

Mining practice in the Kimberley Division of De Beers Consolidated Mines Limited

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SYNOPSIS

This paper highlights the changes that are taking place at De Beers Consolidated Mines Limited in mining methods, mechanization, and employment practices. De Beers has four underground mines in Kimberley itself (Dutoitspan, Bultfontein, De Beers, and Wesselton), and two open-cast mines outside Kimberley (Finsch and Koffiefontein). Emphasis is placed on the new method envisaged for Koffiefontein when it becomes an underground mine. The paper also includes notes on chemical drilling and on winding practices.

SAMEVATTING

Hierdie referaat laat die klem val op die veranderinge wat by De Beers Consolidated Mines Beperk aan die gang is wat betref mynboumetodes, meganisering en indiensnemingspraktyke. De Beers het vier ondergrondse myne in Kimberley self (Dutoitspan, Bultfontein, De Beers en Wesselton) en twee oopgroefmyne buite Kimberley (Finsch en Koffiefontein). Die nuwe metode wat vir Koffiefontein beoog word wanneer dit 'n ondergrondse myn word, word veral benadruk. Die referaat sluit ook aantekeninge oor chemiese boorwerk en oor wenasgebruike in.

INTRODUCTION

The Kimberley Division of De Beers Consolidated Mines Limited is composed of four underground mines in Kimberley itself (Dutoitspan, Bultfontein, De Beers, and Wesselton) and two open-cast mines outside Kimberley (Finsch and Koffiefontein). Fig. 1 is a map of the Kimberley district showing the location of these mines. Finsch is 150 km west of Kimberley and Koffiefontein is 80 km south of Kimberley. Kimberley also con-

tains the head office of the De Beers Company, and the diamond-sorting and trading companies for goods produced in Southern Africa, which are newly housed in a building specially designed for the purpose.

Diamonds were first discovered in the Kimberley area at Hopetown in 1866 on the banks of the Orange River. This and later finds triggered the start of the great 'diamond rush'. In 1870 the Jagersfontein (now closed) and Dutoitspan pipes were discovered, followed by those at Bultfontein, De Beers, and Kimberley (now closed) in 1871, Koffiefontein in 1880, and Wesselton in

1891. Finsch Mine was a more recent discovery by A. T. Fincham in 1961.

The 'Big Hole' remains as a monument to the Kimberley Mine, which was closed in 1913, and today is a great South African tourist attraction, second only to the Kruger National Park. De Beers have built a museum at the edge of the 'Big Hole', which preserves buildings, mining equipment, shops, and goods from the early Kimberley days.

Finsch and Koffiefontein Mines are both open-pit bench mines and are fully self-contained, with their own treatment plants, housing, ser-

*De Beers Consolidated Mines Limited.

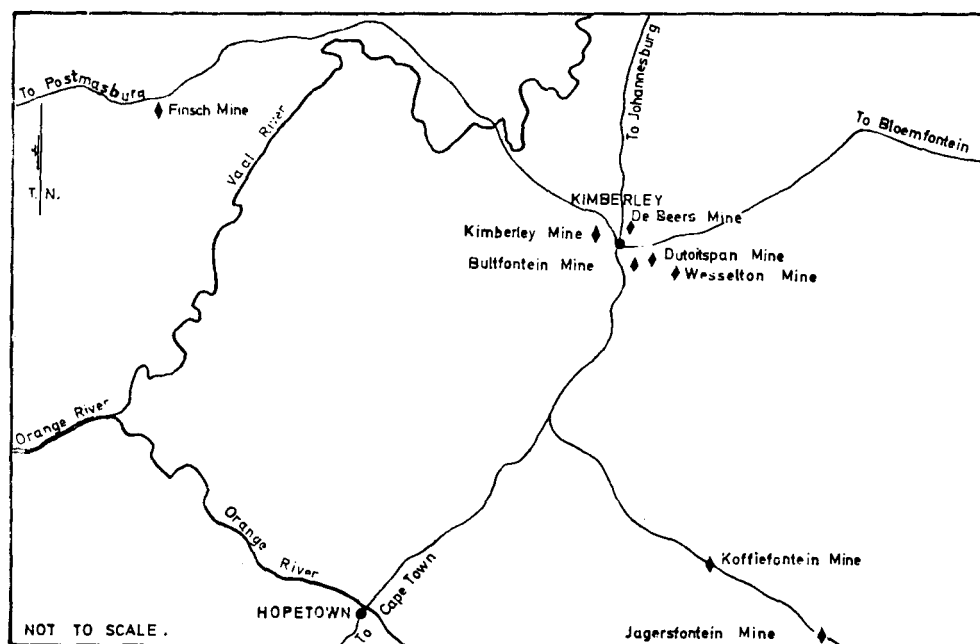


Fig. 1—Map showing the location of the mines of De Beers Consolidated Mines Limited in the Kimberley area

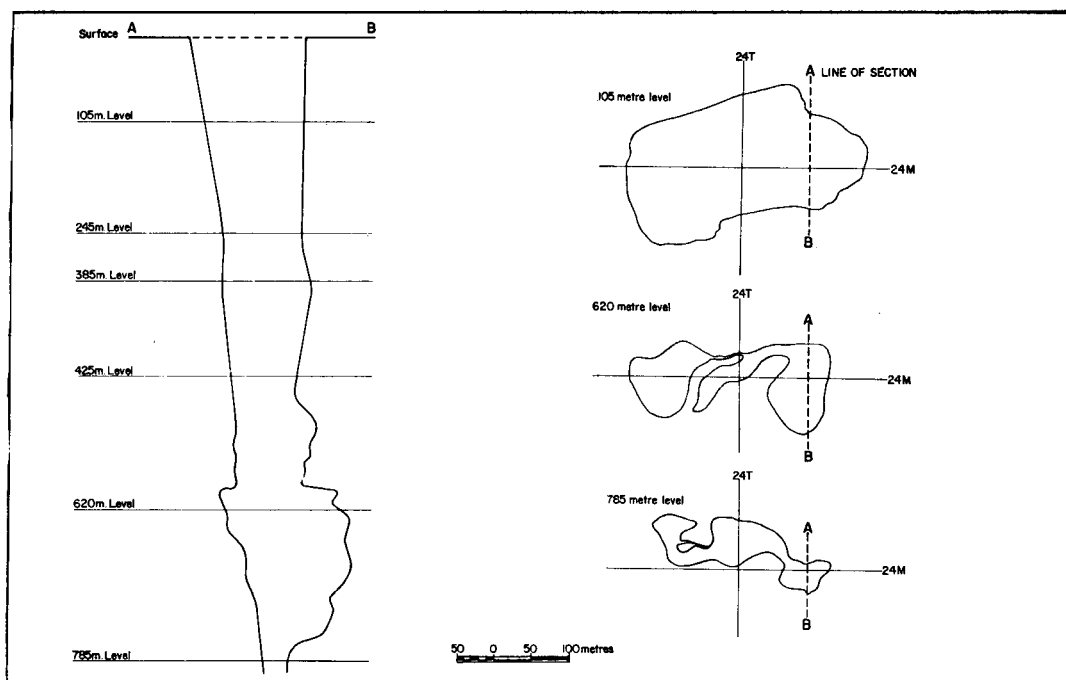


Fig. 2—Plans and section showing the downward taper of the upper part, and the irregularity of the lower part, of a typical Kimberley pipe

vices, etc. The four mines in Kimberley feed ore by conveyor to a central treatment plant and enjoy joint services. These mines utilize the block-caving method of mining, which has almost entirely replaced the previous method of chambering. The combined tonnage of ore produced by the Kimberley Mines is 15 000 t per day, Koffiefontein 12 000 t per day, and Finsch 13 000 t per day.

GEOLOGY

The Kimberley pipes are approximately circular or oval in shape, and are deeply eroded volcanic necks, which, at the present land surface, have average diameters ranging from approximately 230 to 500 m. The upper parts of all the pipes decrease in area with increasing depth, the average dip of the contacts being approximately 80°. At depths of more than 500 m, this regular decrease in area is not always maintained, and the pipes assume highly irregular forms. The contact between the pipe and the wall-rock varies in inclination rapidly over short vertical intervals, and a marked effect of the kimberlite intrusion by structures in the wall-rock is evident. This irregularity at depth is most pronounced at the

Wesselton, De Beers, and Dutoitspan pipes (Fig. 2).

From surface to approximately 100 m, the wallrocks around the Kimberley group of pipes consist of black and grey shales of the Karoo System, containing numerous and generally concordant dolerite sills. The shales overlie a variable thickness (300 to 1000 m) of Ventersdorp lavas with intervening quartzite horizons. The Ventersdorp rock rests uncomfortably on Basement granite-gneiss. The Ventersdorp System is not present at Koffiefontein, where Karoo shales, with intervening dolerite sills, rest directly on granite-gneiss at a depth of approximately 250 m. The wallrocks at Finsch consist of 130 m of banded ironstone separated from the underlying dolomite, limestone, dolomitic shale, and chert of the Dolomite Stage by a sequence of passage beds approximately 25 m thick. The passage beds consist of colourful, rhythmic alternations of thin dolomitic, siliceous, and iron-rich layers.

Several stages of intrusion have resulted in the presence of separate columns of kimberlite and kimberlite breccias within the four Kimberley pipes. Internal contacts between these separate phases are

generally sharp, indicating that earlier intrusions were completely consolidated prior to the intrusion of later material. At Finsch and Koffiefontein Mines only one major phase of intrusion is apparent at the present levels of exposure, but, as in the Kimberley pipes, included fragments of earlier kimberlite indicate a more complex intrusive history.

Most varieties of kimberlite are characteristically hygroscopic, swelling and crumbling when wet owing to the absorption of water by clay minerals in the rocks. This characteristic makes wet drilling of kimberlite extremely difficult, and additional complications arise from the generally incompetent, low-strength characteristics of kimberlite, compounded by the abundance of structural discontinuities (shear planes and fractures). These features make kimberlite difficult to support but are an advantage during block-cave mining operations since they facilitate collapse and break-up of ground within the caves.

The occurrence of diamonds in these pipes and separate intrusions within individual pipes ranges from approximately 1 part in 8 million to less than 1 part in 100 million. The distribution of diamonds is

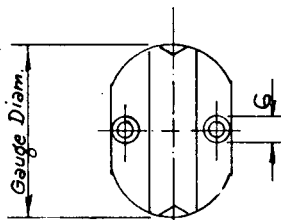
extremely complex. At any level in the pipes, the grade of separate intrusions may be similar or may vary by a factor of five or more. Similarly, pronounced changes in horizontal grade may occur within a single intrusive phase. Vertical variations of grades within individual intrusions may be relatively limited, may change in more or less linear fashion, or may vary

irregularly over successive depth zones. In general, however, both the grade and the average size of diamonds decrease with depth.

