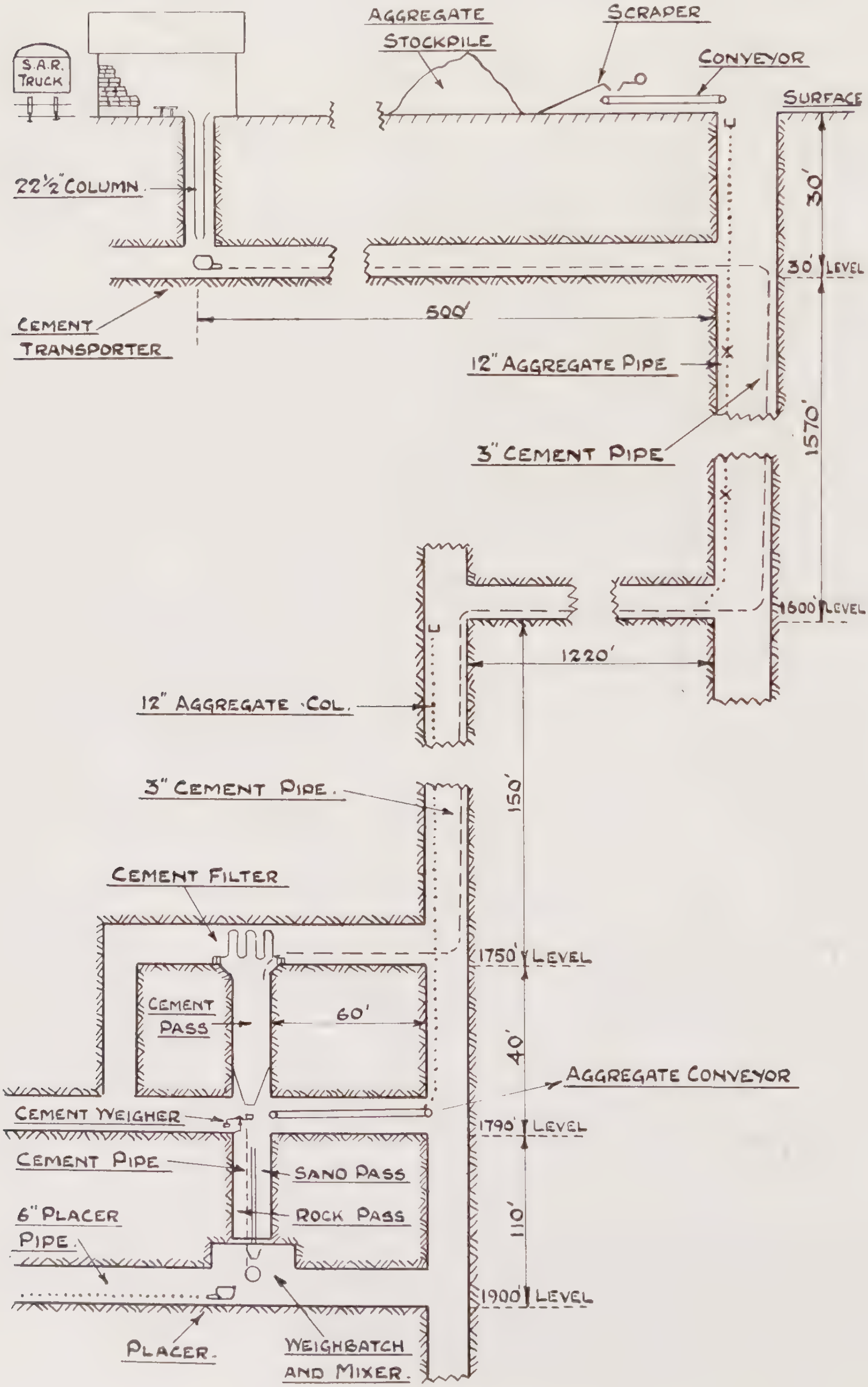

Fig. 7 Cement and aggregate handling layout

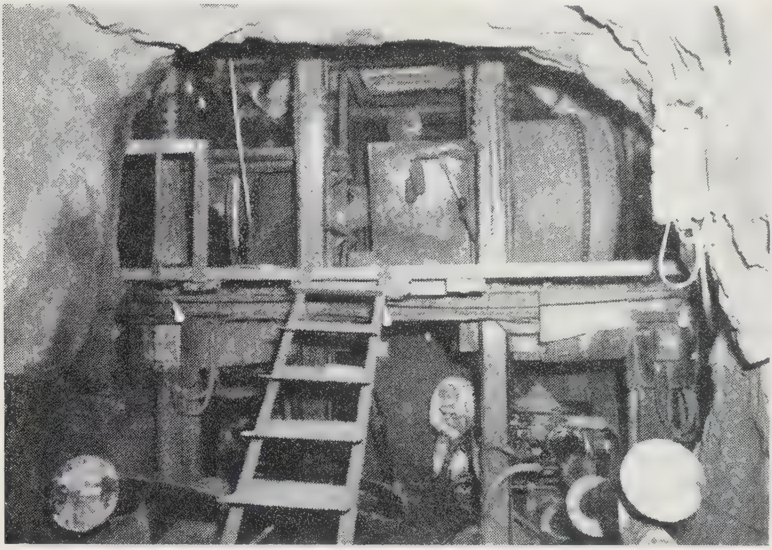

Fig. 8—View of placing chamber showing mixers, placers and inclined delivery pipes

Water is added from a measuring tank and 2·2 oz of lissapol and 1 lb of calcium chloride are added to each mix.

The mixers are able to supply either of the two ½ cu.yd. capacity, Pressweld pneumatic concrete placers. The output of each placer is 10 cu.yd. an hour. The concrete is drawn out of the placer in a series of "plugs." It is important that the compaction of each plug be obtained as soon as possible after leaving the placer. This is achieved by placing a frictional load on the concrete as it leaves the placer by locating the placer in a low sump with an inclined discharge pipe.

A standard 1 : 2 : 4 mix is used with a 0·65 water-cement ratio. A slump of 3 in.—4 in. at the placer is aimed at and to maintain the quality of the concrete, daily compressive tests are carried out. Seven-day compressive tests usually give results in the range of 2,500 to 3,500 lb sq.in.

D. Shuttering

During the early stages of block-caving, various drift shapes were tried. The stresses involved were such, however, that cracking of the concrete occurred with all types. The shape of drift selected was thus chosen purely for ease of operation and repair.

The following are details of the accepted drift standards (Fig. 9):—

Dimensions: 6 ft. by 6 ft. inside concrete.

Thickness of concrete: Sides 2½ ft. or 3 ft.; roof 3 ft.; floors 2 ft.

The majority of drifts have not been reinforced. It was the opinion of both Canadian and American operators that the use of reinforcing, where blasting frequently occurred, was not desirable. Several drifts have been reinforced with 6 in. by 6 in. high-tensile steel mesh but the results obtained have not been conclusive as to the effectiveness thereof. As an experiment, two 45 ft. lengths of drift are to be heavily reinforced.

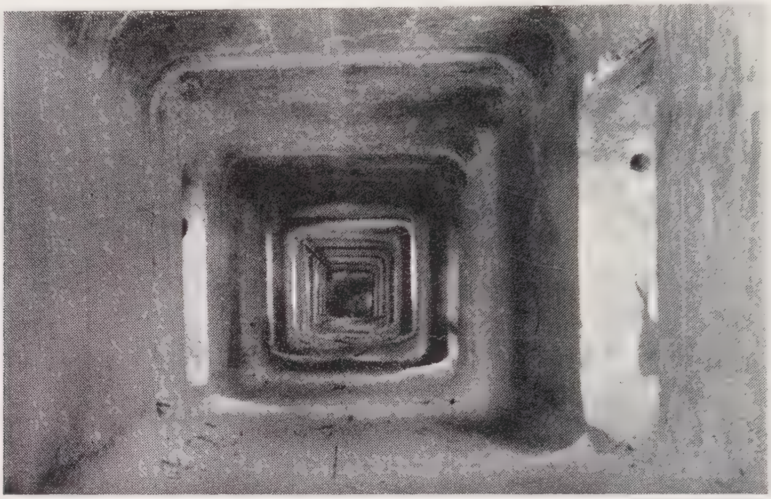

Fig. 9—View of completed drift

Six-inch steam piping is used to convey the concrete from the placer to the shuttering: Y-pieces are used for the bends, the "straight through" limb being closed with a blank flange. This type of bend minimizes wear. Flanges are jig-welded to the pipes, and pipes are of standard lengths, thus ensuring interchangeability.

The steel shuttering for the concrete is constructed from 3/16 in. mild steel plate braced by angle iron. It consists of four sections, roof, two sides and a skeleton floor. The sides are hinged to the base and slide in slots, on rollers, in the roof section. The shuttering is collapsed by operating two centre screws between the sides which draw them inwards and, through a linkage system, draw the roof down a few inches. The base is mounted on skids and it is thus possible to draw the shuttering forward to the next position.

A second type of shuttering in use has a key-piece in the roof section (Fig. 10). During the stripping of the shuttering the key-piece is removed. The sides spring slightly inwards thus allowing the shuttering to be moved.

The front end shuttering consists of steel plates which are bolted to the main shuttering. The outer edge is wired back to old jumpers projecting from the completed concrete. The gap between the edge of the front shuttering and the sidewall is scribed with 9 in. by $1\frac{1}{2}$ in. timber.

The floor section is kept 12 ft. ahead of the shell (i.e. walls and roof). The central 8 ft. of the floor is cast 6 in. higher than the sides to improve the joint between shell and floor. Joints between floors and shells do not coincide.

The length of the shuttering is 12 ft. which allows a 9 in. overlap between the shuttering and the previous shell. The drawpoint shuttering is bolted to the outside of the main shuttering and results in a 4 ft. by 4 ft. opening in the concrete at 40 degrees to the horizontal. Drawpoints are at 11 ft. 3 in. centres staggered on opposite sides of the drifts. Thus, with scraper drifts at 45 ft. centres, each drawpoint serves an area of $22\frac{1}{2}$ ft. square.

The 6 in. concrete delivery pipes are laid in the completed portion of the drift. When placing a shell, the pipes make a 180 degree vertical turn and deliver the

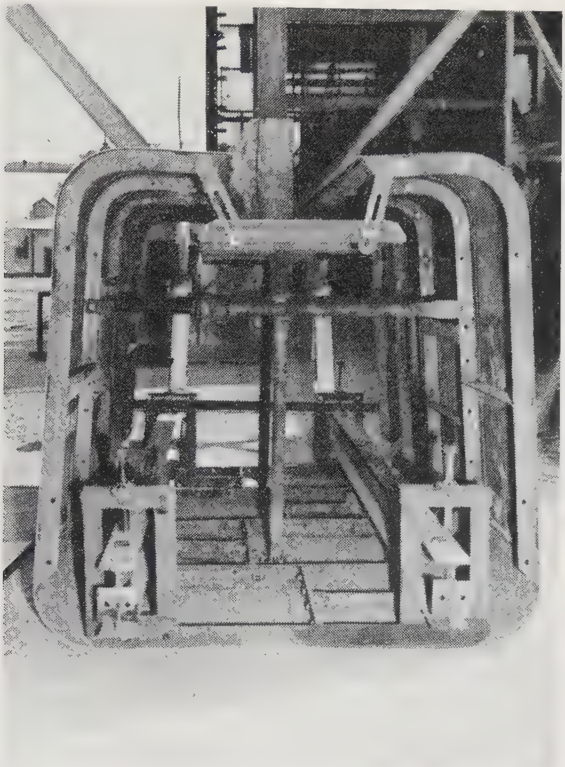

Fig. 10—Shuttering in collapsed position

concrete through a discharge box into the shuttering, at a level above the front shuttering (Fig. 11). This leaves a space above the completed concrete.

Efforts are now being made to fill this space. At the completion of pouring the shell, the discharge box is removed and concrete is blown through to the space above the previous shell (Fig. 12).

The shuttering for the floor section is constructed from steel plate or 9 in. by $1\frac{1}{2}$ in. timber spragged into position.

E. Concreting routine

The drifts are developed as 5 ft. by 7 ft. ends at the footwall elevation of the final drift. Sliping and concreting begin at the return sheave end of the drift and retreat towards the winch chamber.

Sliping is kept approximately 14 ft. ahead of the last completed shell. The sliping consists in widening the 5 ft. by 7 ft. end to the dimensions of 14 ft. by 13 ft. After each sliping blast, roof bolts are installed.

To modify the effects of abutment loading on the drifts, a compressible fill is installed between the outer side of concrete and the solid blue ground. The drifts are sliped 1 ft. or $1\frac{1}{2}$ ft. wider on each side than required for concreting. Bratticing is erected behind which broken blue ground is packed, after which the shell may be poured.

Fig. 11—View of shuttering and concrete delivery pipe. Side packing can also be seen

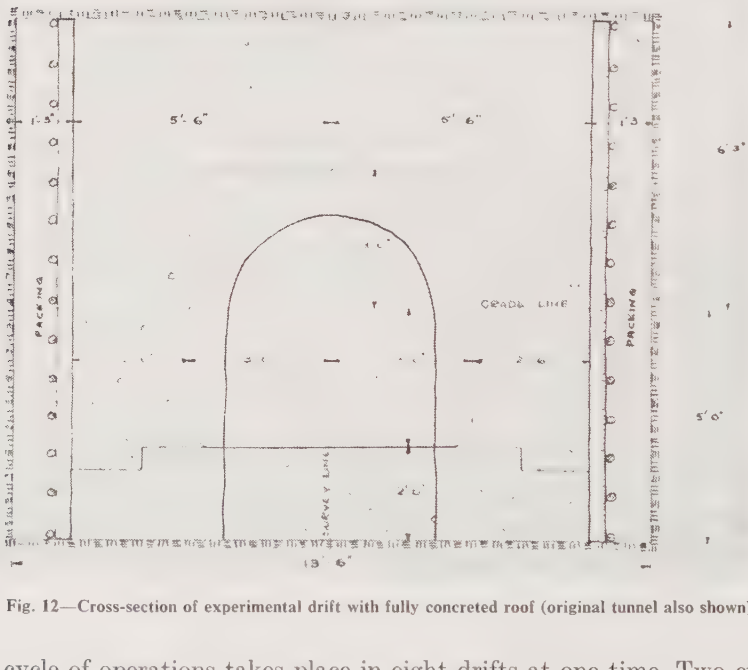

Fig. 12—Cross-section of experimental drift with fully concreted roof (original tunnel also shown)

The cycle of operations takes place in eight drifts at one time. Two crews have four sets of shuttering each. On any day the procedure would be to place concrete in one of the drifts, erect shuttering in the second, clean and slipe in the third, and sidefill the strip shuttering in the fourth. An advance of 11 ft. 3 in. is obtained every fourth day in each drift, the overall daily advance being $22\frac{1}{2}$ ft.

The drifts extend from country rock on one side of the pipe to that on the other side. Scraping is normally done from both ends of the drift, except in the case of drifts less than 200 ft. in length.

10. CRACKING OF THE CONCRETE

Some difficulty has been experienced by the cracking of the concrete in the scraper drifts. Few failures have occurred at joints in the concrete. The majority of these failures may be attributed to one of two causes:—

(*a*) The effect of abutment loading during undercutting. When failure occurs it is usually on the side of the drifts at the drawpoints. This effect has been modified by the following measures:—

- (i) The distance between the drifts and the undercut above has been increased from 19 to 29 ft.
- (ii) The size of individual undercut blasts has been decreased.
- (iii) The compressible fill on the sides reduces point loading. In addition, efforts are now being made to fill the space between the top of the concrete and the hanging wall.

(*b*) The effect of large caved lumps of ground during the early stages of caving.

The solid ground left *in situ* above the drifts gradually breaks away exposing the concrete below. Very large lumps, which break away during the initial stages of caving, may come to rest on the concrete roof, or on the sides or tops of drawpoints, leading to point loading of high intensity which may result in failure. The effect decreases as the caving moves up the pipe.

The raising of the undercut by 10 ft. and the filling of the space between the concrete and the hanging wall should delay the break-up of the solid blue above the drift.

11. YIELDABLE STEEL ARCHES

As a result of the shear cracking of concrete drifts, often leading to further deterioration, an experimental 150 ft. of drift has been equipped with yieldable steel arches (Fig. 13).

They are manufactured in Germany and consist of a three-piece arch formed from special steel sections. The sections consist of heavily flanged U-shaped rolled steel designed to nest into one another at points of overlap where they are clamped together with heavy U-bolts to provide the yieldable feature. The joints are drawn tight enough to hold fast under normal loading but under heavy loading they permit the nested segments to slide or yield before deformation can occur. The load is thus relieved and the structural integrity of the arch is maintained.

As the arches will be subjected to high stresses and will be required to stand for several years, a heavy steel section was chosen. The spacing of the arches is 2 ft. 3 in. which conforms to our drawpoint spacing of 11 ft. 3 in. The arches are separated by seven steel struts to maintain the spacing and to provide lateral stability to the structure. The internal size of the arches is 9 ft. 3 in. wide by 8 ft. high.

The internal dimensions of the drawpoint are 4 ft. by 4 ft. the sides being formed by the legs of two arches. The brow of the drawpoint consists of a heavy steel beam bolted to these two legs, and supporting the intermediate arch.

Fig. 13—Yieldable arches

The legs of the arches rest on channel irons. The floor of the drift has five sections of 45 lb rails laid longitudinally at 18 in. centres to provide the floor for scraping. These rails are held in a concrete mat of maximum thickness 12 in. The space between the arches is tightly packed with 8 in. diameter round timber. The arches have the advantage that the initial excavation is smaller and also that they provide immediate support around the periphery of the drift. No results have, as yet, been obtained as the abutment load has not yet approached this drift.

12. UNDERCUTTING

This consists of mining out a 7 ft. high horizontal slot over the entire area of the pipe. The footwall of the undercut is 29 ft. above the concrete floor of the scraper drifts.

Undercutting is done in sections, 202½ ft. in width, across the pipe from one rock contact to the other, starting at one extremity of the block cave project. The undercut face is carried at right angles to the drifts and the method consists of progressively dividing the undercut section into pillars followed by pillar-wrecking with long-hole drilling and blasting. Undercutting commences over the first four drifts when the concreting of some eight drifts has been completed.

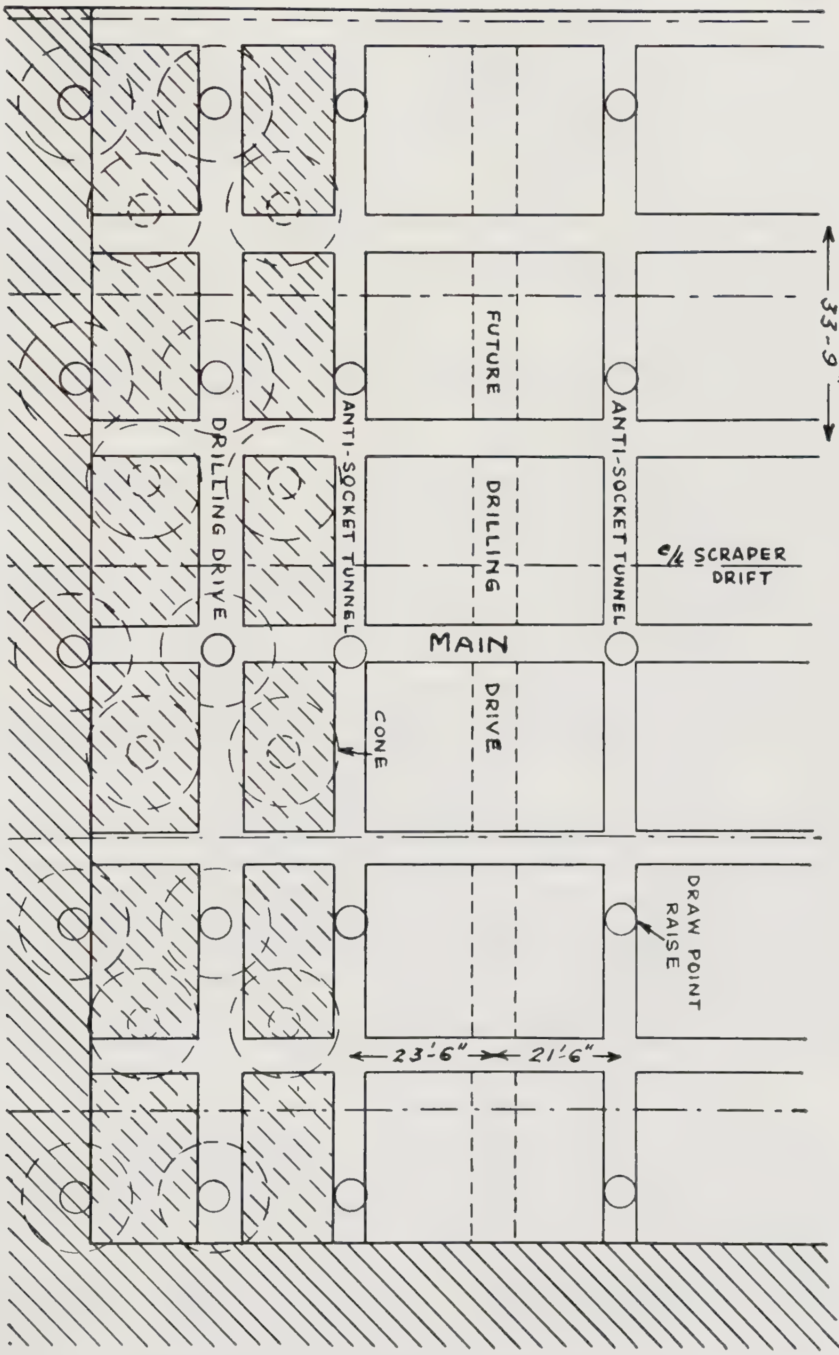

Fig. 14—Plan of undercut level prior to a blast. Pillars to be blasted are shown cross-hatched with broken lines

Access to the undercut horizon is by way of raises in the country rock. The initial development consists of "main" tunnels at $33\frac{3}{4}$ ft. centres, parallel to the drifts, with anti-socket tunnels at right angles to the mains at 45 ft. centres (Fig. 14). The anti-socket tunnels eliminate the possibility of drilling into misfires when long-holing. Mains and anti-socket tunnels are carried at $4\frac{1}{2}$ ft. by 7 ft. A limited number of drawpoint raises are put through from the drifts below for cleaning of the undercut development ground. These also ventilate the undercut level.

Each undercut blast, which takes place at four-week intervals, consists in blasting the pillars between seven mains and two anti-socket tunnels. This represents an area of $202\frac{1}{2}$ ft. by 45 ft. corresponding to 18 drawpoints. The percentage support at the time of pillar-wrecking is 64.

As small pillars take weight and deteriorate rapidly, the 7 ft. by 7 ft. drilling drive is only put through when the other development is complete. The drilling drive interconnects the seven mains and is approximately midway between the anti-socket tunnels.

As soon as the drilling drive has holed, 16 ft. flexible steel holes are drilled between the drilling drive and the anti-socket tunnel on either side. The holes are drilled using drill carriages, the machines being mounted in cradles with manual screw feed. Three holes are drilled per row, the burden being 2 ft. 3 in.

The flexible steel has the normal T.C. chisel bit but the cross-section of the stem is rectangular (approximately $\frac{1}{2}$ in. by 1 in.). It is made of a special alloy which gives the steel considerable flexibility allowing holes to be drilled longer than the width of the drive that drilling is done from. Standard lengths of the steel give 2 ft. 7 in. steel changes up to the required hole length.

Prior to the undercut blast, all the drawpoint raises, not used in undercut cleaning, are raised to within 7 ft. of the undercut footwall. The final round is drilled as well as the coning holes for enlarging the raise to a cone of base diameter 19 ft. (Fig. 15).

The undercut is charged and blasted, using millisecond delay detonators, and in the following week the raises and the coning holes are blasted.

The size of the individual undercut blasts has been progressively decreased since the first experimental blocks. Initially the entire area over four drifts would be undercut in a single blast. However, abutment loading and the concussive effects of the blast proved to be excessive. Further, the undercut face is now carried at right angles to the drift, whereas it used to be carried parallel to the drifts. This avoids putting the whole of a single drift under abutment loading at one time.

13. VENTILATION

Air is drawn in through the main shaft and haulages and passes into the drifts through the chinamen and winch access tunnels. At the end of each drift is a vertical pass offset to the side of the drift, and with a connection into the drift through the concrete sidewall.

Double-sided drifts have either a 4 ft. 6 in. diameter pass or two 3 ft. 6 in. diameter passes. Single-sided drifts have a single 3 ft. 6 in. diameter pass. All passes are concrete lined. A fan at the bottom of each pass exhausts air from the drift into a timbered return airway situated 30 ft. below the drifts. (3 ft. 6 in. passes—19 in. fan delivering 5,000 c.f.m. at $1\frac{1}{2}$ in. w.g; 4 ft. 6 in. passes—24 in. fan delivering 10,000 c.f.m. at $1\frac{1}{2}$ in. w.g.)

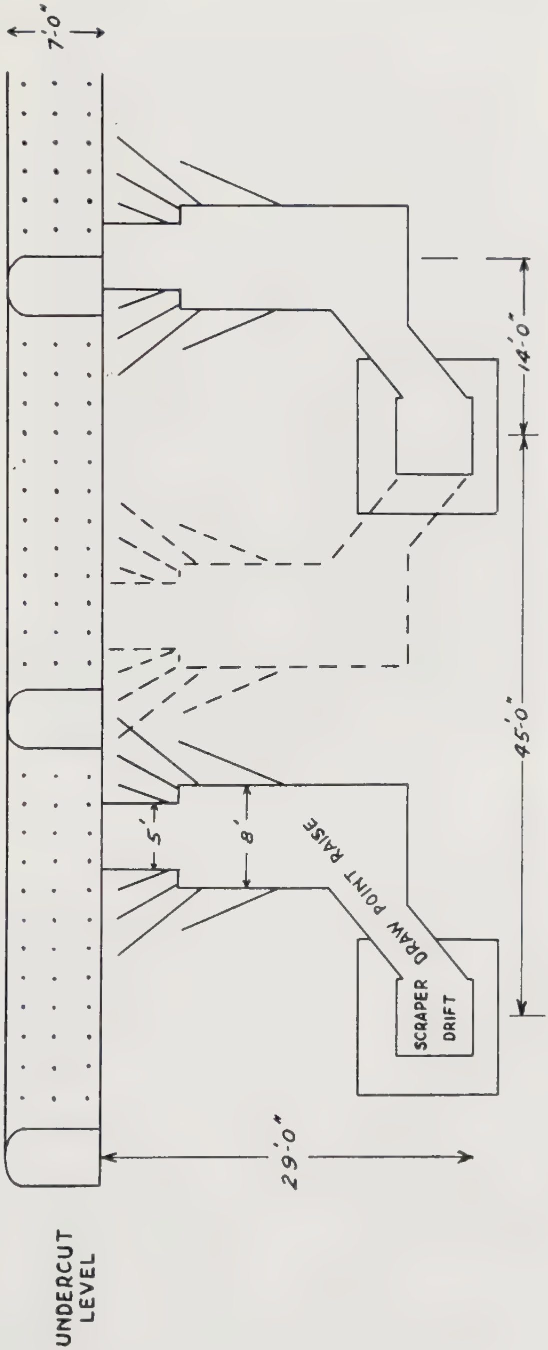

Fig. 15—Cross-section showing draw-point raises and coning holes

The air travels along the return airways to the return air passes which lead to the upcast shaft. The main exhaust fan is situated either on surface or underground.

14. CAVING

After an undercut blast, the ground from the blast is pulled slowly, during which time caving starts. Initially the caving is slow and the caved ground tends to break away in large blocks. Drawing at this stage must be slow and it takes 12 to 18 months to build up to a drawing rate of 12 tons per drawpoint per day. This corresponds to the removal of 30 per cent of the ground caved at a maximum caving rate of 1 ft. per day. The level of the caved and caving ground thus slowly moves up the pipe until the cave breaks through to the overburden above.

Draw control is strictly applied so that the cave face over the entire pipe is at approximately the same level. Alternatively, draw control can be applied so that the cave breaks through to the overburden at approximately the same profile as the solid-overburden contact.

Once the cave has broken through it is possible to draw at a much higher rate. In the experimental section at Bultfontein the drawing rate is now 36 tons per drawpoint per day.

During the initial stages of caving the ground is very coarse. As the cave moves up, however, the ground becomes finer as the large lumps are broken up by mutual attrition in their downward passage to the drawpoints.

Drawing continues until overburden appears at the drawpoints. At this stage there is still a considerable amount of blue ground mixed with the waste so that drawing will continue until the mixed waste-blue ceases to be economic.

15. COLLECTION OF GROUND

Three classes of operators are employed on the scraper drift level. The scraperman is responsible for routine secondary blasting and for the tally and, depending on local conditions, he may take charge of from 6 to 14 drifts.

The actionman is concerned only with the drilling and blasting of large lumps that threaten to damage the concrete in the drifts. The third class is the drift repairman, who repairs drift sections that have cracked and deteriorated.

In all drifts the standardised equipment is as follows:—75 h.p. Joy-Sullivan winch with a 50 h.p. motor and with air-operated thrusters; 15/16 in. pull rope (old hoisting rope); $\frac{3}{4}$ in. tail rope; two-ton V-back scoop; 6 in. diameter tail rope snatch blocks at 45 ft. centres and a 32-volt loco-type, sealed-beam drift light.

In addition, the floor section of the drifts in the country rock is developed 2 ft. below the drift floor level in the "blue." This serves as a stockpiling area for about 15 tons, immediately in front of the chinamen.

The drift light and coloured reflectors hanging in the drift enable the winch driver to tally accurately the number of scoops drawn from each drawpoint. A lighting system enables the winch driver to signal to the haulage crew when a train is required at his chinaman.

The usual method of operation is for the winch driver to fill his stockpile by scraping from drawpoints in the drift, then to signal for a loco, and scrape the stockpiled ground, through the chinaman, into Granbys (Fig. 16).

Fig. 16—Scraping in a drift

The scraperman and his drilling and barring crew make ground in the drifts not being scraped.

Electric locos convey the ground from the pipe to the main hoisting where the tips, underground crusher and main storage passes are situated. Fig. 17 shows an underground tipping scene.

Scraping directly into Granbys, in combination with the underground crusher, reduces secondary blasting to a minimum and eliminates grizzly development.

16. DRAW CONTROL

Draw control is strictly applied to avoid funnelling and premature dilution with the overburden. The winch driver records the source of each scoop and the number of Granby cars taken from the drift. A check on the latter is kept by the haulage crew.

Totals are taken out at fortnightly intervals and a record is kept of progressive tons scraped from each drawpoint in the project.

Drawpoints that are "too high" in relation to the others in the drift are closed for the following fortnight, and drifts that have a tendency to get too far ahead are given a limited quota (in Granbys) for the fortnight.

Drawpoints that are not to be pulled are closed by means of a light pole, thin wire and a lead seal.

A draw control officer is responsible for this work and for the checking of the accuracy of tallying.

17. PRODUCTION AND LABOUR

	Bultfontein 1,220 ft.	Dutoitspan 1,350 ft.	Jagersfontein
Daily tonnage	2,400	1,600	4,800
European labour	2	8	35
Native labour	28	61	285

All block cave production comes from two shifts. Due to local conditions, the production figures for Jagersfontein are lower than the corresponding figures for Kimberley.

The labour totals include all labour employed in extraction and haulage to the ore passes in the country rock. The European labour total excludes supervisory staff.

18. HOISTING

A significant change resulting from the introduction of block-caving at Bultfontein and Dutoitspan has been the centralization of hoisting at one shaft. Under chambering these two mines operated completely independently of one another and each mine had its own hoisting shaft and boiler plant for the operation of steam hoists and compressors.

With the application of block cave methods this is no longer necessary, due to the decreased labour and material requirements. The ground from both mines is now hoisted up the Bultfontein shaft.

In order to achieve this centralization a haulage, 2,220 ft. long, had to be driven from the pipe at Dutoitspan to the hoisting shaft at Bultfontein on the 1,900 ft. level. Three conveyor belts were installed at Dutoitspan to deliver the ground from the peripheral passes to a central pass down which the ground is tipped to the 1,900 ft. level.

On this level an overhead electric traction system connects both pipes with the hoisting shaft. Set tramming circuits are laid out, operated by spring-loaded switches and automatic signalling, thus reducing labour requirements to a minimum.

Ground tipped at the joint shaft passes onto an apron feeder which delivers to an inclined grizzly. Undersize passes directly through to the storage passes while the oversize is crushed in a 48 in. by 42 in. Allis-Chalmers jaw crusher with 6 in. setting. This crusher was installed to cater for the large lumps mined under the block cave system and it is capable of crushing 330 tons per hour, at 6 in. setting.

The Bultfontein shaft had to be completely renovated so that the increased tonnage of 12,000 tons per day could be hoisted through it. All the timber in the shaft was replaced by steel buntons. In the skipways the bunton ends were secured in continuous concrete strips, while in the manways they were concreted into "cousin jack" hitches which had been grouted and bolted to the sidewalls. The steam plants at both mines were shut down. At Bultfontein an a.c. electric winder has been installed for rock winding. This winder is equipped with twin 2,020 h.p. motors and compensated dynamic braking. Thirteen-ton bottom dump skips are now used, replacing the former overturning type. At the same time the man-hoisting engine was converted to electrical operation. A small electric winder, to be used in emergencies only, has been installed at Dutoitspan Mine.

19. ADVANTAGES OF BLOCK-CAVING AS COMPARED WITH CHAMBERING

(*a*) The introduction of block-caving has resulted in lower labour requirements.

For a mine hoisting 4,800 tons per day, the following is the estimated total mining department labour requirements, and productivity, for a mine on chambering as compared with block-caving. The latter includes a preparation allowance for the drop down to the next block-cave horizon.

	European labour	Native labour
Chambering	80	750
Block caving	60	330
Percentage saving	25	56

Productivity	Tons per European shift	Tons per native shift
Chambering	60	6·4
Block caving	80	14·6
Block caving excluding preparation allowance	90	17·1

(*b*) Economics of block-caving

These figures are based on the Dutoitspan 1,900 ft. project.

	Total £	Pence per ton
Total additional costs (working and capital) of preparation work for block-caving (including crusher) compared to preparing for chambering	460,000	5
Estimated saving in working costs of mining by block-cave instead of chambering	1,300,000	15
Net overall saving of block-cave	840,000	10

The total cost of block-cave preparation is equal to an average of £950 per draw-point. This includes the cost of the underground crusher but does not include the cost of the haulage from shaft to pipe.

Current working costs and estimated working costs for a mine on full caving per ton are:—

	Chambering s. d.	Caving s. d.
Mining	6 0	4 9
Surface and general	3 6	3 6
	9 6	8 3

This reduction of 1s. 3d. per ton represents a saving in mining costs of 21 per cent.

(*c*) Chambering tends to become more difficult with increased depth due to crushing of the large number of small pillars formed under this system. On the other hand, caving should take place more readily with increased depth.

(*d*) The annual estimated saving from the joint shaft, serving both Dutoitspan and Bultfontein Mines is £112,000. This is made up of saving on underground haulages, the shaft, electric hoist, elimination of steam plants, etc.

With only one mine in production this annual saving is estimated at £38,000.

(*e*) All workings are concentrated on one level. This reduces overheads and also results in improved supervision.

(*f*) Security will be improved as the hand-loading of ground is replaced by mechanical loading.

(*g*) Ventilation conditions are improved.

(*h*) Greater safety.

In conclusion the authors wish to thank Mr G. S. Giles, Consulting Engineer of the Anglo American Corporation, for permission to publish this paper.

TREATMENT AND RECOVERY PRACTICE AT KIMBERLEY MINES OF DE BEERS CONSOLIDATED MINES, LIMITED

By **E. COLVIN*** (Visitor) and **H. S. SIMPSON†** (Member)

INTRODUCTION

Of the five major kimberlite pipes in the vicinity of Kimberley only three are now being exploited, these being the Bultfontein, Wesselton and Dutoitspan mines. The Bultfontein and Dutoitspan mines are interconnected by a haulage on the 1,900 ft. level and are served by a joint hoisting shaft. Wesselton mine forms a separate entity.

Ground from the mines is passed through three main stages of treatment before the final product of diamonds is obtained:

(1) crushing plant, (2) washing plant, and (3) recovery plant.

For the first few years after the discovery of the "dry diggings" concentration methods were crude; the first major improvements were the introduction of water in the concentration process and the development of the rotary washing pan in 1874. The latter, with certain modifications, is still used and is a highly efficient method of eliminating some 98 per cent of the original ore as tailings.

From about the year 1888, when the first vertical shafts came into operation, the ore was fed from the shaft bins to 20 cu.ft. trucks. These were trammed by endless rope haulages to large areas adjacent to the mines known as "floors." Here the blue ground was spread out and watered and harrowed until the process of disintegration was complete, then reloaded and sent by endless rope haulage to a series of small washing plants for concentration. The process necessitated a very large labour force, as many as 10,000 natives being required. The serious security hazard that this method presented can well be imagined.

In 1925 the direct method of treatment was adopted which involved the use of 9K Allis-Chalmers gyratory crushers and the building of two large washing plants to replace the numerous smaller units. Crushing plants were built on the haulage routes, being fed by endless rope haulages with ground direct from the shafts (−18 in. lumps). The method achieved a large reduction in labour.

From 1932 onward, however, crushing plants were built on modern lines, directly connected to the headgear bins of each hoisting shaft. The ultimate $-1\frac{1}{4}$ in. product was then delivered to one central washing plant. The transport of this ore, by endless rope haulage, was accomplished by approximately two thousand 20 cu.ft. trucks. These haulages have in recent years been replaced by a conveyor belt system which will be referred to later in the paper.

CRUSHING PLANT

Current practice is outlined in the attached flow sheet (Fig. 1). Ore is hoisted in 10 or 13 ton bottom discharge skips and delivered directly to the crushing plant situated at the hoisting shaft of each mine.

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*Superintendent, Treatment Plant.
†Manager, Recovery Plant.

Vibrating feeders deliver the ore from the shaft bins on to conveyors which pass through metal detectors and over magnetic pulleys for the removal of tramp iron.

Primary sizing takes place over single deck vibrating screens of $2\frac{1}{2}$ in. screening. The $+2\frac{1}{2}$ in. product is passed to picking belts for hand sorting to remove waste rock which forms approximately $2\frac{1}{2}$ per cent of the feed. The picking belts deliver to two 9K Allis-Chalmers crushers set to $2\frac{1}{2}$ in.

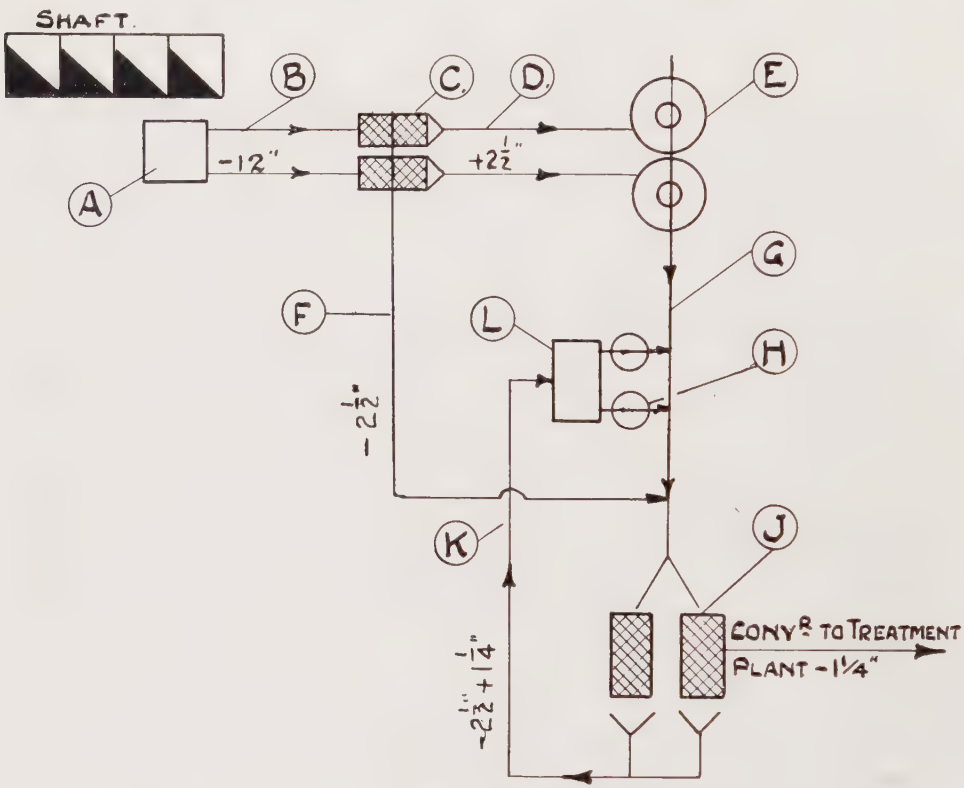

Fig. 1—Legend

A	Headgear bin	B	2—36 in. conveyors—12 in.
C	2—vibrating screens—$2\frac{1}{2}$ in. space	D	2—42 in. picking belts
E	2—9K Allis-Chalmers crushers (primary)	F	1—36 in. conveyor
G	1—42 in. conveyor	H	2—$4\frac{1}{4}$ ft. Symons rushers (secondary)
J	2—Single deck vibrating screens $1\frac{1}{4}$ in. space	K	1—36 in. conveyor
L	1—Secondary crusher surge bin		

Flow diagram. One mine—sorting and crushing plant

The product of these crushers joins the undersize of the primary screens and is passed to two single deck vibrating screens ($1\frac{1}{4}$ in. mesh). Undersize passes to a surge bin while the oversize is crushed to $-1\frac{1}{4}$ in. by two $4\frac{1}{4}$ ft. Symons crushers in closed circuit with the secondary screens.