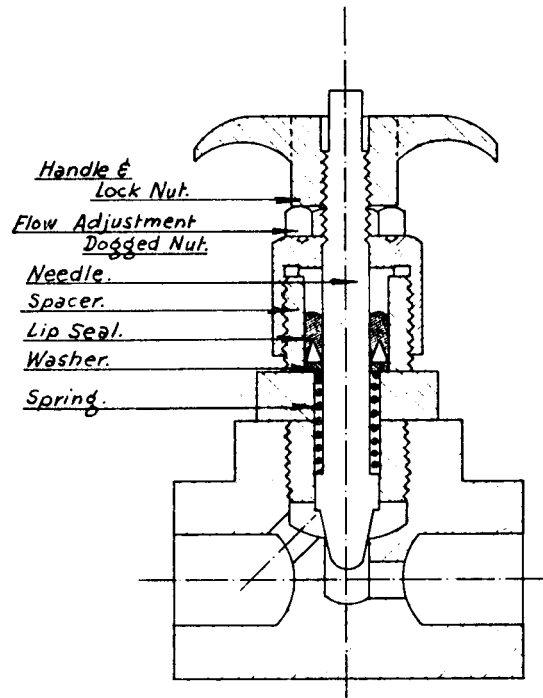
CHEMICAL DRILLING

The problem of dust during drilling operations has long been one of the biggest single mining hazards. Unlike other mining operations, where water flushing through the

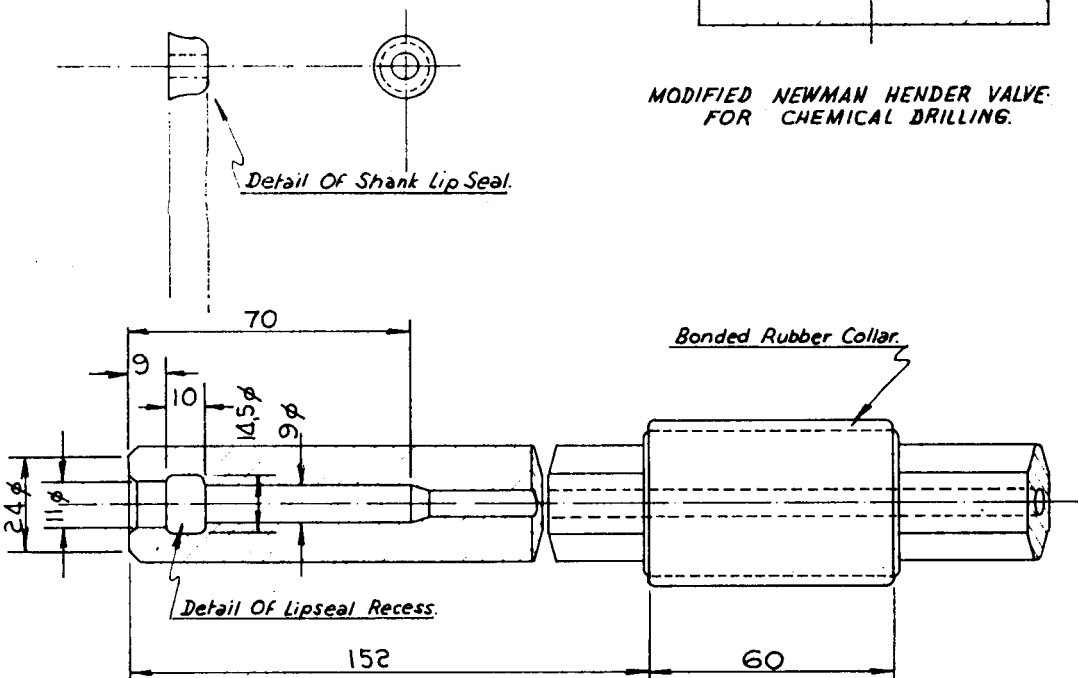
axial hole of the drill steel effectively allays dust, water alone cannot be used during the drilling of kimberlite because it coagulates the dust particles into a thick clay that causes the drill steel to stick in the hole. In addition, the roof, sidewalls, and footwall are weakened by water and therefore become difficult to keep safe. Fortunately, however, there is no free silica in kimberlite,



FRONT VIEW OF DRILL BIT.



MODIFIED NEWMAN HENDER VALVE FOR CHEMICAL DRILLING.



DETAIL OF 25mm. HEX DRILL SHANK.

Fig. 3—The valve, drill bit, drill-steel shank, and lip seal used in chemical drilling

and thus the dust hazard is much smaller than in some mines. Most of the drilling in kimberlite has hitherto been dry, but mining regulations introduced in 1965 called for a dust count of less than 500 particles per millilitre. Attempts were made as far back as 1949 to suppress the dust, and various types of dust-extraction filters, mobile dust cars, and chamber dust cars were used with a fair measure of success. However, it was eventually found that greater efficiency could be obtained from the use of a chemical dispersant introduced into a small amount of flushing water.

During the development phase, experiments on various chemicals were conducted, but it was eventually found that a dispersant called Dispex N40 was the most effective and economical. It was also possible to drill both soft and hard kimberlite with Dispex, and the overall penetration rates were faster as a result. The number of dust particles per millilitre at the drilling face with the chemical method is 85, compared with 244 for dry drilling with dust-extraction units, and 500 as the permissible standard. The cost is approximately R0,01 per metre.

A valve having the following characteristics is necessary to control the mix of water and Dispex to the machine:

- (a) an adjustable flowrate but with a set drilling position,
- (b) a set drilling position that cannot be easily or inadvertently altered by the machine crew,
- (c) a closed position to prevent waste of solution during delays, and
- (d) a flushing action that alters the set flowrate only while flushing is required.

It was found that Newman Hender control valves modified to inject sufficient chemical solution to give the desired consistency of sludge at average penetration rates (Fig. 3) were the most suitable.

The drilling fluid is mixed in and reticulated from a mixing tank that is calibrated and set to give a 0,5 per cent concentration of Dispex. The 'chemical' rock drill has a longer hard-chromed water tube, which requires a recess and a seal

in the shank of the drill steel to allow the water tube to penetrate into the steel for approximately 58 mm (Fig. 3). The head of the drill steel has two holes drilled adjacent to the T.C. insert that intersect the axial hole to ensure proper wetting and reduce blockages.

Encouraging results have been achieved with standard non-vented rock drills. Special permission was obtained from the Mines Department for an experiment, the advantages being

- (1) less leakage due to increased pressure on the seal,
- (2) no air blowing onto the face from the front-head release ports, thus reducing dust formation, and
- (3) no 'fogging' in the event of a faulty seal or water tube.

Because Dispex N40 is a dispersant, any dirt in the pipes and mixing tanks is loosened, causing blocked water tubes and thus rock-drill breakdowns. However, this is avoided by ensuring that the inline filters on the pipe column are kept in good condition and are cleaned regularly. Dispex N40 also affects the machine lubricant, which has necessitated the addition of an additive to reduce its detergent action.

WINDING PRACTICE

The shafts, for the sake of convenience, can be divided into the categories of main shaft for the hoisting of rock, men, and materials, and service shafts for the hoisting of men and materials. Each main shaft is equipped with a rock winder and a man and materials winder, and each service shaft with a man and materials winder.

The rock winders are conventional drum winders with Ward Leonard controls designed for depths up to 1200 m and a speed of 17 m/s. They are on automatic operation, i.e., the skips are automatically loaded from weight-batched flasks. Winding-engine drivers are used only for start-up, shut-down, and re-set operations, apart from rope, winder, and shaft inspections. For these purposes they are drawn from a drivers' pool at the Bultfontein Mine or from the Wesselton Mine man and materials winder.

Jeto bottom-dump skips are used at De Beers and Bultfontein Mines, and Sala bottom-dump skips at Wesselton Mine.

The man and material winders on the Bultfontein and De Beers main shafts (and, from 1976, the Wesselton main shaft) are automatic elevators with a payload of 5000 kg and a speed of 7,5 m/s, and are designed for a maximum winding depth of 975 m. In operation, they are identical to an automatic elevator in a building, no driver or operator being required. The station gates and conveyance doors are operated manually, but are electrically interlocked to ensure that they are closed before the conveyance moves, and mechanically interlocked to prevent them from being opened except when the conveyance has decked at its destination.

The conveyance is called to a station by the pressing of a call button as in a building. During shift times and material-handling times, the call buttons are made inoperative by the turn of a key switch, and control is then limited to the conveyance control panel only, which means that the person in charge retains complete control.

Owing to the depth and speed of wind, the use of trailing signal cable is unpractical, and a radio-type 23-channel signalling system is installed, with the signal generator mounted in the conveyance and powered by a battery. A signal-receiving set is installed in the headgear. This set receives, filters, and amplifies the signals received and feeds them to the control system. For safety, a minimum number of two signals is required to be correctly received before the control panel is activated. In the case of an emergency, the emergency stop button in the conveyance is pressed, which cuts the carrier signal and thus prevents any other signals from being transmitted.

A feature of these elevators is the accurate decking achieved, the maximum variance being about 25 mm. This is maintained by the mounting of magnetic switches at each station. They are activated by a vane on the conveyance that can be adjusted to very close tolerances. For shaft inspection, the machine

is controlled from the inspection platform mounted above the conveyance, the speed being limited to a pre-adjusted inspection speed. The same applies to rope inspection, which is controlled from the bank.

Because of the difficulty of slinging long materials above or below an elevator cage, the conveyance for these machines has been built to accommodate materials up to 7 m long inside the conveyance. An air-driven chain hoist is permanently

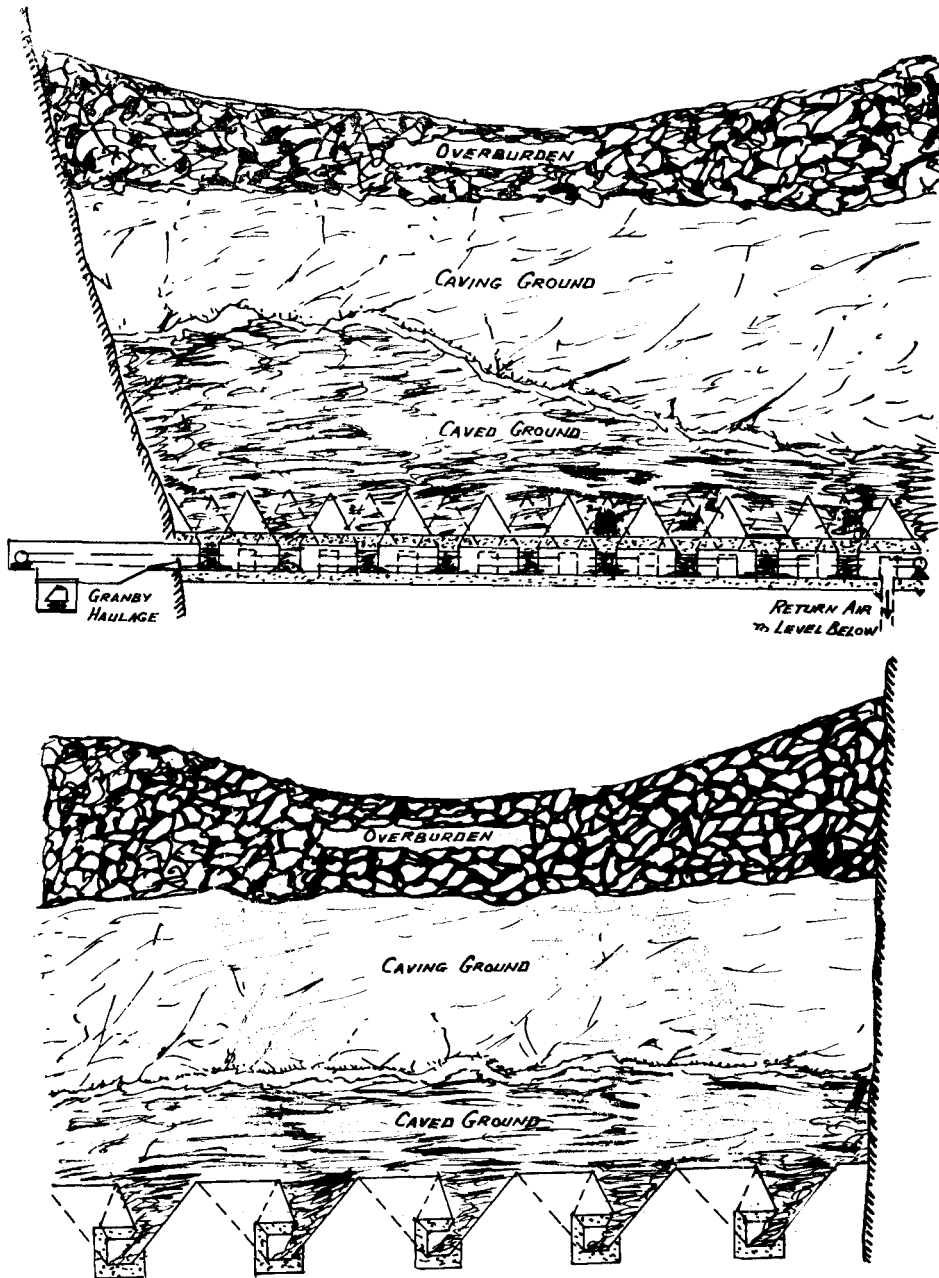
mounted on the bridle to facilitate loading and unloading.