From the surge bin the ground is fed onto a conveyor belt system which transports it to the central washing plant. There are two of these belts and each one handles approximately 600 tons per hour.

WASHING PLANT (Fig. 2)

DESIGN AND CONSTRUCTION

The present plant, which was built at a cost of £1,400,000, replaces the "old plant" which had operated over a period of 30 years. The plant was opened in 1958 and is capable of treating 20,000 tons per day. It is interesting to note that the plant commenced operation only $10\frac{1}{2}$ months after the start of construction.

The new plant was justified on the following grounds:

(a) Its central situation has made it possible to transport the ore to the plant by means of conveyors as opposed to the old method of endless rope haulages.

The resulting labour saving has been 16 Europeans and 170 Natives. A saving in operating costs of approximately £60,000 per annum is also effected.

Fig. 2—View of the new washing plant

(b) The capacity of the new plant enables larger tonnages to be treated. This is particularly advantageous in view of the decreasing grade at depth of the mines.

(c) Improved recovery has resulted from incorporating a tertiary circuit in addition to the primary and secondary concentrating pans. The increase in production has been approximately 12,000 carats a month and this may rise to 14,000 carats per month. These diamonds are, however, of inferior quality. Continuous concentrate removal, better density control and improved techniques for feeding the pans have also been effective in improving recovery.

(d) In the old plant crushing was done in open circuit by means of rolls between the primary and secondary pans. In the new plant the crushing between the first and second stage concentrating pans is done in closed circuit by Symons cone crushers. This has resulted in more efficient crushing and the release of more fine diamonds from the blue ground matrix. Further, the cost of operating these crushers has been substantially lower than that of the rolls.

After the last war exhaustive experiments were conducted in Kimberley with a view to replacing pan washing with more modern methods such as heavy media separation.

The washing media (puddle) in pan plants consists of a suspension of clayey material and fines in water. This puddle forming material has almost invariably been available in the ore from the Kimberley pipes. The ore is comparatively soft and contains a high proportion of fines. The cost of removing the clay and fines from the ore together with the cost of replacing the natural medium with an artificial one, outweighed any potential advantages of heavy media separation and established pan washing as the most economic process. It was therefore adopted for the new plant.

DESCRIPTION OF PLANT (Fig. 3)

Ore from the mines is delivered to the control bin at the plant from where it is distributed either to the washing section in the required quantity, or to the stockpile for later reclamation. The stockpile not only provides a cushion between the mines and the plant, but also permits the plant to accept ground at a rate higher than it can at the time treat. At certain periods two mines are delivering ground at a rate greater than plant capacity. The surplus is stockpiled and treated on the following shift when only one mine is hoisting. Further, a test section has been provided in the stockpile which permits the accumulation of ground from one mine and its subsequent treatment separately as a bulk test to determine the grade of the mine.

From the control bin the ground is conveyed to two 100 ton bins at the top of the washing section. From the bins it is distributed equally to the 12 sections of the plant by comet belt feeders. A rheostat, common to all feeders, gives a variation in the feed rate of 15 per cent.

Primary concentration is done in three stages. Ore fed to the primary, or coarse, pans is unsized below $1\frac{1}{4}$ in. It is estimated that 80 per cent of the diamond recovery takes place in these pans. Prior to entering the primary pans the ground is mixed with initial puddle (S.G. 1·26 to 1·28) at the rate of 230 gallons per load. (1 load = 16 cu.ft.). Clean make-up water (range 80 to 110 gallons per load) is also added at this stage in order to puddle the incoming fines.

The lighter constituents of the feed, overflowing the inner weir of the primary pans, pass over $\frac{3}{8}$ in. rod-deck screens, the underflow of which constitutes the feed to the secondary pans.

The oversize of the $\frac{3}{8}$ in. screens is conveyed to a recrush section (Fig. 4). Here it is crushed in closed circuit to $-\frac{3}{8}$ in. by $5\frac{1}{2}$ ft. Symons shorthead cone crushers, and again introduced into the head feed of the primary pans. This recrush material constitutes as much as 40 per cent of the initial feed so that the rated capacity of the primary pans is exceeded, but any possible loss of diamonds at this stage is overcome in the second stage of concentration.

The overflow from the secondary pans is passed over six mesh screens. The +6 mesh material constitutes the tailings while the screen underflow is pumped by five 10 in. rubber-lined pumps through 42 in. hydrocyclone classifiers which give a

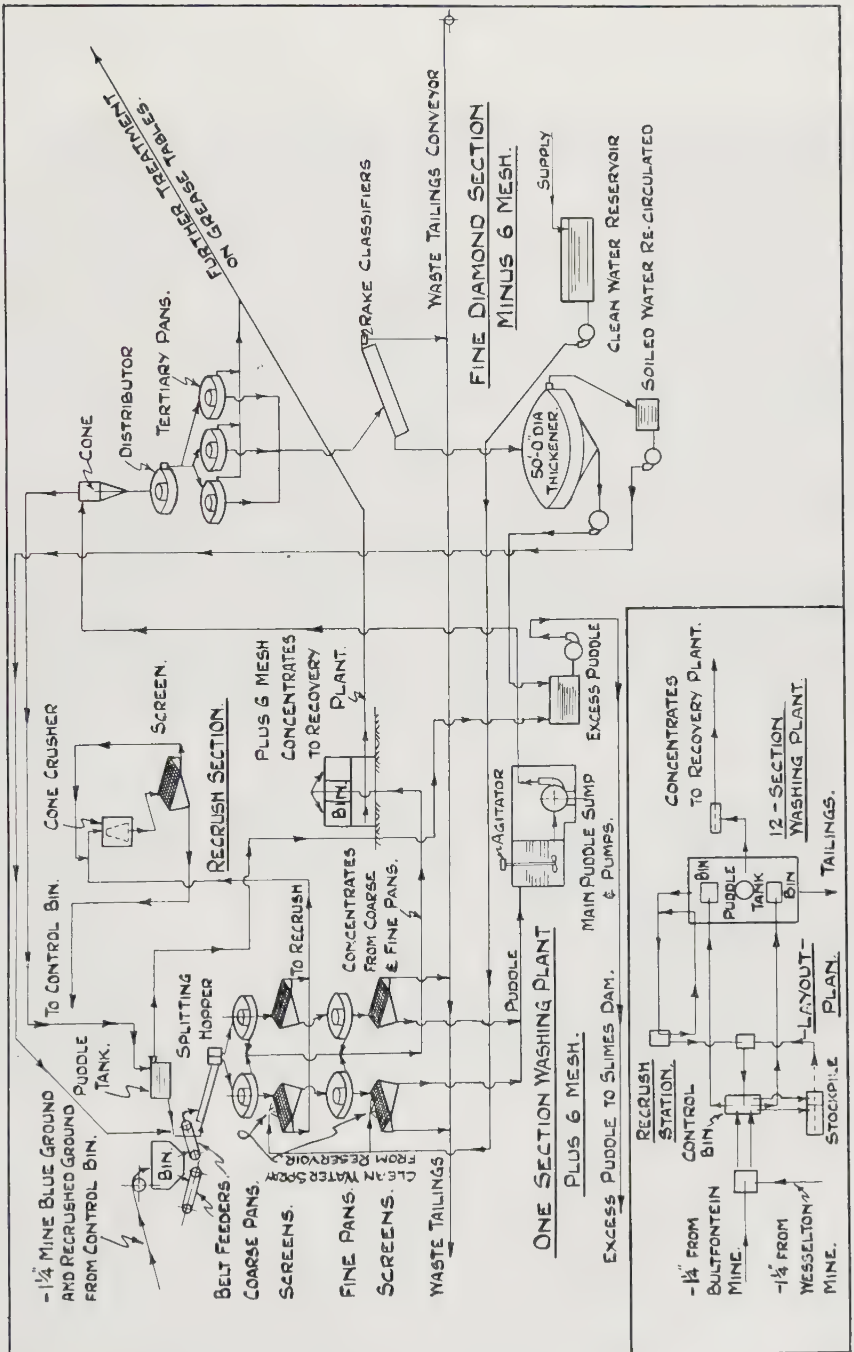

Fig. 3—De Beers Consolidated Mines Limited, Kimberley. Treatment plant washing section flow sheet

separation of from −6 mesh to −28 mesh. This thickened pulp is then fed to the tertiary pans for the final stage of primary concentration. The overflow of the cyclones constitutes the puddle which is recirculated through the plant.

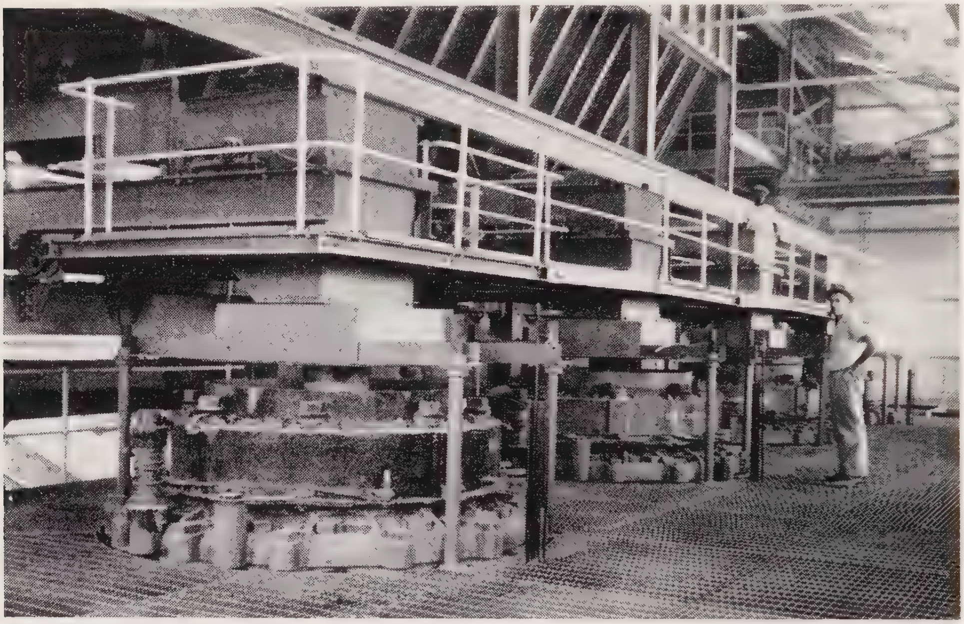

Fig. 4—Recrush section

THE ROTARY WASHING PAN

As this forms the basis of primary concentration, it will be described in greater detail.

Construction (Figs. 5 and 6).

The washing pan is an annular-shaped vessel formed by two concentric walls constructed of steel plate. The outer wall is 14 ft. in diameter and 20 in. high and the inner wall is 6 ft. in diameter and 12 in. high. Between these walls the annular space has a flat cast iron bottom with renewable chilled cast iron liner plates. A sliding door fitted in this bottom, permits the complete cleaning 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 by $1\frac{1}{2}$ in by 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 of each tooth points in the direction of rotation with one of the flat surfaces of the tooth facing the centre of the pan at a tangent to the circle of motion. The outer surface of the tooth in plan is thus at an angle to the tangent and therefore causes an outward thrust towards the periphery of the pan 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 of the pan 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 curve ends and is there for the purpose of moving the settled concentrates to the discharge opening in the outer wall.

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.

The feed inlet is set tangentially to the periphery. At a point 270° from the inlet an overflow weir is cut in the inner circle to a height of $3\frac{1}{4}$ in., from floor, and with a chord length of 28 in. on the primary pans. On the secondary pans these dimensions are $4\frac{1}{2}$ in. and 37 in. respectively.

The driving shaft revolves at 8 r.p.m. giving a peripheral speed at the outer rim of 5.87 ft. per second and at the innner rim of 2·52 ft. per second. The power consumption of the pans is from 7 to 8 h.p. with a rated capacity of 40 loads of 16 cu.ft. each per hour.

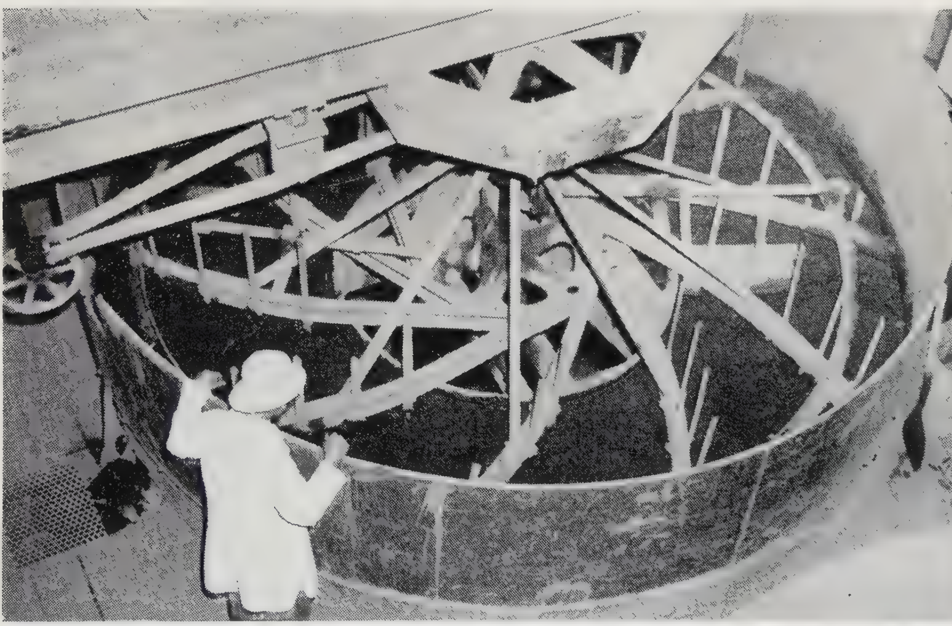

Fig. 5—The rotary washing pan

Operation

The relatively high specific gravity of the diamond (3·5) together with the comparatively small quantities of other heavy minerals, such as ilmenite (4·5), garnet, diopside, etc., permits of a relatively easy separation from the low density gangue minerals by means of these rotary pans.

The feed is continuously introduced tangentially to the outer circumference of the pan. The launder by which the ground and puddle enter is inclined to give the feed a velocity approximately the same as the revolving mass in the pan.

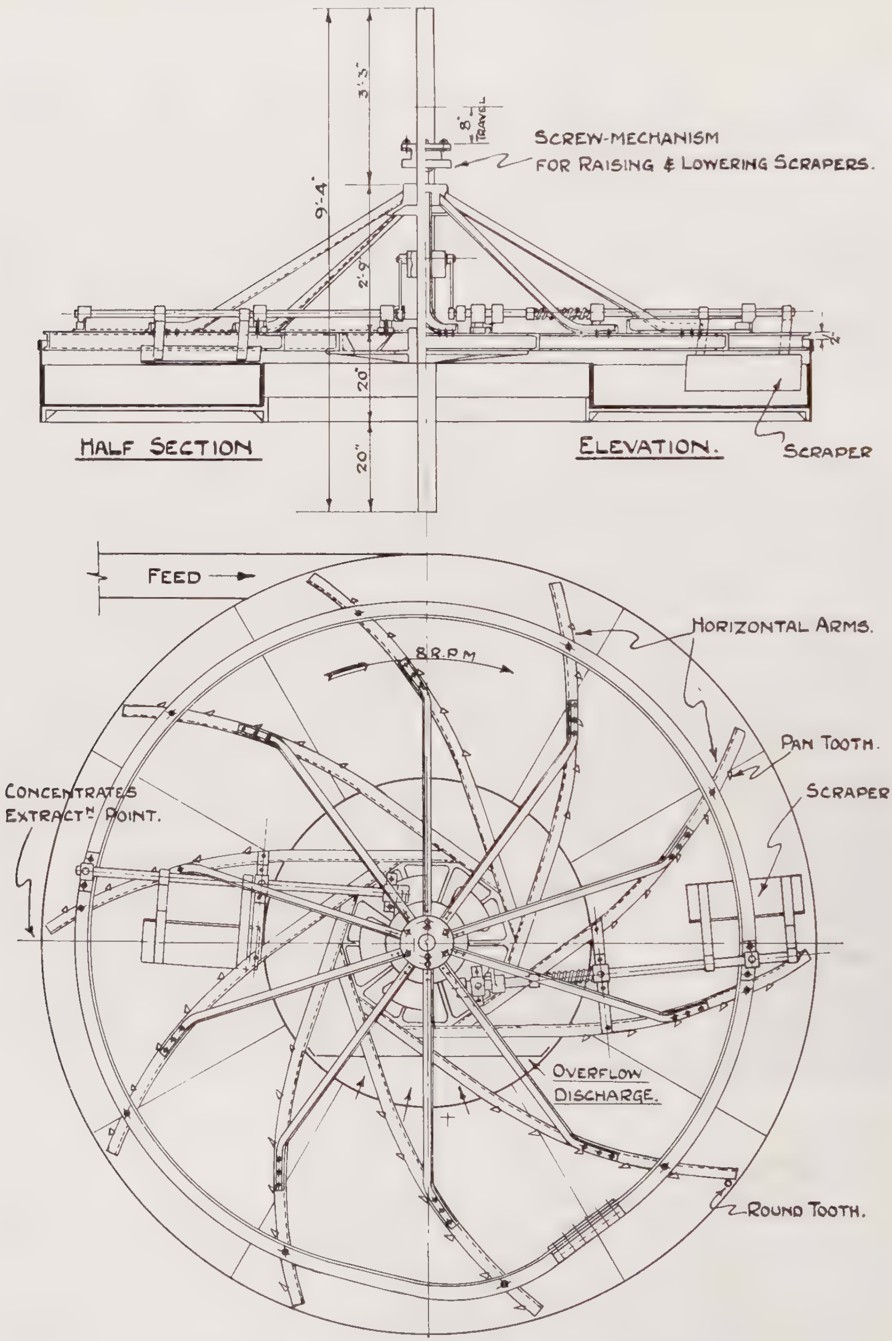

Fig. 6—De Beers Consolidated Mines Limited, Kimberley. 14 ft. diameter washing pan

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. When a pan is washing at the rate of 40 loads per hour the discharge tailings run about $1\frac{1}{2}$ in. high over the outlet weir. At the outer circumference the level is some 4 in. higher. The specific gravity of the mass in the pan (approximately 1·5 to 1·6) is controlled by the regulation of make-up water until the required electrical loading is obtained.

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 outward thrust of the pan teeth becomes more effective thus forcing the concentrates to the outer circumference of the pan.

Control

As can be seen from the foregoing, the control of the concentration process is achieved by the lowering and raising of the density of the mass in the pans by the regulation of clean make-up water to the pan feed. This is achieved by means of hydraulically controlled valves, remotely controlled from a central operating panel, the electrical loading on the pan motors being indicative of the density (Fig. 7). Extraneous loading on the pans, due to mechanical faults or broken teeth, immediately becomes apparent by unsteady ammeter pointers.

Other indicators show the control room operator the quantity of ground available in the bins and the amount of puddle available for distribution. Push button

Fig. 7—View of central operating panel

alarms and lights indicate to remote operators the requirements of the control room operator, who is also able to bring the whole, or parts, of the plant to a complete halt in an emergency.

It is interesting to note that, up until 1932, the density of the pans was controlled by visual observation. From 1932 until 1957 control was exercised by electrical loading on ammeters, one operator manually regulating the valves for eight pans. On the new plant one operator hydraulically controls remote valves of 48 pans and could operate still more if the capacity of the plant should be increased.

Extraction of Concentrates

Concentrates in kimberlite of specific gravity approaching and exceeding that of the diamond only constitute 0·25 per cent of the ore. However, with a valuable product, such as the diamond, a high factor of safety is allowed. A primary concentrate of 2 per cent of the original feed is obtained, heavy media separation and grease tables being used for the final concentration.

The concentrates, collecting on the floor of the pan at the outer circumference, are extracted through a discharge opening in the wall, being pushed out in a steady stream by the single round tooth. The concentrates are then lifted above the puddle level by means of a spiral conveyor. This discharges on to a 30 in. deep trough conveyor belt delivering to the recovery plant for further concentration.

DISPOSAL OF TAILINGS

As in all concentrating plants, the disposal of tailings constitutes a problem, more especially as the $-\frac{3}{8}$ in. tailings contain up to 25 per cent moisture. The coarser tailings from the secondary pans ($-\frac{3}{8}$ in. + 6 mesh), together with the tertiary pan tailings (−6 mesh) which are dewatered in six 8 ft. rake classifiers, are conveyed to the tailings dump for disposal by flingers which are short centre, high speed conveyors travelling at 2,400 ft. per minute. These flingers deposit the tailings a distance of 40 ft. away in the desired direction (Fig. 8).

The overflow of the rake classifiers is pumped to a 50 ft. thickener for desilting before re-use as make-up water in the pans. The underflow of the thickener joins the excess puddle generated in the plant, which overflows the puddle distribution tank, and is pumped to the slimes dam. A small recovery of water is made from the slimes dam.

It is of interest that, prior to 1932, tailings disposal from the washing plant was done by running all the tailings, including slime, from the secondary pans to a central sump. From here the mixture was loaded into 10 ton skips and these were raised on an inclined headgear to a screening plant on a dump 100 to 120 ft. high. (Fig. 9). After screening over $\frac{3}{8}$ in. static screens, the oversize was trammed away on endless rope haulages with 20 cu.ft. trucks and dumped, and the $-\frac{3}{8}$ in. product laundered to a site adjacent to the headgear.

In 1933 this headgear and screening plant system was scrapped and the screening done on the plant, the dewatered tailings being lifted to a bin on the old dump, and trammed away on the haulages for disposal.

In 1952 the haulages on the tailings dump were abolished and a system of conveyors and flingers was installed. With the introduction of this system of tailings disposal a saving of 18 Europeans and 150 Natives was effected for the disposal of approximately the same tonnage.

Fig. 8—Tailings disposal by flinger

LABOUR

The present labour complement, including staff, is as follows:

	Europeans	Natives
Crushers and Conveyors	11	105
Washing Plant (including Tailings disposal)	35	203

A comparison with the figures for 1950 shows the large labour reductions made.

	Europeans	Natives
Crushers and haulages	34	319
Washing Plant	76	471

RECOVERY PLANT

In the production of diamonds, recovery, which is the final concentration of the ore following the initial concentration by crushing and washing, has always been kept in a separate department. This has been mainly for security reasons as the very high concentration effected in the washing process gives a rich concentrate with visible values, requiring treatment under careful security control.

HISTORY

The first evidence of recovery in the diamond industry was immediately after the introduction of the rotary washing pan. Dilute concentrates in the form of gravel of varying sizes were tapped from the pans and hand-gravitated in a riddle in tubs of water. With the reciprocating motion imparted to the riddle, the heavier minerals.

including the diamonds, settled to the bottom. An oscillating and tilting motion then swept them to the centre. The riddle was then removed from the water, and with a quick movement which did not disturb the contents, was turned over and dumped on to a sorting table. On removal of the riddle, diamonds present were easily seen at the top centre of the pile of gravel, and picked out. This process is still seen today on the alluvial diggings.

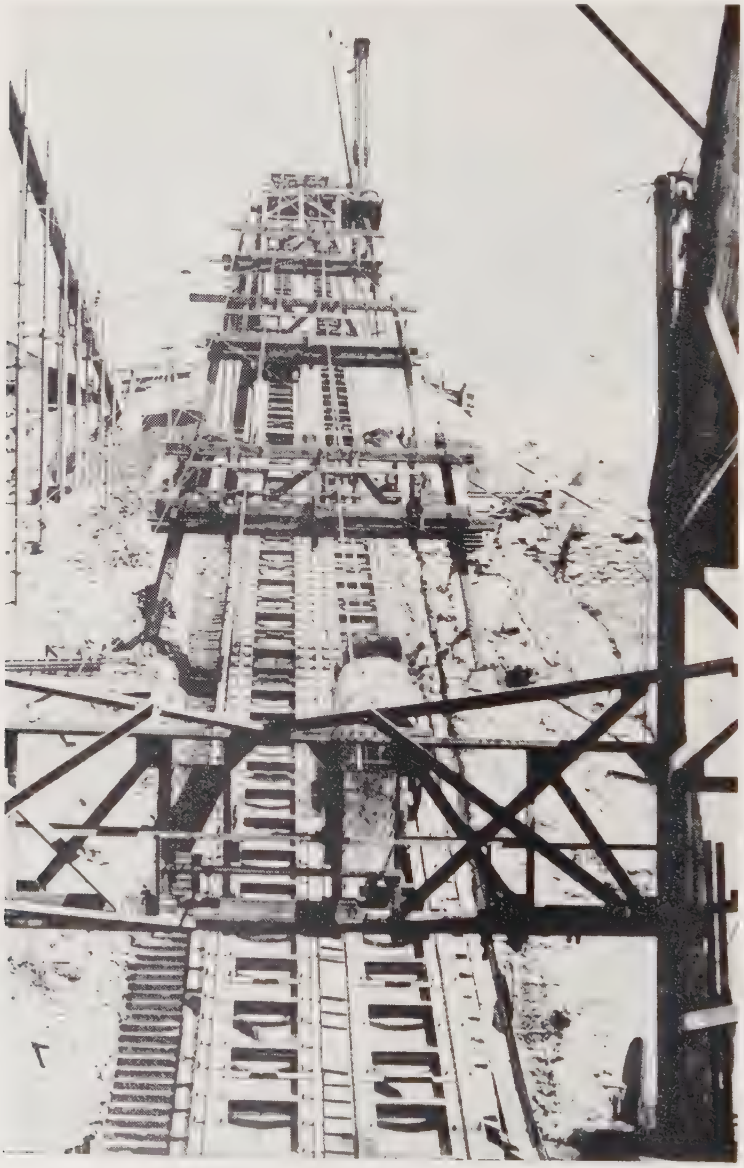

Fig. 9—Skip disposa l system

With the consolidation of the various diamond mining companies into De Beers Consolidated Mines in 1888, production which had previously taken place at scattered points round the kimberlite bearing pipes was centralized into one plant per mine and greater measures of mechanization were introduced.

On the recovery side this consisted of the use of Harz jigs. Close sizing of the feed to the jigs was adopted and the sizing was performed by a long trommel erected over a battery of jigs, the screen sizes being $\frac{1}{8}$ in., $\frac{3}{16}$ in., $\frac{1}{4}$ in. and $\frac{3}{8}$ in. diameter. Before entering the jig, the $\frac{1}{8}$ in. material was again screened in a trommel to remove the $\frac{1}{16}$ in. size which was stockpiled, as, except for a small percentage of outstanding gem diamonds, there was no sale for this size of diamond at the time.

The over $\frac{3}{8}$ in size of material and the hutch product of the jigs was hand-sorted by European labour and gleaned by Native convict labour. Gleaning of the sortings was a job that in the days before consolidation of the mining companies was carried out by wives and children of employees on a commission basis, with some little profit to themselves.

Harz jigs being called "pulsator" jigs at the time, the jigging and sorting plant was named the "Pulsator."

The year 1897 saw a great step forward with the complete mechanization of the recovery process by the invention and use of the grease table or "automatic sorter" as it was then called.

In this year experiments were being carried out in concentrating the $\frac{1}{16}$ in. stockpiled materials with a Wilfley table. The operator of the table noticed that diamonds in the concentrate were adhering to lubricating grease accidently smeared on portions of the surface, whereas the rest of the concentrate was washed off the grease. This discovery led to the development of the first grease table.

In 1907, a new "Pulsator" was built at a site situated centrally to the five operating mines, each of which had its own depositing floors and washing plant from which the concentrates were conveyed by four-fifths of a ton cars hauled on an 18 in. gauge railway by steam locomotives.

The plant consisted of six sections, four of which treated the concentrates from the washing plants in a single shift each day. The fifth and sixth sections were for lay-off maintenance and test purposes.

The concentrates were elevated to each section by 6 in. bucket elevators and were then closely sized by trommels into seven sizes ranging from $-\frac{1}{16}$ in. $+ 0$ to $-1\frac{1}{4}$ in. $+ \frac{5}{8}$in., all through round hole perforated plate screening. Each size was treated in a separate Harz jig and the 25 per cent concentrate from the hutches of the jigs elevated by 4 in. bucket elevators to side shaking grease tables on which the diamonds were recovered (Figs. 10 and 11).

The plant contained 40 double compartment 1 ft. 9 in. by 6 ft. 6 in. Harz jigs and 51 grease tables 2 ft. 3 in. wide by 13 ft. long. Tailings from the jigs and grease tables were collected on 18 in. conveyor belts leading to a dump for disposal. Water for the jigs and grease tables was circulated through settling dams for clarification.

With the closing down of the Kimberley and De Beers Mines by 1914, and the replacement of the depositing floors of the remaining mines in 1925 by crushing plants, later replaced by crushing plants at the shafts of each mine, and the building of a new washing plant to which the ore from each mine was conveyed by one ton cars on an 18 in. gauge endless rope haulage, the Pulsator was no longer conveniently situated. The plant also, although efficient, had become expensive to operate and maintain and it was decided in 1952 to build a modern recovery plant adjacent to the washing plant.

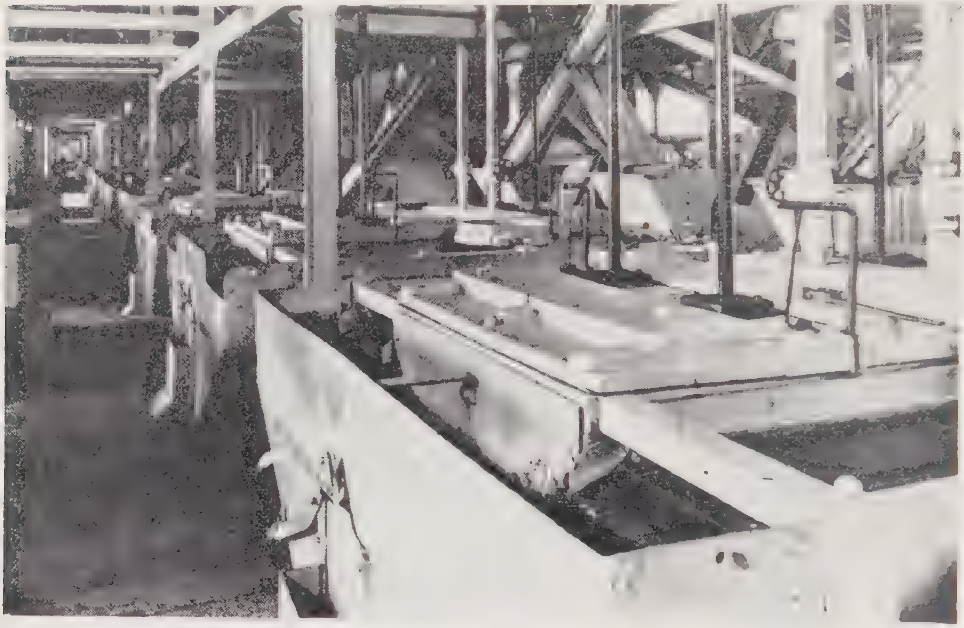

Fig. 10.—Pulsator jig platform

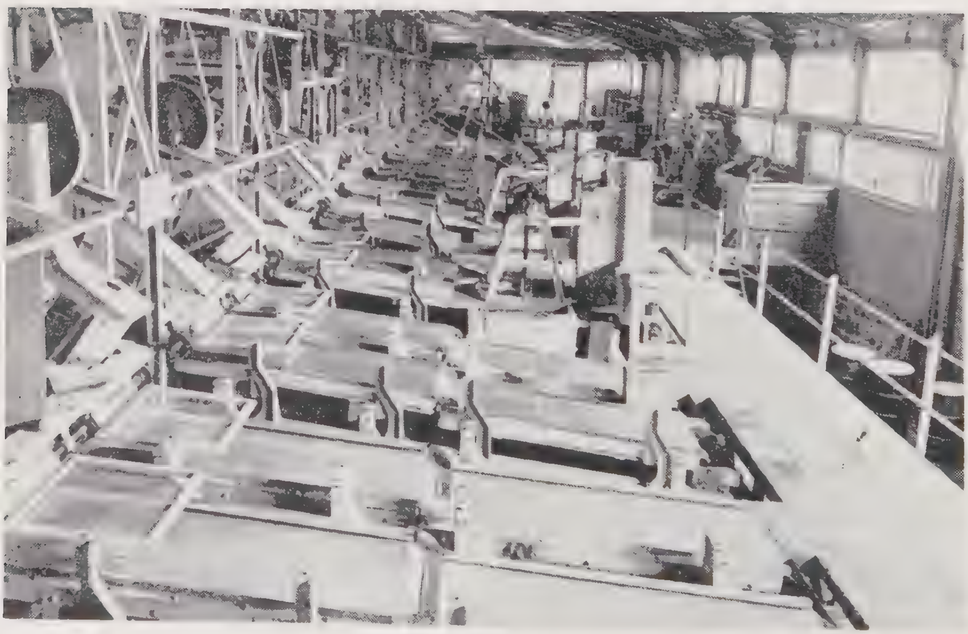

Fig. 11

Extensive research work prior to this established that heavy media separation of the washing plant concentrates produced results equal to that of Harz jigs, with lower operating costs, and at much less capital expenditure. At the same time, research into improving grease tabling methods proved profitable in producing a grease table operating on a contra-flow vibrating principle with a very much greater capacity than the former grease table.

In the recovery plant which was built, concentrates amounting to 200 tons per day were received on 24 in. conveyor belts from the washing plant, and separated into two sizes, plus and minus 10 mesh Tyler. The minus ten mesh fraction, amounting to 20 per cent of the total received, was dewatered in an 8 ft. rake classifier and concentrated further in two 24 in. by 36 in. Denver duplex mineral jigs, producing a 10 per cent concentrate, which was then treated on two vibrating grease tables.

The plus ten mesh fraction was concentrated in a 6 ft. diameter heavy media separation cone using ferro-silicon medium at 2·8 top-of-cone density. The 20 per cent concentrate from the cone was treated on two vibrating and one side-shaking grease tables.

In 1958, with the building of a modern washing plant at a site more centrally situated to the producing mines, the recovery plant was transferred to the new site, and although the heavy media separation process for the plus ten mesh concentrates remained the same, the process for the minus ten mesh concentrate was altered to cater for the greater production of fine diamonds due to the higher efficiency of the new washing plant (Fig. 12).

The recovery plant now consists of two sections, the main recovery plant, where the greatest values are recovered, over a single shift, separate from the washing plant, and the "fine diamond recovery" section, integral with the washing plant, working concurrently with the washing plant over two shifts daily (Fig. 13).

FINE DIAMOND RECOVERY

The "fine diamond" section consists of seven 2 ft. 6 in. by 5 ft. electromagnetic vibrating screens with 28 mesh stainless steel screen cloth, each followed by two vibrating grease tables (Figs. 14 and 15). Concentrates from the tertiary pans, all

Fig. 12—Exterior view of recovery plant

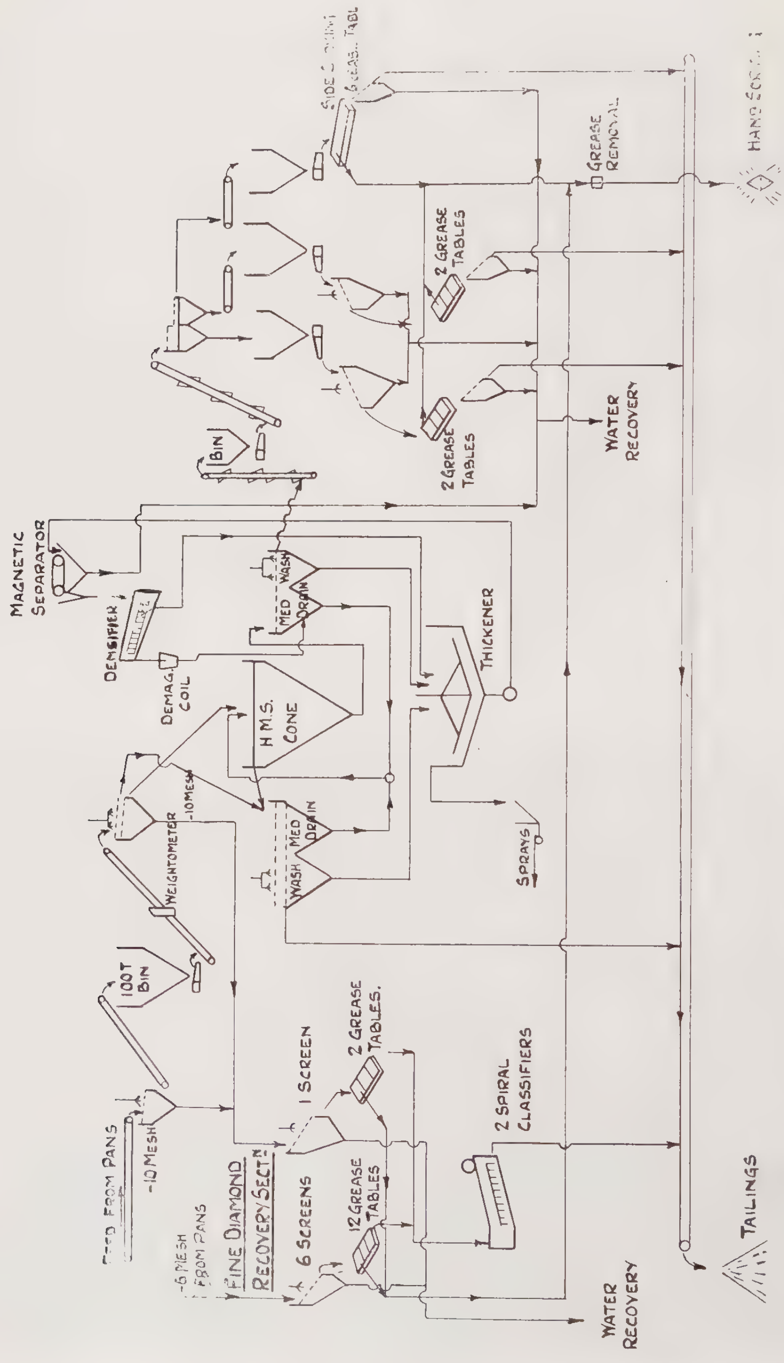

Fig. 13—De Beers Consolidated Mines Limited, Kimberley. Treatment plant recovery section flow sheet

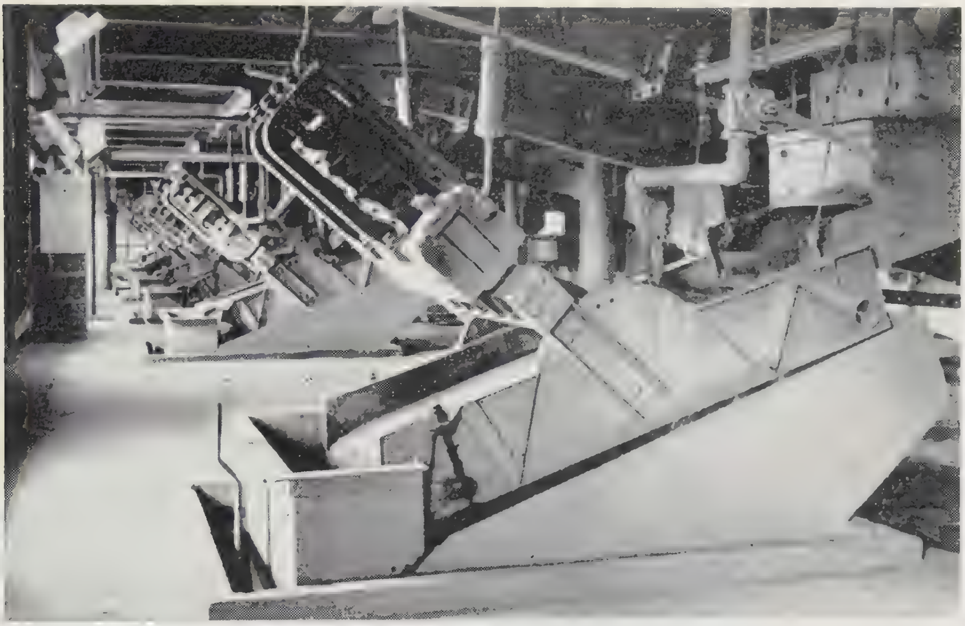

Fig. 14—Fine diamond section washing screens

Fig. 15—Fine diamond section grease tables

minus 6 mesh material, are washed on six of the vibrating screens, and are then treated for diamond recovery on 12 of the grease tables. Tailings from the grease tables after dewatering in two 4 ft. Akins classifiers are led to the washing plant tailings conveyor for disposal on the tailings dump. The wash water from the vibrating screens is laundered to a 50 ft. thickener, the clarified water from the thickener being gravitated to the soiled water sump of the washing plant for re-use, while the underflow is pumped to the excess puddle sump for disposal to the slimes dam with excess puddle from the plant. 400 tons of concentrates are treated daily in this section, each grease table treating 1·75 tons per hour.

The 400 tons daily of $-1\frac{1}{4}$ in. + 0 concentrates from the primary and secondary pans of the washing plant are collected over two shifts on 30 in. deep troughing conveyor belts and led to two 3 ft. by 6 ft. by 10 mesh stainless steel screen cloth vibrating screens for washing. The undersize of the screens is pumped to an electromagnetic vibrating screen in the fine diamond section and is followed by diamond recovery on two grease tables.

The oversize of the two 3 ft. by 6 ft. concentrate wash screens is conveyed by 18 in. inclined conveyor to two 100 ton storage bins. Electromagnetic vibrating feeders, 18 in. by 30 in. trays, feed the concentrates from below the bins to an 18 in. inclined conveyor which delivers them to a 5 ft. by 10 ft. double deck Allis Chalmers low-head vibrating screen, $\frac{1}{4}$ in. mesh high carbon steel top deck, and 10 mesh stainless steel lower deck. The $\frac{1}{4}$ in. mesh on the top deck is a relieving screen, thorough washing being carried out on the 10 mesh screen cloth by sprays fitted on the first half of the screens. The second half of the screen acts as a draining section so that the product leaving the screen has a moisture content of less than 10 per cent. The undersize from the screen is pumped to the fine diamond section for treatment on the tables taking the undersize of the concentrates wash screens. The oversize of the screen is led to the 7 ft. diameter heavy media separation cone (Fig. 16).

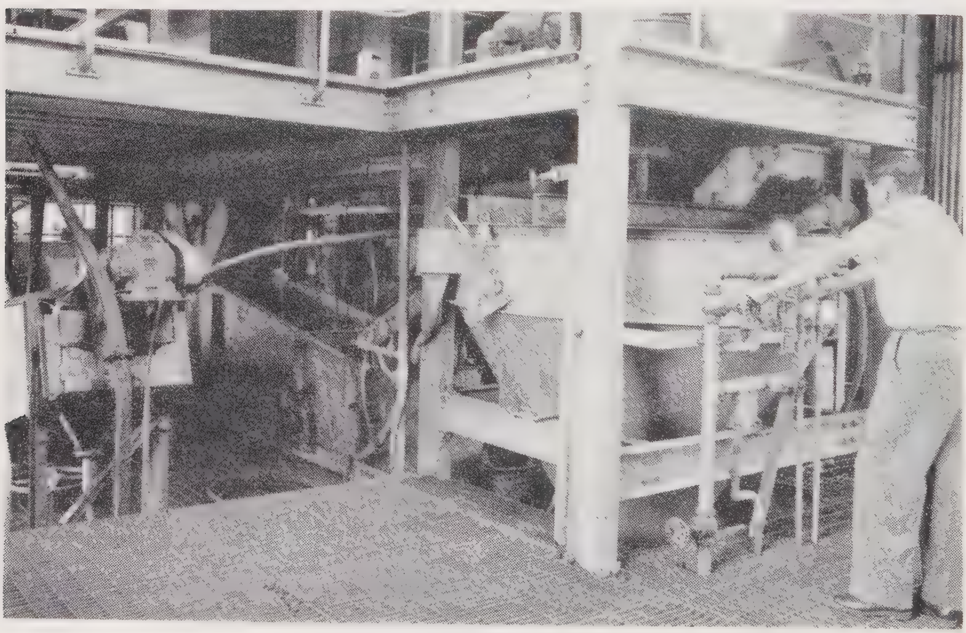

Fig. 16

MAIN RECOVERY PLANT

The main recovery plant operates on one shift only, during which 320 tons of plus 10 mesh concentrates are treated at a rate of 45 tons dry weight per hour. The cone is charged and dropped at the beginning and end of each shift, the time taken for each operation being half-an-hour.

Heavy media separation

The cone is operated at 2·85 density, top, and 3·12 density, bottom, using 65 D ground ferro-silicon, and produces a 20 per cent sink fraction. The float product is drained and washed on a 3 ft. by 16 ft. double deck Allis Chalmers low-head vibrating screen, $\frac{1}{4}$ in. mesh top relieving screen and 10 mesh stainless steel 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. by 10 ft. single deck Allis Chalmers low-head 10 mesh vibrating screen. The first portion of each screen drains the medium from the feed, while sprays fitted on the second portion of each screen thoroughly wash the product to remove any ferro-silicon adhering to the gravels.

The medium drains from each screen into a hopper and gravitates to a 3 in. metal lined Wilfley pump which returns it to the centre of the cone. The washed product from the washing section of each screen is gravitated to a 10 ft. Dorr thickener for thickening before the ferro-silicon recovery circuit. The washed product passing through each screen is received in a horizontally sliding chute which can be positioned under each screen to take any desired cut from the drained medium and divert it to the ferro-silicon recovery circuit with the washed product. The delivery chute to the thickener passes through an Alnico permanent magnet to magnetize ferro-silicon present, flocculating it for easier settling in the thickener.

The thickener underflow is pumped by a $1\frac{1}{2}$ in. E.R.E. metal lined pump to a 24 in. wide wet type belt electromagnetic separator. Here, clean ferro-silicon is removed from slime which is discarded to the fine diamond section. The ferrosilicon, with water, is gravitated to a densifier in the form of a 2 ft. Akins classifier for de-watering and is then admitted to the cone via the medium return pump, first passing through a de-magnetizing coil for dispersal.

Revolving rubber scrapers keep the sides of the cone free from settled ferro-silicon. These scrapers are carried on cross arms from a central shaft which is rotated.

Medium return to the cone is by way of a cylindrical vessel welded round the scraper shaft at the surface level of the cone. Four pipes of different lengths pass downwards from the bottom of this vessel and admit the medium at different depths in the cone. Adjustment of the lengths of these pipes effects the control of rising or downward currents in the cone.

Control of depth of overflow level of medium from the cone is by adjustment of the width of weir, and balance of the return medium with overflow and underflow is effected by adjusting the cut from the medium drained on the screen to the ferro-silicon recovery circuit.

Control of medium density in the cone is by increasing or decreasing the amount of ferro-silicon delivered from the densifier.

The float product is delivered to an 18 in. tailings conveyor belt and the sink product to two 10 ton surge bins.

One hundred and eighty tons per hour of medium is circulated through the cone, containing 137 tons of ferro-silicon, although the cone only holds eight tons of ferro-silicon in suspension in the medium. Ten tons per hour of ferro-silicon is sent to the cleaning circuit. Ferro-silicon consumption taken over the last year amounted to 0·29 lb per ton of cone feed.

Compressed air at 65 lb/sq.in. for the air lift is obtained from the mine supply, but a 180 cu.ft. per minute air compressor is kept as a standby.

Good operating results are obtained running at 2·85 density, top-of-cone, and viscosity of medium which is indicated continuously on a Meyer Viscosimeter remains steady at 19-20 Centipoises.

From the 10 ton surge bins the sink concentrate is fed by Syntron vibrating feeders (15 in. by 18 in. trays) to two 8 in. inclined bucket elevators. The elevators discharge on to a 3 ft. by 6 ft. double deck Tyrock sizing screen with $\frac{7}{8}$ in and $\frac{3}{8}$ in. square mesh screen cloth. The concentrate is separated here into three sizes, $-1\frac{1}{4}$ in $+\frac{7}{8}$ in., $-\frac{7}{8}$ in. $+\frac{3}{8}$ in. and $-\frac{3}{8}$ in. $+$ 10 mesh.

Grease tables

Each size gravitates to a 5 ton storage bin and Syntron vibrating feeders (15 in. by 30 in. trays) under these bins deliver the $-1\frac{1}{4}$ in. $+\frac{7}{8}$ in. size direct to a 2 ft. 3 in. by 6 ft. 6 in. side shaking grease table and the other two each to a 3 ft. by 6 ft. 6 in. Vibrex single deck screen with 10 mesh stainless steel screen cloth for final washing before the grease tables. The product from each of these screens gravitates to a vibrating grease table (Fig. 17). Each table has a stand-by table to allow for removal of diamonds and dressing of the grease surface. Tailings from the grease tables are dewatered on static screens and are conveyed by 18 in. belt conveyor to join the float product from the heavy media section, where, with no further treatment, they are conveyed to the washing plant tailings conveyors for disposal to the tailings dump.

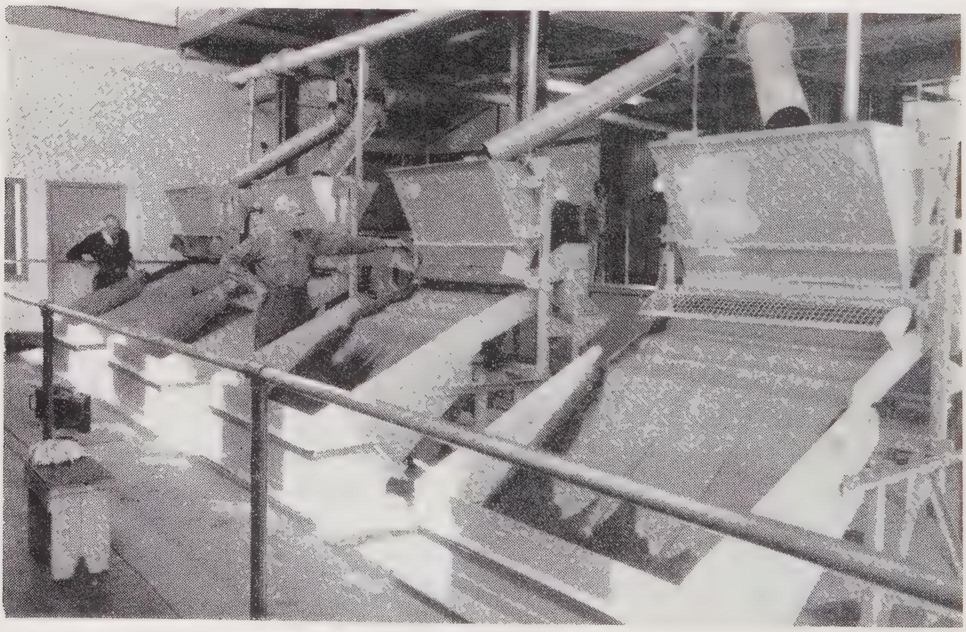

Fig. 17—Vibrating grease tables

Nine tons per hour of sink concentrate is treated on the grease tables, distributed as follows: $-1\frac{1}{4}$ in. $+\frac{7}{8}$ in., 1 ton per hour, $-\frac{7}{8}$ in. $+\frac{3}{8}$ in., 3 tons per hour, and $-\frac{3}{8}$ in. + 10 mesh, 5 tons per hour.

Grease table water amounts to 200 gallons per minute and is circulated through a storage tank below the tables. A 6 in. Gwynne pump is used for this purpose. As the water is kept at a constant temperature of 70°F. summer and winter, the storage tank is provided with steam circulating coils. Steam is raised in a boiler with 750 lb per hour evaporative capacity.

The grease used on the grease tables is a highly refined petrolatum and is purchased in four grades of different penetration point. A different grade is used for each size of concentrate treated, and blending of grades is sometimes resorted to, in order to obtain the correct hardness required for a particular size of concentrate.

The diamonds, which adhere to the grease with about an equal quantity of gangue material, are removed by scraping with a large knife and the surface of the table is re-dressed with a layer of new petrolatum. The scrapings are placed in 33 mesh stainless steel screen cloth covered containers which are then agitated in boiling water. The grease melts away and the containers are taken to the sorting office where the contents are delivered for final cleaning (Fig. 18). An amount of 1,250 lb of petrolatum is used monthly.

Final cleaning

For the final cleaning, the grease table concentrates are dried and sieved into two sizes, plus and minus ten mesh. The gangue is easily removed from the plus ten mesh diamonds by hand, but with the minus ten mesh size, further treatment is required before final checking by hand. This treatment consists of wet differential milling in

Fig. 18—Sorting office

a 6 in. diam. by 10 in. long ball mill run at 75 revolutions per minute, using $\frac{5}{8}$ in. diameter steel balls, followed by de-sliming on a 48 mesh screen, and de-greasing in a chromic acid bath. Thereafter a process of skin flotation of the diamonds from the gangue in dilute hydrochloric acid is adopted, or after drying, an electrostatic separation in an 8 in. long roll laboratory separator, or by both processes.

The daily production of diamonds amounts to 2,800 carats or 1·25 lb weight. This amount of diamonds is recovered from 13,000 tons of kimberlite so that the reduction is approximately 20 million to one.

Summary

In the main recovery plant, the 7 ft. diameter heavy media separation cone has replaced 40 double compartment 1 ft. 9 in. by 6 ft. 6 in. Harz jigs of the pulsator, and three, two vibrating and one side shaking, grease tables have replaced 51 of the old type.

The recovery plant has effected a reduction in labour over the Pulsator complement of 65 per cent in skilled and unskilled European and native labour. The working costs have been reduced by 50 per cent.

The present staff comprises a manager, 10 Europeans and 16 Natives.

In conclusion, the authors wish to thank the Consulting Engineer and Consulting Metallurgist, Anglo American Corporation of South Africa, Limited, and the General Manager, De Beers Consolidated Mines, Ltd., for permission to publish this paper.

Contribution to discussion

W. G. Stockden (Visitor): Mr Colvin and Mr Simpson have given us a most interesting paper, "Treatment and recovery practice at the Kimberley mines of De Beers Consolidated Mines, Limited," and I have great pleasure in making a contribution to it.

From a metallurgical viewpoint it was most enlightening to hear about the old methods of conveying the kimberlite from the mines to the floors and washing plants by endless rope haulages, the crude treatment of the blue ground in the washing and recovery processes, and the large labour force required at great cost and security risk for the recovery of diamonds.

They have described in detail the design of the new plant in Kimberley and have proved, with modern machinery and technique, the extent to which labour and costs can be reduced.

I would like to elaborate on certain of the more important improvements incorporated in the new plant in Kimberley, which have proved most successful both in saving of labour and the cutting of costs to a minimum, with improved efficiency:—

(1) The installation of continuous extractors on the washing pans has proved to be a most efficient means of extracting concentrates and has also reduced the risk of theft of diamonds.
Previously, the concentrates were tapped from the pans by manual opening of a shutter over the extract hole at two hourly intervals, with the result that diamonds were retained in the pans for periods up to this time, with the risk of the finer diamonds being accidentally floated out. With contin-

uous extractors, however, diamonds entering the pans make two to three revolutions of the pan in a matter of minutes before being extracted, with greatly reduced risk of loss.

The Jagersfontein mine plant has retained manual extracting from the pans, but fortunately the low yield of finer diamonds, 0·50 carats per 100 loads from this mine, minimizes the risk of loss of values by this method of extracting.

(2) Deeptroughing conveyors for the conveyance of concentrates from the pans to the wash screens and so to the recovery plant are completely enclosed with wire mesh for the prevention of theft. Maintenance on these conveyors is negligible in comparison with the old type scraper conveyors, which are still in operation at Jagersfontein, and which require frequent attention to prevent breakdowns during the shift, and are most expensive to maintain and operate. Power consumption to these units is exceptionally high for the number of loads transported.

(3) The re-crush section, consisting of $5\frac{1}{2}$ ft. short head Symons crushers is an improvement of major importance. When treating kimberlite with a high content of fine diamonds, as is the case with both Wesselton and Bultfontein mines, it is essential that the product from the re-crush section be reduced to the required size for the release of fine diamonds. In the older plants operating with rolls, which are not in closed circuit, it is most difficult and costly to crush to a uniform size.

(4) I would like to mention here the importance of mechanization in the handling of heavy equipment, particularly where maintenance is concerned. In the past the chain block was frequently used but now where tonnages have been stepped up, and it is necessary to keep all available units in operation, the electric hoists, travelling on monorails which have been installed in the new plant at Kimberley, have proved a great asset. They enable rapid maintenance jobs to be carried out during the shift without loss of tally and in a much more expeditious manner.

The changing of a set of rolls on the older plants took two to three days with a large labour force, whereas now on the new plant in Kimberley, where the $5\frac{1}{2}$ ft. Symons crushers have replaced the rolls, a complete crusher can be changed comfortably in one shift. In this connection nine sets of rolls were previously required to do the work now done by three Symons crushers.

An important innovation at Jagersfontein has been the installation of an underground crusher, with the result that finer ground is now being hoisted and many minor problems and stoppages have been overcome:—

(*a*) Battering of stockpile and headgear bin chutes by large lumps causing frequent repairs to liners, has been reduced to a minimum.

(*b*) Fewer repairs to conveyor belts are required when handling finer material, and longer life is ensured.

(*c*) In the past many stoppages were experienced by large lumps choking the stock pile chutes and blasting had to be resorted to. These stoppages caused long delays which prevented the stockpile being emptied by the end of the shift. Under present operating conditions a constant and even flow of ground is maintained throughout the shift and consequently a higher tonnage is being treated in the same number of operating hours.

Thorough investigations were carried out and tests conducted on many of the old concentrates and tailings dumps surrounding the Jagersfontein mine. Many of these dumps proved payable and it was decided to feed this material through the plant at the rate of 700 loads per day, together with the run-of-mine ground. Owing to the friability of kimberlite these old tailings have completely broken down, and it was felt that many fine diamonds could be recovered from this source. Tests were initially conducted with an 8 ft. tertiary pan treating the wet spigot discharge from one hydrocyclone. Owing to the encouraging results obtained from these tests a 14 ft. tertiary pan has recently been installed to treat the wet spigot discharge from all four hydrocyclones. The increased production is estimated at being 1,000 to 1,500 carats per month of −9 mesh diamonds.

For correct operation of the tertiary pans it is necessary that the feed should be slime free, and to obtain this feed from the spigots of the 42 in. hydrocyclones, Sala valves are used on the underflow discharge.

The valves, made by the Sala Company of Sweden are 4 in. internal diameter rubber cylinders carried in cast iron housings. Hydraulic pressure is applied to the outside walls of the cylinders, deforming the rubber, and partially or completely closing the valve depending on the pressure applied.

By regulating these valves according to the percentage solids in the puddle circulating through the hydrocyclones a reasonably dry rope discharge can be maintained at all times. The discharge is slime free, the slime overflowing the hydrocyclones for use as circulating puddle in the primary and secondary pans.

Fig. 1 is a general view of a 42 in. hydrocyclone, while Fig. 2 is a close-up view of a Sala valve.

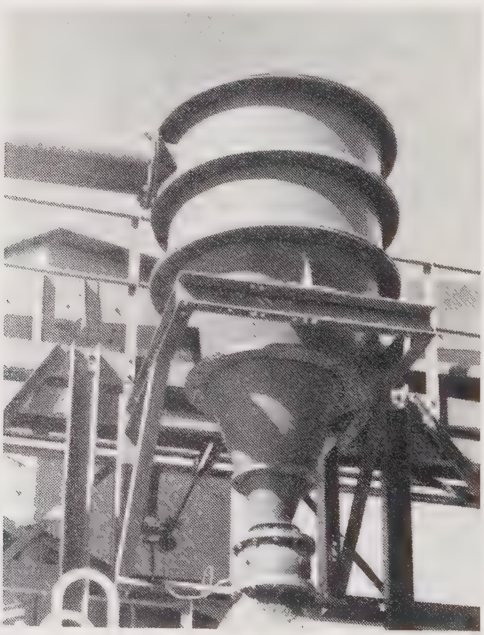

Fig. 1

Tests conducted on the old tailings dumps also indicated that, due to exposure, many of the diamonds had become refractory, and would not adhere to the grease tables. To avoid the loss of the larger stones, Harz jigs have been installed in the recovery plant to treat the $-1\frac{1}{4}$ in. $+\frac{5}{8}$ in. and $-\frac{5}{8}$ in. $+\frac{3}{8}$ in. materials. The jig concentrate, which is approximately 20 per cent. of the feed, is hand sorted. Hand sorting of these quantities is not completely satisfactory and therefore a conditioner and grease belt is being installed to handle the concentrate. This should be in operation in the near future.

An interesting experiment was carried out on the recovery of refractory diamonds by means of an optical separator. Unfortunately, at its present state of development, it did not appear to be suitable to treat the Jagersfontein concentrates particularly as, on the one hand, it was being triggered off by light-coloured particles in the concentrate, whereas certain badly discoloured diamonds passed through the light trap without affecting the unit. This problem is, however, being worked on by the Diamond Research Laboratory.

In conclusion I would like to congratulate the authors, both on their paper and the hard work they must have put in at De Beers, in bringing the new plant into production, and express my pleasure at being invited to submit a contribution to this valuable paper.

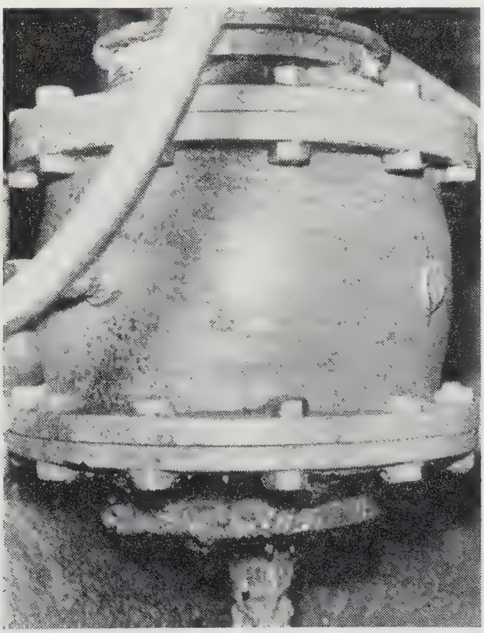

Fig. 2

NATIVE ADMINISTRATION IN THE KIMBERLEY DIAMOND MINES

By W. S. Gallagher*, J. S. Sandilands† and T. V. Howell‡ (Visitors)

The housing of African labour on the diamond mines dates back to 1886 when the "closed" compound system was introduced by "Matabele" Thompson in an effort to eradicate the illegal trading between the African mineworkers and the illicit diamond buyers.

Although the foundations of this system have remained the same throughout the years, the whole attitude towards the housing and administration of our African employees has undergone radical changes with modern methods and ideas being used to their full advantage.

TYPE OF LABOUR

To-day, due to improved techniques and mechanization in the mines and surface plants, a labour force of only 3,050 is required to meet our needs. These 3,000 Africans are voluntarily engaged through the Labour Bureau of the Bantu Affairs Department. Fifty-six per cent of them come from Basutoland, 23 per cent from Bechuanaland and the Northern Cape, 8 per cent from the Transkei, and 3 per cent from Natal, while 10 per cent are local detribalised Africans. This trend will change with the advent of further mechanization when it is likely that a smaller labour force will be required, and when more use can be made of local or detribalised labour whose physical standards are lower than those of the tribal African.

METHOD OF ENGAGEMENT

Because a certain number of our African employees complete their contracts and return home each day, the constant engagement of African workers is necessary to replace this wastage or loss of labour and so keep the labour force at the required strength.

The engagement of the African workers, at an approximate rate of 20 per day, is the responsibility of the training officer, who is advised of wastage requirements according to job category by the hostels. Generally there are more Africans seeking work than we can employ and 95 per cent of all available labour has had previous mining experience. Seventy-five per cent of the Africans seeking employment are old employees of the De Beers Company. Due regard is paid to those old employees requiring sheltered employment.

All ex-employees are in possession of a discharge card which reflects:—

(1) Mine record number

(2) Name

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*General Manager.

†Native Administration Officer, Wesselton Hostel.

‡Native Administration Officer, Dutoitspan Hostel.

(3) Aptitude rating
(4) National identity number
(5) Previous type of work
(6) Character
(7) Medical category.

In addition, specialists are in possession of a pink card which guarantees immediate re-employment. The high percentage of specialists who returned in 1959 is indicated by the following figures:—

Job category	Wastage	Number trained	Per cent trained
Boss boys	210	30	14·3
Machine boys	244	25	10·2
Pipes and tracks	91	48	52·7
Loco drivers	50	42	84·0
Winch drivers	51	49	96·0

The two latter categories include revisionary training and it is expected that a higher percentage of loco drivers will return to the company once the quotas for block cave are standardized. All the above categories comprise wastage due to completion of contracts, pension, black listing and reclassification on medical grounds.

DOCUMENTATION AND CLASSIFICATION

The previous records of re-engaged ex-employees are checked against new attestations and are brought up to date, while new medical records are compiled following radiological and medical examinations. Record cards are compiled for all new employees and all Africans, not previously tested, are sent to the aptitude centre before proceeding to the labour reserve pool at the hostel.

APTITUDE TESTING

The aptitude tests are carried out by the training officer or his deputy. Performance type tests, designed for group administration, were introduced in 1956 to co-ordinate occupational classification of Africans with a very low standard of education. General intelligence, mechanical skill and supervisory ability are measured. The men are then allocated to jobs previously evaluated according to their aptitude rating. Job preference and a follow-up programme form an integral part of the whole system.

Educational tests have also been introduced in order to utilize the African with a minimum of Standard Six School Certificate.

It has been found that detribalised Africans reveal a higher standard of intelligence and mechanical ability, but lack leadership and physical capabilities. Sixty-five per cent of all Africans have proved mechanical ability but only 7 per cent qualify for supervisory training.

LABOUR RESERVE POOL

While awaiting job allocation, all newly engaged Africans go to what is known as the labour reserve pool. Generally they are allocated to jobs within a few days, but in the case of trained specialists the period of waiting may be slightly longer. During this period the men are fed and housed and are given induction talks on personal hygiene, safety, hostel discipline and routine.

HOSTELS AND THEIR AMENITIES

Initially the African mineworkers sign a contract to work on the mines for a period of six months. The majority renew their contracts immediately they expire, the average length of stay being nine months and the maximum permissable length of stay 14 months. All these Africans live in the hostels and proceed directly to and from their place of work. They are only allowed out of the mining area on completion of their term of service after being screened and searched.

The two hostels at Kimberley, namely Dutoitspan and Wesselton, have recently undergone complete renovation. Each of these hostels can house 1,750 men. They have been built on a rectangular plan with the main buildings occupying the borders of the rectangle which covers several acres. The large space at the centre is occupied by lawns, gardens, a swimming pool, skittle alleys, sports fields, an open-air cinema, cooking and other facilities (see Fig. 1).

The bedrooms, which are fitted with double-tier beds, steel lockers, tables and benches, accommodate between 16 and 20 men and are conveniently situated in relation to the wash and shower rooms, all of which are suitably equipped.

Fig. 1—The Administration Block at Dutoitspan Hostel with the mine headgear in the background

SELF-FEEDING

Since the inception of the hostels on the diamond mines, the African mineworkers have preferred to feed themselves. Because self-feeding is the order of the day, all wages are paid in full and nothing in kind is provided. However, certain basic foodstuffs are heavily subsidized by the Company and are consequently sold at greatly reduced prices. The aggregate amount spent by each man per month on foodstuffs is £2 6s. 6d., the average daily calorie intake being 3,500. All the amenities for the cooking of food are provided and the stoves are kept alight by hostel employees. In addition, cooked fish and chips are sold and a free issue of fortified soup is available each day, while $2\frac{1}{2}$ lb of fresh vegetables are supplied to each man weekly.

Each hostel has a butcher shop, general store, soup kitchen, fish shop and cafe, so that everything the Africans may require can be bought on the premises (see Fig. 2).

Fig. 2—The general store at Wesselton Hostel

MAGOU

Magou, a beverage made from mealie meal and unsifted flour, is made and sold in the hostels at a nominal price, the profits therefrom being paid into a benevolent fund. Kaffir beer has as yet not been supplied, but this question remains an open one. Consensus of opinion taken from time to time indicates that beer is not recommended. Illicit brewing does, however, take place but active measures are taken to suppress it.

WELFARE AND RECREATION

Since the African spends all his leisure time within the hostel precincts, welfare and recreation are important functions of hostel administration. A European official is in charge of a small, trained, African staff who conduct classes in woodwork, tailoring, reading and writing, cobbling and handicrafts (see Figs. 3 and 4). Film shows are put on once a week and soccer matches and other competitive sports, as well as talent contests, are arranged.

Fig. 3—A hostel schoolroom

Each hostel has its own chapel, which is inter-denominational, and church services are conducted on Sundays, while lay preachers amongst the hostel inmates conduct services during the week.

A special aspect of hostel administration on the diamond mines is that, because of the necessity for stringent security measures, European supervision is required throughout the 24 hours of the day.

MANNING AND JOB ALLOCATION

Besides providing suitable and congenial living quarters for the Africans, it is the duty of the hostel administrators to ensure that the necessary labour is available for the mines and surface plants. In order to utilize our African labour force with the maximum efficiency a visual labour control system is in operation. Colour codes and flags are used to show the job card, workers card, medical category, educational and training standard of each African. Absentees and specialists who are surplus to the establishment are also indicated as well as total working strength and gang strength (see Figs. 5 and 6).

Fig. 4 – The tailoring school in the hostel

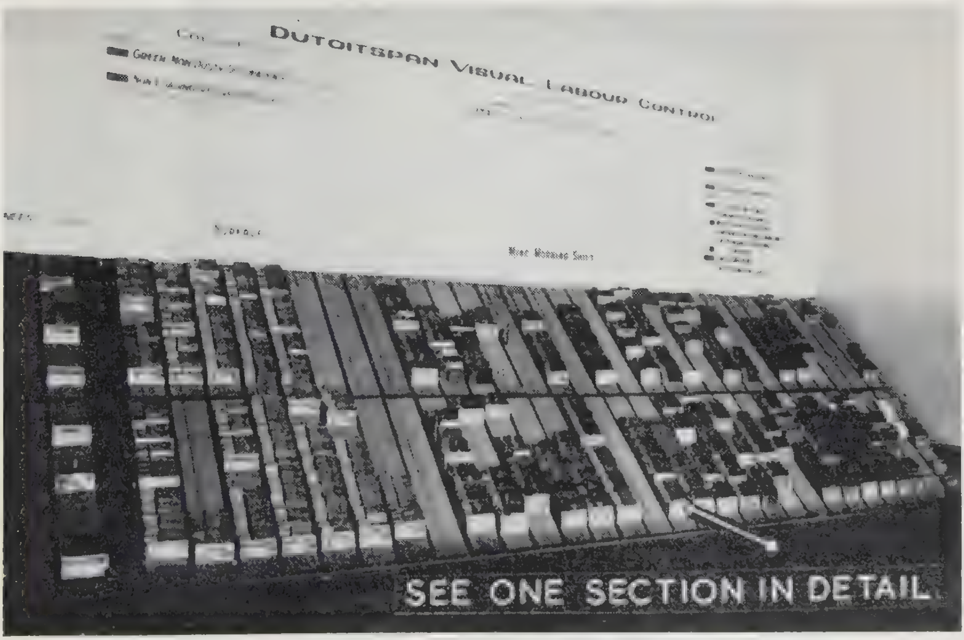


Fig. 5—The visual labour control at Dutoitspan Hostel

The labour strength for the various jobs or projects is laid down by the study department and no increase or decrease in labour strength may be made unless authorized by this department in writing. Any wastage is replaced from the reserve pool as soon as possible.

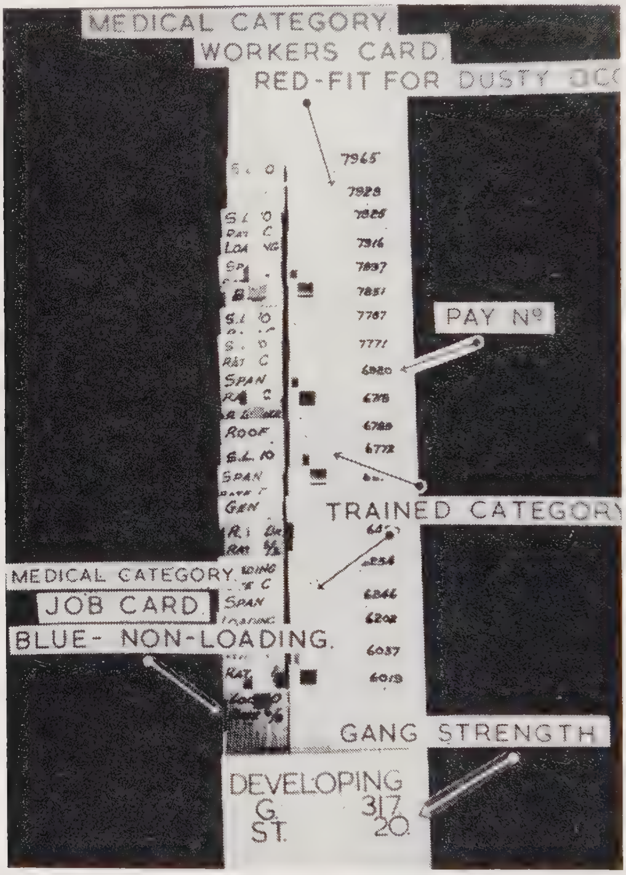


Fig. 6—A close up view of a section of visual labour control

All Africans are allocated to jobs according to the evaluation of that job, the suitability of their qualifications to the job and their state of health. Trained specialists, when required, are allocated from training gangs or from the reserve pool if available. Non-specialists are allocated to jobs according to the manning sheet.

On re-engagement all previously qualified boss boys are given some revisionary training which ranges from three days to two weeks in the case of boss boys with three years intermittent service. Following this they are either placed as boss boys or are allocated to another job, preferably machines, and are flagged on the visual control so that they may assume boss boy's duties as soon as a vacancy occurs. Similarly all trained specialists, who are not immediately placed in the job for which they have been trained are specially flagged for correct job allocation at the earliest opportunity. Trained gold mine labourers are also flagged on the section of the visual control set aside for this purpose.

No African may be transferred from one working gang to another without the consent of the study department except when the labour complement of a gang is changed. In the event of transfer of a trained specialist, reference must be made to the training officer.

This visual control system is extremely simple to operate. It requires no previous knowledge to be able to deduce from it all the required facts relating to labour requirements, availability of labour, shortages and surpluses. It ensures that our African labour is used to the best advantage and greatly assists in the selective engagement of our African employees.

TRAINING

Because of the high percentage of ex-employees who return to us for further periods of service, it is to our advantage to train as many African mineworkers as possible. Provision is, therefore, made for the training of Africans in all work categories. Courses are given for winch drivers, jack hammers, pipes and tracks, pipe-threading machines, timbering, electrical and diesel locomotives.

All training is the direct responsibility of the training officer who works in close liaison with the Native administration officers and attends all meetings appertaining to African labour. He is also responsible for the engagement of the Africans, manning and aptitude-testing.

The training officer is in a position to know, three weeks in advance, what the requirements for trained Africans will be due to wastage through discharges, etc. Normally a high percentage of previously trained specialists, who need only a short period of revisionary training, are available from the reserve pool. However, during the planting and reaping seasons a shortage of these specialists is common and during this period it is necessary to provide full training courses. Some over-training is done to ease the position at this time so that trained men, not previously allocated to specialist jobs, may assume the duties of trained men who seek discharge.

Voluntary candidates, suitable for supervisory training, are earmarked and are allocated to as many different jobs as possible during which time their progress is closely followed before the final selection of boss boys is made. This supervisory training lasts 30 days, 23 days being spent in underground and job training, five days on man management and two days being tested.

The duration of the training courses, which are held at the underground training levels, is as follows:—

Loco drivers	10 shifts
Winch drivers	7 shifts
Machine boys	10 shifts
Pipes and tracks	12 shifts
Timbering	6 shifts
Pipe-threading machine	7 shifts

Training in timbering is always given on the job and all trainees for pipe-threading machines are chosen from men trained in pipes and tracks. However, excluding supervisory training, the period of training required in the above categories is flexible due to the large numbers of certificated workers from the gold mines who desire employment in the diamond mines.

The examination and testing of trainees are carried out by the department concerned and not by the training officer.

In training, extensive use is made of audio-visual methods of instruction. The use of colour slides showing job practice is current and a start has been made by our photographic department in compiling 16 mm. training films on certain aspects of diamond mining, e.g. block cave.

PAY, COSTING AND TIMEKEEPING

For pay purposes the Africans fall into two groups: those on a fixed daily wage and those who are on contract, their daily wage varying in proportion to the amount of work done. Africans on a fixed rate are paid on the "absentee system," i.e. they are assumed to be at work and are paid accordingly unless they are reported to be absent by the hostels. All Africans on contract are paid by positive booking. The supervisor returns a ticket recording earnings which are positively entered on earnings cards. The hostel labour supervisor is responsible for advising the time office of all variations in rates of pay, absentees picked up in the crush, etc.

All overtime must be filled in on a prescribed form and countersigned by the head of the section. The hostel is responsible for answering any queries which can be checked against a duplicate copy and colour chart on which different colour pencil marks indicate the time at which the African returned through the crush.

INCREMENTS, BONUSES AND PENSIONS

Service increments are paid on the aggregate number of years an African has worked for the Company. Those with over three years service receive an increment of 3d. per shift. Those with six years service receive 6d. per shift, and those with ten years service receive 1s. per shift. Furthermore, all Africans in possession of a valid first-aid certificate are paid a bonus of 3d. per shift and any African finding and handing in a diamond receives a bonus of up to £500, a larger reward being paid at the discretion of the Board of Directors.

A monthly pension is paid to all old employees who have completed a qualifying period of service. On their death their widows receive half this amount.

HEALTH AND ACCIDENT STATISTICS

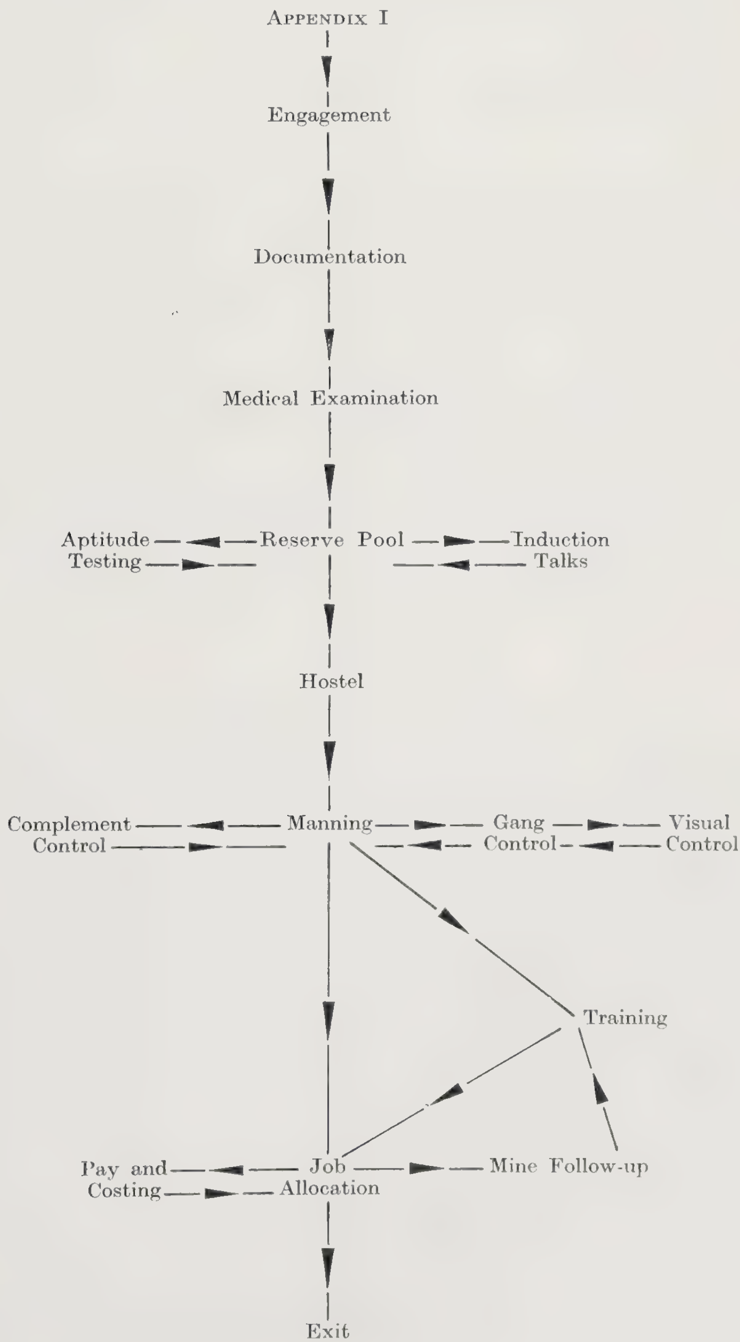
Effective health measures can be more fully enforced under our hostel system, since close supervision over the Africans is ensured for six months at least. Selective engagement and job allocation according to physical fitness allow for the maintenance of a high standard of good health especially as all applicants are given a thorough medical examination and X-ray and are inoculated against typhoid and pneumonia before they enter the hostels. Intensive training and instruction in safety help to reduce the morbidity rate and the number of shifts lost due to accidents and injury.