The man and material winders on the service shafts are all automatic elevators with a payload of about 2700 kg and a speed of 2,5 m/s at a maximum length of wind of 360 m. In operation, they are identical to the machines on the main shafts, except that they are controlled by a trailing cable. For the development phase on these shafts, a skip and cage combination

was installed with the skip having a payload of 3 t.

Further progress in automation is being planned for Koffiefontein Mine, where a new shaft is nearing completion. This will be equipped with three fully automatic winders, which will not be equipped with the normal driver's control console, and no winding engine drivers will be required or be on call.

The rock winder will be a thyristor drive, three-rope Koepe with



SECTION ACROSS SCRAPER DRIFT

Fig. 4—The general layout of a block cave

a sheave of 5 m diameter. In operation, it will be completely automatic, i.e., when switched on, it will hoist ore continuously provided that it is available in the underground storage bins and the headgear bins are able to receive it. For shaft inspection, it will be controlled at inspection speed only, from the platform above the skip. For rope examination, it will be controlled at examination speed from a position in the headgear, the bank, or the shaft bottom (for tail ropes). Controls are also provided for skip inspections and starting up after an unscheduled stop. This machine will be licensed as an elevator despite the fact that it is powered by 3850 kW and has a payload of 23 t.

The man and materials winder will be an automatic elevator with the same general specification as the existing main-shaft elevators. It is designed for a maximum length of wind of 1000 m.

The third winder is a heavy-materials winder, which is capable of raising or lowering a load of 30 t at a speed of 1,25 m/s. Its duty is very much the same as that of a crane. It will be operated from a pendant control, with control buttons for 'Up', 'Down', 'Creep Up',

'Creep Down', and 'Stop', that can be plugged in at any station. In addition, it will be possible to despatch the load to a specific station.

The gradual automation of winders over the past thirteen years has resulted in a significant reduction in the complements of winding engine drivers, bankmen, onsetters, and cage assistants. Automated machines are safer as the controls operate only within predetermined limits that cannot readily be bypassed. Although the maintenance of automated winders is more complex than that of manually operated machines, very little difficulty has been experienced in training artisans for this duty.

BLOCK-CAVING PRACTICE

The introduction of block caving is described in a paper published in 1960, by W. S. Gallagher and W. K. B. Loftus. Fig. 4 shows a general layout of a block cave. The present paper deals with the subsequent weight problems encountered in the Bultfontein 580 m block cave, changes in layout and design to overcome the weight problem, and brief descriptions of the subsequent layouts at all four of the Kimberley Mines.

Weight Problems

The general layout of the 580 m block-cave section, Bultfontein Mine, is shown in Fig. 5. Early practice was to carry out undercutting by pillar wrecking with long holes, which required prior installation of the concreted drifts to permit removal of the broken ground from the undercut horizon. This resulted in serious abutment damage to the drifts as the undercut area increased in size. It was planned to bring the southern half of the block into production initially in 1960, followed by the northern half some three years later. However, installation of the completed drifts in the northern half continued ahead of the undercutting in the southern half, and by the time undercutting was completed in the southern section up to Drift No. 8, serious abutment damage was occurring as far as Drift No. 13 in the northern section.

Initially, the undercut floor was carried 6 m above the drift floor, but subsequently this distance was increased to 7,4 m to lessen abutment damage. However, initial production difficulties led to the tops of the cones being slipped off by long holes to reduce the distance

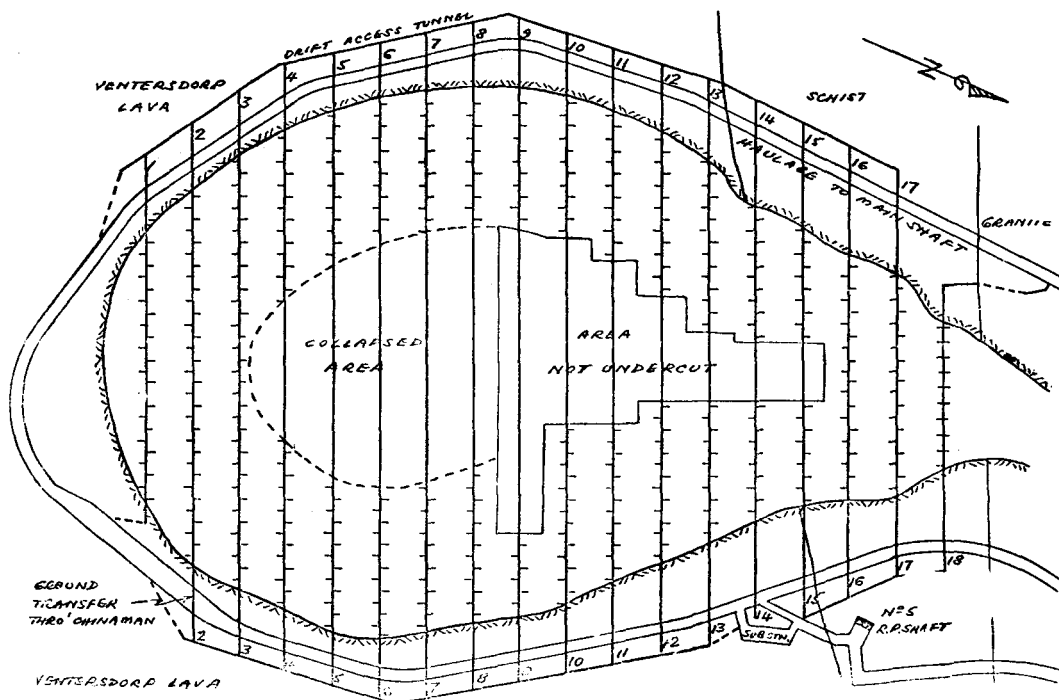


Fig. 5—The layout of the 580 m block cave, Bultfontein Mine

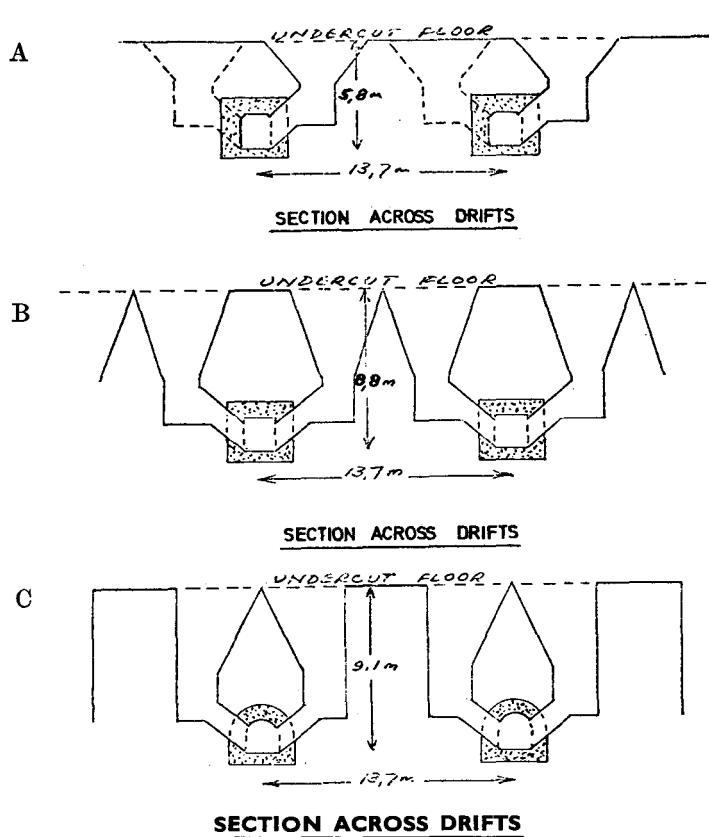


Fig. 6

from floor of drift to top of cone back to approximately 6 m. A section of the cone layout is shown in Fig. 6(A).

Soon after the southern section attained its full production of 4320 t/d, the central area, shown dotted in Fig. 5, collapsed. Concerted and continued drift-repair efforts over several years failed to regain this area, as pressures and weight were excessive. As short lengths of drifts were regained, excessive pressure crushed the concrete lining of the drifts almost overnight, and finally all regaining work in this area was abandoned.

In 1964 work was started to bring the northern half of the 580 m level into production as the ore reserves in the southern half were nearing depletion. All the drifts in the northern half had to be reconstructed as they had deteriorated severely since initial installation. Then, when undercutting of this section was started, it was decided to leave a pillar, not undercut, in the centre, approximating to the area that had collapsed in the southern section. This proved to be a costly decision.

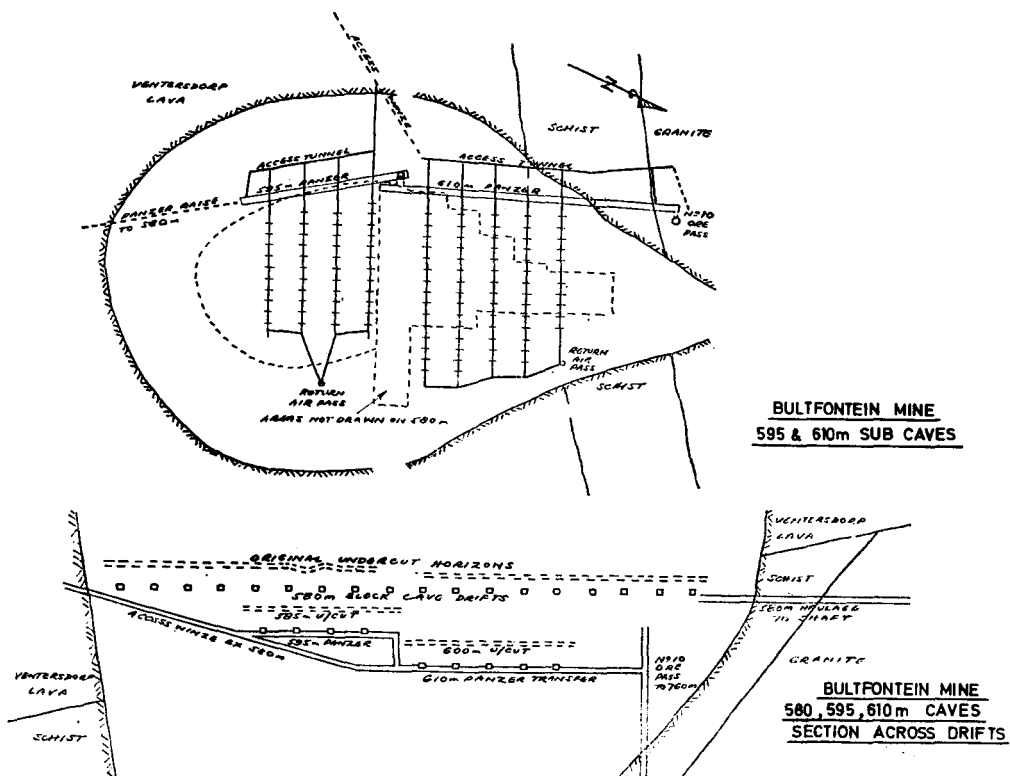


Fig. 7

As full production of the northern section was being reached, the central pillar became a focus of intense pressure. The ends of the drifts adjoining the pillar were pushed back laterally, sometimes at the rate of a few millimetres per day, and intense top and side pressure on the drifts occurred approximately halfway between the pillar and the rock contact. Determined efforts to advance to the pillar in order to undercut it failed. At the same time, a large number of drift-repair crews made little progress in repairing the damaged drifts. Ore reserves on the southern section were running out, and call production could not be achieved on the northern section. The call production was then reduced from 4320 to 2540 t/d, and plans were made for a rescue operation.

Installation of Sub-level Caves

It was decided to construct small block-cave sections below the central collapsed area of the southern section on the 595 m level, and below the pillar area of the northern section on the 610 m level, to recover the lost ore reserves in these areas. On the south side it was decided to make use of existing ventilation level development for the undercut horizon on the 585 m level, with the drifts on the 595 m level, and to elevate the ore by incline to the 580 m level haulage.

Because of the intense pressure exerted by the pillar on the north side, it was decided to site the sub-cave lower down, with the undercut on the 600 m level and the drifts on the 610 m level, 30 m below the 580 m level drifts. There the ore would be conveyed horizontally to an existing ore pass, No. 10, in the country rock. The plan and section view of the sub-caves is shown in Fig. 7.

At the same time, some radical re-thinking took place in the design and procedures for block caving.

Summary of Changes in Design and Procedures

The approach to the weight problem was two-pronged: firstly, to design a system where the weight on the drifts would be minimized, and, secondly, to make the drift

lining as strong as possible to withstand the peripheral stress on the drifts.

In the first category, it was obviously desirable to site the undercut horizon as high as possible above the drift horizon commensurate with practical production requirements. Also, if the undercut could be stopped prior to the installation of the drifts, undercut abutment damage to the drifts could be avoided. It was decided to increase the undercut floor height from 6 to 10 m, as it was felt that 10 m was the limit of height at which 'action' work could be effectively carried out. 'Action' work is the name given to the operation of breaking or bringing down

large lumps that block the top of the cones on the undercut floor horizon and require the 'action' crews to enter the drawpoints either to drill and blast the lumps, or to place large pack blasts to displace the lumps. Cone geometry could also be varied. They could be sited so as to converge over the drifts, leaving a pillar between the drifts (Fig. 6C) or be located between the drifts so as to leave a pillar over the drifts (Fig. 6B). The latter problem was submitted to the Department of Mining of the University of the Witwatersrand, who carried out photo-elastic stress studies with plastic two-dimensional models of the two layouts. These studies

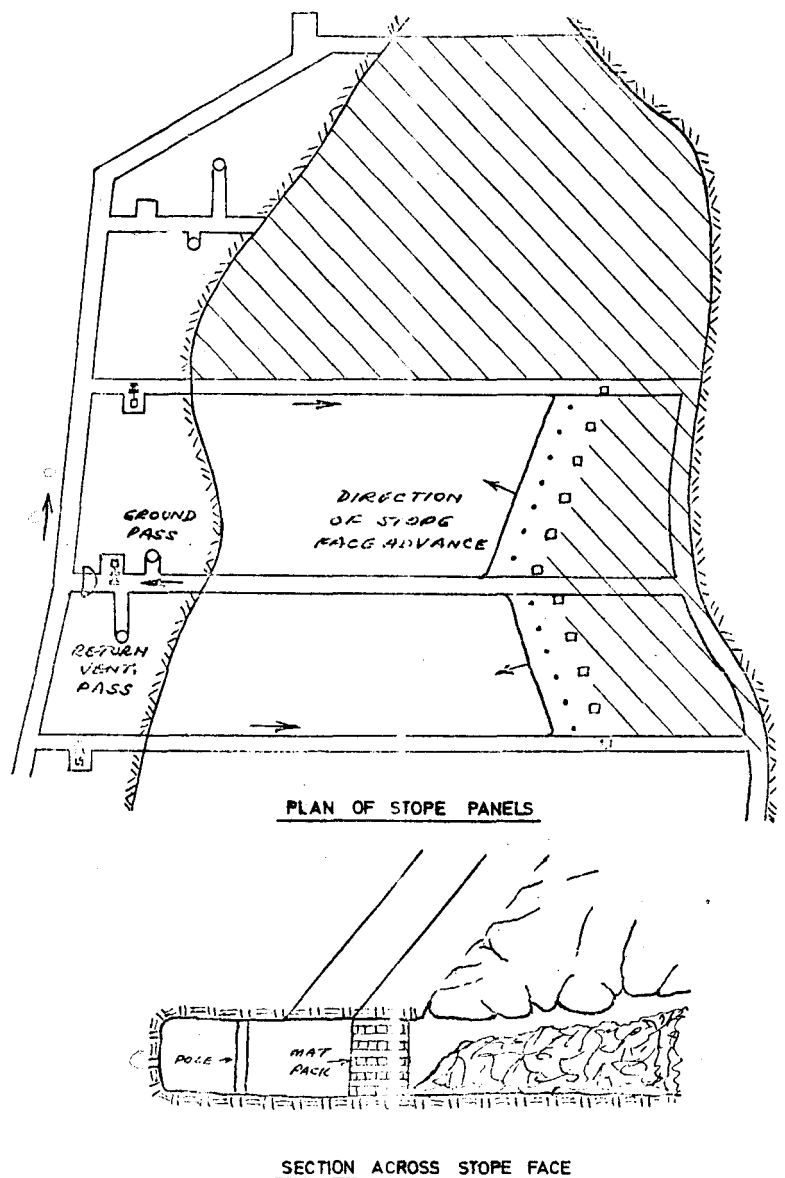


Fig. 8—A typical stope layout

clearly indicated that stresses on the margins of the drifts would be much lower where the cones converged over the drifts (Fig. 6C), and this layout was adopted.

Stoping of the undercut operation could be carried out by breast-stopping panels of suitable width and scraping the broken ground back to ore passes situated in the country rock. As the kimberlite in the Kimberley mines is comparatively weak and incompetent—crushing strengths vary from 15 to 85 MPa with 55 MPa being considered a reasonable average—it was considered advisable to limit face lengths to about 30 m, and to induce early caving behind the face by drilling and blasting 5 m hanging-wall wrecking holes at 50° into the hangingwall. Stope width was 1,7 m. Fig. 8 shows a typical stoping layout. Special attention had to be paid to dense mat pack and prop support close to the face to prevent hanging collapse at the face, and to complete removal of support behind the face where hangingwall wrecking was carried out. Undercutting by stoping was completely successful and eliminated all abutment damage to drifts. In addition, pressure on drifts prior to concreting was considerably reduced as the undercutting tended to de-stress the drift horizon.

In the second approach to the problem, pneumatic placing of the concrete was replaced by pumping, to avoid segregation of the concrete and to enable a concrete of lower water-cement ratio, and hence greater strength, to be transported. Samples of the aggregates were submitted to the Portland Cement Institute, who designed the requisite mix for pumping, to achieve a minimum 28 day strength of 44 MPa. Strict quality control was maintained on the supplies of aggregates by sieving analyses, and on the quality of concrete by slump and cube tests. A double-acting concrete pump is shown in Fig. 9, and diagrammatic sections of a batching plant and the supply of aggregates and cement are given in Fig. 10. Aggregate supplies were stored above the pump and fed to paddle mixers by modern automated weight-batching equipment.

Because of the weak nature of the kimberlite, drifts were initially developed as pilot tunnels 1,4 by 2,1 m across the pipe and then slipped for concreting to 3,5 by 3,3 m. Slipping was carried some 10 to 12 m ahead of the finished concrete, which permitted a 5 m long concrete shell, 1 m thick, to be poured on an existing concrete floor together with a further 5 m concrete floor ahead of it for the next shell. The inside dimensions of the finished drift were 1,72 by 1,83 m. Special attention was paid to obtain smooth-blasted periphery in the slipped portion, and roof-bolt support was installed to prevent fall out.

Drifts were slipped to a normal arch shape, and the concrete filling was also arch-shaped. Special efforts were made to obtain a complete concrete fill to the roof of the excavation. The shuttering was changed from the early monolithic type to a simple shuttering of small plates supported on standard rigid arches. The monolithic type became very difficult to install because of early distortion due to normal wear and tear. Although arch-shaped drifts were considered desirable in earlier practice, they could not be attained because it was not always possible to control the profile of the slipped excavation in the weak kimberlite, nor was it possible to fill to the roof with concrete that was pneumatically placed into the shuttering. Fig. 11 is a view of shuttering in place ahead of completed concrete, but with the front masking removed. Fig. 12 shows a completed drift.

One further problem had to be overcome in the design of the two sub-level caves: the transport of the ore from the 595 m drift to the 580 m haulage for the one cave, and from the 610 m drifts to No. 10 pass in the country rock for the second cave section. As the transport tunnels in both cases were situated in the weak kimberlite, and the finished tunnel would require a 1 m thick concrete lining, there was obviously a limit to the size of the raw excavation that could be supported and adequately controlled prior to concreting, and hence to the finished size of the concrete tunnel. Obviously, a finished tunnel

to carry Granby cars was completely impracticable. In addition, in the conventional block-cave system, all lumps of ore that could pass through the drawpoints (1,2 by 1,2 m) could be handled by the scraping and tramming equipment, and were eventually reduced in size by the large underground crusher, so avoiding the necessity of costly grizzly blasting. If this feature was to be retained, and this was most desirable, conventional conveyor belts could not be used for the transport process.

It was decided to equip the transport tunnels with heavy-duty Panzer or armoured conveyors, as used in longwall coal mining in Europe. Although expensive in initial cost, these proved to be extremely successful for the purpose required. They were robust and could handle all the large lumps delivered by the scraper scoops, as well as being economical to operate. The Panzer conveyor on 610 m level is shown in Fig. 13.

Production from the Sub-level Caves

Production commenced from these two sub-level caves (595 m and 610 m) some 18 months after the start of the whole operation. Initially, six drawpoints in each drift in line were opened for production. Drawpoints were 5 m apart, and thus the width of the producing strip was 30 m. No problems were encountered. After pulling from the strip for about two months, during which a cushion of broken ground built up above the producing drawpoints, the strip was slowly moved across the pipe, by first closing the rear drawpoint and then opening the next drawpoint in front of the strip. After a further two months of production in this position, the process was repeated, so that gradually the producing strip was moved across the whole cave area. This rescue operation proved to be completely successful. No damage occurred, and the production from these small layouts reached 1600 t/d, when the total production from the mine was suspended for a few years because of a recession in diamond sales. It was evident that all the ore re-