The following comparison in the morbidity rates and the number of shifts lost in the Kimberley diamond mines and the gold mines of the Anglo American Group of Companies is given. (See tables, p. 502).

The De Beers rate for the number of shifts lost through disease during 1956, 1957 and 1958 shows an increase due to the influenza epidemic.

During the period 1955 to 1959 only 15 cases of tuberculosis without pneumoconiosis, two cases of pneumoconiosis without tuberculosis, and one case of pneumoconiosis with tuberculosis were accepted by the Bureau.

FLOW SHEET OF NATIVE LABOUR ADMINISTRATION



MORBIDITY RATES PER 1,000 AFRICANS PER ANNUM

Year	Mining accidents			Disease		
	De Beers	Reef	O.F.S.	De Beers	Reef	O.F.S.
1954 ...	48	175	273	80	404	494
1955 ...	50	172	228	116	376	481
1956 ...	48	173	214	209	401	479
1957 ...	47	155	168	287	459	528
1958 ...	50	162	179	188	446	484

LOST SHIFTS PER 1,000 AFRICANS PER ANNUM

Year	Mining accidents			Disease		
	De Beers	Reef	O.F.S.	De Beers	Reef	O.F.S.
1954 ...	301	1,179	2,549	616	2,019	2,217
1955 ...	425	1,262	2,495	980	2,236	2,815
1956 ...	305	1,275	2,806	1,591	2,428	3,209
1957 ...	414	1,219	2,499	1,749	2,529	3,601
1958 ...	596	1,203	2,383	1,591	2,542	4,151

DISCHARGE

Three weeks before an African terminates his contract, the training department and records office are notified of his intention by the hostel authorities. All records concerning job category, number of shifts worked, etc., are brought up to date and the African is radiologically screened. He is then issued with his wages, savings bank book and any money he may have handed in for safe keeping. Any complaints are settled and a discharge card together with a pink card in the case of specialists and a white card in the case of non-specialists is handed to him. He is then discharged and escorted out of the mining area, having been instructed on how to go about seeking re-employment with the Company should he desire it.

CONCLUSION

It is our endeavour to maintain as high a standard of administration in our hostels as possible in order to ensure the constant supply of voluntary labour we have been fortunate enough to have up to the present. The only way to do this is to keep in touch with the Africans themselves, for they are the ones who will decide whether we succeed or fail in our efforts to make their stay on the diamond mines both happy and enriching from a cultural as well as a monetary point of view.

MINING PRACTICE AT THE PREMIER DIAMOND MINE

H. F. HODGSON* (Member) and **E. J. B. SEWEL*** (Visitor)

INTRODUCTION

The Premier Mine was discovered through normal field prospecting in 1902 and the company was formed in 1903 (with a capital of £80,000).

The property is 23 miles east of Pretoria and the mine consists of a diamondiferous mass known as kimberlite or blueground, in the form of an elliptical, vertical pipe of volcanic origin, 78 acres in extent, measuring 2,800 ft. by 1,250 ft. There are several pipes in the district but only one of economic importance.

The country rock is felsite in good condition, resulting in a minimum amount of scaling of pipe walls. There is a mass of Waterberg quartzite in the centre of the pipe which, being in the form of a wedge, fades out at depth.

Exploratory drilling and later development have disclosed a non-diamondiferous gabbro sill, 250 ft. thick at 1,310 ft. below the surface, which dips slightly to the north and cuts through the kimberlite pipe and the surrounding country rock.

In this paper, it is intended to give a general description of what could be described as three stages of mining practice at the Premier Mine.

MINING PERIOD 1903-1932

The mining method was confined to open-cast mining in the form of four open cuts serviced by inclined rope haulage traction.

Figs. 1 and 2 depict the start of a mining cut in 1903 and the result of ten years mining.

Fig. 3 is a typical plan view of the layout which was repeated several times as the mining levels automatically deepened with the removal of enormous volumes of blueground. Up to 1932, when the mine closed down, the working level had reached 610 ft. below surface accounting for the removal of 107 million tons of blueground out of a 78 acre hole.

The open cuts were developed by means of step-bench mining, the blasted holes having been initially drilled manually and later by jack hammer machines, drilling dry.

As the cuts opened out, levels were established at 50 ft. vertical intervals, each level having a shelf some 50 ft. wide. These landings were graded to allow 18 in. gauge tracks to come off a high point on the inclined haulage with sufficient gradient for one-ton trucks to gravitate round the cut to a lower collecting point on the incline haulage.

The knowledge and efficient operation of rope haulage traction were of amazingly high standard.

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* General Manager and Assistant General Manager, respectively, Premier (Transvaal) Diamond Mining Company, Limited.

Fig. 1—Start of a mining cut in 1903

Fig. 2—The result of 10 years of mining (approximately 1913)

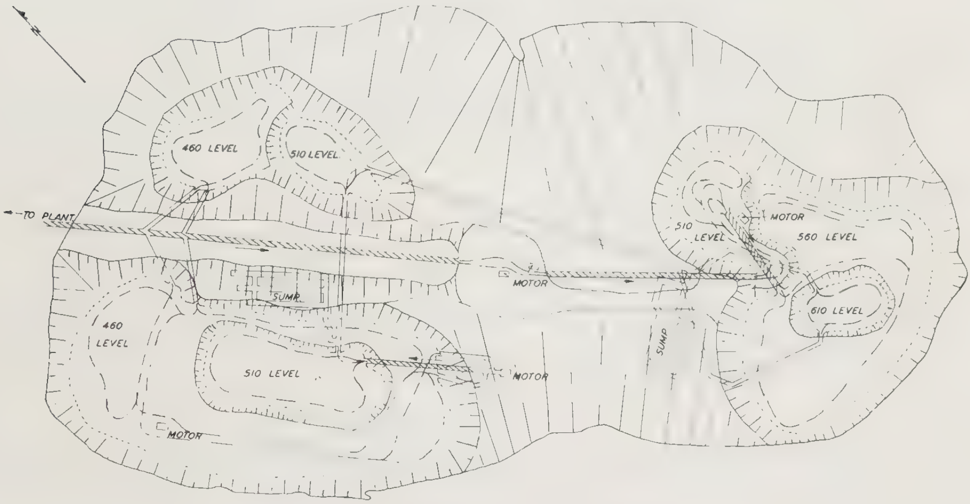

Fig. 3—Schematic plan of old type workings

This can be appreciated when it is stated that an inclined haulage 22 ft. wide, 2,500 ft. long, equipped with four sets of 45 lb track, hauling one-ton cars up a 1 : 20 gradient handled up to 24,000 tons of blueground per day for a total of $8\frac{1}{2}$ million tons in the year 1913.

The following figures of production are interesting:

Year	Tons treated	Carats of diamond recovered
1903	61,500	99,208
Average per year between 1907-1914	$6\frac{3}{4}$ million	2 million
Total to 1932	107 million	29 million
Total to date	148 million	$42\frac{1}{2}$ million

The haulage driving units consisted of 1,000 h.p. Belliss and Morcom, triple expansion, steam-driven engines.

The overall operating cost at this time was $2/10\frac{1}{2}$d. per ton treated.

This system of mining created many problems, such as:

1. Locking up large masses of blueground to carry and protect the elaborate haulage system.
2. Heavy rainfall caused most difficult operation.
3. Danger to personnel from rock falls.
4. Irregular profile, creating a difficult mining problem, when considering underground mining layout.

The latter condition was aggravated by selective mining, a policy forced on the company by the lack of a sales market for the bulk of the industrial and the poorer gem production, but probably more important the low price paid for diamonds up to 1937. The diamond production was made up of 20 per cent gem and 80 per cent industrial. Under the circumstances it was not possible to mine the large areas of mixed rock and blueground.

MINING PERIOD 1945-1963

When it was decided to re-open the mine after 13½ years of stoppage two major problems arose:

1. A method of mining the blueground.
2. A method of treating the blueground.

In the first instance, it was decided to abandon open-cast mining in favour of a complete underground system. Although several different phases of the operation have been described over the last six years, it is felt a general description of the complete operation will be of value.

GENERAL LAYOUT

Fig. 4 is a diagrammatic sketch of the pipe showing the location of the three shafts serving the pipe.

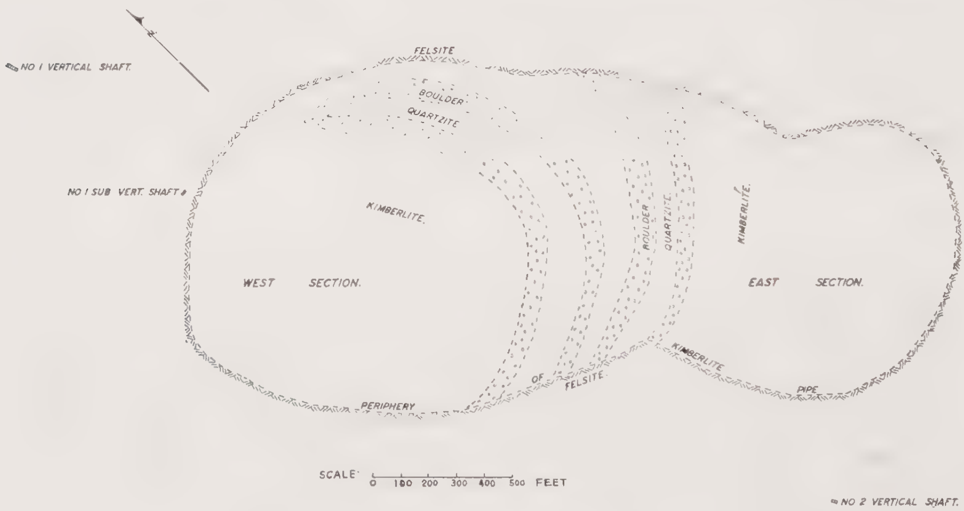

Fig. 4—Diagrammatic layout of pipes and shafts

Figs. 5 and 6 show the mining layout for working the blueground down to the 810 ft. level; also the initial development required to mine out the blueground to the top of the gabbro sill.

It should be noted that the cone, grizzly and transfer level layout on the western section above the 1,060 ft. haulageway differs from the eastern section. This will be explained later.

SHAFT SYSTEM

No. 1 Shaft, situated 800 ft. from the perimeter, is a rectangular five compartment, vertical shaft, equipped with steel buntons and runners, and it is the main downcast ventilation shaft.

The shaft is serviced by two 3,240 h.p. semi-automatic Ward Leonard winders and a small single drum Scott winder.

The winder, serving Nos. 1 and 2 compartments, pulls the mine production. The winder in Nos. 3 and 4 compartments is a dual-purpose unit handling European personnel, material and rock. The man conveyance and rock skip run as a permanent fixture in a common bridle.

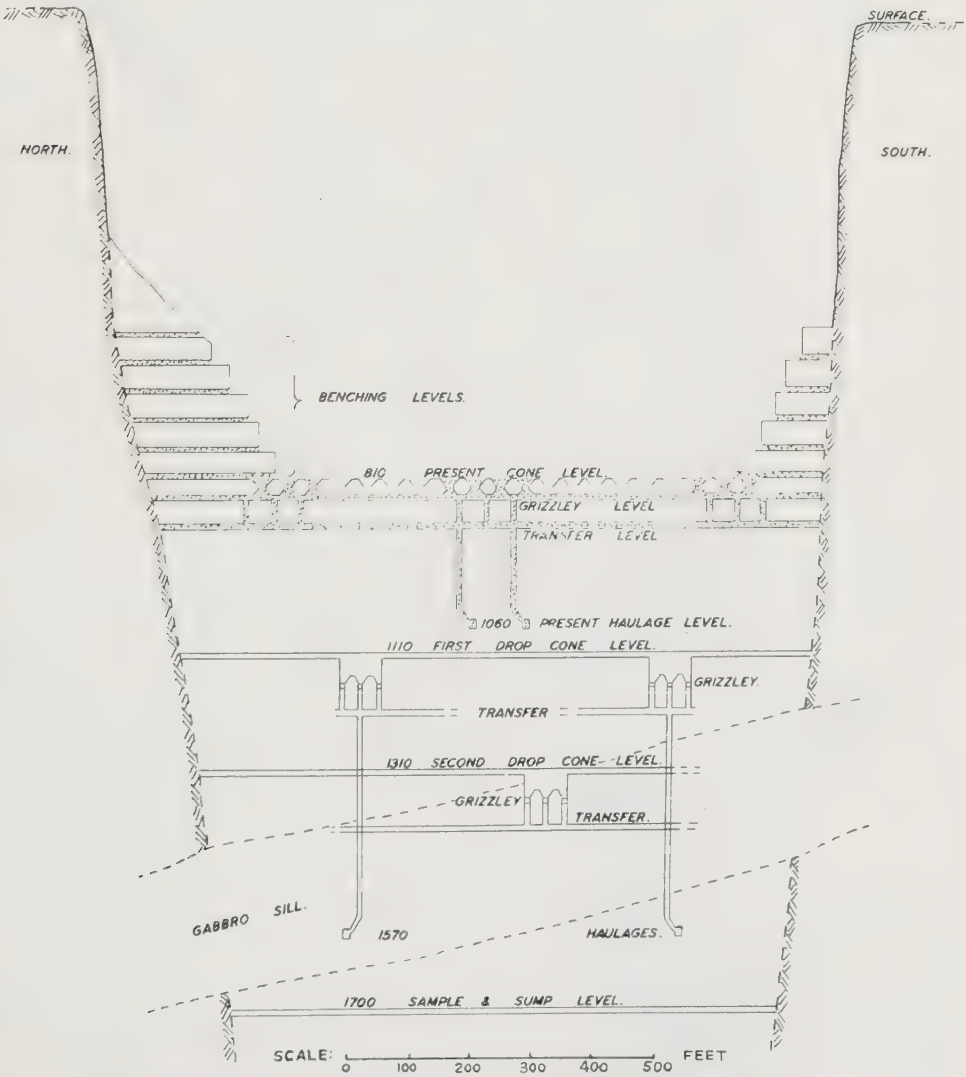

Fig. 5—Diagrammatic cross-section through west mine (initial development shown dotted)

The shaft was originally sunk to 1,352 ft. below the collar and was recently deepened to 1,902 ft. The method of sinking was described by Borchers[1] in 1959.

The shaft is connected with the pipe on three elevations, namely, the 610 ft., 1,060 ft. and 1,570 ft. levels.

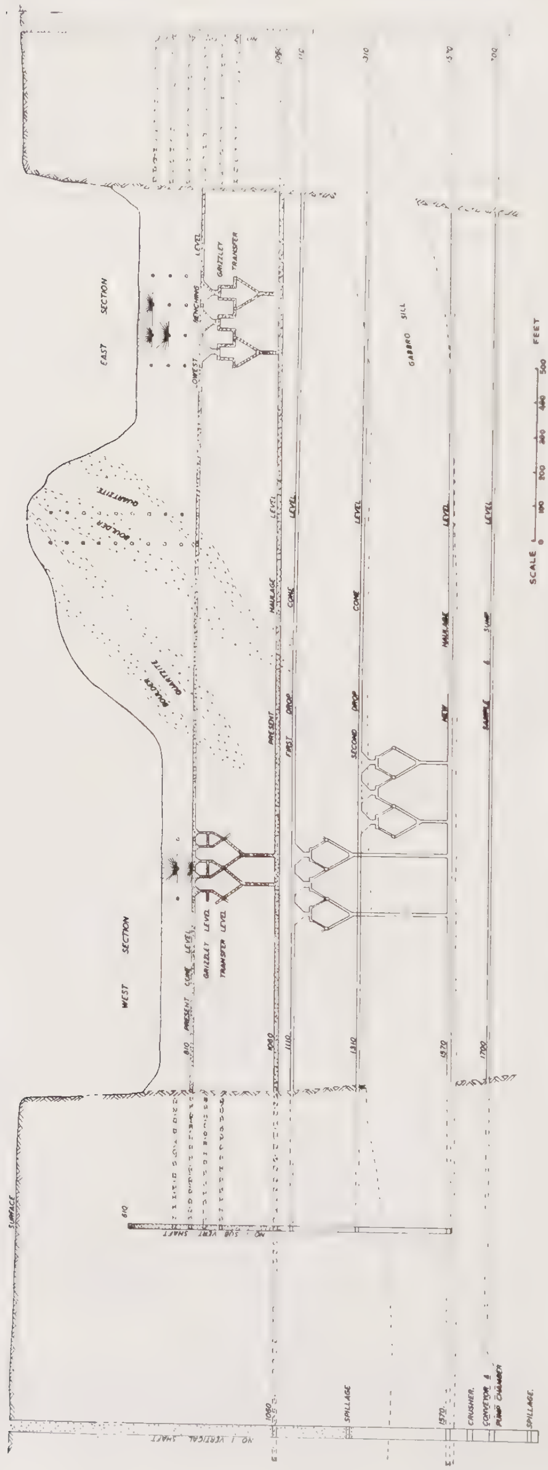

Fig. 6—Diagrammatic longitudinal section of mine (initial development shown dotted)

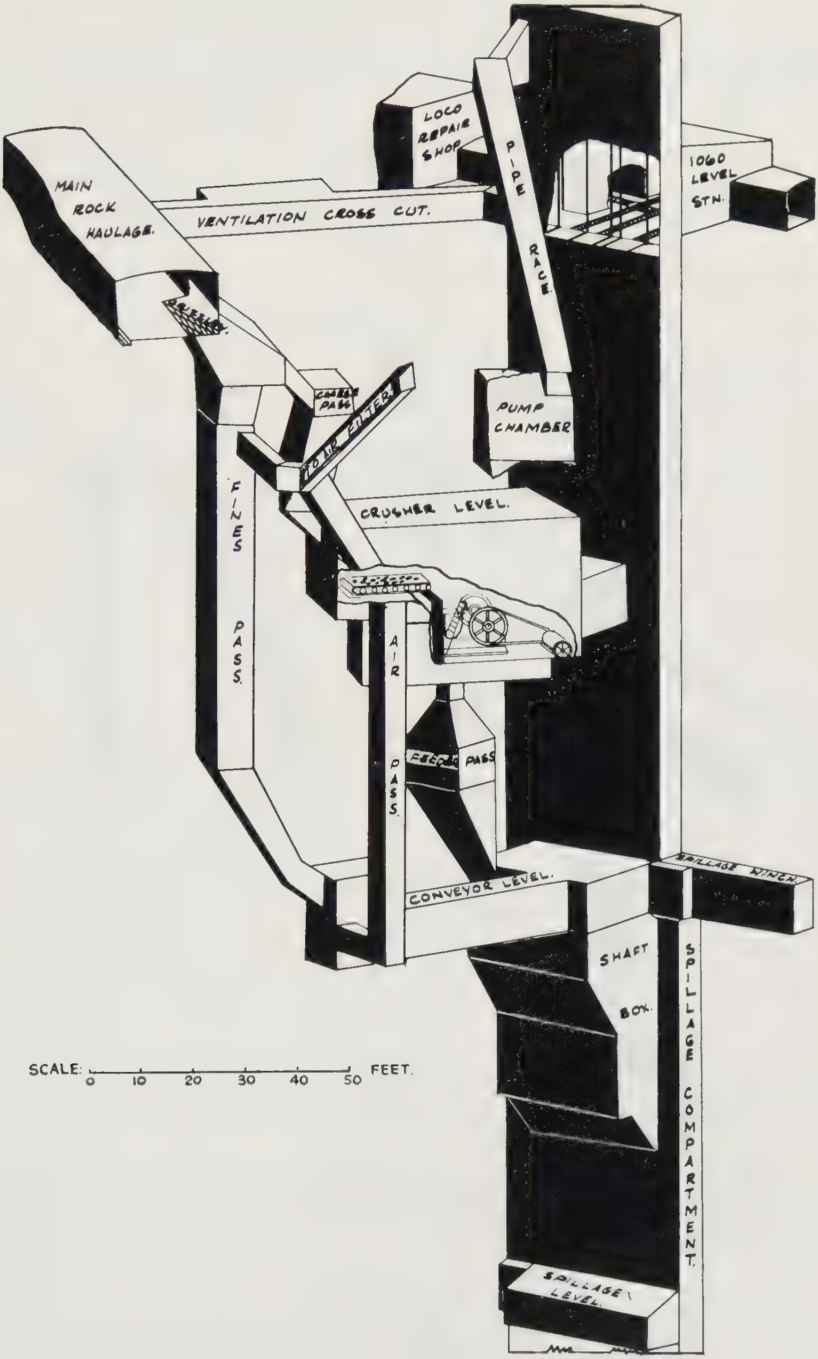

Fig. 7—Three-dimensional view of No. 1 vertical shaft for mining down to 810 ft. level

Fig. 7 is a diagrammatical three-dimensional view of the shaft arrangements used in connection with the mining of the pipe to the 810 ft. level.

Fig. 8 is a similar three-dimensional view of the new shaft arrangements to be used for mining the blueground between 810 ft. level and the top of the gabbro sill.

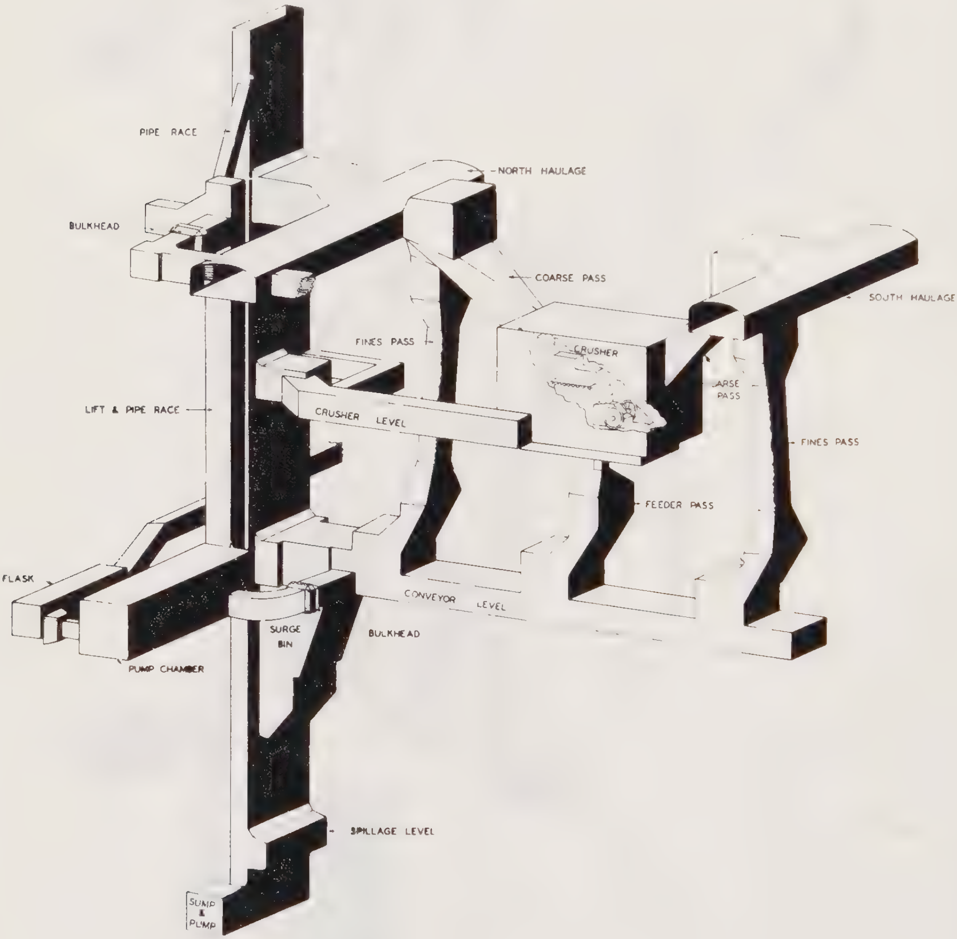

Fig. 8—Three-dimensional view of No. 1 vertical shaft for mining below 810 ft. level

No. 2 Shaft, situated 300 ft. from the perimeter within the compound area, is a rectangular two compartment, vertical shaft, used for hoisting Native labour and material. It is the downcast ventilation shaft for the eastern section. The shaft extends 1,769 ft. below collar and is connected to all working levels.

Sub-vertical shaft, situated 50 ft. from the perimeter, is a rectangular, two compartment, vertical shaft, used as an intake airway and service shaft to all working levels on the western section. The shaft extends between the 610 ft. and 1,570 ft. levels.

It may be convenient at this point to state that all modifications to follow will apply to the 500 ft. section of the pipe now being developed below the working haulage level. This section will represent the third stage production period, namely, 1963 to 1990.

Referring to Figs. 7 and 8 it will be noted that there is a complete modification of the shaft layout, the main features being:

1. Twin haulageways replacing single haulage line between pipe rim and tip (Figs. 9 and 10).

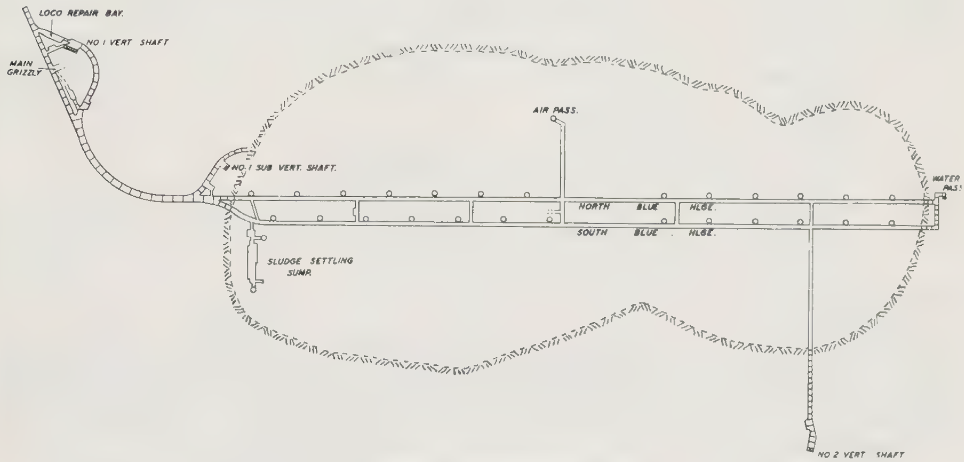

Fig. 9—Plan of 1060 main haulage layout

2. Two grizzly tips and undersize ore passes (—6 in. rock) in place of one.
3. Conveyor belt transfer from ore passes has been duplicated.
4. Four feeder points from ore passes in place of one.
5. Steel surge bin, V-shaped, installed in place of vertical concrete wall. Steel bin floated on rubber pads to facilitate the use of vibrating mechanism at bottom of bin.

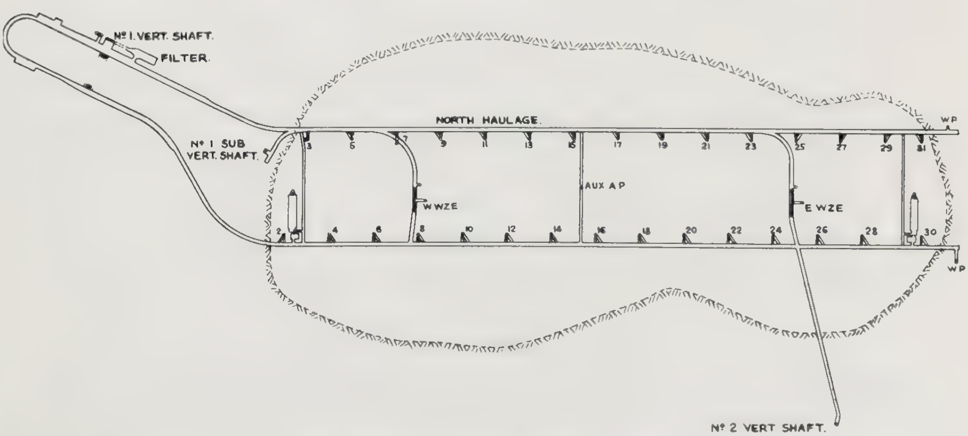

Fig. 10—Plan of 1570 main haulage layout

6. Two 48 in. by 42 in. Allis Chalmers jaw crushers in place of one.
7. Double control doors (air-operated) on surge bin chutes.
8. Shaft bottom spillage, sludge and pumping arrangements.

While on shafts it may be of interest to note a modification in the shaft telephone system. The magneto type system has been replaced by a new type evolved for use in the shafts. The layout consists of an ordinary hand set housed in a box as shown in Fig. 11. The equipment consists of a steel box, 12½ ohm induction coils, terminal board and two push buttons. By pressing a button, the bells (24 volt a.c.) ring on all phones in the system. The other button, when depressed, allows speech assisted by a standard 4½ volt dry cell.

The advantages over the magneto type phone are:

1. Elimination of moving parts.
2. Simple and foolproof.
3. Minimizes faults and requires less maintenance.
4. Saving in cost. A standard type magneto phone costs £50 and requires two-core cable at 1/4½d. per foot, whereas the Premier Mine phone costs £10 and uses four-core cable at 1/6½d. per foot.

Skips

When hoisting takes place from the lower levels it is intended to replace the existing overturning 11·4 ton skips with similar size bottom-dump skips, which will allow the present payload to be maintained at a lower depth. It is believed that the fouling of skips during the wet season will be eliminated. There is a possibility of a lower maintenance cost.

These modifications are the outcome of operational experience, which proved that the handling of 16,000 tons of rock in a day with a single line shaft system is a high-pressure operation, and in the wet season an impossibility.

The main difficulties encountered in the old system were:

1. Haulage delays due to derailment, breakdown blocking track and loss of the time controlling traffic at junction of north and south haulageways.
2. Blocking of grizzly, coarse pass, crusher jaws due mainly to large rocks and wet blueground sticking.
3. Single feed point out of undersize pass feeding a single conveyor transfer. Damp ground caused considerable delay in the hoisting.
4. Lost time in surge bin due to restricted angle at the bottom of the bin causing damp blueground to breach over the chute opening.
5. Single control in chutes loading the measuring flasks accounted for increased decking time, due to the difficulty in ensuring a full load or by occasional overloading.
6. The extra labour required to keep the system fully operative during the five-month wet season.
7. All major repair and maintenance must necessarily be done on overtime.

HAULAGEWAYS

Figs. 9 and 10 represent the old and new underground haulageway layout. The noticeable features of the new system are:

1. The independent haulage and tip serving north and south sides of the mine.

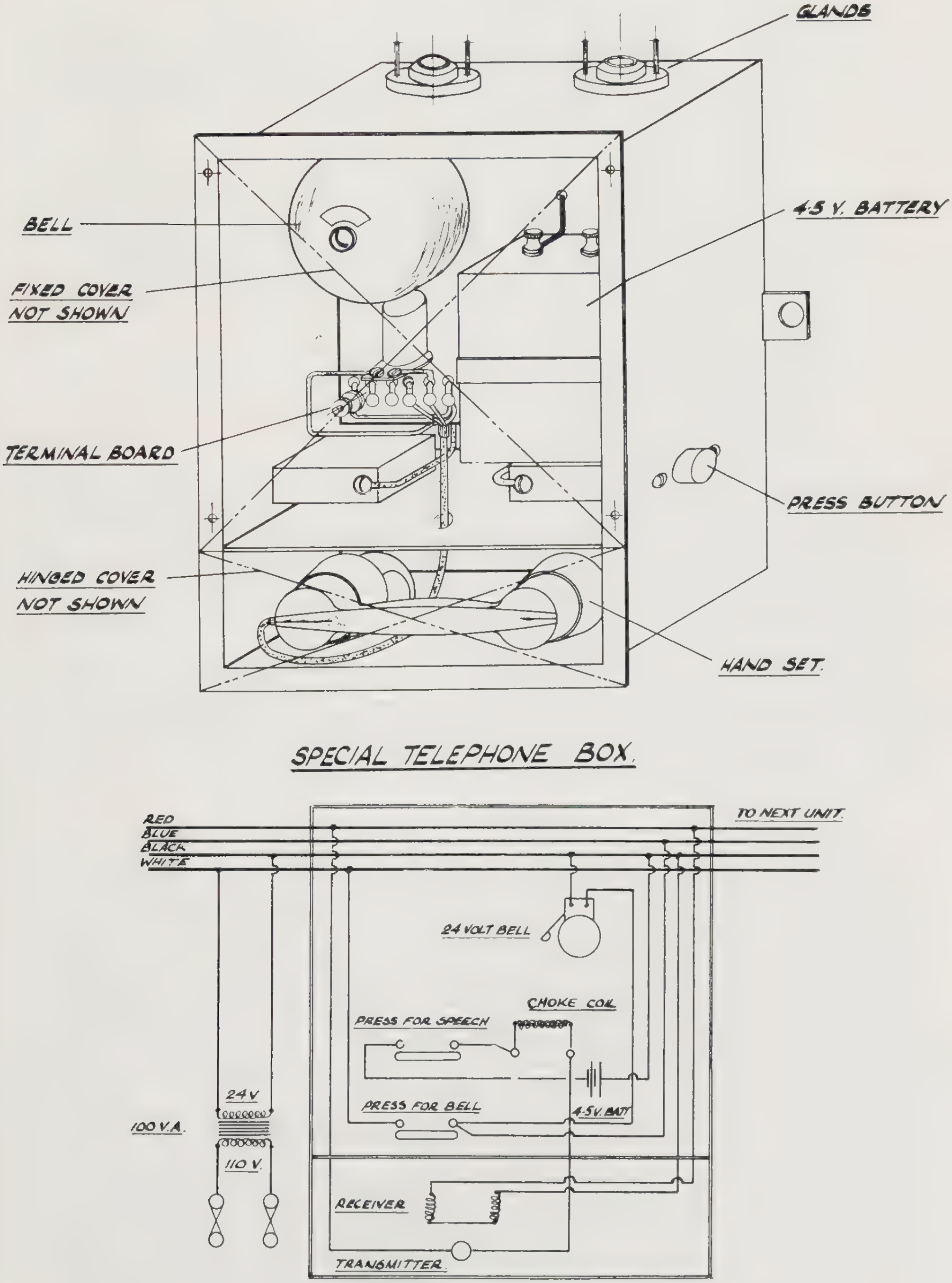

Fig. 11—Special telephone box and wiring diagram

2. Each tunnel has a repair bay inter-connected. The north bay is equipped for major repair and overhaul for either haulage, while the south bay is laid out for daily inspection and minor repair.
3. In the north tunnel a centralized, high capacity dust filter plant is shown. This layout, with a capacity of 75,000 c.f.m. will deal with both tipping points and the complete shaft system.
4. The location of the main upcast airway to facilitate ventilation control.
5. Inside the east and west pipe rim two sludge settling sumps will be cut, the clear water discharging to water (ex sampling) tunnels with a capacity of four million gallons, the settled sludge pumped direct to surface. These water tunnels were cut in blueground, on a grid system, initially for the purpose of obtaining a bulk sample of blueground for the valuation of the 1,700 ft. level.

The sampling tunnels serve three purposes namely:

1. To provide a representative sample of blueground to be treated in a special sampling plant.
2. The tunnels provide a large water storage sump.
3. The layout is conveniently arranged for the tunnels to serve as mining production tunnels when mining reaches the 1,700 ft. elevation. It should be noted that this development, sited in blueground, more than pays for itself.

In the new haulage system, each tunnel will be equipped with three locos, two existing 13 ton G.E.C.—and one 11 ton Goodman trolley wire locos, 30-180 cu.ft. Granby cars, 81 lb rail section laid at 3 ft. gauge on 10 in. by 5 in. by 6 ft. Rhodesian teak sleepers and 80 lb "elastic" type rail spikes.

The loading box, of which 30 will be installed, was described by Borchers[2] in 1957.

The Granby cars are each fitted with an automatic scraper which has proved invaluable in keeping the cars clean during the wet season. This car cleaner is most effective with a negligible maintenance cost.

MINING LAYOUT

In 1946 the development of the western section of the pipe commenced.

Figs. 5 and 6 already referred to, give a general picture of the mining system.

To initiate this system of mining it is necessary to cut a grid of tunnels on each mining level, a grizzly cross cut level, transfer level and the collecting haulage level.

Each mining level requires two main tunnels, cut for convenient layout approximately 270 ft. either side of the centre line of the pipe, and parallel to the longitudinal axis, and a slot cutting tunnel on the longitudinal axis. There is also a complete grid of tunnels, spaced 90 ft. apart, cut at right angles to the slot tunnel and each connecting with the mains.

Figs. 5 and 6 indicate the primary development required for handling the broken blueground from the mining levels.

In the layout of the western section a cone and grizzly grid system was cut on 45 ft. centres on the 810 ft. and 843 ft. levels, respectively. A slot 45 ft. wide, 300 ft. in depth was cut through the centre of the long axis of the pipe to the 810 ft. level.

The mining levels which are cut 50 ft. apart, vertically, allow the slot walls to be stepped in the form of benches 36 ft. wide, forming a slope of 65°.

Fig. 12 is a three-dimensional view showing the cones, finger raises and grizzly layout.

The cones and grizzlys were cut on 45 ft. centres to produce a maximum number of collecting and draw points, at the earliest possible date. This was essential as the initial production was to come from one-third of the pipe area.

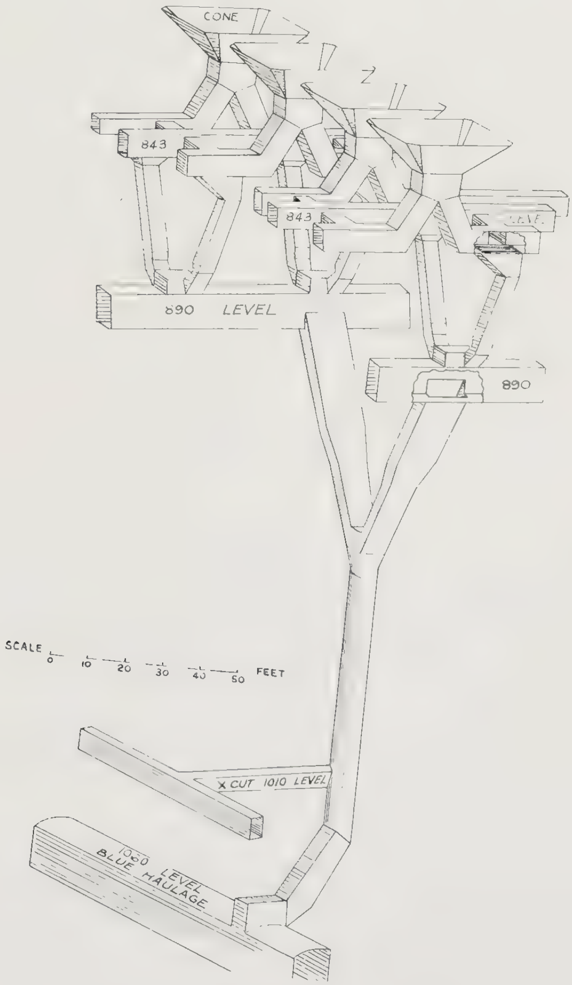

Fig. 12—Three-dimensional view of pass system

The main reasons for the above decision were:

(*a*) Minimum capital expenditure to production stage.