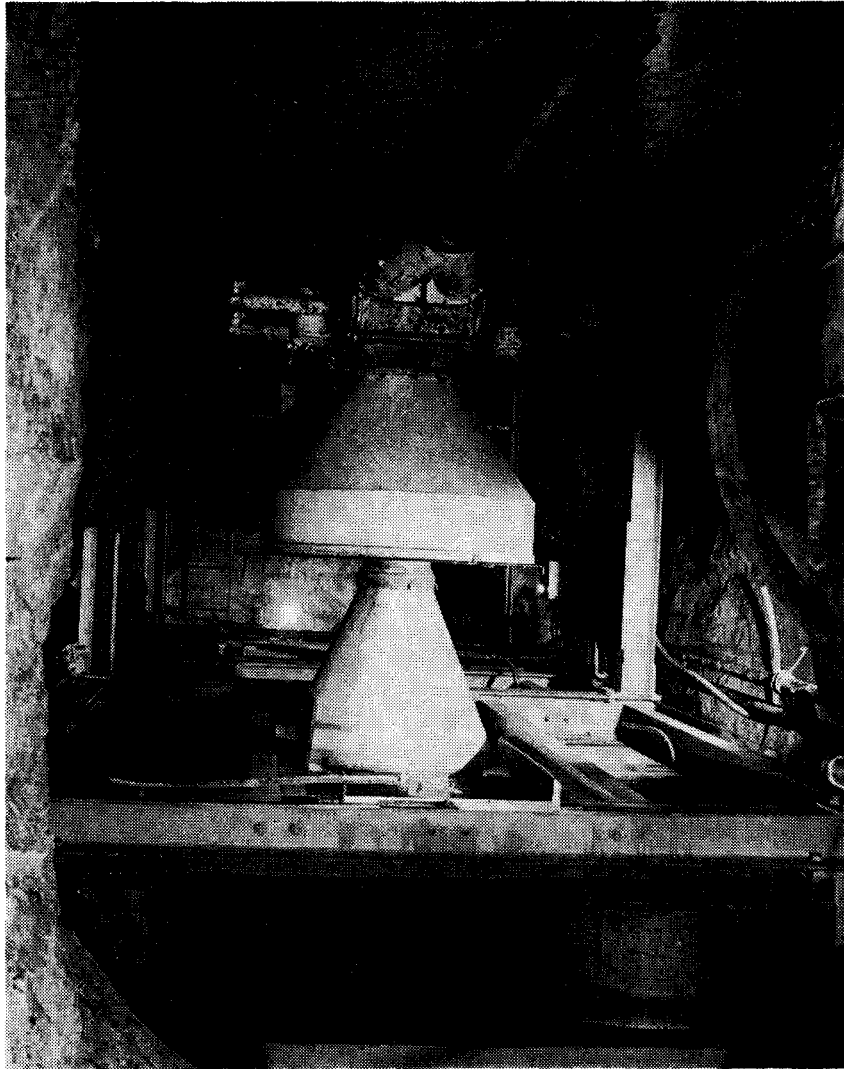


Fig. 9—A double-acting concrete pump

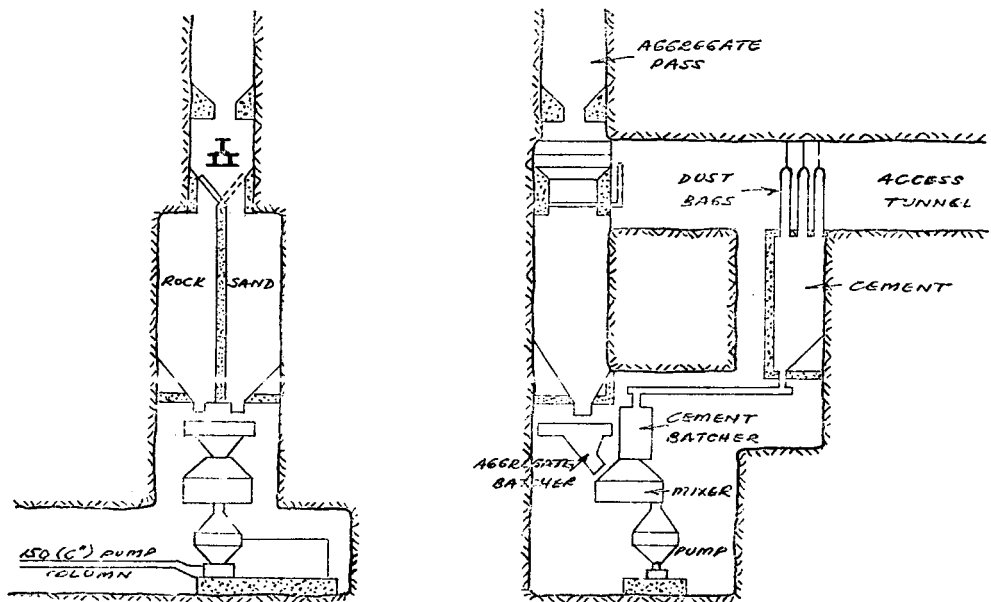


Fig. 10—A batching plant for pumping concrete

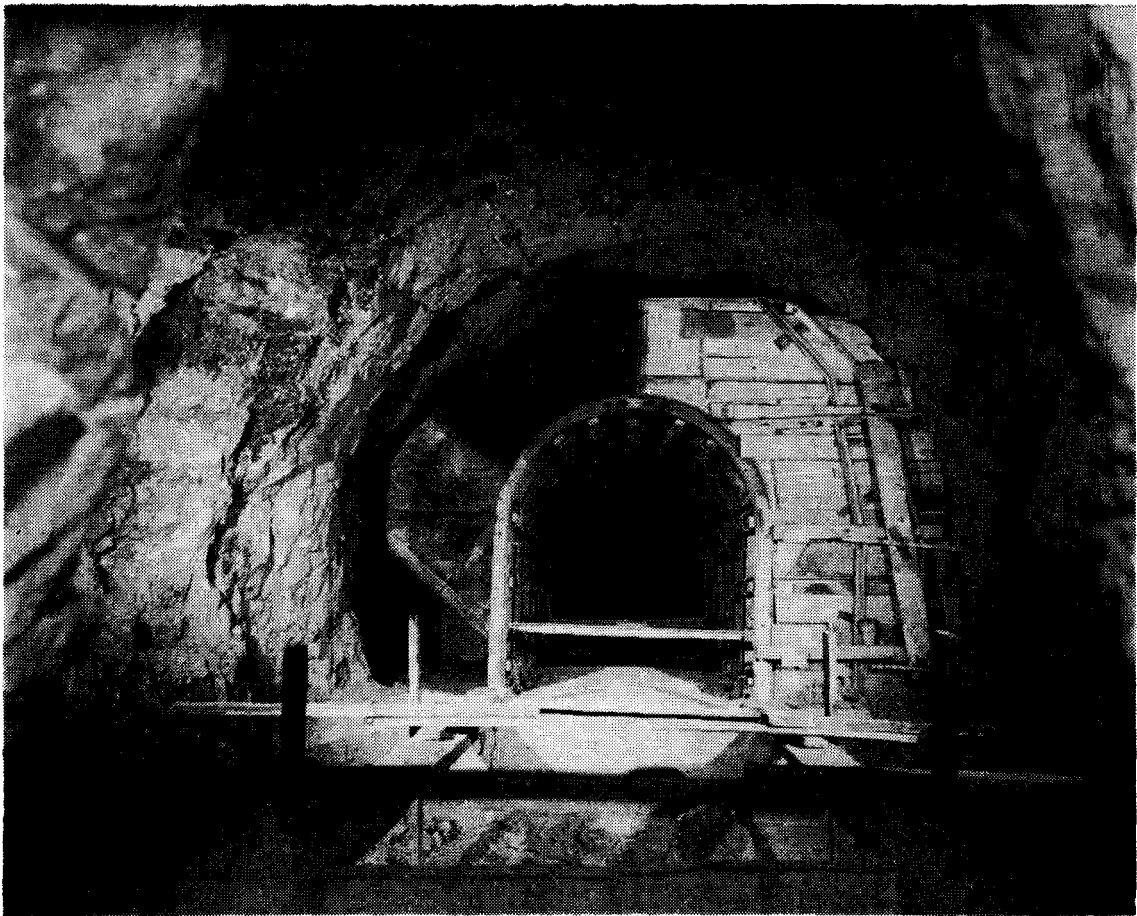


Fig. 11—Concrete shuttering with front masking partially removed



Fig. 12—View of a completed block-cave drift



Fig. 13—View of a Panzer conveyor on 610 m level, Bultfontein Mine

serves lost in the centre of the 580 m level would be successfully recovered. It should be noted that effective draw control, which is essential for optimum recovery of ore reserves, can be achieved only where weight can be controlled and severe drift damage avoided.

Change in Basic Design of the 700 m Block Cave

Whilst the construction phase of the sub-level caves was in progress, development of a conventional block-cave layout on the 700 m level, Bultfontein was proceeding. The haulage from the main shaft to the pipe had been completed, and the haulage tunnel together with the access tunnel round the pipe was being developed in the country rock. However, the country rock from the shaft to the pipe, a competent

granite gneiss, soon changed to a very incompetent chlorite-talc schist as the haulages and accompanying access tunnels on each side commenced to encircle the pipe. Both haulages and access tunnels were supported with yieldable steel arches, the haulages with 29 kg/m sections and the access tunnels with 16 kg/m sections, each spaced 0,6 m apart. As this development proceeded, despite keeping the steel arch support right up to the face, it became increasingly evident that these excavations would not remain sufficiently stable to serve as extract tunnels for the 700 m production. It was at this stage that the design of the 595 m and 620 m sub-caves and the operation of the Panzer conveyors were seen to be entirely successful. Accordingly, the design of the 700 m layout was changed

to embody the successful features of the sub-caves. As the schist country rock on the 700 m level was less competent than the kimberlite, it was decided to use Panzer conveyors in concrete-lined tunnels in the kimberlite as ore conveyors, instead of trolley locomotive trains in the country rock. Accordingly, the arch-lined haulages in the country rock were concrete lined to form circular access tunnels, the existing conventional access tunnels were waste filled, and Panzer conveyors were installed in concrete-lined drifts in the kimberlite 10 m below the production drifts, with short passes equipped with chain-link doors feeding down from the production drifts. Production drifts 15 m apart were installed across the pipe from the now-circular concrete-lined access tunnel, after the undercut had been stoped. The drift layout is shown in plan in Fig. 14, and Fig. 15 shows the plan view of the Panzer transfer level. Fig. 16 shows a section through the transfer pass. Initial production from this cave had just commenced when the production of the mine was suspended, as mentioned above. However, no serious problems are envisaged when production is again commenced from this section.

760 m Layout, Dutoitspan Mine

The main development of the 760 m block cave layout at Dutoitspan was being carried out slightly ahead of that of the 700 m at Bultfontein. This was also a conventional layout. The country rock on the south side was a competent granite-gneiss, whereas that on the north side was a weak schist, although stronger than the Bultfontein schist. The north-side haulage, access tunnels, and winch chambers had been supported with yieldable arches. Although these excavations appeared to be reasonably stable, it was decided to reinforce them with concrete after the experience with the Bultfontein 700 m layout. The haulage was lined inside the arches, permitting just sufficient clearance for the operation of 6 t Granby cars, but not the 6 t trolley locomotives. These trains would be hauled by battery locomotives. The winch chambers were

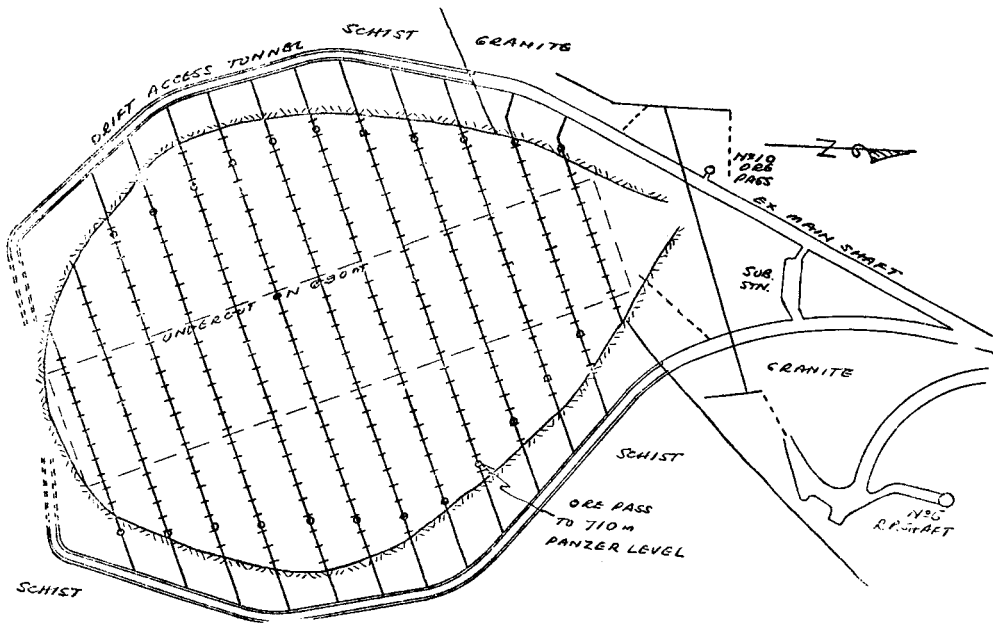


Fig. 14—Plan of 700 m block cave, Bultfontein Mine

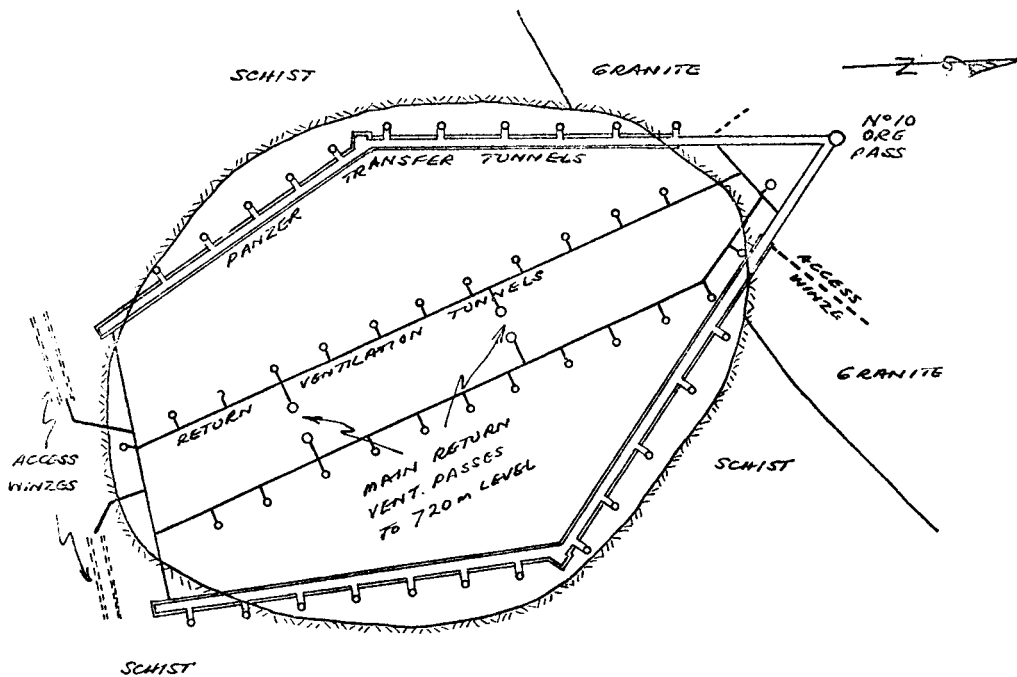


Fig. 15—Plan of 710 m Panzer transfer level, Bultfontein Mine

also concrete-lined inside arches, but not the access tunnels. The latter were heavily injected with Colgrout. The layout is shown in Fig. 17.

785 m Layout, Wesselton Mine

The drift and haulage layout is shown in Fig. 18. As can be seen, the pipe was split into three lobes, but it is of interest that primary sampling development on the 930 m

level shows the lobes to have coalesced into one pipe body again.

Initially, the layout was designed with a single haulage tunnel between the pipe and the main shaft, but the necessity for hauling large tonnages ultimately (about 8300 t/d) required the addition of the line connecting the shaft with the northern part of the pipe, thus completing the haulage loop and making it uni-directional. This would also

facilitate any future automation of the trains.

The general layout in the cave system itself was not changed. However, the following innovations were introduced.

- (a) Pozzolith was used as an additive to the concrete. This resulted in an increase of the final minimum 28-day strength from 44 to 54 MPa.
- (b) On completion of the con-

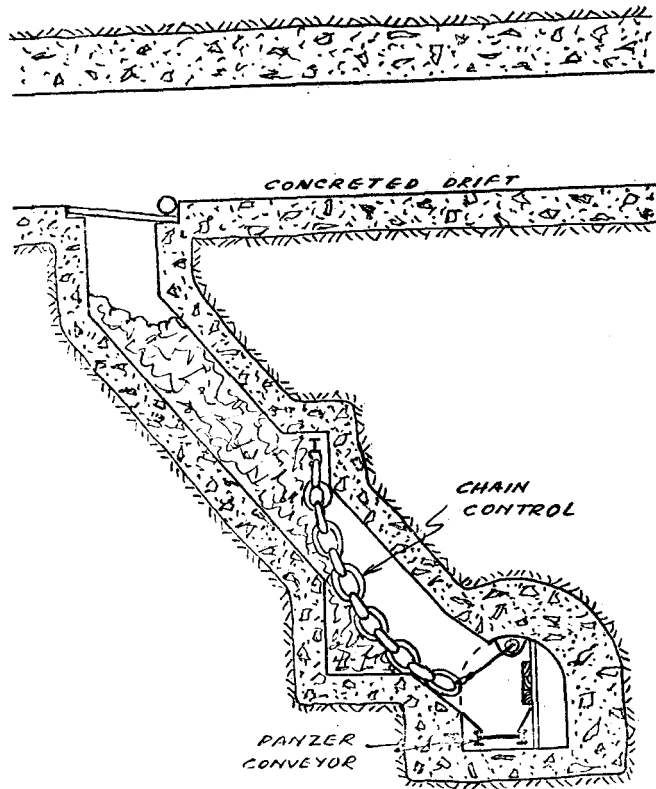


Fig. 16—Section through the transfer pass

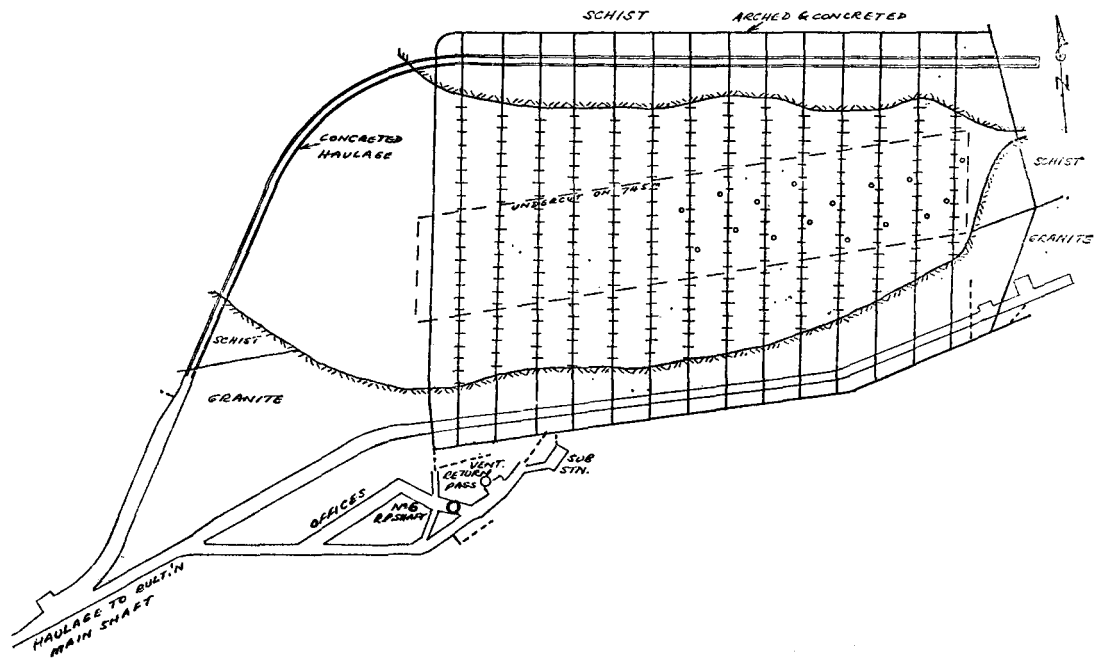


Fig. 17—Layout of 760 m block cave, Dutoitspan Mine

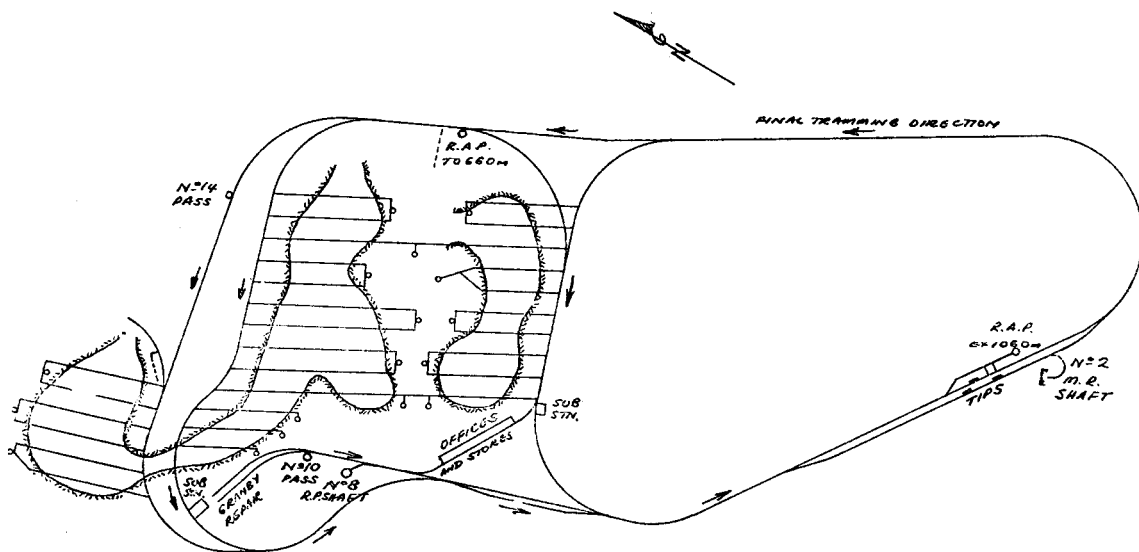


Fig. 18—Layout of 785 m block cave, Wesselton Mine

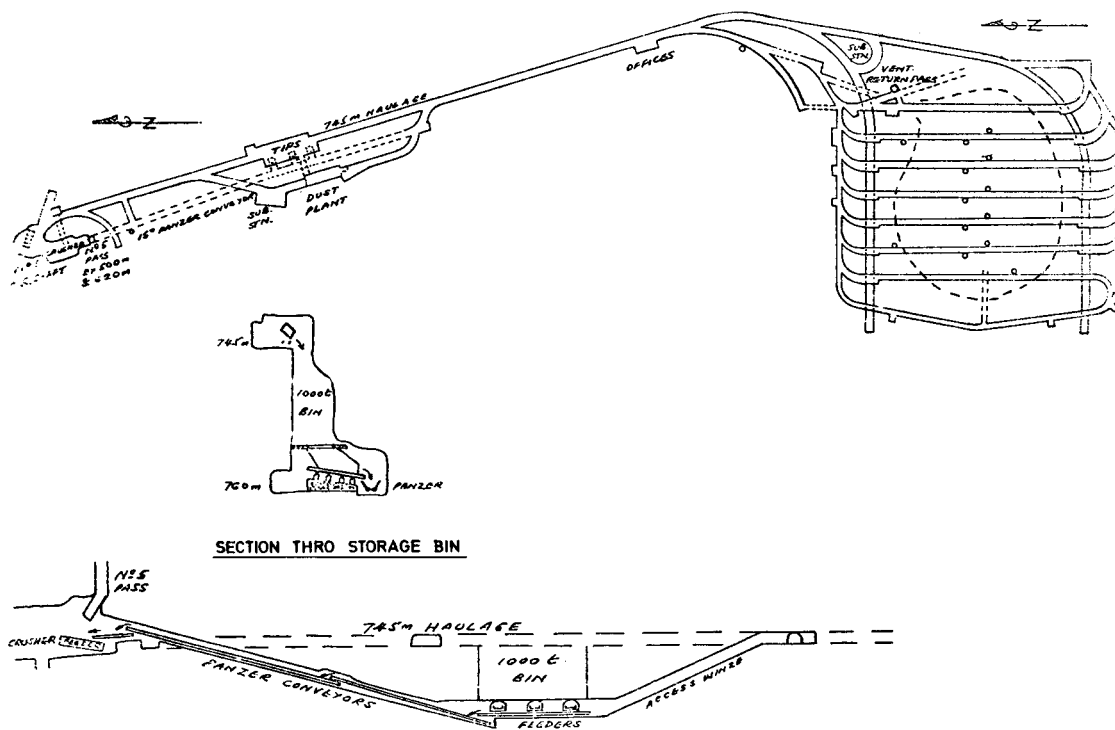


Fig. 19—Layout of proposed 745 m block cave, De Beers Mine

creting of a drift, grout was injected into the roof of each shell to ensure a complete fill between the concrete and the kimberlite.