(*b*) Large-scale production at earliest possible date.

(*c*) As the pipe area had never been sampled it was not possible to assess the potential value of the pipe below 600 ft. level, and from a valuation of previous operation it appeared as though 810 ft. level was the economic bottom of the pipe. Therefore, it was decided to minimize the dead area of the pipe floor at 810 ft. level and cones at 45 ft. centres with a cone mouth of 34 ft. appeared to be the maximum open area for collection without interfering with the strength and durability of cone pillars.

Each tunnel on the transfer level is serviced by two six-ton self-tipping and erecting Hudson cars hauled by a remote control endless rope system, on a 3 ft. gauge 45 lb track, driven by a 15 h.p. Scott winch fitted with a fleeting wheel, fluid-drive coupling and solenoid brake. The remote control system is a push button stop-start at all working points in the tunnel, exact spotting of cars being achieved with the use of the thruster solenoid brake. A safety device, operated by the car, prevents accidental over-run into either the return wheel or the haulage machine itself. Initially, the push-button control was direct onto the 500 volt supply line but the somewhat damp conditions under which these controls work, resulted in continuous maintenance, apart from the danger of a buried 500 volt cable next to an operating track, causing an accident. The control cable was altered to 24 volt operation, the energizing of the 500 volt supply being done by relays at the haulage motor control board. Since this alteration, no trouble has been experienced.

The six-ton cars are being replaced by eight-ton side tipping cars activated by air cylinder. There are three main reasons for this replacement: positive operation, car factor due to wet blueground and maintenance cost.

From experience gained in operating the western section it was decided to modify the eastern section layout of cone, grizzly and transfer levels.

Figs. 13 and 14 show the modified cone, finger raise and grizzly layout.

In this layout it will be noticed the cone opening increases from 34 ft. to 68 ft. and the throat of the finger raise opens from 12 ft. square to 18 ft. by 60 ft. Further, in the western layout a 90 ft. block produced eight finger raises feeding six grizzlys, whereas in the eastern layout six finger raises feed six grizzlys. However, the production potential is accelerated by larger openings to grizzlys, longer finger raise life and more solid pillar area in which to establish working conditions where deterioration or wear of finger raises necessitates retreating into the solid.

Fig. 15 shows a modified grizzly pass loading chute on the 906 ft. transfer level.

It will be noted the passes have been brought onto the same centre line. This feature eliminated considerable survey work, construction excavation, concrete and steel work, without interfering with the productivity.

Fig. 16 is a three-dimensional view of the system.

It may be mentioned that on the south side of the slot the transfer tunnels were equipped with $2\frac{1}{2}$ ton scrapers operated by endless rope haulage system with remote control.

Although the production potential of this system is most advantageous there are several drawbacks, e.g. high initial capital cost and high maintenance cost.

Scraper tunnels are the sump of the pipe area and during the wet season the sticky condition of the blueground seriously interferes with production by blocking main passes, slowing down haulage traffic and holding up the shaft loading and hoisting system.

ROCK BREAKING

1. *Drills and bits*

Due to the relative softness of blueground, it was initially decided that rotary drilling for blastholes of up to 65 ft. in length would be preferable to percussion

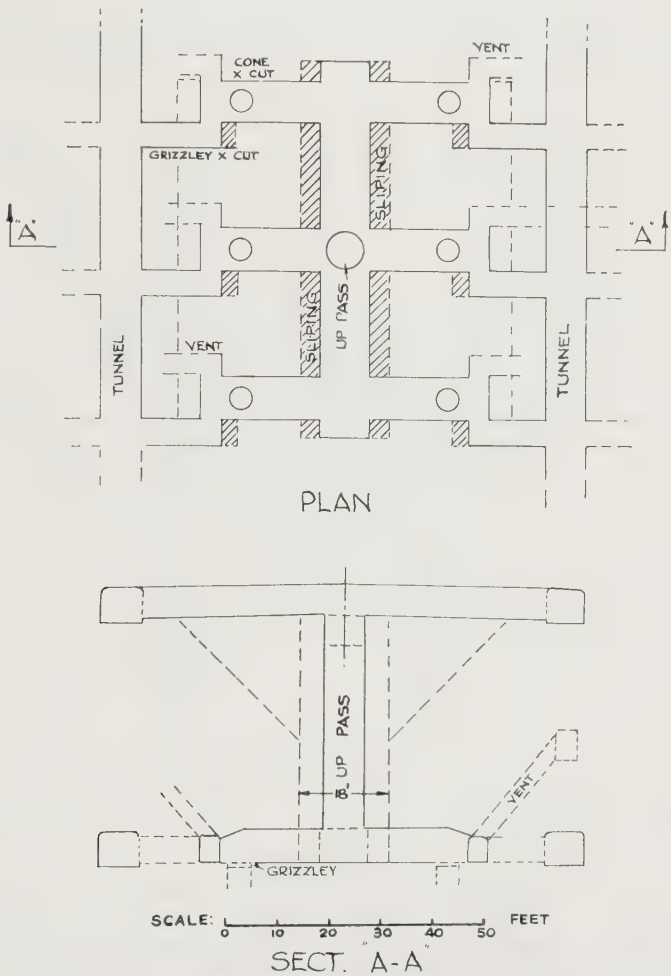

Fig. 13—East and central wedge cone and finger raise

drilling. This was borne out by experiment in 1946 which showed that in percussion drilling the volume of cuttings, due to the fast advance in soft ground, could not be sludged fast enough to prevent the rods jamming in the hole, due to the lack of clearance between the couplings and the diameter of the hole. Increasing the clearance in the hole set up vibrations which damaged the drill steel equipment.

The main problem then revolved around the profile of the crown to be used. The non-coring diamond crowns available at that time proved unsuccessful, as the high rotational speed and fine grinding of the diamond bit produced an extremely fine blueground slime which blocked the waterholes in the crown. Experiments then turned towards the use of tungsten carbide inserts as the cutting medium, and after considerable manufacturing experiments in respect of the percentage of cobalt hardening medium in the tungsten carbide, a three-wing crown was evolved followed by a two-wing twist drill type, as illustrated in Fig. 17. The "EX" or $1\frac{1}{2}$ in. diameter size was used but larger diameters could be used, satisfactorily.

At the same time, hard patches were found in the blueground, and the tungsten carbide crowns were unable to penetrate these with success. With a modified insert the "Mufulira" type coring crown, set with inserts of tungsten carbide impregnated with

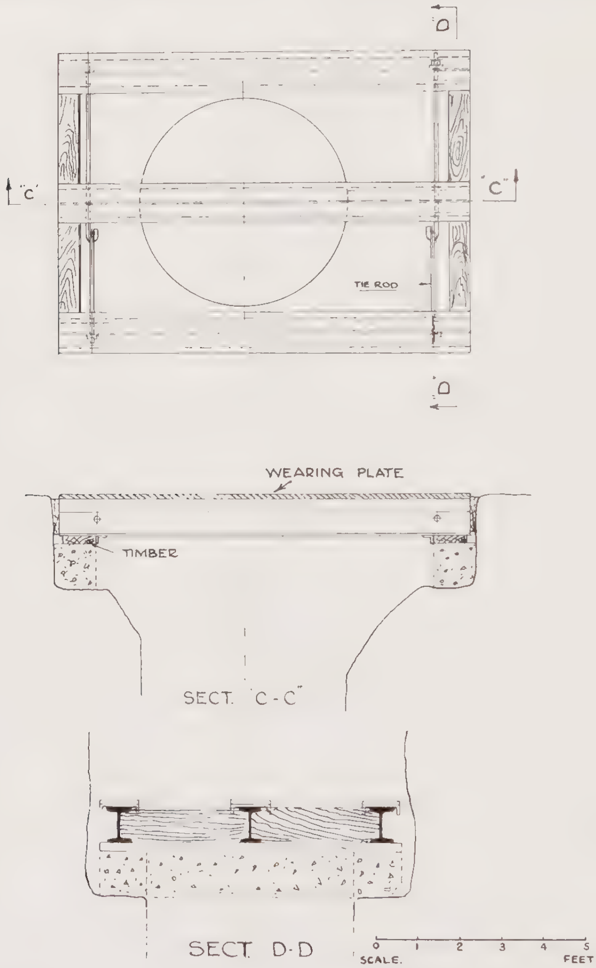

Fig. 14—Standard 10 ft. grizzly

diamond chips as shown in Fig. 18 was used. This crown, though slow drilling, stood up to the hard inclusions, and proved more economical than a coring crown set with individual diamonds where close supervision is required to maintain a reasonable salvage value.

When a hard patch is encountered, the tungsten carbide twist drill crown is changed to a "Mufulira" type until the hard inclusion has been penetrated. To take care of gauge sizes, a hole gauge is supplied which is merely a set of accurately cut holes in a steel plate, each varying by $\frac{1}{32}$ in. diameter.

Various types of rotary machine drills were tested but the common speeds of rotation proved to be too high. The final choice was the CP.55A drill geared to 900 r.p.m. on the feed screw and fitted with four gears, the highest of which was 25 revs./inch equivalent to a penetration of 36 in./min., and the lowest 800 revs./inch which is used for the coring crown.

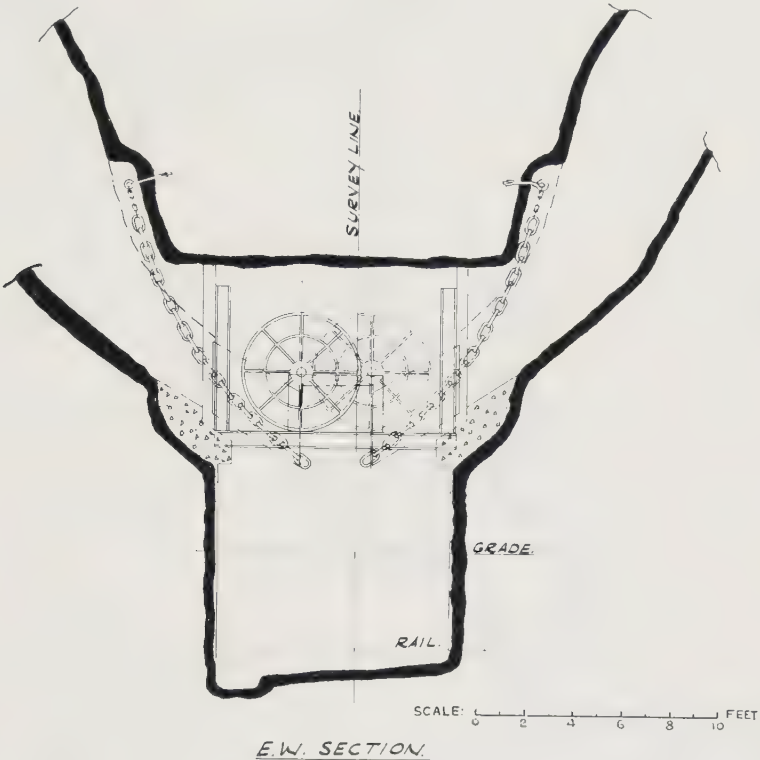

Fig. 15—Standard central east loading chute

The standard drilling crew is 11 Natives for each European driller. Three drills are operated by nine Natives, each drill having one machine boy with two spanner boys for changing the two-ft. extension rods. One rigging boy to rig bars ahead of the drilling assisted by the boss boy who also supervises the drilling technique of his three crews. By means of sidewall survey pegs placed every three rings or 18 ft., bars are rigged ahead, and the rings to be drilled are actually marked on the sidewall by using a protractor which fits the machine clamp. The drilling crews simply change their machine from the finished ring to the new bar rig and commence drilling, immediately, without having to wait for setting up or marking off. It is interesting to note that under normal drilling conditions of 400 ft. or more per underground shift, rod changing can be accomplished in 10 secs. from throttle off to throttle on.

The foregoing procedure was sufficient to drill the blueground, but a different proposition was encountered when the mixed blueground and Waterberg quartzite in the centre of the mine were reached.

Blast-hole drilling was effectively carried out using extension steel and rigged drifters. Due to the size of the couplings, the $1\frac{1}{2}$ in. diameter hole could not be maintained and a $1\frac{7}{8}$ in. diameter bit was used. The standard tungsten carbide chisel bit profile, in the form of a screwed detachable bit, is used with extension rods.

Experiments with cruciform and chisel bits proved the chisel bit to be more economical and efficient, easier to sharpen, and it has not shown any tendency towards deterioration in the quality of the hole.

Under normal operating conditions the average footages drilled with rotary and percussion machines are 400 and 160 per month, respectively. There were two standards of hole diameter $1\frac{1}{2}$ in. and $1\frac{7}{8}$ in. using two sizes of explosive, namely $1\frac{1}{4}$ in. and $1\frac{1}{2}$ in. by 22 in. sticks.

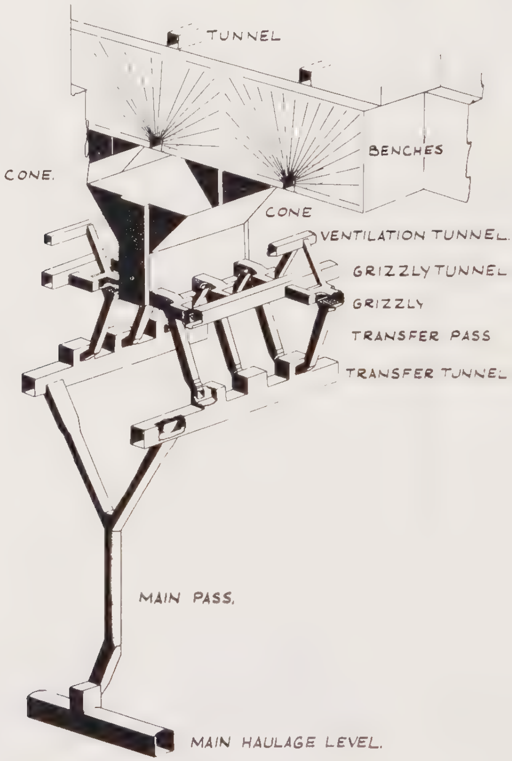

Fig. 16—Three-dimensional view of new system

In keeping with the recent investigation concerning standardization of explosive manufacture, it has been decided to standardize all production drilling at 2 in. diameter hole using $1\frac{3}{4}$ in. by 22 in. explosive.

2. *Drilling technique*

The original mine layout of levels 50 ft. apart, vertically, and drilling tunnels 90 ft. apart, horizontally, has been maintained.

The question of fragmentation in the primary breaking of blueground has been and is a problem for continuous research and experimentation.

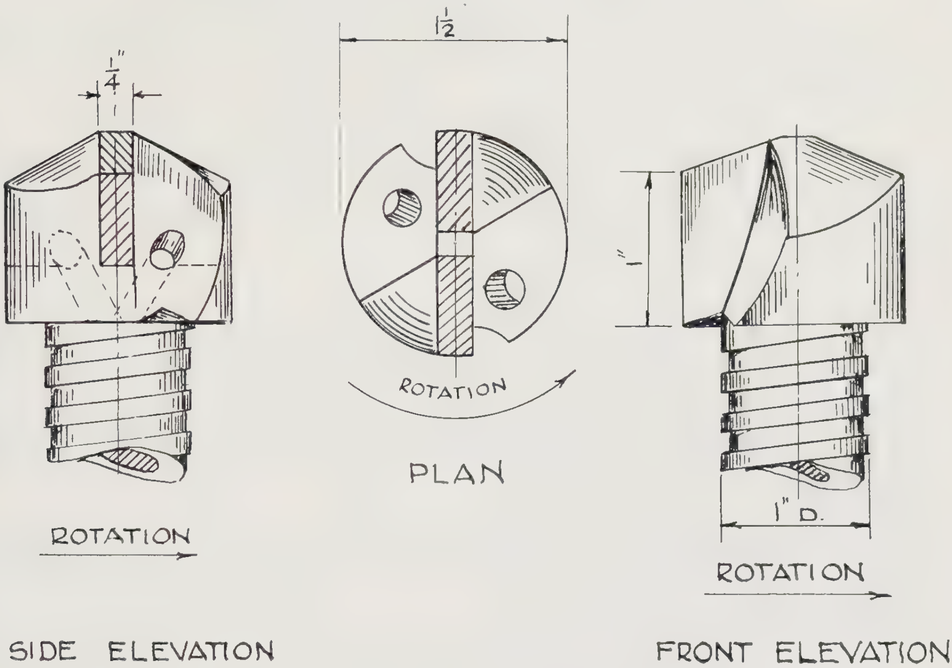

Fig. 17—Twist drill type tungsten carbide insert non-coring crown

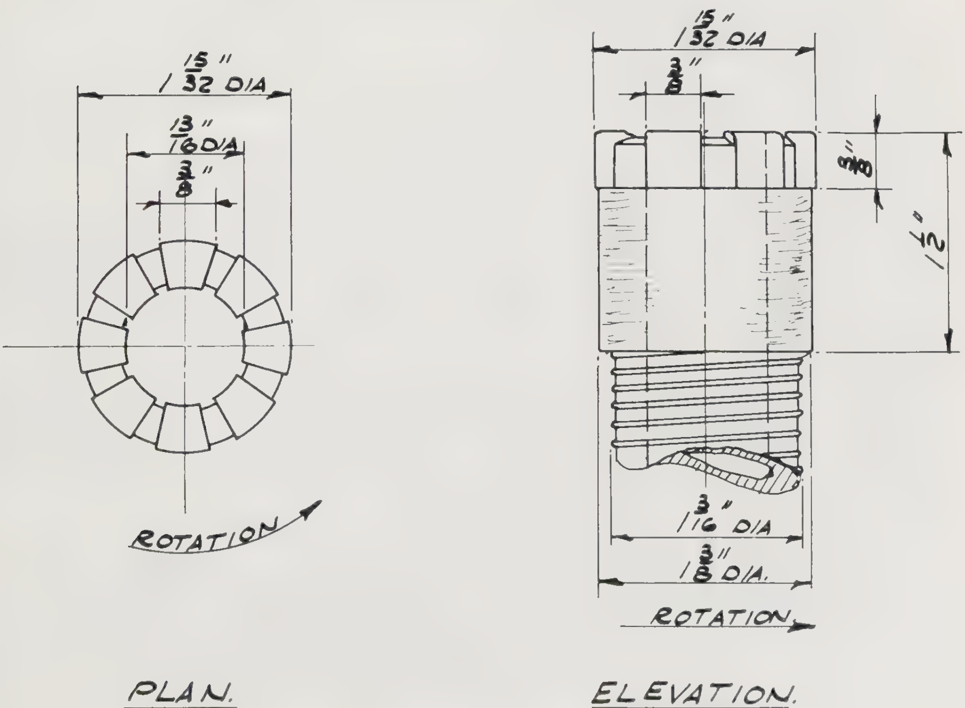

Fig. 18—Mufulira type diamond impregnated tungsten carbide coring crown

Before production in 1950 and subsequently, every possible combination of ring pattern, burden, powder loading, instantaneous and milli-second detonation has been studied and tested.

The initial standard was a 20-hole ring with 8 ft. of burden, this being finally reduced to 6 ft. burden (see Fig. 19).

The crux of the matter is, that the ground is "rubbery" and full of cross slips which makes efficient primary fragmentation most difficult, and to date, this problem has not been satisfactorily solved.

3. *Blasting technique*

In large tonnage production it is necessary to be well ahead of call with drilling, charging and blasting arrangements. As drilling and blasting preparation takes time it is necessary to have large sections of the mine drilled over.

From experience, it was found that many drilled holes could not be successfully charged if left standing. Therefore, all holes are charged as soon as possible, and permission has been obtained for charged holes to stand without blasting for a maximum of 30 days.

Both dynamite and gelignite explosives are used, the latter in wet areas and during the rainy season. Detonating fuse is used throughout the length of each hole charged with explosives. This may appear to be an extravagant procedure but is justified for several reasons:

(*a*) In grizzly work it is important to minimize the chance of undetonated explosive being mixed with the broken ground or even remaining in large unbroken rocks.

(*b*) Small ground movement in holes after charging can break the continuity of the powder.

(*c*) Portions of the hole becoming damp can prevent the complete hole exploding.

(*d*) Cut-offs are reduced to a minimum.

Special gangs are employed on charging holes, the ganger in charge recording any short drilling or loss of hole length due to any other cause.

Charging rods used are 6 ft. lengths of "Nucana" sewer rods specially adapted for blast-hole drilling. As these rods are quite flexible the lengths are not disconnected with each injection of explosive. Rings are usually blasted in panels of four tunnels with four rings in each tunnel; this procedure probably assists the fragmentation. But any number of rings can be blasted to suit the convenience or requirements of tonnage in the slot.

The most baffling problem so far has been the detonation of charges. The ideal condition would be a continuous wave of detonating holes following through the rings being blasted, instead of an instantaneous ring blast which tends to cut a badly fragmented slice from the bench.

In practice the above has not been possible.

A brief summary of detonation practice is as follows:

1. Each hole was connected with individual standard delay detonators.
2. Each hole was connected with the same number delay of standard delay detonator, with the knowledge that considerable scatter existed in the detonators.

   Neither condition 1 or 2 prevented cut-offs.

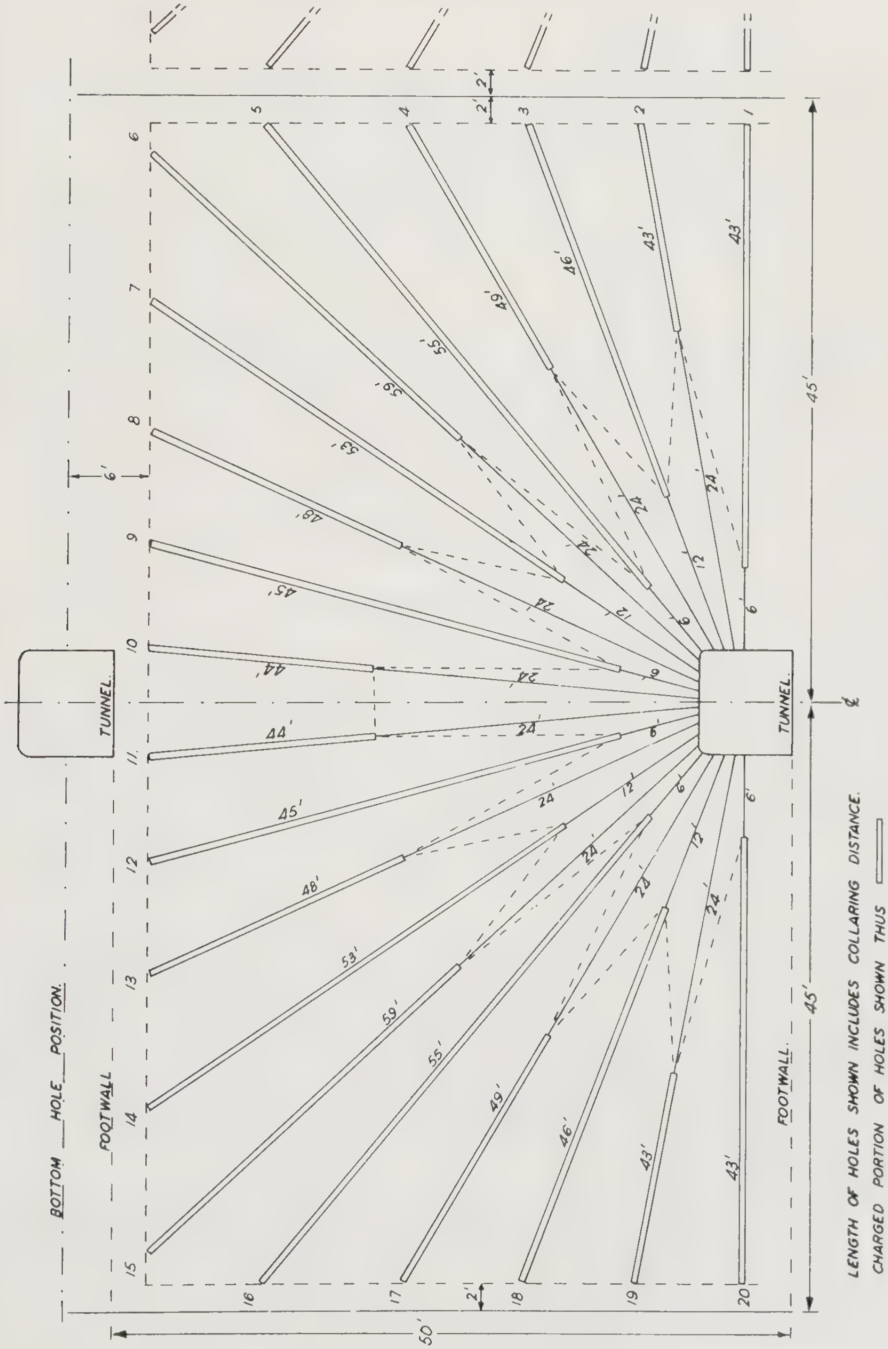

Fig. 19—50 ft. standard ring diagram

3. An extensive series of tests were conducted with 25 milli-second delay detonators in collaboration with African Explosive technicians without any positive result.

It was proved with a testing unit that the so-called 25 milli-second delays had considerable scatter above 17 milli-seconds. Also it was concluded that the ground movement was too sharp for this application.

At this stage, instantaneous blasting of rings was carried out, using a ring main of cordtex with branch connection on the hanging wall to each hole. To avoid abnormal shock wave, created by blasting a large number of rings simultaneously, a different number milli-second delay detonator is used to initiate between the blast in each tunnel.

Tests were again carried out with a "dropping arm" type of blaster encased in a heavy steel box which was developed on the mine. It was possible to adjust the interval to 2·5 milli-second delay, but although a certain measure of success was obtained the instantaneous standard has remained. The ring main of cordtex has been superceded by "cluster" in which all cordtex leads from the holes are brought to the centre of the tunnel for connection to the detonator. The delay between rings is obtained with a suitable length of cordtex.

As large quantities of explosive are used in every ring blast resulting in considerable concussion and air blast, special precautions must be taken in blasting, which is only done at change of shift when all personnel in the working area are out of the mine.

Initiation is done from cubicles situated at the Sub-Vertical and No. 2 Shafts, serving the west and the east sections.

When blasting rings on a level, the blaster proceeds to lace up his "clusters". He then tests his main tunnel firing points which are situated at the junction of each tunnel and the main travelling tunnel, and are carried back to the shaft station by a two-core cable in the main tunnel. Each tunnel has two firing points in parallel connected off this cable. The plug at the station is disconnected from the shaft cable by a locked shorting plug. This is removed and a portable 100-24 volt testing box is plugged to the lighting circuit and to the terminal plug for the two-core cable in the main tunnel. The firing points are then shorted out, one by one, and provided the cable is in order, a 24 volt lamp in the testing box will light, the brilliance being a measure of the electrical efficiency of the lines.

The shorting plug is now replaced and from the tunnel fire point, a portable reel of twin P.V.C. 3/036 wires, shorted by twisting, is run out to the ring to be blasted. The detonator leads are now connected to the real wires and the blaster returns to the tunnel fire point in the main tunnel. The twisted short is removed and connection made to the main tunnel fire point. On returning to the shaft station, the shorting plug is unlocked and the shaft cable plugged in. The blaster then proceeds to the blasting cubicle which has a steel door fitted with as many locks as there are blasters.

When all blasters are assembled, the cubicle is unlocked and the level to be blasted is plugged in, and blasted.

On re-entry next morning to the blasting area, the level cables are shorted by removing feed plugs and inserting and locking the shorting plugs. The blasting leads are then re-rolled onto reel and returned to the electrical shop for repairs and replacement of damaged sections of leads.

The blasting panel (Fig. 20) consists of:

1. Double pole manually-operated knife switch which is weighted to fall into the off position when released.

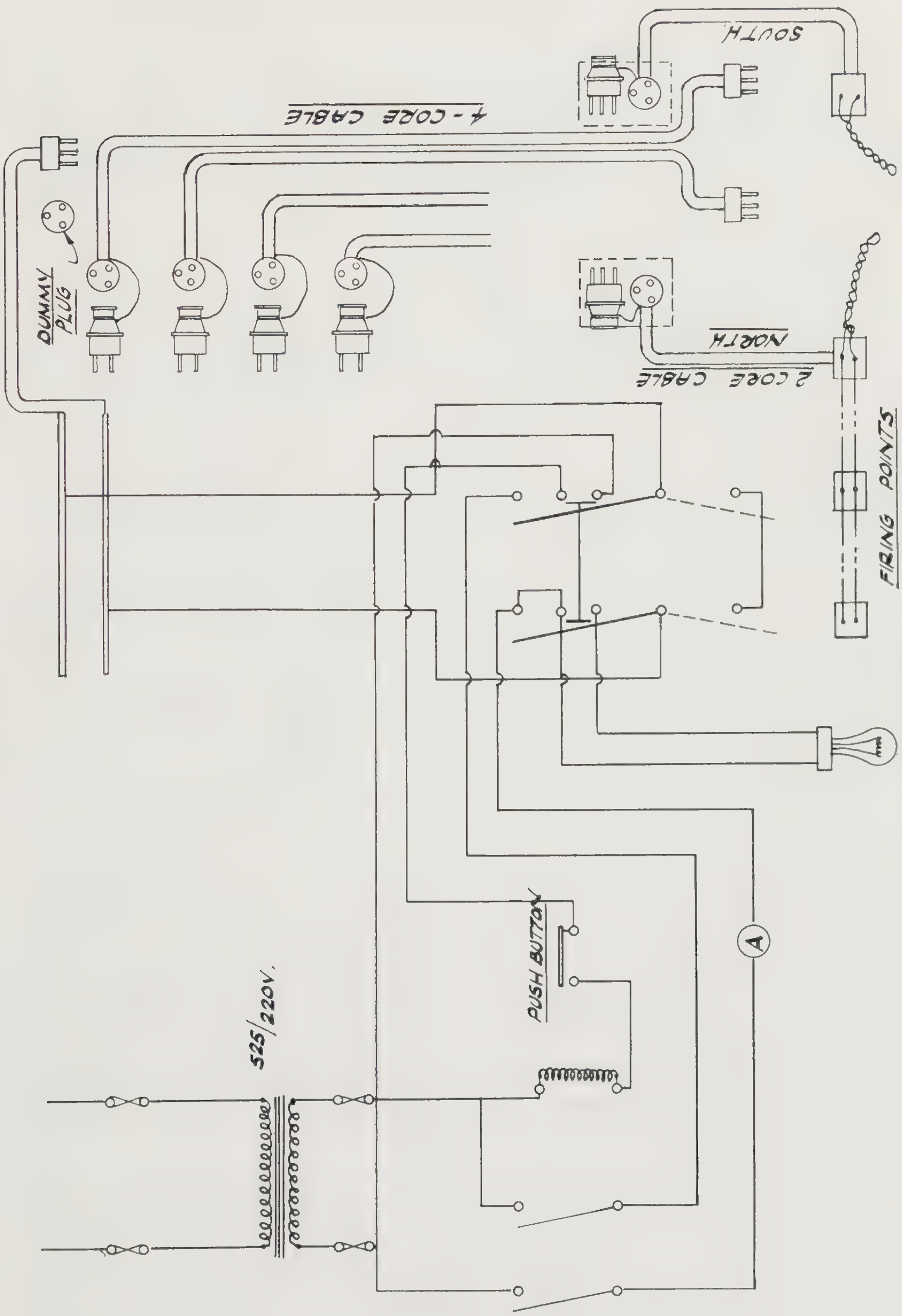

Fig. 20—Centralized blasting cubicle layout

2. One Cutler-hamer 50 amp. contactor.
3. A 525/220 volt transformer.
4. The necessary number of plug tops and bases.

Power at 220 volts is supplied direct to the contactor. The push button operating the contactor is interlocked with auxiliary contacts on the knife switch: thus, when the knife switch is raised on the "on" position the push button can energize the contactor and pass the power via the knife switch to the male portion of the 15 amp. industrial plugs, which are suspended on cab tyre flex. These plugs. when not in use are inserted into dummy bases.

Blasting from this cubicle can be carried out on all levels which, in turn, are divided into north and south, of the mine.

Two 15 amp plug bases are mounted on the cubicle panel and feed via a four-core cable (7/036) to each level, terminating in 15 amp plug tops; adjacent to these plug tops, bases are mounted feeding via two-core 7/036 cables, one to the north and one to the south, of each level.

4. *Secondary grizzly blasting*

The initial arrangement for blasting pop-holes or mud blasts on the grizzlys was by means of 2 ft. safety fuses. 10 ft. fuses being used for bombing large hold-ups in the case of finger raises.

This was considered dangerous when dealing with large bombs and numbers of holes; also it was impossible to count the shots when a number of holes or bombs were blasted at one time. This meant a 30 minute re-entry period after each blast in terms of the mining regulations.

Electric blasting was introduced in the form of six-shot exploders, which was soon replaced by a method of blasting off the electrical system, which has proved most convenient and successful.

Due to the dangers and possible earth shorting of a high volt system, it was decided to use 24 volt for blasting. Experiments have indicated that 70 or more instantaneous detonators can be initiated with this power.

The tunnel cable of two-core construction has firing points at each grizzly, each core being shorted with the other as a safety precaution during the period when no blasting is going on. This condition had been brought about by using 110 volt lighting supply to a 110-24 volt transformer with a "frog-leg" switch on the 24 volt side for blasting. When in the "off" position the first pair of contacts shorted the tunnel cable. When the switch was depressed, the first pair of shorting contacts were taken off the line and the 24 volt current applied by the second pair of contacts, so passing current to the firing point. The whole arrangement was situated in the main tunnel, and was enclosed in a locked steel box in such a way that the three-pin plug supplying the 24 volt power to the blasting switch could not be connected when the door of the box was closed. The grizzlyman responsible for blasting is the only person in possession of the key to the box.

Due to the amount of extra D.C.C. wire being used to connect the tunnel firing points to the detonators in the grizzly chamber, an attempt was made to bring the firing points into the grizzly chamber but any installation was rapidly smashed by the blasting. Eventually, two holes about 12 in. apart were drilled from the tunnel to the grizzly chamber and lead wires brazed to the tunnel cable were threaded through these holes. At the grizzly chamber end, each lead wire was brazed to a 6 in. nail which protruded about 1 in. from the hole, the collar of which was filled with mortar for about 2-3 in. This method has proved effective and stands up to blasting whilst presenting no difficulty in connecting blasting wires as long as the nail is kept reasonably clean. All detonators in each grizzly blast, whether pop-holes or bombs, are connected in series to the fire point, each grizzly being in parallel with the tunnel blasting cable. The odd open-circuit detonator will cause a misfire in any one grizzly but the incidence of this is very low and from 1951 when this system was installed, more than six million detonators have been blasted under conditions of perfect safety.

MINE VENTILATION

The main features of the ventilation layout is that each working place has an independent fresh air supply.

Haulageways. The fresh air intake for the north and south tunnels comes from Nos. 1 and 2 vertical shafts, the return air passing to an upcast ventilation pass in centre of mine (see Figs. 9 and 10).

Transfer levels. 890 west and 906 east and new drop down levels 1210 west and east and 1410 west and east.

Western section. This is ventilated by intake air from No. 1 Shaft, air coursing along haulageway and up the Sub-Vertical Shaft to 890-1210-1410 levels west.

Eastern section. Each level is ventilated by intake air direct from No. 2 Shaft.

The main intake for the western and eastern sections on any operating level is split between the north and south side of each section, and by means of four 65 in. Safanco fans each forcing 80,000 c.f.m., the four areas of the pipe are pressurized at 1 in. w.g.

There are 15 tunnels in each area and by means of regulators each tunnel has a controlled quantity of air flowing.

A small proportion of the air to each tunnel exhausts into a common upcast airway, but the major portion of the intake upcasts through the passes to the grizzly level which prevents blast fumes from the grizzly level contaminating the transfer level.

Grizzly level 843 W., 859 E. and new drop down levels 1160 west and east and 1360 west and east.

The main downcast air is supplied to the grizzly level from the same source as the transfer level. Again the main volume is split between the north and south side of each section.

The distribution to individual tunnels is by necessity rather different. Each tunnel is sealed with double doors and equipped with a low pressure 1 in. w.g. forcing wall fan designed to withstand blast concussion, delivering 10,000 c.f.m.

Each tunnel has an individual exhaust airway arranged so that each working grizzly is independently connected to the intake and exhaust airway.

It was found impossible to maintain a positive ventilation flow when pressurizing each area with a single fan, as on the transfer level. The air blast from continuous blasting of multiple holes and bombing caused the smoke and fumes to flood the level; also the fact that the seal of blueground in the cones is often inadequate, caused short circuiting of the air supply.

GENERAL

Service inclines

Apart from the shaft system, the north and south sides of both the western and eastern sections of the mine are serviced by inclines, cut in the rock on a gradient of 1 : 1·6 and extending from the top working levels to the transfer level.

These inclines are used to distribute ventilation air and material by means of skid ways and monorail transport. It is also a man-way for internal ingress and egress to the levels.

Figs. 21 and 22 illustrate the detail and installation of the transport arrangement.

Vertical raising

The mining layout at the Premier Mine calls for a main pass system from the transfer level to the collecting haulageway. The total footage involved is in the region of 16,000 ft. of vertical passes, 8 ft. in diameter.

An "Alimak" pass climber is commissioned for this work, and has started operating. There is not enough information to hand to comment on this operation.

Indicator light for skip loading

To maintain the high rate of hoisting required from the No. 1 Ward Leonard hoist, amounting to about 16,000 tons per day, from 1,300 ft., it was essential to reduce the decking time to an absolute minimum. The minimum winding cycle that can be obtained on this semi-automatic hoist is 45 sec., made up of 15 sec. acceleration, 20 sec. full speed and 10 sec. deceleration. To obtain the 70 skips per hour required, only 4-5 sec. could be allowed for decking. With the type of ground being handled, the measuring flask will only empty in 9 sec. Thus the loading must commence before the load is fully out of the flask. This was done for many years by a mark fixed to the rope and sighted by the Native operating the measuring flask loading door, through a peephole. This method was unreliable, resulting in excess spillage and double loading. This practice has been replaced by a system of lights operated by the hoist cam gear and set to operate 4 ft. above the loading mark (see Fig. 23). This is about 2 ft. above the point where the chute lip and the skip lip pass each other. When the light goes on, the Native loads and the ground will reach the skip at the same time as the lips pass. As the skip leaves on the upward trip, the light goes out, but should the hoist trip, then the loading light stays on. The possibility of double load is avoided by a red warning light connected to the safety circuit which goes on immediately the hoist trips.

Main pump chamber

There are four features concerning the layout of the pump chamber:

1. The chamber is sealed off from the shaft except for an entrance and exit well above the haulageway.
2. The chamber can be sealed off from the haulageway with a watertight bulkhead.
3. The 3,000 volt power supply is direct from surface to the chamber via the pipe race and out of chamber to main sub-station.
4. Ventilation can be brought into the chamber via the pipe race in the event of flooding.

These features allow the main haulageway system and shaft to flood without endangering the pump station; also the flood waters could be dewatered by the main pumps through a bulkhead pipe on the conveyor level.

In conclusion the authors wish to thank Mr. G. S. Giles, Consulting Engineer, Anglo American Corporation of S.A. Ltd., for permission to publish this paper.

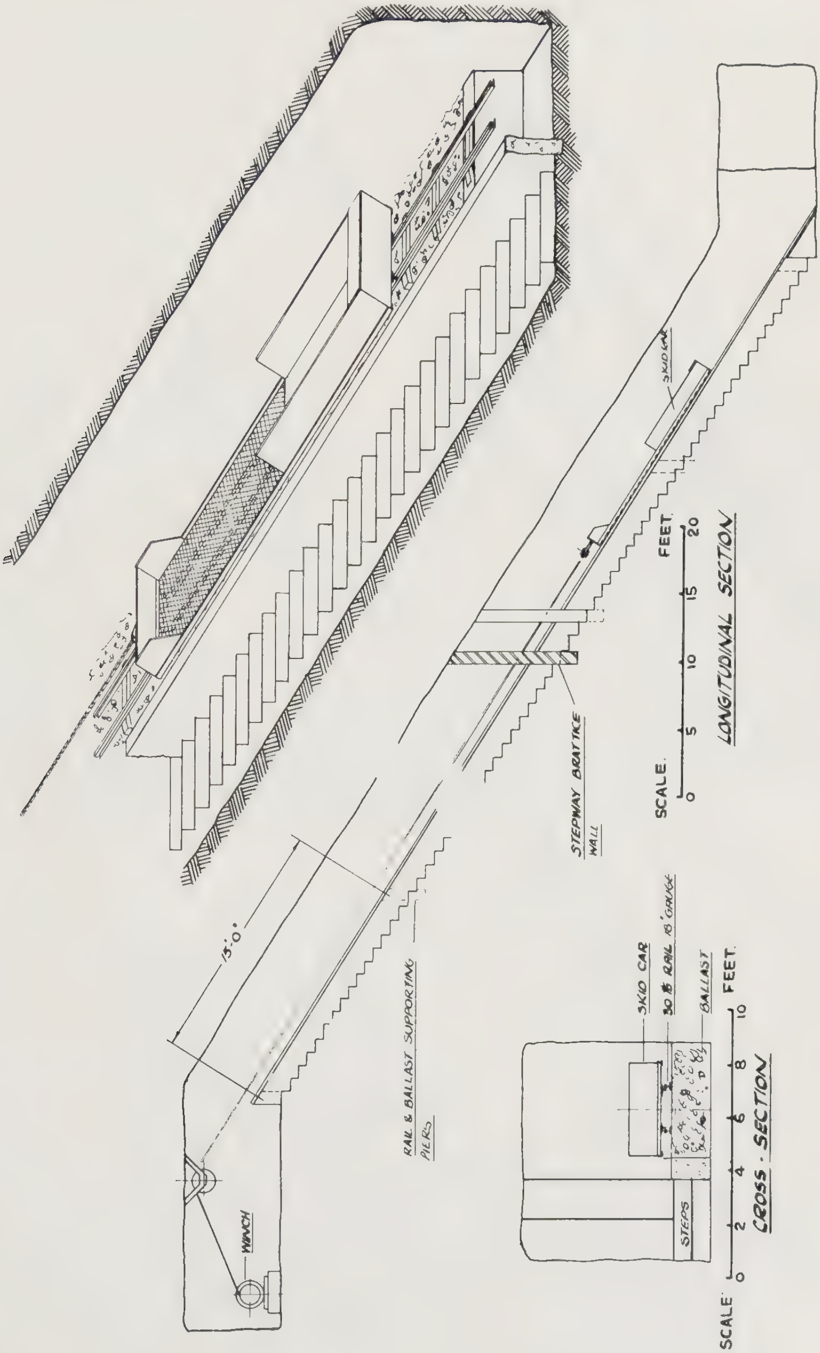

Fig. 21—Skidway

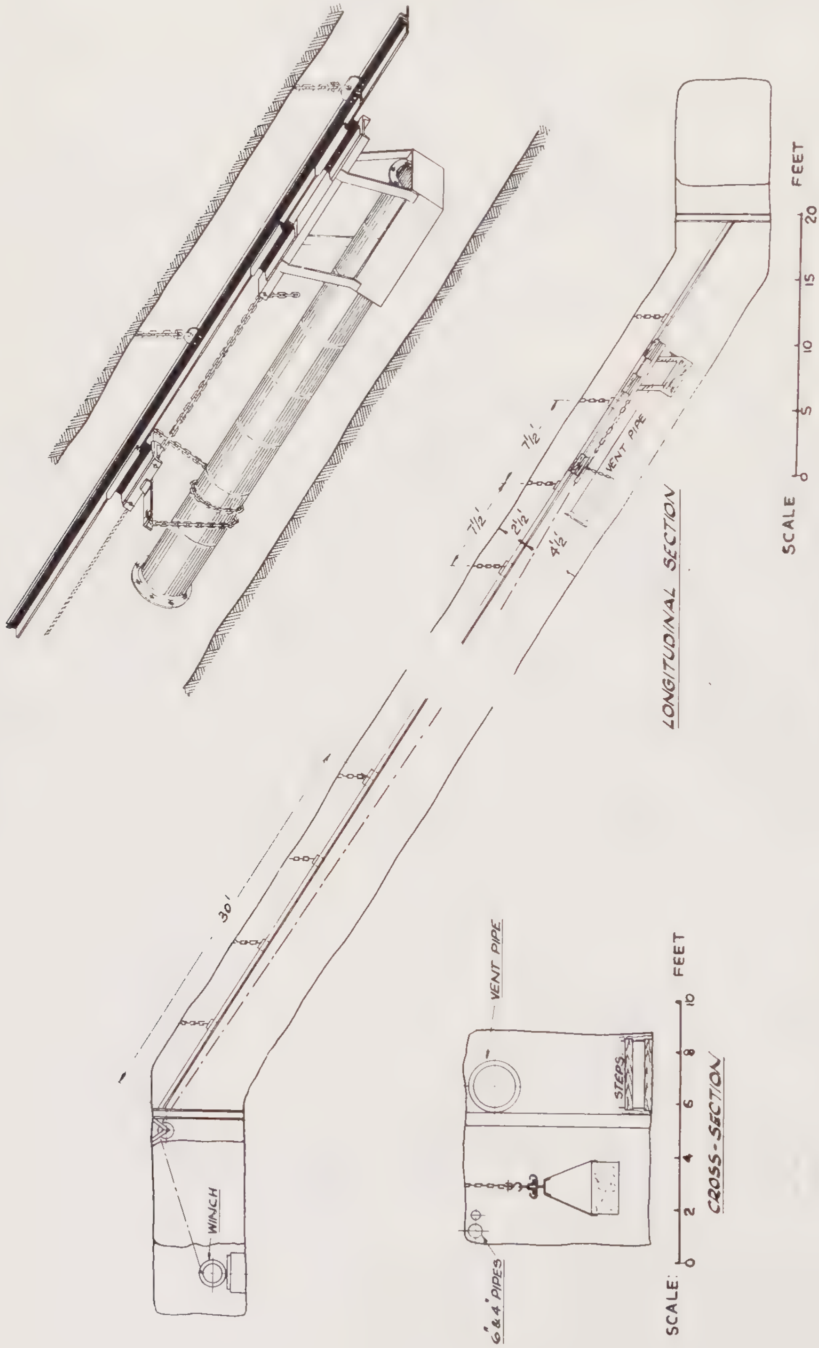

Fig. 22—Monorail

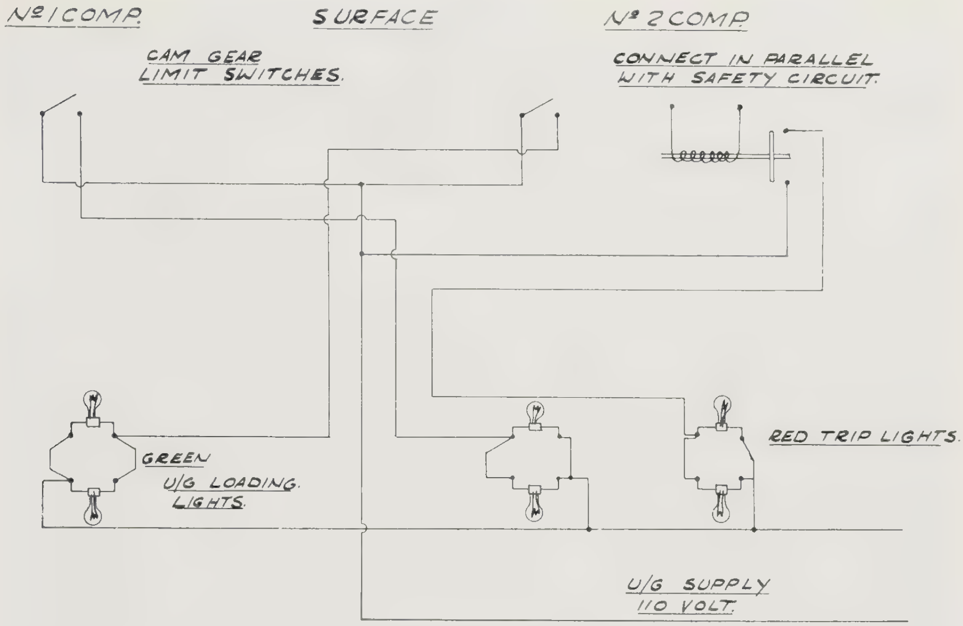

Fig. 23—Indicator lights for skip loading

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TREATMENT PLANT AND DIAMOND RECOVERY AT THE PREMIER MINE

By **R. I. SHANKS***, B.Sc.(Melb.), A.M.I.M.M. (Visitor)

HISTORICAL

The mine was prospected by the late Sir Thomas Cullinan (then Mr Thomas Cullinan) and floated as a company in 1903. According to P. A. Wagner, the mine was prospected by pitting, diamond drilling, and the sinking of a prospect shaft.

The material was treated in a small pan plant, presumably known as No. 1 Gear. No. 2 Gear is not mentioned by Wagner. No. 3 Gear was a five unit plant comprising in each unit a grizzly at $2\frac{1}{2}$ in., two "T" Hadfield and Jacks gyratory crushers crushing to $2\frac{1}{2}$ in. followed by two double sets of 4 ft. fluted rolls crushing respectively to $1\frac{1}{4}$ in. and $\frac{7}{8}$ in., a 5 ft. elevator elevating all the material to four first-treatment washing pans, from which a concentrate was drawn. The tailings from these pans were reduced to $-\frac{3}{16}$ in. in two sets of 6 ft. smooth rolls which delivered into four second-treatment washing pans from which a concentrate was also drawn. The tailings were dewatered and trammed to the dump and portion of the slime was returned to the first-treatment pans as puddle, the remainder being gravitated over the country.

The No. 4 Gear of seven units was similar in crushing arrangements, the pans being replaced by eight coarse roughing jigs and one coarse finishing jig, and after recrushing, eight fine roughing jigs and one fine finishing jig.

These plants have been clearly described by R. J. Adamson[1].

Owing to economic conditions obtaining in 1932 the mine was closed down and was re-opened in 1945. While the mine was being dewatered experiments were being carried out in Kimberley to ascertain the most satisfactory type of treatment plant for the Premier "blueground". Experiments with a 20 in. heavy-medium separation cone proved sufficiently interesting for the Consulting Engineer to decide to erect a pilot plant with a capacity of 100 loads per hour (one load = 0·8 short ton). This plant was erected at the mine and began treatment in 1947.

Fig. 1 is a general view of the pilot plant. On the left in the background is the crushing section. On the right is the H.M.S., recrush and jig section and in the left foreground is the recovery, diamond sorting, laboratory and office.

From this pilot plant the present plant which handles up to 16,000 tons per day was designed.

SHAFT STOCK PILE

The blueground is crushed underground to pass a 6 in. grizzly and is hoisted to a standard three trough stockpile which has a capacity of some 7,000 loads. Chutes from the apices of the three troughs discharge centrally over two parallel 36 in. conveyors with tandem drives and 75 h.p. motors which elevate the ore, at 500 tons per hour each, to two Symons 5 ft. 6 in. by 3 ft. 6 in. vibrating grizzlies at the top of the main crusher building.

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*Plant Superintendent, The Premier (Transvaal) Diamond Mining Co. Ltd.

Fig. 1

SORTING AND CRUSHING PLANT

Fig. 2 is a general view of the plant from the headgear showing the crushing section in the foreground with the mine store on the right. The 100 ft. thickeners and the primary washing and jig section are in the middle distance. In the background are the H.M.S. section in the centre with the recrush on the left and the recovery on the right. The tailings conveyor can be seen running away into the distance.

The grizzlies have bars set at 5 in. and the oversize is discharged to a cross conveyor which in turn discharges to a picking conveyor 42 in. wide and travelling at 90 ft. per minute from which waste, which is easy to recognize, is picked by Natives and dropped into hoppers below, where it is trammed to the waste dump.

The undersize from the grizzlies drops onto two 12 ft. by 6 ft. Gyrex screens with two decks, the top fitted with 3 in. square mesh and the bottom with $1\frac{1}{2}$ in. square mesh. The +3 in. material falls onto two 42 in. waste picking belts. The −3 in. $+1\frac{1}{2}$ in. material is conveyed to two 60 in. wide parallel conveyors, running at 90 ft. per minute for the purpose of hand picking any diamonds large enough to be damaged in the crusher. The $-1\frac{1}{2}$ in. material by-passes the crushers and joins the crusher product on a 42 in. conveyor, inclined at 16°, feeding the 1,500 ton bin.

Two integrating weightometers are installed on this conveyor, one an Adequate Weigher and the other a Philips straingauge weightometer. Tonnages are calculated on the skips hoisted multiplied by the skip factor. The weightometer figures are only used as a check.

The +5 in. material is crushed to −3 in. in two standard coarse $4\frac{1}{4}$ in. Symons cone crushers, the product being conveyed back to the screen floor to a 4 ft. by $8\frac{1}{2}$ ft. Gyrex screen with $1\frac{1}{2}$ in. square mesh apertures. The $-1\frac{1}{2}$ in. material joins that from the 6 ft. by 12 ft. screens and by-passes the crushers, while the $+1\frac{1}{2}$ in. material joins the −3 in. $+1\frac{1}{2}$ in. material to the 60 in. picking conveyors before passing to a short head $5\frac{1}{2}$ ft. Symons cone crusher for comminution to 80 per cent—1 in. slots.

Fig. 2

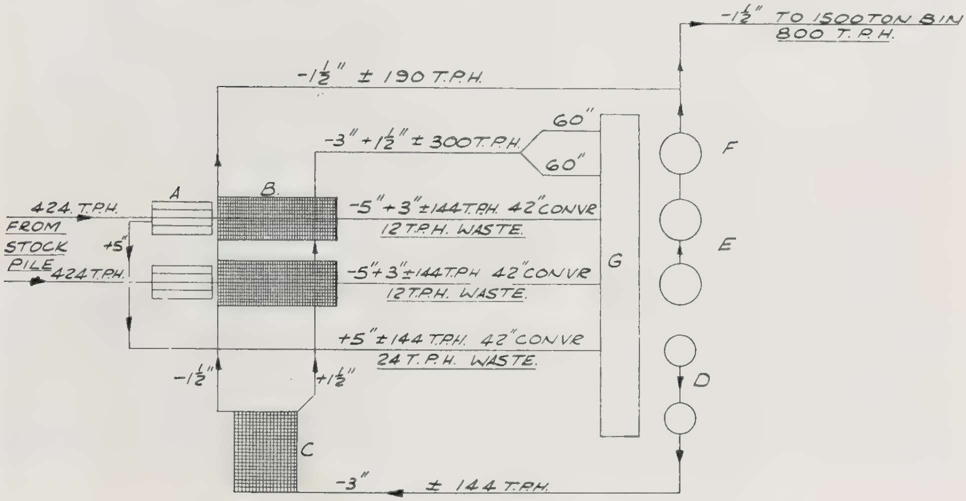
The −5 in. +3 in. material is crushed in two standard $5\frac{1}{2}$ ft. Symons cone crushers to 80 per cent—1 in. slots and all is collected together with the $-1\frac{1}{2}$ in. material and conveyed to the 1,500 ton storage bin, where it is mixed with the recrush return material.

Fig. 3 shows a diagrammatic layout of the primary crushing section.

A dust extraction plant of the water spray, coke-filter, type draws dust from screens, chutes and crushers at a total of 145,000 cu.ft. of air per minute at 6 in. water gauge. The sludge is pumped to the 40 ft. thickeners. Fig. 4 shows a view of the pipes of the dust extraction plant and the spray and coke filter house.

A typical grading of the crusher product to the storage bin is as follows:

$+1$ in.	22·4 per cent
$+\frac{3}{4}$ in.	23·8 ,, ,,
$+\frac{1}{2}$ in.	13·0 ,, ,,
$+\frac{3}{8}$ in.	6·5 ,, ,,
$+\frac{1}{4}$ in.	5·4 ,, ,,
$+\frac{3}{16}$ in.	4·2 ,, ,,
$+\frac{1}{8}$ in.	5·2 ,, ,,
$-\frac{1}{8}$ in.	19·5 ,, ,,



Legend

A. Two grizzlies—5 ft. 6 in. by 3 ft.
B. Two gyrex screens 12 ft. by 6 ft.
C. One gyrex screen 8 ft. 6 in. by 4 ft.
D. Two 4¼ ft. Symons cone crusher
E. Two 5½ ft. Symons cone crusher
F. One 5½ ft. short head cone crusher
G. Crusher bin

Fig. 3

SCREENING AND WASHING PLANT

As the heavy medium process does not operate satisfactorily with very fine material the next process is one of washing. Below the 1,500 ton bin are eight quadrant doors feeding eight 48 in. belt feeders which discharge to eight 5 ft. by 16 ft. Allis Chalmers, low head, double deck screens (two to each unit) fitted with $\frac{1}{2}$ in. square mesh carbon steel screen cloth on the top deck and 7 mesh (·094 in.) stainless steel screen cloth on the bottom deck. The top deck has six sprays and the bottom deck three sprays, all operating at 52 lb per sq.in. pressure of water. Fig. 5 is a general view of the discharge end of the 5 ft. by 16 ft. Allis Chalmers low head primary washing screens. The rate of feed to each screen is approximatley 140 tons per hour of which about 60 per cent is new feed.

Each screen was originally fed by a 36 in. Syntron F55 electromagnetic vibrating pan feeder, which was replaced by belt feeders because of the difficulty of maintaining a uniform feed to the screens, on account of the packing of the wet blueground fines in the pans damping the vibrations. Also, it was impossible to maintain a constant setting of amplitude and frequency in the vibrating mechanism.

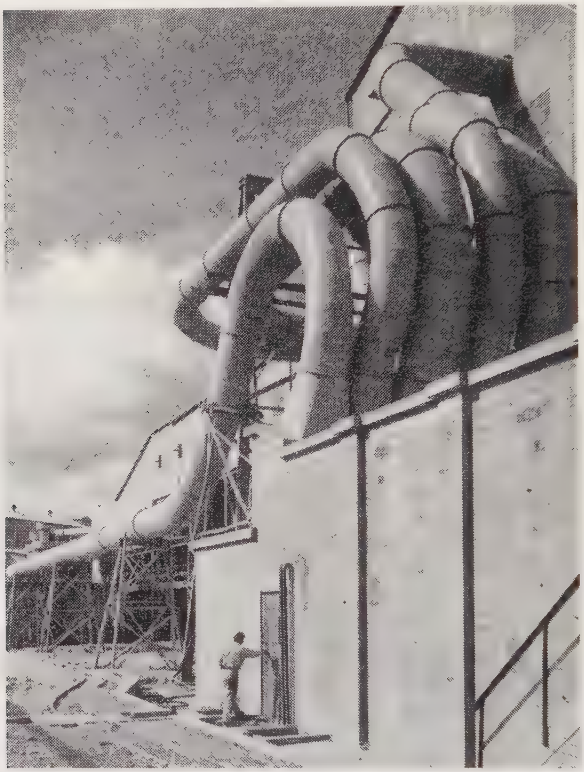


Fig. 4

Fig. 6 shows a diagrammatic layout of one section of the primary washing and jig plant.

JIG PLANT

The −7 mesh undersize from each pair of screens averaging 60 tons per hour is pumped by a 6 in. E.R.G. sand pump with a 70 h.p. motor to an 8 ft. by 18 ft. 6 in. Dorr duplex classifier. The overflow, mostly −28 mesh Tyler and about one-fifth of the total solids, flows to a pair of 100 ft. thickeners from which the overflow is returned as spray water. The thickeners' underflow is pumped away to the slimes dam. The settled sands, about 40 tons per hour, are fed to a pair of duplex 36 in. by 24 in. Denver. Mineral jigs fitted with a 3 mm. wedge-wire screen and equipped with a bedding of alluvial pebbles of average S.G. 3·26.

The jigs are adapted for continuous extraction by piping the hutch product to a watertight bucket elevator and allowing sufficient height above water level to dewater the concentrate. A concentrate of approximately 10 per cent of the feed from the eight primary jigs is fed to two cleaner jigs, where the concentrate runs about 10 per cent of the feed and the tailings are returned to Nos. 5, 6, 7 and 8 primary jigs. The concentrate is taken by conveyor to the recovery bin.

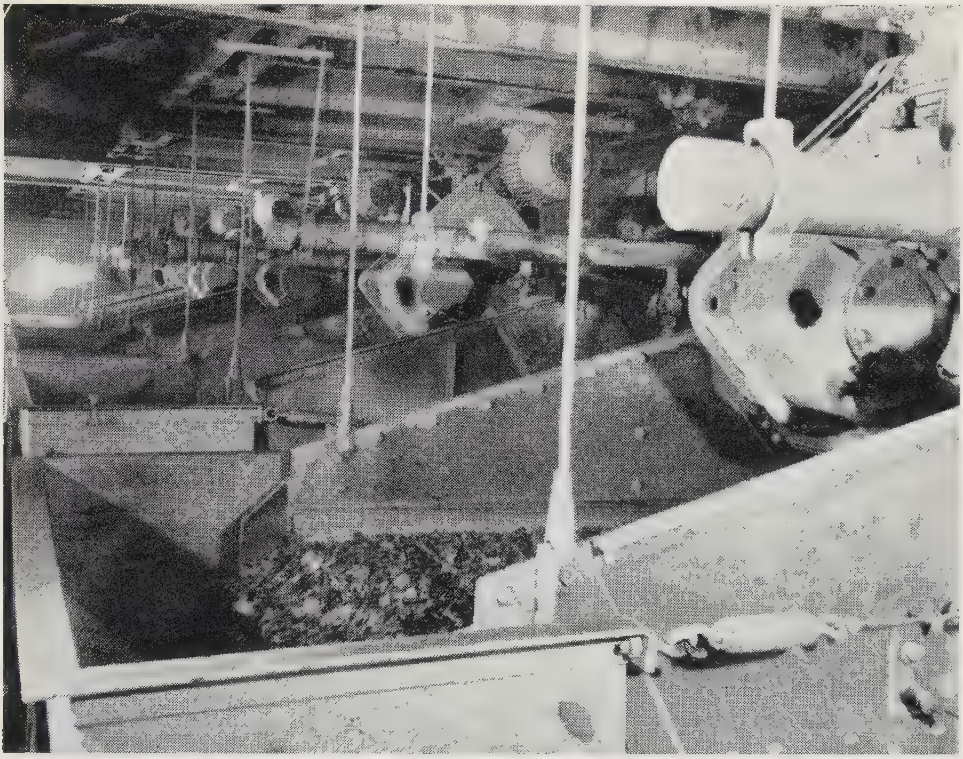
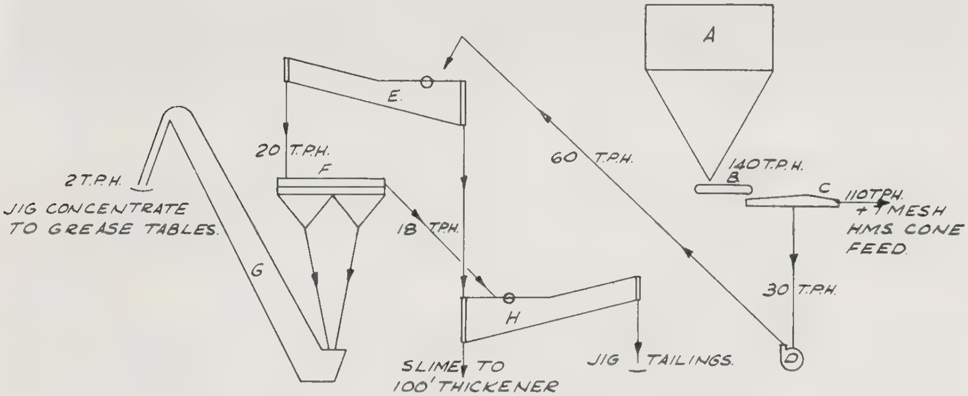


Fig. 5



Legend

A. 1,500 top bin
B. Eight 48 in. belt feeders
C. Eight 16 ft. by 5 ft. Allis Chalmers screens
D. Eight 6 in. sand pumps
E. Four 8 ft. by 18 ft. 6 in. duplex classifiers
F. Eight 36 in. by 24 in. Denver jigs primary
Two 36 in. by 24 in. Denver jigs secondary
G. Eight bucket elevators
H. Four 8 ft. by 18 ft. 6 in. duplex classifiers

Fig. 6

HEAVY-MEDIUM SEPARATION PLANT

Fig. 7 shows a general view of the H.M.S. section control floor and feed conveyors.

The plant consists of four 16 ft. heavy medium separating cones equipped with external airlifts.

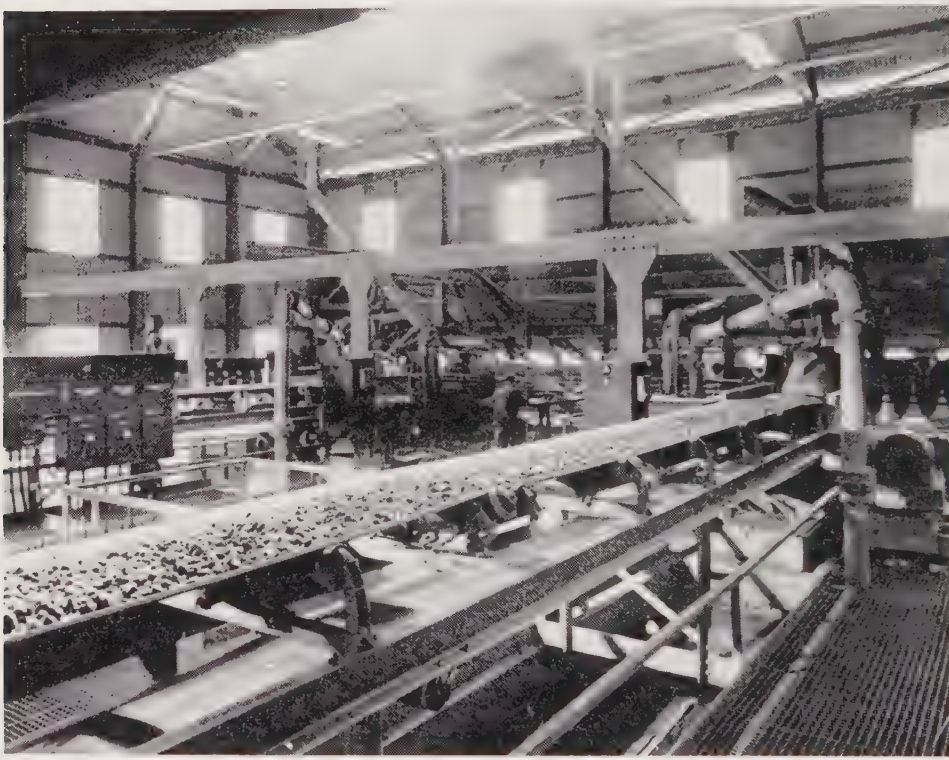


Fig. 7

Typical grading of cone feed, washed primary screen product:

$+1$ in.	21·4 per cent
$+\frac{3}{4}$ in.	18·7 ,, ,,
$+\frac{1}{2}$ in.	17·2 ,, ,,
$+\frac{3}{8}$ in.	13·0 ,, ,,
$+\frac{1}{4}$ in.	12·1 ,, ,,
$+\frac{3}{16}$ in.	5·7 ,, ,,
$+\frac{1}{8}$ in.	5·9 ,, ,,
$-\frac{1}{8}$ in.	6·0 ,, ,,

The washed + 7 mesh material from each pair of primary screens is fed by a 30 in. conveyor belt equipped with a weightometer having an integrating dial, tonnages being read hourly and recorded, to a 16 ft. H.M. cone. The medium is 65 D. ferro-silicon suspension at 2·84 top S.G. and approximately 20-22 centipoise viscosity measured by a modified Stormer viscosimeter. The differential in the cone at this viscosity is about 0·07 between the top and the airlift discharge. The specification of the ferro-silicon is as follows:

Si	14-16 per cent
+48 Mesh Tyler	Nil
+65 Mesh Tyler	± 0·5 per cent
−325 Mesh Tyler	38-43 per cent
Non-magnetics less than 1 per cent	
Specific gravity ± 6·8	

At this specific gravity the sink concentrate averages about 3·5 per cent of the original feed and is discharged from the airlift to a 3 ft. by 10 ft. Allis Chalmers low head screen fitted with 9 mesh stainless steel screen cloth with 0·065 aperture as a sink drain. The medium drains away and, not being diluted, is returned to the cone by the 5 in. Wilfley sink, medium return pump with a 73·5 h.p. motor.

The concentrate then passes to a 3 ft. by 10 ft. Allis Chalmers low head sink wash screen, also fitted with 9 mesh stainless steel cloth, which has two wash sprays at 40 p.s.i. pressure. The concentrate then falls to a 24 in. conveyor belt for transfer to the secondary H.M.S. plant.

The wash water containing ferro-silicon and sand and slime from the sink and float screens is laundered through a magnetic block, to flocculate the ferro-silicon, into a 20 ft. thickener. The overflow from two 20 ft. thickeners passes to one 40 ft. thickener, the overflow from which is used as spray water on the sink and float screens.

The underflow from the 20 ft. and 40 ft. thickeners is pumped to the magnetic separators to free the ferro-silicon from silicates etc. Primary separators are a 48 in. Dings and a 48 in. Memco separator in parallel, and the secondary separator is a 48 in. Dings to each cone unit. The underflow from the secondary separator is returned to the 6 in. E.R.G. sand pumps under the primary screens and is treated in the jigs to ensure that no values escape.

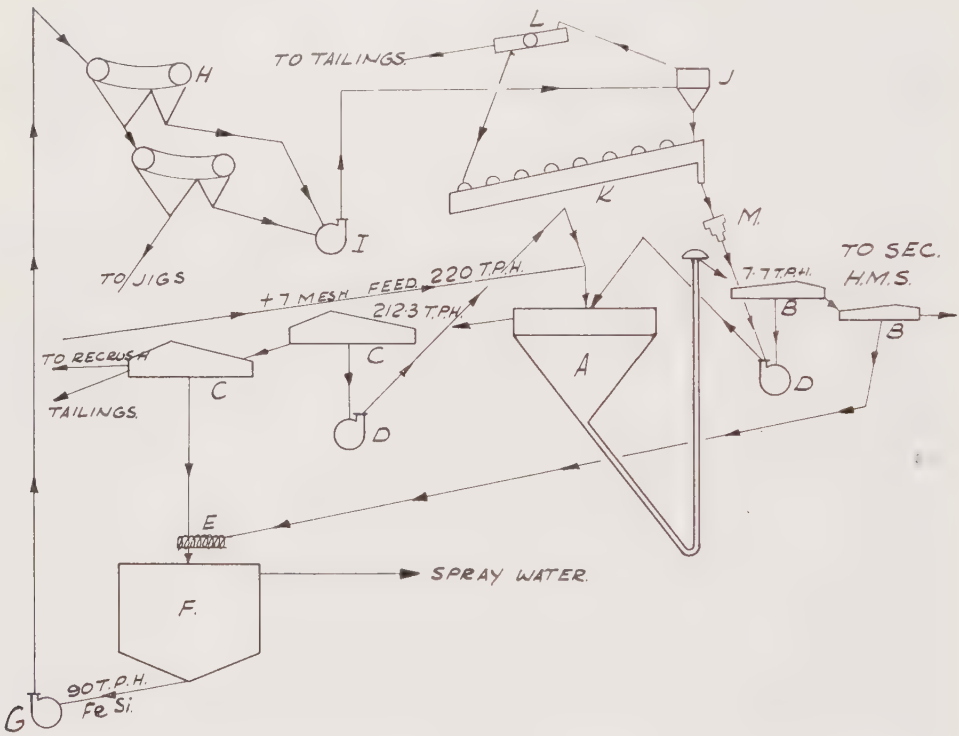
The cleaned ferro-silicon is pumped by a 4 in. Vacseal pump to an 8 in. cyclone at about 2·0 s.g. The spigot discharges clean ferro-silicon at +3·2 s.g. and the cyclone overflow passes to a $1\frac{1}{2}$ ft. by 3 ft. Aero-vibe screen with 24 mesh (0·023 in.) aperture to separate entrapped flat silicates which are discarded and the −24 mesh material drops into the boot of a 48 in. Akins spiral classifier used as a densifier. The cyclone spigot discharge and the densifier discharge are demagnetized before returning to the circuit via the sink medium return pump.

The cone float overflows to two 5 ft. by 14 ft. Allis Chalmers low head screens in parallel, with a top deck of $\frac{3}{8}$ in. by $1\frac{1}{4}$ in. carbon steel screen cloth, and a bottom deck of 9 mesh stainless steel screen cloth. The ferro-silicon drains away and is returned to the cone by the 5 in. Wilfley float medium return pump and an 87 h.p. motor.

The float material then falls onto the pair of 5 ft. by 14 ft. Allis Chalmers low head float wash screens where five top sprays and two bottom sprays wash the ferro-silicon from the float particles. The $+\frac{3}{8}$ in. by $1\frac{1}{4}$ in. material falls to a 30 in. conveyor for conveyance to the recrush plant. The $-\frac{3}{8}$ in. by $1\frac{1}{4}$ in. +9 mesh material falls to a 36 in. conveyor for disposal as waste on the tailings bank.

The $+\frac{3}{8}$ in. by $1\frac{1}{4}$ in. float material is elevated to bins in the recrush plant by 42 in. conveyors and a distributing tripper. Quadrant doors feed onto 30 in. feed conveyors to six $5\frac{1}{2}$ ft. Symons "Short Head" cone crushers set to produce 80 per cent $-\frac{3}{8}$ in. slot material. Each crusher product is discharged to a 30 in. conveyor which feeds two Symons rod-deck screens with ¼ in. rods set to ⅜ in. aperture. The oversize is returned by 30 in. conveyors to the recrush bin and the undersize is conveyed to the 1,500 ton bin by a 30 in. conveyor for recovery of small diamonds released in the recrushing.

Fig. 8 is a diagrammatic representation of a cone circuit.



Legend

A. Four 16 ft. cones
B. Eight 10 ft. by 3 ft. sink screens
C. Sixteen 14 ft. by 5 ft. float screens
D. Eight 5 in. medium return pumps
E. Four Magnetizing blocks
F. Four 20 ft. Thickeners
G. Eight 4 in. underflow pumps
H. Eight prim. 4 sec. mag. sep.
I. Four 4 in. vacseal pumps
J. Four 8 in. cyclones
K. Four 48 in. densifiers
L. Four 36 in. by 18 in. aerovibe screen
M. Four demagnetizing coils

Fig. 8

SECONDARY H.M.S. PLANT

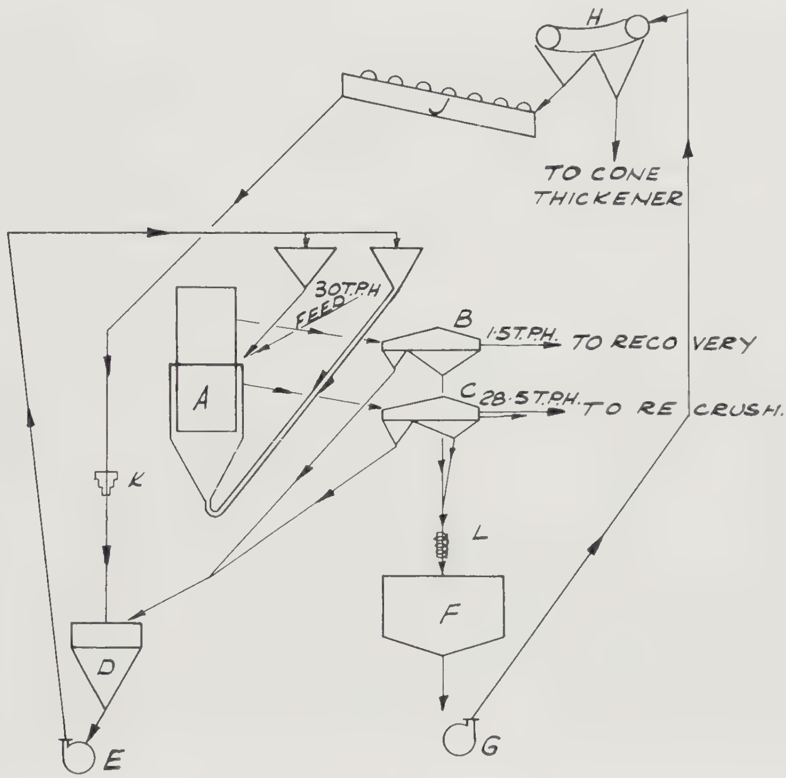
The cone sink concentrate is conveyed to a 4 ft. by 13 ft. "Wedag Heavy Medium" wheel separator in which atomized (spherical) ferro-silicon is used with a medium specific gravity of 3·15 and a viscosity—by the modified Stormer viscosimeter—of between 11 and 18 centipoises. It has been found necessary to stabilize this ferro-silicon by adding slime or bentonite to the medium.

The drum (wheel) produces a sink concentrate of between 10 and 12 per cent of the cone concentrate which is drained and washed on a 3 ft. by 10 ft. Allis Chalmers low head screen fitted with 9 mesh stainless steel screen cloth and is then conveyed to the recovery bin.

The float material is drained and washed on a single 5 ft. by 14 ft. Allis Chalmers low head screen with a top deck of $\frac{3}{8}$ in. by $1\frac{1}{4}$ in. screen and a bottom deck of 9 mesh stainless steel screen cloth. At present all float material is returned to the recrush, though it may be possible after exhaustive tests to discard all this material to waste.

It is proposed to increase the specific gravity of the drum separator medium to 3·35, thereby reducing the quantity of concentrate to the recovery to a minimum.

Fig. 9 is a diagrammatic representation of the secondary H.M.S. drum separator circuit.



Legend

A. Drum wheel separator
B. 10 ft. by 3 ft. sink screen
C. 14 ft. by 5 ft. float screen
D. Medium ret. pump hopper
E. Medium return pump
F. 10 ft. thickener
G. Underflow pump
H. Magnetic separator
J. Densifier 60 in.
K. Demagnetizing coil
L. Magnetic block

Fig. 9

GREASE TABLE RECOVERY PLANT

H.M. concentrate

The cone concentrate is evenly distributed over the length of a 700 load recovery bin by a tripper conveyor. There are five equally spaced draw off points under the bin feeding five 4 ft. by 8 ft. 6 in. Gyrex screens fitted with $\frac{1}{2}$ in. top deck and a 10 square mesh stainless steel bottom deck. The control of feed to the screens is by means of a regulated water jet in the back of the chutes feeding the screens. Each screen discharges its products to four grease tables. The $+\frac{1}{2}$ in. material is fed to one table and the $-\frac{1}{2}$ in. $+10$ mesh material—which contains the bulk of the diamonds—to three tables. The -10 mesh material is pumped to a dewatering tank and is gravitated to the two 3 ft. by 6 ft. Tyrock screens fitted with 24 mesh—0·0237 in. aperture—where the jig concentrate is washed free from slime. This is then fed to eight sand tables.

Fig. 10 is a general view of the grease tables.

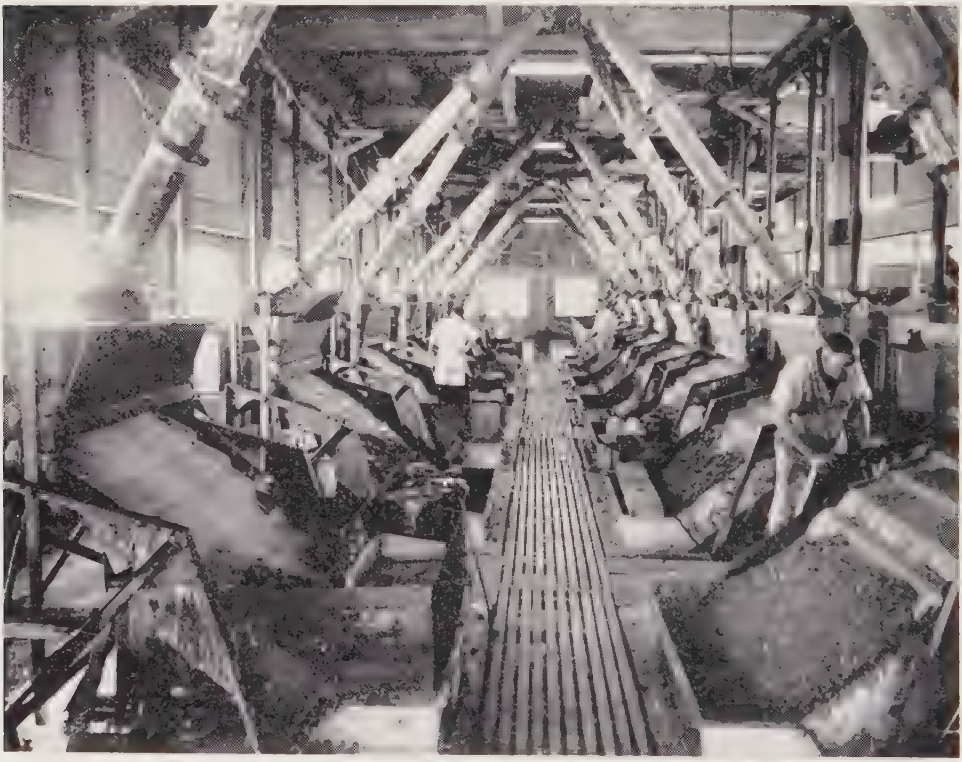


Fig. 10

The grease tables consist of an aluminium casting 4 ft. by 3 ft. divided into four 9 in. steps of 4 in. drop, vibrated by "Locker Traylor M.3" electromagnetic vibrators, acting contra to the flow of material down the tables.