- (c) All block-cave haulages in the Kimberley mines had been equipped with 6 t Granby cars and 6 t trolley d.c. locomotives. It was decided to equip the Wesselton haulage with 10 t Granby cars and 10 t a.c. trolley locomotives.
- (d) In the stoping operation, the 5 m wrecking holes drilled into the hanging and blasted 8 to 10 m behind the face were discontinued. Instead, the stope was carried 1,9 to 2,0 m wide, and 1,5 m holes were drilled at 50° into the hanging behind the face. These were designed, not to 'wreck' the hanging, but to break an additional 1,0 m from the hanging and so increase the stope width. This led to more stable face conditions and would induce earlier caving.

- (e) Klockner-Ferromatik Hydraulic Props were introduced as face support, instead of mat packs. Three rows were installed 1,5 m apart, and props were 1,5 m apart. These were successful and saved timber and labour. However, in friable ground, especially near the rock contacts, mat packs had to be used as well.
- (f) A light two-boom drill rig has recently been introduced and is being used to complete the haulage loop.
- (g) A complete investigation was made into full automation of the haulage (driverless trains and remote control of filling and tipping). However, the cost of automation is not economic at present wage levels.

745 m Layout, De Beers Mine

This is the most recent of the block-cave systems, and the proposed layout is shown in Fig. 19.

A feature of this layout is that

horizontal ore storage has been provided on the same level as the underground crusher, and the ore will be reclaimed from below the horizontal storage bins by a Panzer conveyor that feeds onto an inclined Panzer conveyor, delivering the ore into the crusher. This crusher treats the ore from the existing production levels on the 500 and 620 m levels. If conventional storage in passes above the crusher had been adopted for the new layout, an additional crusher and access cross-cut from the shaft would have been required. The 745 m level has been estimated to be the economic cut-off level of this section of the pipe.

The 745 m haulage from the shaft was commenced with a two-boom medium drill rig (Fig. 20). After initial training of crews and teething troubles, the development has settled down into a smooth operation. The rig is operated by three Blacks, who also carry out the complete cleaning operation with a front-end

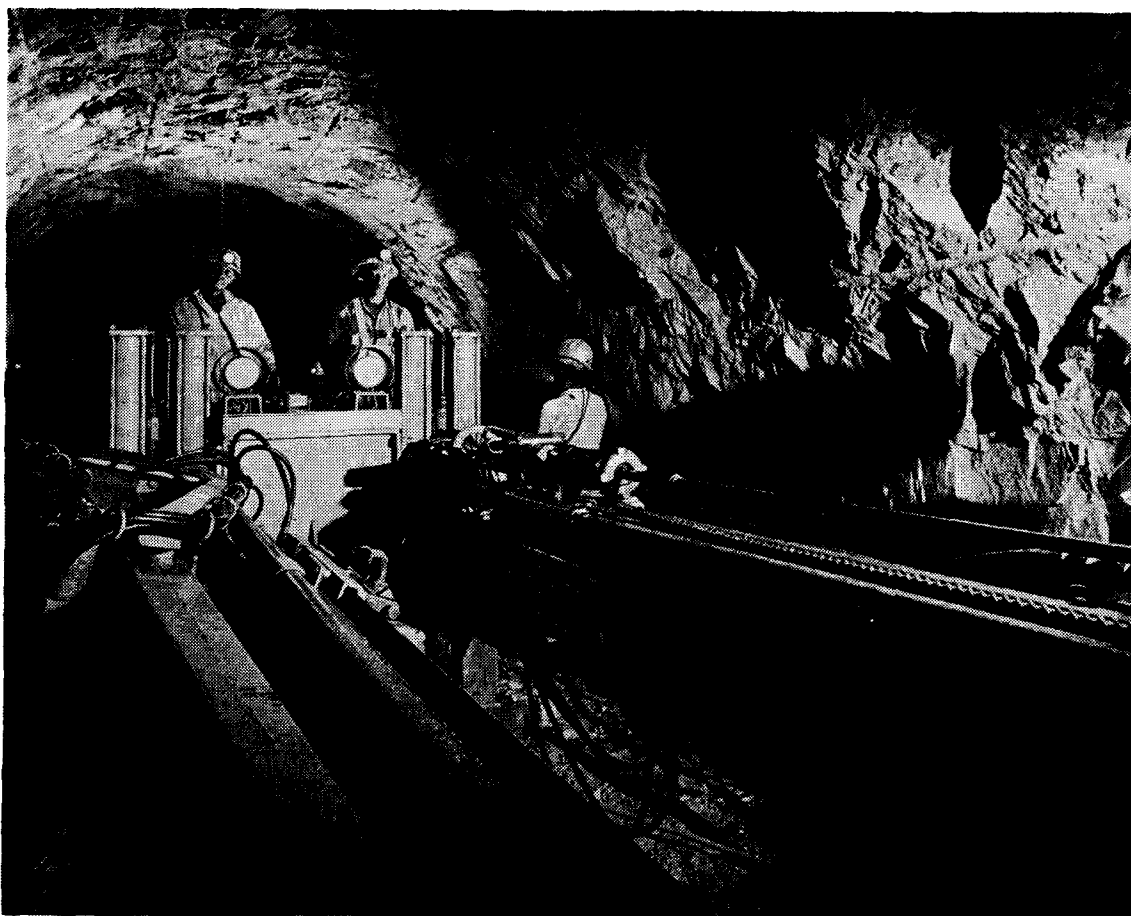


Fig. 20—Twin-boom pneumatic drill rig used at De Beers Mine

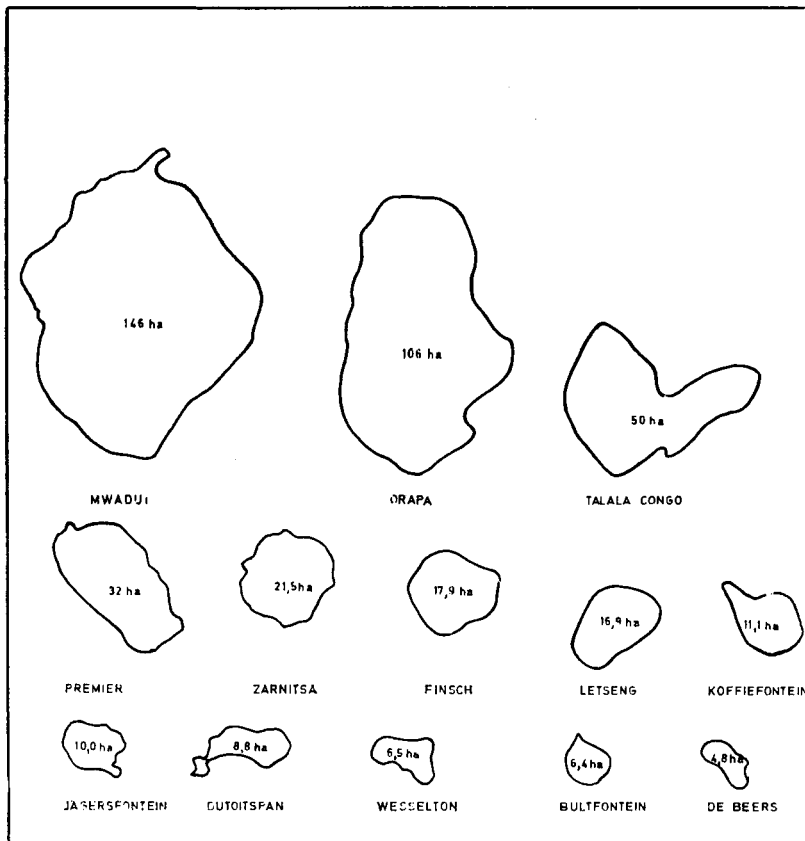


Fig. 21—Relative sizes of some kimberlite pipes

loader and Granby cars. On a three-shift operation, the 3,5 by 3,4 m haulage is advanced 5 m per day. It is also intended to carry out the development around the pipe, haulage, access tunnels, and drifts in kimberlite with the smaller drill rig at present in use at Wesselton.

Research Investigations

The Mining Department of the University has taken a keen interest in a number of aspects for improving the block-caving system, during the course of which they developed a sophisticated and effective stress-strain measuring instrument. As mentioned for the Bultfontein sub-level caves, stress distributions around drifts for different cone configurations were measured using the photo-elastic stress technique. The results led to the adoption of a configuration that has proved highly successful.

At present the Rock Mechanics Department of the University is carrying out a similar investigation of a cone configuration comprising a continuous slot between drifts,

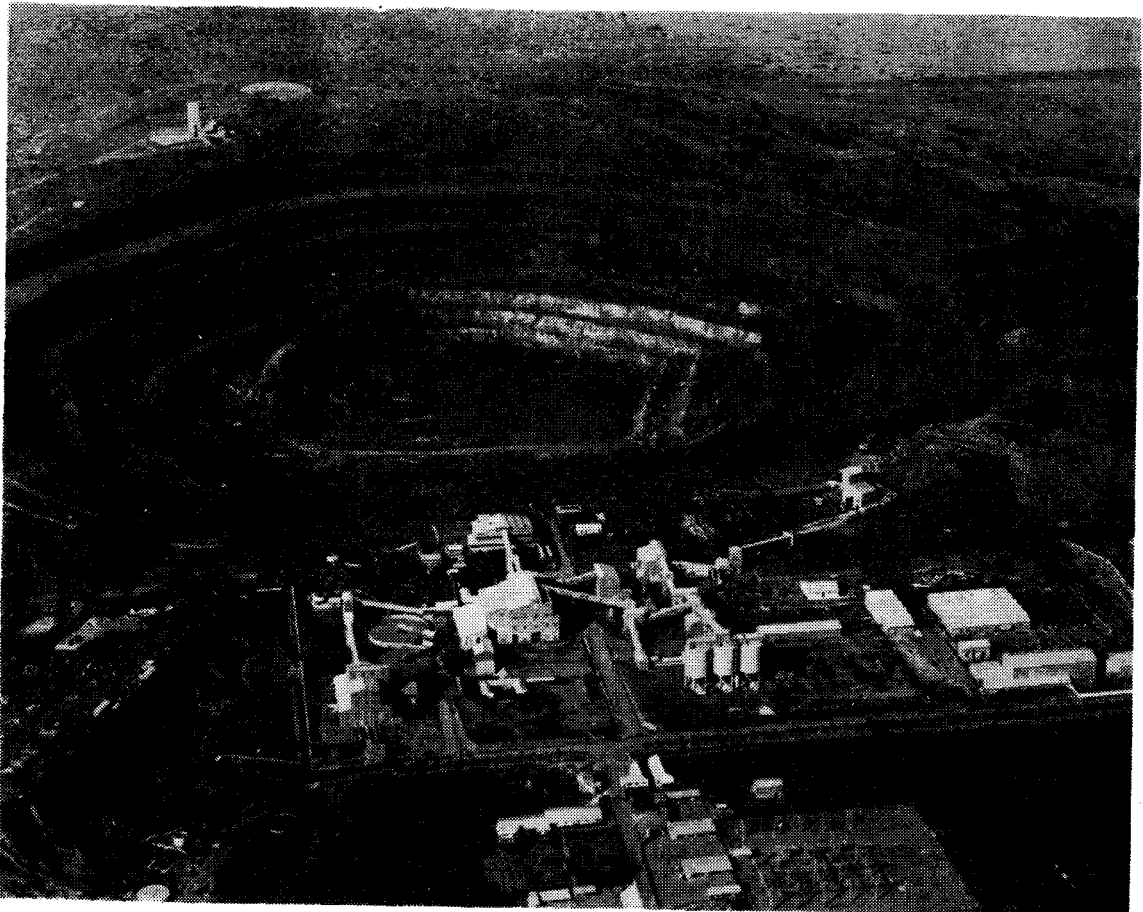


Fig. 22—Aerial view of Finsch Mine

which would be cut by a mechanized rig. The technique of finite element analysis is being used to compare this proposed configuration with the existing successful one.