The table top is covered with base petrolatum covering $\frac{3}{8}$ in. thick over which is spread a thin topping of vaseline and wax, the proportion of each topping mixture is dependent on the circulating water temperature and size of feed.

Concentrate and wash water are fed onto the vibrating grease table, which retains the diamonds because of the water repellent property of the natural mine diamond. Water-wetted particles of other material are washed off the table.

While the intensity of table vibrations is standard throughout, the table inclination is set to accommodate the size of feed, and the volume of wash water flowing varies in terms of the table inclination and feed size.

Fig. 11 is a close-up view of a grease table showing diamonds adhering to the grease.

Fig. 11

The rate of feed of the $-\frac{1}{2}$ in. material to the three tables set at 11° slope is 5-6 loads per hour for cone concentrate and about 1-1$\frac{1}{2}$ loads per hour of drum concentrate. The feed rate of the $+\frac{1}{2}$ in. material to the table set at 25° slope is 10-13 loads per hour for cone concentrate and for the drum concentrate is so little as to be insufficient to load the table. Each table has four vibrating steps and a static step. The first step retaining 90 per cent of the diamonds is skimmed every 45 minutes for cone concentrate and the skimmings are put in a pot with $\frac{1}{16}$ in. perforated walls and a tight fitting lid. When full, the pot is sealed for security and bolted in a cradle which is rotated in boiling water. The grease is melted and runs out through the perforations, collected and re-used. The pot is then passed to the diamond sorting office. The remainder of the steps are skimmed every 1$\frac{1}{2}$ hours and the product retained in separate pots. These pots are numbered and the diamonds recovered from them are recorded separately.

Jig concentrates

The sand from the secondary jigs, together with the undersize of the Gyrex screens, is washed on two 3 ft. by 6 ft. Tyrock screens to remove slime and then fed in a particle thin layer over the eight sand tables similar to the tables previously mentioned but with seven steps, the slope of the tables being 7°.

It may be mentioned that the tailings from the $+\frac{1}{2}$ in. coarse table is returned to the recrush for comminution. All other tailings are discharged to tailings dumps.

TESTING FOR LOSSES

Regular testing of the behaviour of diamonds in the jigs, cones and drum separator is carried out by adding a small diamond, which has been drilled and rendered radioactive by the insertion of a piece of radioactive cobalt wire, to the particular feed. The peregrinations of the radioactive diamond are followed with scintillators until discharged from the unit to either concentrate or tailings where the radioactive diamond is caught in a sample cutter and removed. The time of retention in the unit is also noted. Numerous repetitions of this procedure are made at each machine and the behaviour of the machine assessed therefrom.

There is no really satisfactory method of testing for diamond in tailings as the concentration of diamond in the new blueground to the plant is only one part in 13,000,000, necessitating an enormous sample to be of any value.

The H.M.S. cone tailings are sampled on the way to the tailings bank by inserting a cutter in the stream of a conveyor discharge and elevating the sample so cut to two grease tables in parallel. This test unit is run daily on different shifts. The rate of flow is checked frequently and the time of running kept accurately so that the actual tabled material can be assessed moderately accurately, and the product from each day sorted and weighed so that the loss can be calculated.

The recovery tables tailings are sampled weekly and retabled and the losses calculated. To check this the separate weighing of the 4th and 5th steps of the H.M.S. tables is totalled for the month. From years of experience we assume that an equivalent quantity of diamond has been lost and this figure calculated as a percentage agrees very closely with the weekly sample.

A further advance in testing for diamonds is being developed at the Diamond Research Laboratory. This is a photo-separator. The material to be tested—grease table tailings—is fed to a slow moving conveyor at such a rate to give not more than particle thickness. This conveyor passes through a strip source of light across the belt and under a curtain. The diamond reflects the light beyond the curtain and a photo-electric cell operates a flap in the discharge and removes that portion of the material containing the cause of the flash of light. For the most part the gangue materials do not reflect sufficient light to activate the photo-electric cell.

DIAMOND SORTING

The boiled out diamond pots are slid down a chute to the sorting office where they are opened and the contents removed. The contents are boiled in caustic soda solution to ensure that all the grease is removed and then dried. The dry concentrate is then screened using the diamond sieves for this purpose, mainly for ease in sorting. The finer sizes of the heavy medium concentrate are treated in the electrostatic separator which after half a dozen passes gives a fairly clean diamond concentrate and a fairly diamond free tailing both of which are hand sorted.

Fig. 12 shows the diamond sorters at work.

The sand table concentrate is first degreased by boiling in caustic soda solution and then tube milled in a 6 in. by 12 in. tube mill with $\frac{3}{8}$ in. ball bearings as grinding medium to slime the gangue. The tube mill product is washed on a 60 mesh screen and the undersize discarded. The diamonds are then separated from the other pieces of hard waste by floating on water in a beaker and removed by pouring off the raft of diamonds so floated. The material remaining in the beaker is sorted by hand to remove any few non-floating diamonds.

Fig. 12

The diamonds from each concentrate are then washed in trichlorethylene to remove the last of the grease and are then sized and weighed and the various weights of the sizes recorded. The daily recovery is calculated from these weighings.

At the end of each week the production is allowed to stand in 40 per cent hydrofluoric acid for two days to remove any gangue material adhering to the diamonds. The diamonds are then removed from the hydrofluoric acid, washed with water and boiled in hydrofluoric acid, washed with water again and finally washed in rectified spirits of wine. The parcel is then checked through again to ensure that no extraneous material is contained therein and weighed ready for shipping.

No sorting of gem from industrial diamonds is done on the mine, 100 per cent clean diamond being shipped to the central sorting office in Kimberley.

TAILINGS DISPOSAL

Tailings from the bottom deck of the float screen fall to a 36 in. transverse conveyor which deposits onto a 48 in. conveyor which also receives the sands from the dewatering classifiers in the jig plant. This 48 in. conveyor discharges to a surge bin of sufficient capacity to take the float content of the four cones should there be any stoppage on the single line tailings conveyor system.

Drawing off from the bottom of the surge bin are two 36 in. conveyors in parallel either of which can handle the full load of tailings discharging to a single line 36 in. conveyor travelling at 300 ft. per minute. This conveyor, 3,240 ft. long, is divided into four units each fitted with tandem drive as well as a stand-by motor and gearbox, bearing approximately north. At the head pulley of the fourth conveyor are two transverse conveyors each 36 in. wide with head pulley drive conveying east and west At the eastern and western head pulleys 36 in. conveyors again bear to the north. The eastern conveyor travels downhill at 6° to cross the valley to a ridge on the northern side enclosing an area with a catchment of some 2,000 acres for use as a slimes dam which will last for many years to come.

The tailings are deposited with a Stevens Adamson thrower. They are speeded up from 300 f.p.m. to 600 f.p.m. on short conveyors and thrown down the throat of the thrower unit onto a short endless belt travelling at 2,400 f.p.m. The top of this endless belt is depressed between the head and tail pulleys by a pair of "bull wheels" which gives the tailings a trajectory as in Fig. 13. With a range of about 40 ft. and swivelling through 270° it is possible to build a bank 80 ft. wide at the top, as was done across the valley.

As the tailings bank advances, extensions are made by adding a stringer of 28 ft. at a time next to the thrower assembly and adding 56 ft. of conveyor belt with

Fig. 13

"Comet" fasteners. In order to reduce the number of joints in the belt with consequent leaks into the return belt three extension pieces were used, 56 ft. (one stringer), 112 ft. (two stringers), and 224 ft. (four stringers). When all three pieces had been used a piece of 448 ft. (eight stringers) was vulcanised to the main belt and the process of advancing repeated.

SLIMES DISPOSAL

Owing to the rolling nature of the surrounding country, shown in Fig. 14, and the necessity of preventing slime contaminating streams, the construction of slimes dams was expensive.

Fig. 14

Small walls of approximately 15 ft. high were "bulldozed" from the country soil with wing walls running into the rising ground. This enclosed an area where the slime could be deposited.

Fig. 15 shows the original earth wall with a strong growth of grass and the feebler growth on the slime wall above.

The slime pumped from the underflow of the 100 ft. thickeners with eight Dorrco Triplex diaphragm pumps grades from 28 mesh down to $\frac{1}{2}$ micron but does not behave like any tube mill slime. The coarse particles settle as soon as the velocity drops and the fine particles only appear to settle on evaporation of the water. It was therefore necessary to deposit the coarse particles round the perimeter of the dam where the wall was to be raised. This was done by running the slime through 10 in. "Everite"

Fig. 15

pipes on trestles 3 ft. high. Each Everite pipe was drilled and fitted with a 1 in. spigot onto which was attached a 1 in. hose 12 ft. long, the open end being placed in the dam. As the slime comes out of the hose it fans out, the velocity drops, the coarse particles settle forming a beach and the fine slime is pushed to the back of the dam. Between 60 and 70 discharge hoses handled all the slime pumped and were progressively traversed along the wall at from 3 to 8 pipes per hour, depositing the sand in a beach-form on the perimeter of the wall. When the head of the beach reached the top of the wall, the slime was diverted to the next dam, the pipes were removed and the wall was raised 18 in. to surveyed pegs and stepped in 4 ft. Fairly clean water was drawn off by penstocks and ducted to a central pumping station where it was pumped to a storage dam for use in the plant as make-up water.

The depositing life of these dams is limited to a safe wall height, considered to be about 60 ft. vertical. It was necessary therefore, to consider a more permanent system of containing the slime, a matter of $2\frac{1}{2}$ million gallons of water and 850 tons of slime per day.

A wall (of tailings) was thrown across a blind valley with a catchment water shed of 2,000 acres. The wall is 1,500 ft. long, 80 ft. wide at the top with a maximum depth of 250 ft. Slime is deposited on the upstream side of the wall in the manner described above. The sand and slime settles out with a natural classification from coarse to fine driving the water and micron material to the upstream pool.

Fig. 16 shows the completed tailings wall with the slimes dam upstream on the right.

The beach formed has a gradient 1 in 20 and forms an impervious blanket between the liquid slime and the porous wall of tailings, with a free board of approximately 5 ft. One million gallons of decanted water is returned to the plant from the dam, daily. Fig. 17 is a close-up view of the slime beach.

Fig. 16

Fig. 17

This dam has been the subject of investigation by a team of civil engineers for the past two years.

Appendix I is a break-down of the European and African labour complement required for the running of the plant. Appendix II is a distribution of the plant costs per load and per short ton for the year 1959.

ACKNOWLEDGEMENT

I wish to thank Mr G. S. Giles, Consulting Engineer, Anglo American Corporation of S.A. Ltd., for permission to publish this paper.

REFERENCE

1. Adamson, R. J. Presidential Address, "Some account of diamond winning practices in Southern Africa." *Journal of the South African Institute of Mining and Metallurgy*, Vol. 60, No. 1., August, 1959.

(Appendix I on page 167)

APPENDIX II

Costs:

	Per load treated	Per short ton treated
Primary crushing	2·46d.	3·07d.
Secondary crushing	1·58d.	1·98d.
Primary screens	1·34d.	1·67d.
Sands treatment	1·69d.	2·12d.
Separation	8·76d.	10·95d.
Recovery	1·04d.	1·31d.
Tailings disposal	2·31d.	2·89d.
Diamond sorting	0·17d.	0·21d.
Waste rock disposal	0·30d.	0·37d.
Slimes disposal	2·27d.	2·83d.
	21·92d.	27·40d.

APPENDIX I

LABOUR

PLANT SUPERINTENDENT

Plant Superintendent — Assistant plant superintendent; Diamond sorters, Graders, Clerk, Technical assistant, Beltman; Assistant plant superintendent

	Diamond sorters	Graders	Clerk	Technical assistant	Beltman
	3 1 African	2 6 Africans	1 1 African	1	2 6 Africans

Assistant plant superintendent — Foreman (Primary crusher, Waste rock, Secondary crushers); Foreman (Tailings, Slimes)

Foreman (Primary screens, Jigs, Yardman, H.M.S.)

Assistant plant superintendent — Secondary H.M.S.; Foreman Recovery

	Primary crusher	Waste rock	Secondary crushers	Tailings	Slimes	Primary screens	Jigs	Yardman	H.M.S.	Secondary H.M.S.	Foreman Recovery
Europeans	6	2	3+1 for leave	4+1 for leave	4	3	3	1	13+1 for leave	2	6+1 for leave
Africans	126	24	45	39	36	30	18	12	78		10

TOTAL: 67 Europeans
432 Africans

MAINTENANCE LABOUR:

Europeans	Fitters	Boilermakers	Electricians	Riggers	Welders	Total
Foremen	1	1	1	—	—	40
Artisans	11	6 + 4 Screensmen	5	3	1	
Apprentices ...			7			
Africans	24	23	7	12	2	68

NATIVE LABOUR AND PAYMENT ON PREMIER DIAMOND MINE

By **E. J. B. SEWEL*** (Visitor)

DESCRIPTION OF PROPERTY

Premier Mine lies some 23 miles to the east of Pretoria, at an altitude of about 4,740 ft. above sea level. The mining of the diamondiferous kimberlite or "blueground" is carried on in a vertical pipe of volcanic origin. An average of 370,000 tons per month is hoisted to surface from which over 100,000 carats of diamonds are produced. These diamonds are 80 per cent industrial, but both gem and industrial are of high quality.

About 580 Europeans and 2,500 Natives are employed, of which some 250 Europeans and 1,400 Natives work below surface.

Europeans are housed in quarters which form the village of Cullinan, named after the founder of the company.

Whilst the bulk of work is done on the day shift, production and washing of blueground continues on a treble-shift basis throughout the week.

NATIVE LABOUR SYSTEM

Natives are employed on a basis which differs in three main features as compared with general gold mining conditions in South Africa.

These are:

1. Close compounding.
2. Self feeding.
3. Fortnightly payment of wages.

1. *Close compounding*

For reasons of diamond security, which are obvious in considering the value of a gem diamond as compared with its size and weight, nearly all Natives are kept within the security fence which surrounds the mining area, and stay within this area for the duration of their contract. Fig. 1 shows the outside of the closed compound and Fig. 2

Fig. 1

Fig. 2

*Assistant General Manager, Premier (Transvaal) Diamond Mining Co. Ltd.

the administrative block within. The contract is for four months in the compound, and not for a specific number of shifts worked. It would thus be possible for a Native to be taken ill on admission and do no work whatsoever during his contract. Continuation after the contract period is by desire of the Native concerned and the average stay is about 10 months.

Recruiting of Natives has not been necessary as more than sufficient present themselves for employment, and of these over 60 per cent have previous service at Premier. This, in itself, is a tribute to the conditions of employment.

The compound manager and the local Peri-Urban Labour Bureau work together in facilitating the formal procedure required from Native labourers. They are escorted to the Labour Bureau for the issue of the usual 72 hour work-seeking permits, and should there be no employment available within 72 hours for those who are acceptable, selected labour is returned by the compound manager with a recommendation that the period be extended. Labour is, however, taken on and attested three times per week and it is therefore a simple matter for the compound manager, knowing his forthcoming requirements, to guarantee employment to those whom he has recommended for extension of permit.

Not all the Natives are within the security fence. Approximately 300 are required for village maintenance and to run the large tailings dumps and slimes dams, all of which are outside the fence. These are either accommodated in an open compound adjacent to the "awaiting employment" compound or a small number "squat" on the surrounding farms.

The three main divisions of our labour force are Mpedi, Transvaal Basutos of the Sotho group, followed by Shangaans from P.E.A. of the Tonga group, and Basutos from Basutoland of the N'Guni group. Together with a small number of Xosas from Eastern Province, and a few of the other groups of Southern Africa, the present tribal distribution of the Closed Compound is as follows:

Mpedi	51	per cent
Shangaans	22	,, ,,
Basuto	15	,, ,,
Xosa	8	,, ,,
Other	4	,, ,,
	100	,, ,,

Figs. 3-4 are photographs of typical underground Native mine workers and Fig. 5 is of the surface tailings bank.

2. *Self feeding*

It has generally been the practice in the mining industry to include feeding and accommodation as part of wage payments, but on this mine, feeding is not included with wages, and the contract requires Natives to provide food at their own expense. It would appear that this system is greatly liked, as the tribalised Native much prefers the food which he has prepared himself as compared with any of the balanced diet foods that may be supplied to him. On account of the closed compound, the company operates a shop within the area for the main purpose of enabling the Native to purchase his food cheaply but again being a closed compound, other commodities such as clothing and hardware are also stocked. This shop is run on a non-profit basis, prices being arranged so that any profit earned covers only the cost of running the shop plus the cost of the free "mahewu" and kaffir beer issues that are made. Thus it will be seen that the Native is purchasing his food more or less at cost rather than at retail prices.

Fig. 3

Fig. 4

As food is not included in wages, the Native becomes entitled to cost of living allowance under Government Regulations and this varies from 1/10d. on the basic surface rate of 2/9d. per shift to 4/7d. per shift for those on "incentive" bonus who may be earning as much as 10/- per shift.

Records show that the average amount spent per day on food is 1/4d. as compared with 1/2d. for cost of feeding on the gold mines. In comparison with cost of living allowance it is seen that the Native shows a small profit, yet is spending only a little more to prepare his own food than if he were to be fed by the company. It may be said that the Native will deny himself food if he has to pay for it himself. Experience here has not borne this out, and records show that he is eating a reasonably balanced diet. It can also be appreciated that with the cost motive absent in the shop, it is possible to supply nutritive foods, which may be unpopular eating, at very cut prices and this particularly applies to vegetables. In addition, careful medical checks reveal food deficiencies and watch is kept on any Native who is not feeding himself in a reasonable manner.

In general, self feeding is greatly appreciated by the Native who accompanies his cooking and eating with his favourite hobby, gossip, and one has the opinion that this system makes for a more contented labour force than where food is supplied. Against this, it is seen that the company is actually the loser, for as pointed out previously, the cost of feeding *en masse* as quoted above is about 6d. less than the minimum cost of living allowance which we are obliged to pay.

Fig. 6 shows one of the stoves in the communal kitchen in the busy period at the end of the morning shift.

3. *Fortnightly payment of wages*

Usual mining practice is to pay only when 30 shifts have been worked, but under self feeding, Natives must have money for purchase of food at shorter intervals than this.

Accordingly all Natives are paid fortnightly, irrespective of shifts actually worked, and it is laid down in the contract that such deductions as the company may be entitled to make may not be made if the Native is left with less than 15/- in cash per fortnight.

Fig. 5

Fig. 6

In practice, the pay is divided into two, surface and underground being paid separately on alternate weeks.

Basic rates of pay are 2/9d. per shift on surface and 3/3d. per shift underground Most of the underground complement are on incentive bonus and with cost of living allowance, as mentioned previously, average payments per shift come to 5/4d. for surface and 7/5d. for underground, the overall average being 6/6d. These figures are based on the first three months of 1960.

The usual facilities are provided for savings bank and home remittances and, in this connection, it is interesting to note how much is actually saved from wages in this manner. In the first three months of 1960, a total of £64,033 was paid out in Native wages. Of this amount, £18,279 or 28·5 per cent was placed in savings bank and £4,865 or 7·6 per cent sent home. This illustrates one of the benefits of closed compounding to the Native, where he is not subjected to exploitation by high prices, illicit liquor and prostitution, as obtains around most open compounds. The gold mining industry in 1959, paid £26·2 million in wages and of this £4·75 million was put into voluntary or enforced home remittances and savings banks. This represents 18·2 per cent of the total wage bill as compared with 36·1 per cent for Premier, but it must be remembered that in an open compound as in the gold mines, postal orders and other forms of remittance can be used without going through the official records. Nonetheless, the figure for Premier speaks for itself.

MEDICAL

The usual medical facilities are supplied, including full hospital facilities and 24-hour dressing station attendance in the compound. Very careful watch is kept on weight and the following average figures illustrate the improvement in physique whilst in the compound:

Average weight on admission	137·2 lb
Average weight on discharge	139·0 lb

These figures are based on the first four months of 1960, but have not varied greatly over a number of years.

Increasing attention is being paid to the Native worker by the Pneumoconiosis Board, and X-rays and examination must now take place on engagement and thereafter every six months. Some 1,000 X-rays per month are made, using the miniature method.

Figs. 7-9 illustrate various aspects of the medical procedures followed.

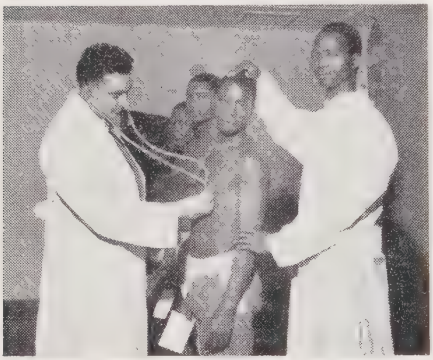


Fig. 7

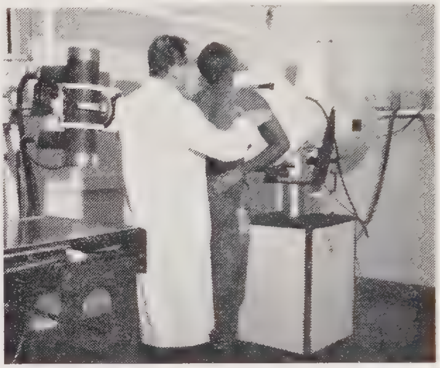


Fig. 8

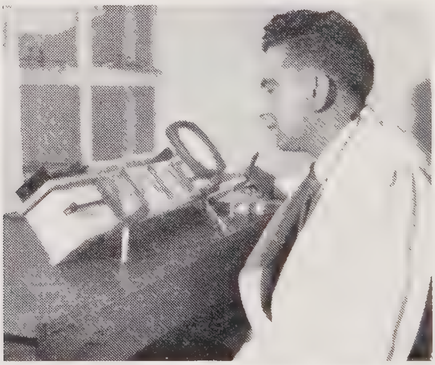


Fig. 9

NATIVE LABOUR CONTROL

The control of Native gangs is obtained through a crush and Native checker system, both of which are under the authority of the study department. The setting of complements for the various gangs is not essentially controlled by the study department, but rather by the various heads of departments with the approval of the general manager or assistant general manager. This has arisen because it has been found that complements required are so standardized that little variation is necessary from the norm: Study does, of course, investigate particular or special cases.

Apart from this, the study department, responsible directly to the assistant general manager, control all gang checking, bonus payments, shift payments and overtime worked, the returns being submitted in such a form that the Time Office is purely an office for machining the payroll. In addition, all heads of departments plus mine overseers and shift bosses receive daily labour files or returns of their daily state of labour together with the set complements. Study department personnel required for Native labour control is as follows:

European

Chief of study and 1st assistant (part time only on labour)

Gang clerk	1	Male in charge of gang office
Asst. gang clerk	1	Male
Bonus clerk	1	Female
Checking clerk	1	Female
Crush supervisors	3	Male, day's pay. Control of crush and also the lamproom.
	7	Excluding chief study officer and assistant.

Native

Boss boy	1	Supervises all aspects of crush and checking
Crush clerks	6	3 in surface crush, 3 underground crush, 24 hours, 1 each per shift.
Underground checkers ...	10	8 underground, 1 surface closed, 1 surface open.
	17	

Crush

The crush itself, being situated in the compound, is supervised by the Native controller who is responsible to the chief study officer for the running of it, and is actually a very convenient base for his normal work. The self-service lamproom is combined with the underground crush and leads direct to No. 2 Vertical Shaft which is used for the raising and lowering of Natives. The surface crush adjoins the underground layout and leads to No. 1 Shaft, where the workshops and treatment plant are situated.

The lane and board system in a "crush" is fairly standard practice in the mining industry. Four lanes are provided for underground labour, one for permanent day shift and three for the treble shift working. All gangs are on removable battens to facilitate movement from lane to lane. Surface labour has a further three lanes for treble shift working.

Each Native has two metal discs above each other on the gang batten, the top one with his number and the bottom one painted red on one side and black on the other. The coloured disc records whether or not he is at work, the red indicating "at work" and the black indicating "in compound". Special discs are used to indicate "hospital" or "sick parade".

As soon as the shift is out, the checkers work out the replacements required and, drawing them from a pool gang, proceed immediately to those gangs requiring to be made up. The pool gangs are operated on all three shifts, the strength being based on average absentees over a long period. In the case of the surface closed compound labour, the pool is 10 for 800 on strength and for underground, is 24 for 1,400 on strength.

Those absent without leave are then marked for parading to the compound manager.

The crush boards are operated by crush clerks, one per operating lane, and assisted by the checkers. Each Native has his personal work card which covers the pay period of two weeks. It is stamped as he goes through the crush with the hour and date, and similarly counterstamped on his return to the compound. This ticket is his receipt for work done, and overtime and bonus measurements are also entered on it. It must be signed by the ganger for whom he works, but more notice is actually taken of date and time stamps by the crush.

After the shift is out, the crush supervisor, in company with the crush clerk, records the absentees for transmission to the study gang office on a special form. Gang numbers of all gangs at work are also recorded as a check in the gang office, otherwise the "absentee only" system is used. Should a Native or a complete gang be called out for overtime before the normal shift time, then the time through the crush—"time out"—is recorded on another form, and the time back through the crush—"time in"—would be "normal", and the overtime hours worked would then be entered from his work card. If the Native or gang work overtime after normal shift, then "hours" and "time in" are recorded as he goes through to the compound. The form is then transmitted to the study department.

STUDY DEPARTMENT AND PAYMENTS

The basis of records in the gang office section of the study department is the gang record. Transfers, discharges and engagements are shown on this sheet but a blank square against number and date counts as a shift worked, only the absentee squares being shown with a letter to indicate the type of absence e.g. H.=hospital, C.=sick parade and missed shift, T.S.=temporary surface (medical), A.=a.w.o.l. etc. The lowest shift worked from the top of the sheet is closed off with a thick line. This is to avoid errors in totalling blank squares, squares for Natives permanently out of the gang being ruled across.

From the daily totals of shifts worked in this record and from the lists of medical and other absentees, the total daily strength is obtained. By applying the engagements and discharges since the previous day, the previous day's strength is found and this must correspond with the figure supplied by the compound. This check ensures that no totalling error has been made and is essential to the system.

At the same time, the bonus clerk is entering the bonus and overtime cards. For overtime, the crush return is entered on each Native's card, the cards being arranged in gang order. To complete the overtime check, all European gangers have to make out an overtime slip. This is passed to the gang office who check on it that the overtime booked on the work card agrees with the slip and add the "time through crush". The slip is now passed back to the official in charge who checks and initials it, using the "time through crush" to make a check on the amount of overtime booked.

Incentive bonuses are divided into three types of payment:

1. Individual payment, such as footage drilled in rotary or percussion drilling.
2. Gang payment, such as development cleaning where the lashing gang shares the footage advanced by a developer blasting up to six ends per day.
3. Sectional payment, based on tonnage hoisted and applying to types of work such as grizzlies or transfer level operation.

In the case of individual and gang payments, the bonus card is entered up from the ganger's returns. In the case of tonnage bonus, the amount is calculated for the fortnightly period, averaged per shift, and recorded for all Natives who are shown in the gang record as being engaged on that particular type of work for the day. The ganger's returns in this type of payment are used as a check against the gang record.

Native time office

With the study department's daily returns of absentees, and the set of bonus and overtime cards applying to the pay periods, it is now seen how the Native time office becomes purely a machining department, converting the returns into payroll form with the use of accounting machines, and two female clerks are more than sufficient to do this.

Basically, each Native has a record card which states amongst other details, his rate of pay and amount for the standard 12 shifts of a pay period, these cards being kept in numerical order.

As the daily absentee lists are received, an absentee card is made out, the purpose of which is to record the amount to be paid at the end of the pay period after absences are deducted. Naturally the bulk of Natives work the full 12 shifts and no absentee card is needed.

When the payroll is to be made up, the absentee cards are first transferred to the record cards in pencil. The bonus and overtime cards from the study department, now sorted into numerical order, are placed on the accounting machine rack, and the payroll is made up by the operator taking the record card from the trolley, inserting it in the machine, typing the standard information and inserting the bonus and overtime as read off from the pile. As each bonus and overtime card is used it is flipped over to reveal the next card. The machine now records the typed information on the record card, the payroll sheet and the pay slip, makes up the pay total for the individual Native and carries the column totals for summarizing at the end of each payroll page. It should also be noted that a record is carried of "dusty occupation" shifts as required by the Pneumoconiosis Board. This is obtained through the study department, who designate each gang as "D.O." or "N.D.O.", depending on type of work done by the gang. (i.e. dusty or non dusty occupation.)

At the completion of the payroll, a comparison is made between total shifts worked as thrown out by the machine and as returned daily by the study department, which is a check on the accuracy of the payroll.

In the actual pay at the compound, the native controller, with the paymaster, makes a list of all pay queries by Natives, and these can, of course, be substantiated on the spot by the production of their daily work cards. This list is then investigated by the chief study officer and timekeeper thus enabling the reason for any error to be discovered as well as the rectifying of it. It is interesting to note that on most occasions, there are no queries for the surface pay, and of an average of half a dozen for the underground pay, only half will be found to require correction.

Distribution of Native pay

For many years now, the distribution of Native wages has been on a fixed percentage basis rather than "day to day" returns and the reasoning behind this is as follows:

If the complement setting is correct and the complement is 100 per cent, then set complements are just as accurate within the statistical limits required as actual shifts worked. As total complements do not vary greatly from month to month, a percentage, based on complement, is sufficient to split the shift payments. Test runs over a period are used to set the percentages for bonus and overtime, which exist as separate totals in the payroll.

To elaborate on the above, the complement setting is continuously checked by the underground and surface checkers and Natives working in their wrong gangs or wrong jobs are immediately reported. As mentioned earlier in the paper, sufficient labour is offering to enable us to work with a full complement at all times.

A complication exists in that the payrolls, being made out fortnightly, do not coincide with the Mine month. This is adjusted by taking a shift pro-rata for the odd days beyond the last completed payroll. This amount is then subtracted from the payroll when completed and forms the start of the new month. In this way, a slight error might exist for the last days of the month but this is always corrected in the following month.

Where, however, a change of policy results in a local shift of labour from one type of work to another, a safeguard is applied to the percentages.

When the gang clerk submits his complement return to the statistician for the monthly figures, a variation of more than 200 shifts worked, which represents about 0·5 per cent of the total, is recalculated as a new percentage of the total and advised to the time office. In this way any big variation in complement is allowed for in the fixed percentage, but the complete figures are not altered, the shift split now being based on more than 100 per cent or less than 100 per cent, depending on the variation advised.

In actual practice, the fixed percentages are recalculated every 6-12 months and under this system, the work involved in distribution is very little.

TRAINING

New engagements

As discussed previously, engagements are made according to the advice of discharge given a week ahead to the compound by the Native.

In the case of surface Natives, engagements are made twice a week. On entry to the compound in the morning, they are lectured by the compound staff on hygiene, use of the shop, issued with any necessary equipment and made acquainted with the general compound rules regarding money matters, including procedure for savings bank and remittances.

In the afternoon, they are lectured on safety at the Native instructor's school and on the following day, work one shift with the Native they are going to replace. Although paid for, this shift is regarded as a training shift and is not charged against the labour strength of the official concerned. On the following shift, the Native for discharge is off work and the "new boy" is on his own.

Although the safety lecture is reasonably comprehensive, and is supported by practical demonstrations in the school, it has been realized that pre-work training of surface Native labour in all the varieties of jobs that are encountered on surface is very difficult, and more reliance is placed on the one shift that is worked as a training shift with the Native that is being discharged.

This is not the case underground where most Native jobs are standardized and the safety drill can be instilled under actual conditions before he starts work on his own.

Underground labour is engaged three times a week and, after going through the same compound procedure advice, as outlined for surface labour, Natives are lectured on general underground safety in the afternoon. On the following day, the new underground labourer is taken on a conducted tour of the workings and, in addition to being shown how the various jobs are done, the safety aspect of each is stressed, together with precautions to be observed in travelling to and from the working place.

The afternoon is taken up with the showing of safety films and on the following day, the recruit is ready for normal work.

Boss boys

All underground boss boys are trained by the Native instructor for two weeks before being certificated by the assistant underground manager. Their selection is made by the underground officials after investigation by the compound staff, due regard being paid to the tribal percentages in the compound as a whole.

The course consists of alternate days' surface and underground instruction, the surface instruction being of a practical as well as a theoretical nature. Use is made of safety films and models, together with a full scale representation of underground tunnels, and may be seen in Figs. 10 and 11.

Fig. 10

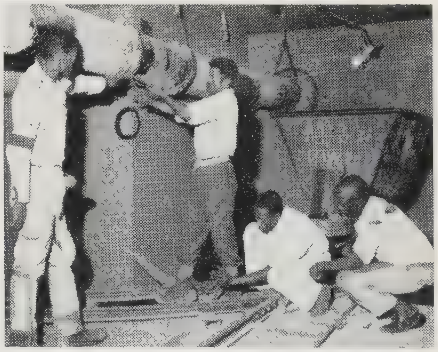


Fig. 11

With extensions to the compound, it is hoped to improve the facilities in this section to cover all forms of underground work at full scale on surface.

The underground instruction is through the medium of boss boy instructors for each type of work, the European Native instructor spending a certain amount of time on each type of work whilst underground.

PROFICIENCY PAY

Proficiency pay is given to many classes of Natives who may not rank as boss boys but who have attained a certain skill in their particular class of work, and these payments are laid down in the scale of standard pay rates.

Whilst this was under the control of the officials concerned, there were always complaints regarding non-payment of what was due. Since it has been placed under the Native controller, and countersigned by the officials concerned, there have been no complaints, and all those due for the increase intended have obtained it without delay.

All Native labour, if returning within six months of discharge, is put back on the rate of pay received immediately prior to discharge and placed on his old job on the first vacancy.

ACKNOWLEDGEMENT

Permission to publish these notes has been given by the Consulting Engineer, Diamond Section, Anglo American Corporation of S.A. Ltd.

MINING A VERTICAL KIMBERLITE FISSURE AT STAR DIAMONDS (PROPRIETARY) LIMITED

*By **H. F. Allan, B.Sc.** (Member)

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INTRODUCTION

The Star Diamond Mine has for several years been producing relatively small gem stones of very high quality and good colour. A high proportion of the diamonds recovered consists of chips or blocks, with boart constituting less than five per cent by weight of the total caratage. The undamaged stone is characterized by its pronounced crystalline structure, the sides of the crystal being highly polished and the angles and corners sharply defined.

The mine is situated on the farm Wynandsfontein 653 in the Winburg district of the Orange Free State, directly east of Theron station on the Kroonstad-Bloemfontein railway, and about seven miles north of the village of Theunissen.

Prospecting for diamonds in the Wynandsfontein area began as far back as 1911 and the first diamond mining company to work there was formed in 1926. Operations were confined to pigrooting the outcrop. In 1930 the market slumped and operations ceased, to be restarted in 1947 under more favourable market conditions.

GEOLOGY

The Star Mine is one of the line of old mines, including Sweet Home, Thor, Driehoek, Phoenix, Theron, Monteleo, Lion Hill and others, all of which have been worked on a small scale by syndicates at various times in the past. These mines lie in a straight line running due east and west on either side of Theron railway station for a total distance of some ten miles. Apparently they all exploited a single dyke, or "fissure", or a set of closely spaced parallel dykes, and "pipes" which represent local enlargements, or "blows", of the main dyke.

At Star, work is at present confined to "fissure" mining. The fissure consists of typical kimberlite "blue ground", a serpentinized olivine-pyroxene-phlogopite peridotite, often with a marked brecciated appearance and with both occidental and cognate inclusions. In places, however, the rock is highly micaceous, this variety evidently

*Assistant Consulting Engineer, Gold Fields of South Africa Limited.

having been produced either by a process of local segregation or by contamination with country rock as it occurs characteristically both near the contacts of the dyke and around the larger impounded fragments.

The fissure cuts through shales belonging to the lower portion of the Beaufort Series of the Karroo System, and through a thick Karroo dolerite sheet which overlies the shales. These latter lie more or less horizontally, and although the dolerite is a sill in the regional sense, it locally cuts across the bedding of the shales and dips east at varying angles as high as 45 degrees.

The dip of the fissure is essentially vertical and is controlled by a set of pronounced east-west joints in the country rocks. There are, in fact, a few fissures oriented roughly parallel and occurring in what appears to be an *en echelon* pattern, so that where one fissure pinches out, another starts developing in a position offset some distance to the side.

The maximum thickness of the main fissure is about 40 in. There is no immediate change in fissure thicknesses when passing from shale into dolerite. The contacts in both types of country rock are sharp and smooth and if anything the fissure is even more regular in the dolerite than in the shale.

To date the mine has exploited some 6,000 ft. of fissure strike length down to a maximum depth of 850 ft. About three-quarters of the mining has taken place with dolerite as country rock and the balance with shale as country rock.

MINING

Shaft system

When mining operations were restarted in 1947, two vertical shafts were sunk, No. 1 to the 3rd Level (400 ft. below collar) and No. 2 to the 2nd Level (300 ft. below collar). These two shafts did not provide adequate capacity and were replaced by the Main Shaft and No. 3 Shaft, each of three compartments, which were designed to form the main arteries of the mine for the handling of all men and the bulk of material and ore and to serve as downcast airways for ventilation. These two shafts have now reached the 6th Level at a vertical depth of 850 ft. below collar. No. 1 Shaft has been retained in commission to assist in the handling of material and ore only and to facilitate exploration of the eastern section of the mine below No. 2 Level.

It is not the policy to sink the two main shafts to any final depth since it is not known to what extent grade will persist as the fissure penetrates deeper into the shale country rock. Instead, these shafts are deepened by four levels at a time. During sinking, stations are cut at 120 ft. intervals and when the lowest required level has been reached the complete ore pass system is cut and the four new levels developed.

At present the ore, which becomes extremely sticky when wet, is hoisted in 16 cu.ft. cars in cages. It is expected that there will be less water in stopes at depth and that hoisting can then be done in bottom discharge skips.

Development layout

In the early stages of the underground life of the mine tramming drives were developed on the fissure itself. This necessitated leaving extensive pillars above and below the drives when the ground was stoped out. The nature of the kimberlite filling the fissure is such that it weathers very rapidly and tends to become self mining after a short period of time and consequently conditions in the drives quickly became hazardous and eventually the drives themselves were lost. In order to overcome this difficulty a development technique involving the advance of twin ends was introduced, in which the main 8 ft. by 8 ft. tramming drive is located in country rock 15 to 20 ft. from the fissure, the small fissure drive being developed as far as possible at fissure width only. Crosscuts between the two drives are put through at intervals of 40 ft. (Plate I).

Plate I—Fissure drive

Owing to local changes in the orientation of the fissure it is the practice to keep the fissure drive in advance of the tramming drive so that the direction of the latter may be made to conform with that of the former thereby enabling the pillar between the two to be maintained at a reasonably constant width.

On completion of a shaft sinking cycle, development immediately begins on each of the four levels thereby made available. Raise and ore pass connections are then put through as rapidly as possible to enable stoping operations to begin and ore to be passed to the lowest or haulage level. (See Fig. 1).

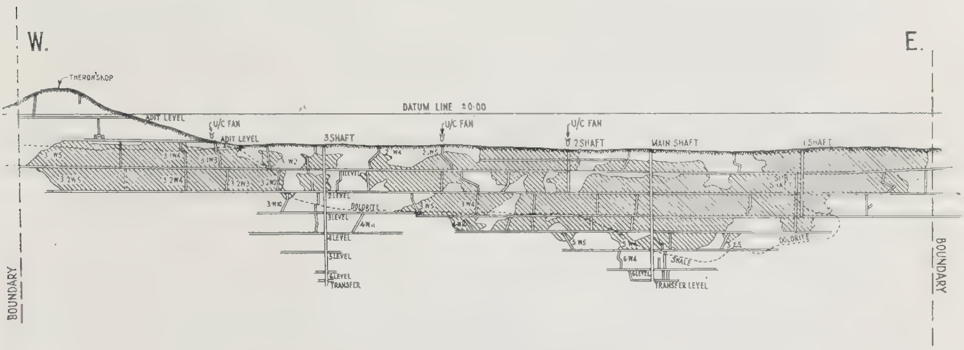

Fig. 1—Vertical projection through mine

It is policy to extend raises between levels only as starting points for stoping operations. Additional raises are developed only when breaks or pinches in fissure continuity make them necessary or when faults cannot be negotiated by stoping.

As stoping proceeds in the newly developed section so storage space is provided for waste from tramming level development. Thus the support required in the stopes, in addition to the initial packing is readily to hand. Development and stoping then continue normally and operations are synchronized so as to have sufficient development waste storage space available at all times in the stoping areas. It follows therefore that in the initial stages of opening up a new four-level section the aim is to develop and stope the two lower levels as rapidly as possible so as to provide early tipping space. It will be appreciated that once tipping space has been made available by stoping it is necessary to hoist waste development rock from the lowest level only.

The 6th Level which is now being driven along the strike of the mine will be equipped as a main haulage and collecting level. Future planning of the mine layout allows for the 10th Level as the next main haulage horizon.

Stoping methods

(i) *Original shrinkage—overhand*

In the early underground history of the mine the overhand shrinkage method of mining was employed. The average fissure width at that time was approximately 30 in. and stoping widths were in the order of 36 in. It appears that great difficulty was experienced in withdrawing the stope shrinkage owing to the effect of surface waters on the kimberlite which has the property of swelling and compacting when wet. Also it led to hazardous conditions in the stopes where the ore was liable to hang up.

(ii) *Underhand method*

In an attempt to overcome these difficulties the underhand method of stoping was resorted to, with tramming drives on fissure. This necessitated leaving safety pillars above and below these drives. Support in the stopes themselves consisted of props and matpacks and no waste filling was attempted. Although this stoping method met with some success and was an improvement on that previously practised, it was still bedevilled by the unfortunate property of the kimberlite, already referred to, of weathering rapidly and thereby tending to become self mining. As a result, underfoot conditions quickly became dangerous and difficulty was experienced in keeping the fissure drives open. To counteract these disabilities, temporarily at least, it was necessary to increase considerably the size of the safety pillars, particularly those covering stopes under previously mined areas. This called for excessive stope development; also, in spite of the increased size of the pillars they eventually disintegrated, closed the tramming drives, and rendered the ground irrecoverable. It was at this stage that modifications to the mining method were introduced which resulted in the system now in use.

(iii) *Modified underhand method*

The present stoping system has been employed for approximately four years and is a considerable improvement on its predecessors. The general layout as envisaged for the future is indicated in Fig. 2 which shows a longwall

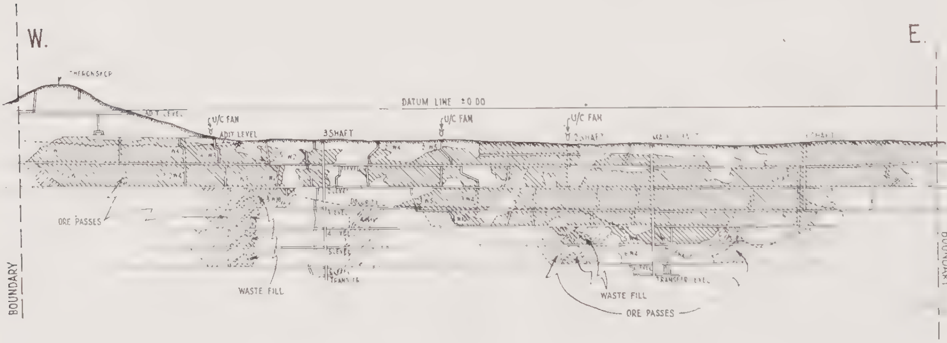

Fig. 2—Vertical projection through mine showing future layout

system of stope faces carried in V formation. Details of the method are shown in Fig. 3. The stope faces are carried underhand as before at an angle of approximately 25 degrees to the horizontal which permits easy lashing and safe travelling. The tramming drives are placed in country rock 15 to 20 ft. from the fissure and no stope safety pillars are left. Instead, double rows of 2 ft. by 2 ft. matpacks, at 8 ft. centres on strike and 6 ft. centres on dip, are installed above and below the fissure drive. These are stulled over with 8 ft. by 4 in. laggings to hold the waste packing and to serve as protection for persons working below; they are permanent installations. The double row above the drive is installed ahead of the advancing stope face, the fissure drive having been sliped out to a height of 20 ft. to accommodate it and to provide sufficient capacity for the stope boxes which deliver at points opposite to the crosscuts connecting the tramming and fissure drives.

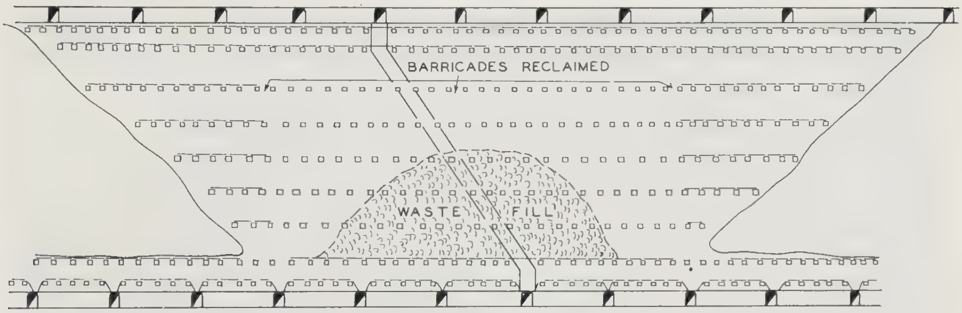

Fig. 3—Details of underhand stoping method

Support in the stope itself consists initially of 2 ft. by 2 ft. matpacks at 8 ft. centres on strike and 15 ft. centres on dip supplemented by props. These matpacks are stulled over with 8 ft. by 4 in. laggings and loaded to provide overhead protection. As the face advances the stulls are reclaimed and waste fill is run into the back area of the stope, completely filling it. This waste fill is supported by the double row of stulled-over matpacks above the fissure drive already described. All timber is treated in a surface pickling plant as a precaution against rot.

The cycle of opening up a stoping area is normal in every way except that stope raises are usually driven at an angle of 55 degrees to the horizontal to facilitate travelling. Underhand stoping is then begun from the top of the raise under the protection of the overhead double barricade. The broken ore is lashed down the face (and in the early stages into the raise) to a normal type stope box erected adjacent to the crosscut from the tramming level.

Owing to the difficulties experienced when the kimberlite becomes wet, and because the ore contains a very low percentage of silica, all drilling in the stopes and fissure drives is done dry, and as far as possible general drainage water is diverted away from the stoping area. Holes are drilled to an approximate length of 30 in. with an average burden of 18 in. The relatively short length of hole is aimed at preventing damage to the side walls, particularly on the contact of the dolerite—shale horizon, and to avoid too great a concentration of explosive (40 per cent dynamite) which might damage diamonds in the vicinity of the blast. Also as protection for the contained stones it is normal practice to attempt to break the ore as coarsely as can be conveniently handled.

The average stoping width attained with the mining method described is 32 in. from an average fissure width of 22 in.

Experience of fissure mining has shown that the grade of ore is subject to appreciable and unpredictable fluctuations, the daily recovery of diamonds varying between wide limits. Estimates of payability and decisions to stope any area are therefore based upon the apparent width of the fissure. (Plate II).

(iv) *Experimental shrinkage method—overhand*

The vertical depth below surface at which stoping is now taking place and the fact that it is now possible to conduct dry mining operations have led to the recommencement of shrinkage stoping in certain sections of the mine

Plate 2—An open stope

and experimental stopes in dry areas have been mined by this method. In addition to general safety, security from loss of large stones, etc., the practice of shrinkage mining should result in a more even flow of ore to the treatment plant and greater flexibility because of the greater number of points from which ore could be selected, thus providing a certain measure of grade control.

The central section of the mine, which is the deepest and where the presence of water has diminished somewhat, has entered the shale horizon lying below the dolerite sill and it is proposed again to experiment with full shrinkage stoping there.

The present system of development will be retained as will the inclined raise connections at 55 degrees to the horizontal. The fissure drive will be sliped to provide sufficient height for the erection of a double barricade and normal overhand shrinkage mining will take place (see Fig. 4). All water from the upper levels will be channelled away from the shrinkage area and drilling will be dry. It is proposed not to leave pillars and shrinkage will be drawn off as rapidly as possible once the opposing stope faces are sufficiently advanced. Instead of matpacks for internal support, 4 in. to 6 in. props are proposed to prevent the slabbing of large rocks prior to removal of the shrinkage and to avoid hanging up of the shrinkage caused by too rapid pinching of the sidewalls.

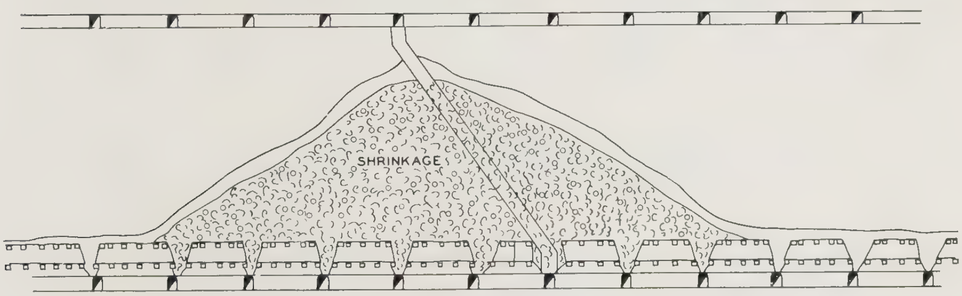

Fig. 4—Overhand shrinkage stope

VENTILATION ARRANGEMENTS

The mine is classed as a fiery mine owing to the presence of methane and the appropriate precautions are taken. Three upcast fans are situated on surface with the main hoisting and travelling shafts downcasting to the lowest levels and the working faces forming the return airways.

TREATMENT

The treatment of the diamondiferous ore follows conventional methods which have been dealt with in other publications. The ore is crushed in two stages, trommelled and concentrated in two stages of diamond pans, the latter stage in closed circuit with a third stage of crushing. The concentrates are poured over grease tables for the final recovery of the diamonds.

SECURITY

The question of security which is always an important one in diamond mining militates against the provision of stock piles on surface. Apart from ore passes underground there are no facilities for ore storage.

The mine is policed by a staff of one European detective and 26 Bantu. There is a security fence around the property and only authorized persons are permitted to enter. All persons employed on the mine are allowed to enter or leave at will but are liable to be searched if considered necessary. A very favourable reward system for persons finding and handing in diamonds is well publicized and is believed to be an effective deterrent against theft. Rewards of up to £200 have been paid.

STATISTICS

The following are salient features of operations for the year ended 30th June, 1960:—

Loads (16 cu.ft.) treated —Mine ore	91,630
—Old tailings	4,441
Total loads treated	96,071
Cost per load treated	40s.
Total carats recovered	36,820
Carats per 100 loads of mine ore	38·326

The writer acknowledges with thanks the permission of the directors of Star Diamonds (Proprietary) Limited, and of Selected Mining Holdings Limited (of which Star Diamonds (Pty.) Limited is a wholly-owned subsidiary) to publish this paper.

MODIFICATION OF A DIAPHRAGM JIG TO TREAT LARGE TONNAGES OF DIAMOND BEARING KIMBERLITE

By **R. G. WEAVIND** (Member) and **D. F. C. McLACHLAN** (Visitor)

SYNOPSIS

Because the jigs at Premier treat —2mm. kimberlite at a very high feed rate it was found necessary to modify them to maintain an efficient minerals separation. The accuracy of the separation was deduced from the behaviour of radio-active diamond tracers in the jig.

INTRODUCTION

The Premier Mine jig plant treats some 160 t.p.h. of classified —2mm. feed. The plant comprises eight primary and two secondary diaphragm type jigs.

This high feed rate of 20 t.p.h. to each primary jig is considerably more than the jig was originally designed to handle and accordingly, the jig characteristics were studied in order to modify the operating variables to treat the large tonnage most effectively.

This was done by studying the primary jigs treating the normal production feed. A feature of the method of study was the use of radio-active diamond tracers to determine the behaviour of diamonds in the jigging process.

ESTIMATION OF JIG PERFORMANCE

Two methods were used to assess the efficiency of minerals separation in the jig viz. radio-active diamond tracers and the Tromp curve.

(i) *Radio-active diamond tracers*

Use of the radio-active tracer to test the efficiency of diamond concentrators was first reported by Adamson*. The behaviour of diamonds in the jig was studied by introducing a tracer diamond into the feed and following its path through the jig. The time of residence was also noted.

Each tracer was prepared by cementing a short length of radio-active cobalt wire into a 0.5 mm. hole drilled into a diamond of selected size. The specific gravity of the tracer stone was kept at 3.5 by counterbalancing the effects of the cobalt and the cement. The sizes of the two diamonds used in the tests were —9+10 and —10+14 mesh Tyler. The shape of the larger stone was octahedral but the smaller stone was relatively flat.

(ii) *Tromp curve*

The tromp curve needs no introduction to the coal dresser but its use in mineral dressing has been somewhat limited up to now. It is simply a graph of the percentage recovery of particles of one product e.g. the hutch product of the jig, plotted against

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*Adamson, R. J. "Some Account of Diamond Winning Practices in Southern Africa", presidential address to the *S.A. Inst. of Min. & Met.*, August 1959.

the particle specific gravity; a specimen Tromp curve is given later in the paper. The curve therefore shows how efficiently the lighter mineral fractions were rejected from the jig and how efficiently the heavier minerals including the diamond were recovered in the hutch product.

The shape of the Tromp curve provides an estimate of the efficiency of minerals separation and is defined by a single parameter, the Belugou imperfection coefficient, B,

$$\text{where } B = \frac{\rho_{75} - \rho_{25}}{2\,(\rho_{50} - 1)}$$

and where ρ_x is the s.g. of a particle of which the recovery to the hutch product was x per cent.

It will therefore be deduced from the above that $(\rho_{75} - \rho_{25})$ is the statisticians' interquartile range and that ρ_{50} is the effective density of separation in the jigging process. Tromp curve results for the beneficiation of coal and iron in the size range —9+10 mesh Tyler i.e. —2+1 mm. show how the imperfection coefficient reflects the accuracy of separation:

0.20 poor; 0.10 fair; 0.06 good; 0.02 excellent.

The advantages of the Tromp curve lie in the fact that a single parameter is sufficient to define the accuracy of separation, and that its value is broadly independent of the density composition of the feed and the concentration ratio in the jigging process. Perhaps the main objection to the Tromp curve is that it is influenced by the shape of the particles i.e. a flat particle may behave like a particle of lower s.g. This point has not been thoroughly investigated but it may be noted that the results showed that when the Tromp curve indicated a recovery of 87 per cent for —10+14 mesh kimberlite of density 3.5, the recovery of the flattish tracer diamond was 89 per cent.

Assessment of jig performance was, therefore, made by both the Tromp curve and the radio-active tracers. However, the tracers were used for most of the testing because they provided a more rapid estimate of the jig efficiency.

THE JIG CIRCUIT

Feed to the jig circuit at Premier comprises about 160 t.p.h. (dry weight) of a —9 mesh Tyler product from rake classifiers that nominally reject all —28 mesh material.

All jigs at Premier are the 24 by 36 in. "Denver duplex" mineral jigs. Eight primary jigs each receive about 20 t.p.h. new feed and the combined hutch products are distributed to two secondary jigs. Tailings from the primary jigs are rejected but all the secondary jig tailings are recycled to the feed of four of the primary jigs.

Concentrates from the secondary jigs are further concentrated over vibrating grease tables on which the diamonds are recovered.

DESCRIPTION OF THE STANDARD JIG

During the course of the investigation it was found necessary to modify the jig but the following is a description of the standard "Denver duplex" before being modified.

The duplex jig comprises two compartments, in series. The screen over each compartment is 24in. wide and 36in. long and is made of 3 mm. aperture wedge wire. Water is pulsed through the screens by two top driven diaphragms connected to a pivoted walking beam that is in turn driven by an eccentric. Adjustment of the eccentric allows stroke variation of 0 to $\frac{3}{4}$ in., and the speed is 290—300 strokes per minute. The eccentric shaft also drives a rotating water valve that limits water addition so as to reduce the intensity of the suction stroke. The bedding material is steel shot.

TEST PROCEDURE

The test procedure was generally to change each of the variables, one at a time, and to note the effect on the diamond recovery as determined by the radio-active tracer method. Of course, the diamond recovery was very dependent on the concentration ratio and whenever this ratio did show much change as a result of altering a variable, this was taken into account—if only empirically. Wider use of the Tromp curve would therefore have allowed a more elegant assessment of the results and a more rapid method of Tromp curve analysis is now being devised for use in any future test work.

Each of the jig variables was controlled as follows:

(i) The feed rate was normally held at about 20 t.p.h. through the *r.o.m* control and most testing was carried out under these conditions.

(ii) No water was added to the feed—the moisture content of the feed was 25 per cent. Water addition to each hutch was controlled by valves and the combined water consumption to the two hutches was measured.

(iii) The bedding originally comprised steel shot but this was later replaced by alluvial gravels.

(iv) Stroke frequency was controlled by a variable speed drive.

(v) Stroke amplitude was varied up to a maximum of $\frac{3}{4}$ in.

The effects of the variables were mainly determined by using the smallest radio-active tracer viz. the —10+14 mesh flat shaped diamond. The period of residence of this diamond in the jig bed and its recovery from either the first or the second compartment were noted.

EFFECT OF THE VARIABLES ON THE JIG PERFORMANCE

Detailed experimental test results of the effect of the variables are not given but Table I does illustrate how the radio-active tracer diamond was used to assess the effect of some of the variables. It may be noted that the accuracy of the percentages given in the table is limited because only about thirty tracer additions were used in each test e.g. the table shows that under one particular set of conditions, the apparent recovery of the —10+14 mesh tracer was 100 per cent, yet it was later shown that under the same conditions, the tracer was recovered only 89 times after 100 additions. This therefore showed that although thirty tracer additions did give an indication of the recovery efficiency, a larger number were needed to give an accurate estimate. The question is clearly one of statistical probability.

Examination of the effect of the variables was further complicated by the fact that it was not always valid to assume the condition *ceteris paribus*. For instance, jig operators will readily appreciate that although a particular stroke frequency may give optimum results at one stroke length, a different stroke length may be required at some other frequency.

(i) *The feed*

An irregular feed to the jig was caused by the necessarily irregular discharge from the rake classifier and in fact, these surges tended to scour the jig bed away

TABLE I

SOME TEST RESULTS WITH THE RADIO-ACTIVE TRACER DIAMONDS

Feed rate (t.p.h.)	Rotary valve	Water flow (g.p.h.)	Bedding		Stroke		Concentrate (per cent)	Diamonds reported in first hutch (per cent)	Per cent diamond recovery from both hutches	
			Type	Weight per jig (lb)	Amplitude (in.)	Frequency (c.p.m.)			—9+10 mesh Tyler	—10+14 mesh Tyler
19·9	Yes	7,600	Steel shot	620	$\frac{5}{16}$	290—300	12·8	32	78	—
								30	—	72
20·0	Yes	7,600	Alluvial gravels	300	$\frac{5}{16}$	290—300	10·5	50	100	—
								45	—	92
20·0	No	11,500	Steel shot	460	$\frac{3}{8}$	290—300	12·9	72	92	—
								38	—	84
20·1	No	9,500	Alluvial gravels	260	$\frac{1}{2}$	270	10·3	85	100	—
								80	—	100

from the feed end. The difficulty was overcome simply by mounting a plate about 3 in. above the feed weir so that the feed piled up slightly behind this barrier and flowed smoothly through the gap. Fig. 1 shows how the barrier plate was located.

It was assumed that the optimum feed rate was 20 tons per hour and the jig variables were adjusted to handle this tonnage.

(ii) *Water flow*

The high feed rate necessitated a comparatively large amount of water in order to produce compaction and dilation of material in the jig bed. However, too much water caused turbulence and tended to sweep away the heavy minerals indiscriminately with the light. Valves were fitted to regulate the water to each hutch.

The optimum flow of water to the two hutches was determined to be 11,500 g.p.h. when a bed of steel shot was used and 9,500 g.p.h. with the bed of alluvial gravel.

The rotating water valve restricted the water flow to 7,600 g.p.h. and it was therefore found that the accuracy of separation in the jig was much better without the rotating water valve than with it. The fact that the rotating water valve reduced the intensity of the suction stroke was perhaps a disadvantage in itself. Williams* reports that the Denver jig operated better without the rotating water valve and Charbonnier†, in a theoretical analysis of jig action, draws the conclusion that it is mainly during the suction cycle that true gravity concentration takes place.

(iii) *Bedding*

Because the specific gravity of the diamond (3.5) and that of most of the associated kimberlite minerals (2.7) are much less than steel (7.2), the bed of steel shot was replaced with a bed of heavy alluvial gravels (3.2) from the Consolidated Diamond Mines of S.W.A. Limited. The gravel was sized in the range —10+6 mm. and 130 lb was used in each compartment to produce a depth of 2 to 3 in. Fig. 2 shows both the steel shot and the gravel bedding.

*Williams F. A., Author's reply to discussion of "Recovery of semi-heavy minerals in jigs", *Inst* of *Min. and Met.*, Vol. 68, part 9, pp. 423-456.

†Charbonnier R. P., "A study of a simplified theory of gravity concentration in coal jigs", *Canadian Min. and Met. Bull.*, June 1959, pp. 385-388.

Fig. 1 Photograph showing divisions in jig screen compartments and feed distribution plate, as viewed from the tailings end

However, owing to the low s.g. of this bed, it tended to displace to the discharge end of the jig screen and to the side of the screen furthest from the diaphragm. This bed movement was successfully countered by dividing the screen compartment into 12 sections of 9 by 8 in., each section 4 to 5 in. deep. Fig. 1 shows these divisions, which were effected with strips of screen cloth having apertures $\frac{3}{4}$ in. by $\frac{3}{16}$ in.

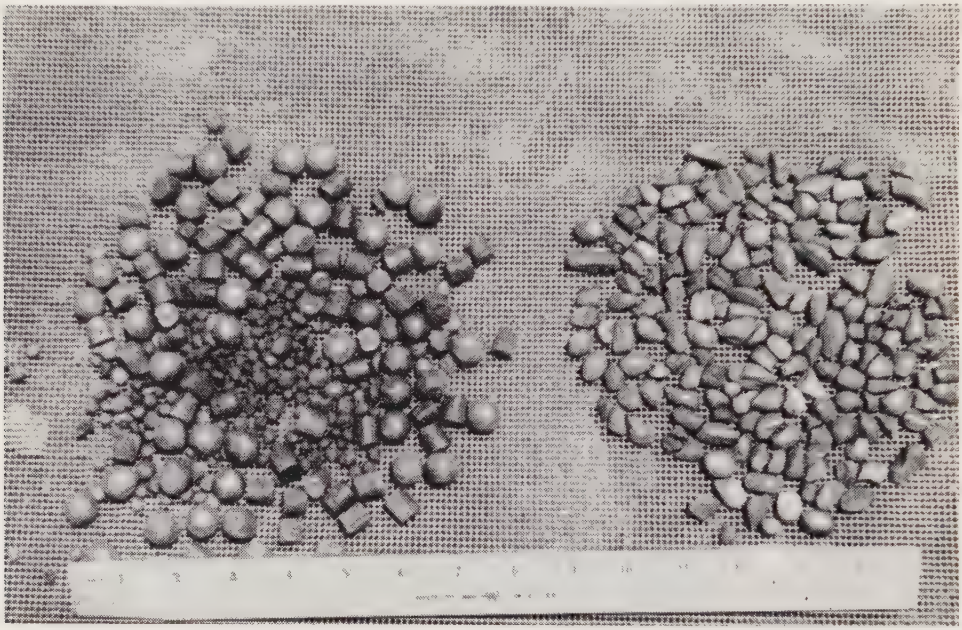

Fig. 2—Steel shot (left), and gravel bedding

This bed of alluvial gravels resulted in a marked improvement in the accuracy of separation.

(iv) *Stroke frequency*

With the bed of alluvial gravels, a stroke frequency greater than 300 cycles per minute produced a permanent state of dilation of the bed, whilst a frequency less than 250 cycles per minute was not sufficient to dilate the bed evenly. The speed finally selected was 270 cycles per minute.

(v) *Stroke amplitude*

At 270 cycles per minute the optimum amplitude was found to be $\frac{1}{2}$ in. When much less than this e.g. $\frac{1}{4}$ in., the bed tended to compact permanently whilst an amplitude up to $\frac{3}{4}$ in. caused excessive turbulence in the bed.

RESULT OF JIGGING AT 20 T.P.H.

The efficiency of the jig was determined under the following conditions:

(i) Feed rate, 20 t.p.h.
(ii) Water flow, 9,500 g.p.h. per compartment.
(iii) Bedding, 130 lb of —10+6 mm. alluvial gravels per compartment.
(iv) Stroke frequency, 270 cycles per minute.
(v) Stroke length, $\frac{1}{2}$ in.

Before these modifications were made, the Tromp curve imperfection value for the concentration was 0.11 for —10+14 mesh particles but after the above conditions had been established, the imperfection value in the same size range was improved to 0.07.

The Tromp curve given in Fig. 3 for the separation under these modified operating conditions also showed that the recovery of —10+14 mesh particles of s.g. 3.5 was 87 per cent. This value was confirmed by adding the —10+14 mesh radio-active tracer diamond to the feed 100 times, when on 89 occasions it reported to the concentrates. It may be added that the octahedral shaped —9+10 mesh tracer reported to the concentrates 90 times out of 90.

CONCLUSIONS

Estimation of the result of a diamond concentration process is seldom conclusive simply because the diamond is very rare and because it is not amenable to chemical assay. Nevertheless, estimation of diamond recovery by the radio-active tracer method appears to be reliable. The Tromp curve procedure provides a complete picture of the behaviour of all the minerals present but it sufferes the disadvantages of being laborious to produce and of assuming that the diamond porosity and shape are the same as those of the average kimberlite mineral of density 3.5. Just how

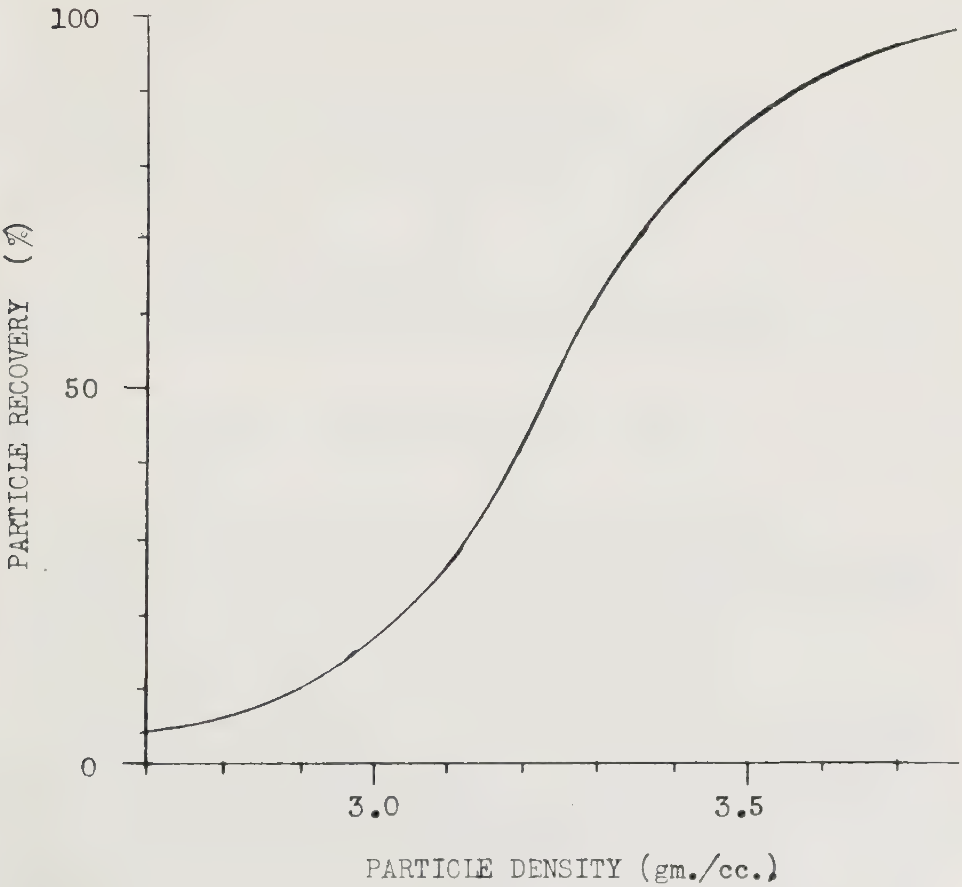

Fig. 3—Tromp curve showing the accuracy of separation in the modified jig

valid these assumptions are, is a question that has not yet been studied in full, but the available results definitely indicate that the average kimberlite mineral of density 3.5 behaves similarly to a flattish diamond having the same mesh size.

The feed rate was a variable that could not be changed much without changing the number of operating jigs, and in modifying the jig to handle the large tonnage, the two most important alterations were to the jig bed and to the water flow. The quantity of water flow to each compartment was found to be critical and the test results pointed to the value of maintaining a strong suction stroke; it was for these reasons that the rotating water valve was discarded. The substitution of an alluvial gravel bed for the steel shot resulted perhaps, in the most pronounced improvement of the jig efficiency.

At 20 t.p.h. the improved jig efficiency resulted in a marked increase of diamond recovery with a diminution of the quantity of final concentrates, but it may be added that even under the improved operating conditions, this efficiency showed a pronounced decline at a feed rate much in excess of 20 t.p.h.

Contribution to discussion

R. C. M. van der Spuy (Associate Member): It was with great interest that I read this paper by Messrs Weavind and McLachlan. In South West Africa and for the past six years on Rooiberg I have been operating jigs to recover cassiterite. At Rooiberg it has been possible to increase the efficiency of a similar mineral jig considerably by making certain alterations.

1. REMOVAL OF ROTARY VALVE

The removal of the rotary valve resulted in far more efficient operation, and improved recovery. A point of importance is that the removal of this valve reduced the number of wearing parts from 27 to 18.

2. DEPTH OF BED
(Height of tailing board above the screen)

It is felt that this is a very important operating variable which the authors have apparently overlooked.

The mineral jig under consideration was primarily designed for closed grinding circuits where a deep bed is necessary because of the fineness of material treated. When the jig is operating on much coarser feed, however, it has been found that a shallower bed results in an increase in recovery.

In the paper "A study of the motion of particles in a jig bed" which was presented by G. O. Lill and H. G. Smith at the International Mineral Processing Congress in London last year[1] the authors determined an optimum bed thickness for a given stroke and speed by observing the rate at which the heavy particles penetrated beds of different thicknesses.

Tests were carried out in 4 in. by 4 in. transparent compartments and the authors state that: "With a bed thickness of 4·5 cm. . . . the particles moved very slowly through the top layers of the bed, but on reaching a point approximately 3·2 cm. above the jigging screen a sharp increase in downward velocity occurred. In tests

1. G. O. Lill and H. G. Smith. A study of the motion of particles in a jig bed. International Mineral Processing Congress, London, April, 1960.

with a bed 3·1 cm. deep no tendency for particles to stick in the top layers was observed, but with a bed 3·8 cm. deep both types of behaviour were noted. It was concluded that the loosening wave (during pulsion stroke) only reached a point between 3·1 cm. and 3·8 cm. above the screen."

On this subject Taggart[2] makes the following statements: "Practice as to depth of bed varies. A deep bed requires much water which sends the fine free mineral to the latter cells."

"When a clean finished product is desired a deep bed is necessary."

"A large difference between specific gravities of the mineral species to be separated makes a relatively shallow bed permissible."

"A large proportion of heavy mineral permits the use of a shallower bed."

"With fine feeds (2 mm.) the minimum depth is about 20 times the grain diameter."

The following list shows how practice varies as to the depth of bed in different jigs.

Cooley jig (Taggart) 9—5 in. Collem jig (Taggart) 2·75—3 in. Woodbury jig (Taggart) $2\frac{1}{2}$ in. deep. Rouss jig (Taggart) 4 in. Standard 26 in. Pan American jig, $2\frac{1}{4}$ in. and 42 in. model $5\frac{1}{2}$ in. Denver mineral jig 4—$7\frac{1}{4}$ in.

The procedure followed by the writer when installing a jig is that the amplitude of the pulse is set equal to three times the maximum particle size and the plunger velocity is then calculated from Monroe's formula.[3]

$V = 26{\cdot}32\ D(S-1)$ where V = plunger velocity in mms./sec. D = diameter of largest particle in mms. S = S.G. of gangue mineral. The depth of bed is then altered by raising or lowering the screen until maximum efficiency is obtained.

On this basis, therefore, the Premier Mine jigs should operate at 246 r.p.m. with a $\frac{1}{4}$ in. stroke. Because of the lower specific gravity of the diamonds and the slightly finer feed, a bed depth of $4\frac{1}{2}$—5 in. is suggested. A shallower bed also reduces the water consumption.

At Rooiberg, on a minus $\frac{1}{8}$ in. feed optimum results have been obtained when the jig is operated at 238 r.p.m. with a 0·32 in. stroke, bed depth $3\frac{3}{4}$ to $4\frac{1}{4}$ in. This bed depth includes ragging, the depth of which varies from $\frac{1}{2}$ to $\frac{3}{4}$ in.

3. ELIMINATION OF EDDIES IN THE BED

The four corners of the screen baskets of these jigs protruded approximately one inch into the bed. This caused eddies to be formed resulting in holes in the four corners. On Rooiberg the jigs have been altered so that there is a straight flow through the compartments.

4. FEED

Another point is the angle at which the feed falls into the bed. If the feed to the first bed does not contain too much water, it falls naturally very nearly perpendicularly into the bed. Due to additional dilution the pulp entering the second bed does so at an acute angle and the surface of the first few inches of bed is subjected to a scouring action which reduces the effective jigging area. A baffle plate can be installed which obviates this scouring action and diverts the material vertically into the bed.

2. A. F. Taggart. *Handbook of Mineral Dressing.*
3. Richards and Locke. *Textbook of Ore Dressing.*

5. SCREEN

It has also been found that there is much less tendency for the wedge wires to blind when they are at right angles to the direction of flow. When the wedge wires are parallel to the direction of flow, blinding starts from the sides and works inwards towards the centre of the bed. The outside edges of the bed become dead and the material flowing along the outer edges of the bed is not subjected to any jigging action. With the wedge wires at right angles to the flow, the blinding is not pronounced and when it does occur it starts from the front and back of the bed (i.e. the sides parallel to the wedge wire).

Blinding occurring here results in a reduction of the effective area of the bed but the material is still subjected to a jigging action over the complete width of the bed.

6. METHOD OF DRIVING THE PLUNGERS

The transmission of the stroke through a walking beam has one great disadvantage in that the spring-bolts do not rotate in their bushes but rock a few degrees in either direction. This makes lubrication of the three spring-bolt bushes and the walking beam bush very ineffective and these parts are inclined to wear fairly rapidly. Wear soon leads to the stroke diminishing and loss of efficiency. By placing a shaft across the centre of the plungers parallel to the direction of flow and connecting each of the plunger plates to an eccentric on this shaft with a longer plunger rod the jig works just as effectively. The wearing parts are then virtually all roller and ball bearings and the number of moving parts is further reduced from 18 to 9.

7. CONCLUSION

In the Leeuwpoort concentrator of the Rooiberg Minerals Development Company the original mineral jigs were only responsible for 25 per cent of the total cassiterite recovered by the mill prior to modification. As a result of the alterations enumerated in this discussion the jigs now recover 78 per cent of the total cassiterite.

Recent tests have shown that the modified jig is more efficient than another standard type of jig operating in this mill.

VALEDICTORY ADDRESS

By **W. S. FINDLAY** (President)

The symposium on diamond mining and recovery in Southern Africa has formed a particularly valuable contribution to our transactions. The practices disclosed in the papers presented indicate very clearly that diamond mining and recovery procedures embrace the most modern methods available. As in the past, the diamond mining industry is well in the van of progress and we are very pleased that its light is no longer hidden under a bushel as far as the Institute is concerned.

In his opening address, Mr Giles made reference to the great value of the marketing arrangements devised and implemented by the late Sir Ernest Oppenheimer. The brief account given by Mr Giles could be greatly amplified and I am sure that, if a paper were to be presented dealing with the complex ramifications of the Central Selling Organization, members would have a far greater appreciation of the debt this country owes to Sir Ernest.

Sir Ernest was the pioneer of planned production and marketing in the mining industry. The soundness of his philosophy of co-operation as opposed to unbridled competition is reflected in the continued stability and growth of the diamond industry and in the remarkable technical progress recorded in the papers presented in the course of this symposium.

The benefits of rationalized marketing accruing to producers, consumers and of course to the economy of the countries concerned have already been mentioned.

The Central Selling Organization reports that its sales of gem diamonds in 1960 created a new record at £63,450,000 while its combined sales of gem and industrials at £89,700,000 were only 1·5 per cent less than the record sales of 1959. This decrease was entirely due to supplies of industrial diamonds not being sufficient to meet the demand. In this connection it is worth recording that the weight of industrial diamonds sold was approximately five and a half times the weight of gems.

Mention of diamonds invariably brings to mind a picture of glittering gems, which are alleged to be "woman's best friend" and little thought is ever given to the enormous part being played in the world today by "man's best friend," the industrial diamond. Unsuitable for use as gem because of its colour and impurities, but of great value because of its unequalled hardness, this quality of diamond has found increasing uses in industry, particularly since the last war and the advent of the jet and space age.

Industrial diamonds can be divided roughly into two divisions: the industrial or better quality stone that is used for lathe tools, wire drawing dies, drill stones and wheel dressers and the lower quality crushing boart, as it is called, which is crushed and graded into mesh sizes for use in wheels for grinding tungsten carbide tools, non-ferrous metals, hard plastics and glass and saws for cutting marble and granite. To drilling diamonds, of course, apart from their importance in mining, we owe the ability to probe the secrets of deep seated wealth in our own country and much of the world's supply of oil.

It would take too long to enumerate the many hundreds of applications of industrial diamonds in our daily life, but I might mention a few of the more interesting ones. A major use is in the machining of motor car parts, and windows of most cars are pencil-edged with diamond wheels. It is quite startling to know that it is claimed that without the use of 25 cents worth of diamond the American car would cost twice as much! Diamond mesh is used in rotary saws for cutting expansion joints in hundreds of miles of concrete roads in America and in large diamond-impregnated rollers to smooth concrete runways for jet aircraft. Several million sapphire and ruby watch stones are sawn and polished in Switzerland annually with diamond powder. A diamond die will draw accurately well over 200 miles of copper wire. Today, in fact, diamonds are vital in all precision engineering including jet engines, electronics and even satellites. With their help, the phrase "reaching for the moon" no longer indicates impossibility.

To quote some figures, the sales of 1,500,000 carats of crushing boart in 1939 rose to 13,000,000 carats in 1960. In addition, three million carats of drilling material and half a million carats of the better quality industrial stones were sold last year.

Although known reserves of gem diamonds in South Africa are limited, it is comforting to know that they are sufficient to maintain our present rate of production for many years to come. It is also reassuring to know that De Beers, looking further ahead, has established costly but efficient organizations to prospect for new diamond fields, to extend and improve utilization of industrial diamonds and to conduct research in the synthetic sphere.

It is well known to you that synthesis of industrial diamonds on a commercial basis was successfully achieved in South Africa in 1959, another example of the foresight and confidence displayed by Sir Ernest. This development is one of the many being energetically pursued by Mr Harry Oppenheimer and, as you are probably aware, a factory for the production of synthetic industrial diamonds is soon to be commissioned at Springs.

Prospecting for diamonds is in progress on an unprecedented scale, using the most modern equipment and techniques available. This work extends beyond the Union and South West Africa into Basutoland, Bechuanaland, Rhodesia, Nyasaland and Tanganyika.

In the Diamond Research Laboratory, yet another facility provided by Sir Ernest, provision is made for fundamental research into the physical properties of diamonds, their application to new spheres in industry, their more efficient performance in drilling and all branches of engineering and their more economic methods of production at the mines.

To sum up, the diamond-industry initiated industrial development in South Africa, has been of inestimable value in providing capital for development of the country's natural resources and continues to set an example which could well be emulated in other industries.

Gentlemen, I must express, on behalf of the Institute, our appreciation to the authors of the various papers for the most interesting and valuable record they have presented of South Africa's diamond mining industry.

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