General Statistics

Output per White underground shift	318 t
Output per Black underground shift	20 t
Output per man (Black and White)	19 t
Output per case of explosives	250 t
Output per winch per shift	100 t
Cost per centare stoped	R17,00
Cost per cone developed	R300,00
Cost per metre of concrete drift	R250,00
Cost per tonne, including treatment and general expenses	R2,60

OPEN-CAST MINING AT FINSCH AND KOFFIEFONTEIN

Finsch Mine has now been in production for almost eleven years, during which 43 000 000 t of waste rock have been removed and 24 000 000 t of diamondiferous ore have been treated. The pipe is roughly circular and is approximately 500 m in diameter with a surface area of 17,9 hectares. The size of the Finsch pipe relative to other major pipes can be seen in Fig. 21.

The mining layout is a series of 12 m benches, which were originally cut at an extremely shallow angle but which have gradually been steepened to an angle of 45°. The shape of the pit is polygonal, roughly approximating the shape of the pipe (Fig. 22). To eliminate the number of corners inherent in a zig-zag roadway, an 18 m width spiral roadway has been laid out at a gradient of 8 per cent (1 in 12). To date, the pit has reached a depth of 148 m below the surface.

Koffiefontein Mine was re-opened in January 1970 after having lain dormant for 40 years. Since then, 37 000 000 t of waste have been mined, and 11 000 000 t of ore have been treated (Fig. 23). Initially, Koffiefontein was regarded as a marginal proposition, but more recently viable reserves have been proven for a life in excess of twenty

years. It became obvious that Koffiefontein would not be mined throughout its life as an open pit, and major planning exercises were initiated to cater for the eventual drop down to an underground mining operation. Nevertheless, in the operation of the open pit, the equipment, final slope angles, and the ultimate open-pit bottom require constant re-assessment. This extremely interesting phase in Koffiefontein's life will extend until about 1981, when the transition from open-cast to underground mining will be complete.

Equipment

The equipment currently used at Finsch and Koffiefontein is as shown in Table 1.

TABLE 1 EQUIPMENT USED AT FINSCH AND KOFFIEFONTEIN MINES		
<i>Finsch</i>		<i>Koffiefontein</i>
	<i>Drilling</i>	
4 rotary drill rigs		6 rotary drill rigs 2 rotary percussive drill rigs
	<i>Loading</i>	
1 6 yd ³ electric shovel 3 4,5 yd ³ electric shovels 1 10 yd ³ front-end loader		4 6½ yd ³ electric shovels 4 10 yd ³ front-end loaders
	<i>Hauling</i>	
10 70 t haul units		11 70 t haul units 7 50 t haul units 5 35 t haul units
	<i>Service and Sundry</i>	
1 crawler mounted dozer 2 rubber-tyred dozers 3 graders 2 water carts 1 padding truck 1 explosive truck 1 Anflex truck		1 crawler mounted dozer 3 rubber-tyred dozers 2 graders 2 water carts 1 padding truck 1 explosive truck 1 Anflex truck 1 hydraulic breaker mounted on a hydraulic excavator.

Where possible, standardization of equipment has been attempted, although in certain instances this has not been possible owing to poor delivery dates.

An interesting development has been the recent introduction, at Koffiefontein, of a hydraulic impact breaker mounted on a hydraulic excavator (Fig. 24). One of the major problems at Koffiefontein has been the poor fragmentation of the dolerite and the soft kimberlite, resulting in a disproportionate amount of secondary blasting. Until recently, this duty was carried out by rotary percussive drill rigs and has been expensive in terms of men and materials, and disrupting to the pit cycle because of blasting and the consequent dozing of fly rock. The introduction of the hydraulic impact breaker has been very impressive. Delays in the pit and

plant have been reduced, the number of men employed on secondary breaking being reduced from four to one.

Selection of Equipment

During the past decade there has been a tendency to move towards larger-capacity equipment. The primary advantages in this escalation in size have been a reduction in the number of units required during any specific phase of the life of the pit, and an increase in the rated duty of units such as drill rigs. Thus the number of operators has tended to decrease, the maintenance requirements have tended to become easier, and the traffic density in the pit has fallen.

However, larger units have some

disadvantages such as a reduction in the flexibility of the operation, and the mismatching of the larger units with permanent or semi-permanent equipment such as the treatment plant and the shovel fleet. It is extremely important, therefore, to consider the entire system in which a unit will operate, and not the unit in isolation.

The present selection techniques are based on three broad factors: (1) operational effectiveness or 'Will it work in the situation?' (2) operational efficiency or 'How much will it cost?' (3) distributor service or 'Will spare parts and technical advice be available from the distributor/manufacturer?'

To effect this type of technique, the units under consideration are studied by three departments: the Mining, Engineering, and Operations Re-

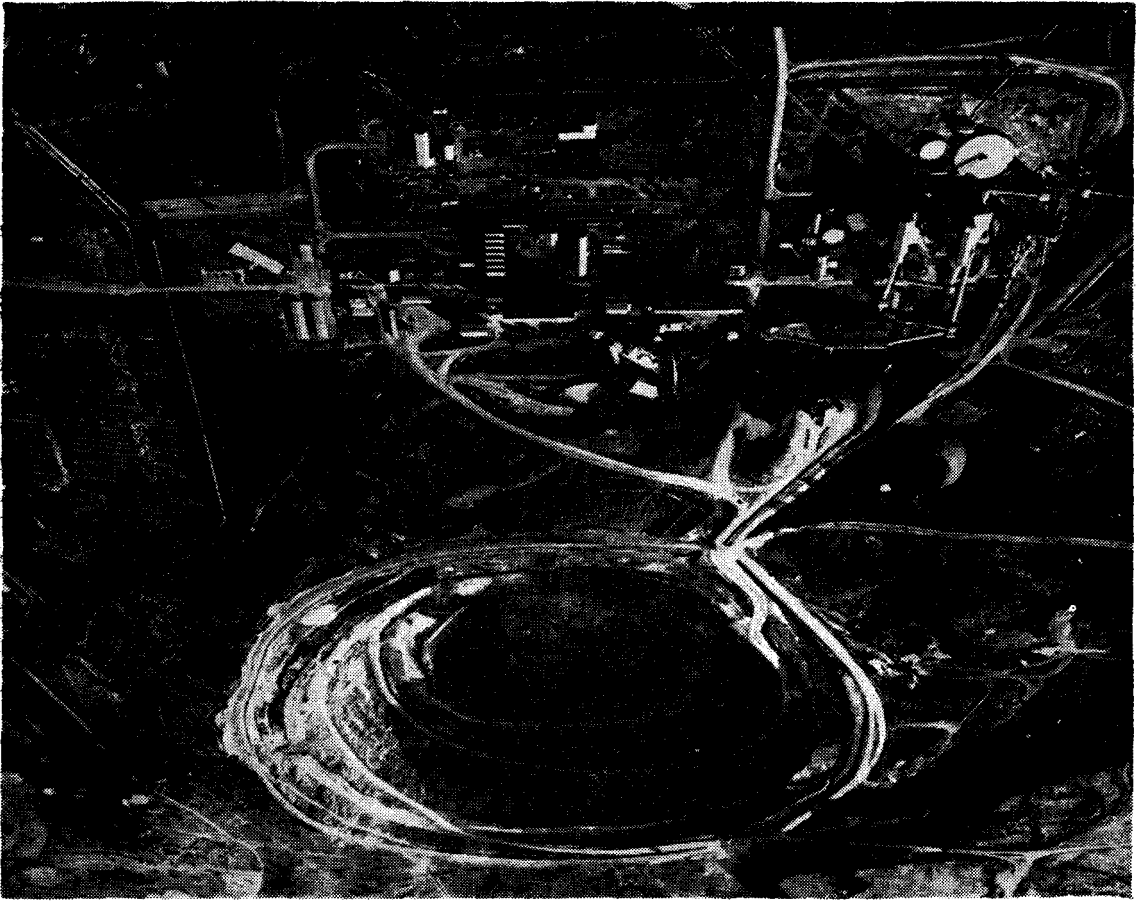


Fig. 23—Aerial view of Koffiefontein Mine



Fig. 24—Hydraulic rock breaker used instead of secondary blasting

search Departments.

The Mining Department is required to comment on the operation of each unit. Where possible, personnel are sent to look at each unit under operational conditions to pick up information on performance and general characteristics. The Engineering Department is required to comment on the probable maintenance costs based on previous experience, interpolation of results achieved by others, and a knowledge of stores holding and workshop facilities. The Operations Research Department is required to analyse the information and simulations received from manufacturers and identify any significant differences in unit performance or cost.

The most important selection in the future will be that of haul units. The shovel fleet is relatively young so that, unless the decision is taken to purchase shovels with larger bucket capacity, it will probably not require replacement. Similarly, the drill fleet is young, with an average age of 2000 hours. The haul fleet, however, requires continual replacement, and an exercise is currently being carried out to determine the truck best suited to the operation. The boundaries of the system in which these haul units will operate are the $4\frac{1}{2}$ yd³ (approximately 9 t) electric shovels and the 450 t/h treatment plants, limiting the choice to the 50 to 70 t range.

Development of a Stripping Philosophy

When Finsch Mine was originally planned, considerable thought was given to the rate at which waste stripping should be effected. The choice lay between the two extremes of mining each waste bench to its final limit before commencing the mining of the underlying bench, or stripping at the instantaneous stripping rate, which means mining only enough waste to liberate the ore required. The problem was made more difficult by the fact that the final slope angles, and thus the ultimate pit configuration, could only be guessed. To gain experience in the necessary mining techniques and to allow time to establish the final slope angle, the decision was

made to sub-divide the mining sequence into four main stages.

The first stage involved mining within the orebody, and thus no stripping was involved. The second stage required waste stripping to be carried to final pit limits, the slope angle being kept fairly shallow (approximately 22°). During this stage it was envisaged that work would be carried out to determine the final slope angle. The stripping ratio was fixed at 1.37 to 1, which was considerably higher than the minimum stripping ratio and allowed a margin for error. The third stage necessitated the mining back of the waste benches to the final limits set by the slope angle, and stage four was the mining back of the flat ore benches to the maximum safe ore slope angle.

The mining pattern has followed the original plan fairly closely, but during the past two years the major disadvantage of mining to an average stripping ratio, that of a detrimental cash flow, has become accentuated by spiralling rates of inflation. This disadvantage is compounded by indications that the final safe slope angle may be as steep as 60° in the ironstone. Thus, the stripping ratio has been gradually reduced to the present ratio of 1 to 1 and could be reduced further when current investigations determine the final slope angle.

As this ratio is reduced and the cash flow enhanced, there will, of course, be a price to pay in loss of flexibility in the mining sequence and the possibility that ore production could be lost if waste stripping were held up. Another disadvantage will be the requirement of an increasingly large haul fleet as the pit deepens.

The final decision will probably be a compromise that will follow the lead given by the original planning team. However, it should be possible to reduce insurance margins because of improved knowledge of the properties of the country rock, and experience in the mining operation.

Economic Open-pit Depth at Finsch

Initial exercises to determine the ultimate depth of the open pit

used the 'ratio of areas' rule, which propounds that

- (i) the instantaneous stripping ratio is equal to the ratio of the respective plan areas of waste stripping and ore mining, and
- (ii) thus, if the surface waste stripping limits are fixed, the instantaneous stripping ratio is the same whatever pit slope is taken.

As the area of the orebody was known and the economic stripping ratio (the value of the instantaneous stripping ratio when the cost of open-cast mining is equal to the cost of underground mining) could be calculated, then the total surface area of waste stripping could be determined. The final slope angles were assumed to be 45° in waste and 35° in ore, and the ultimate open-pit depth was estimated to be 275 m.

The significant factors in the determination of final open-pit depth are the final slope angles and the relative costs of open-pit and underground mining. Costs are a constantly changing variable and necessitate continuous updating of the pit-bottom calculations.

Currently, work is under way on a detailed examination of the properties of the country rock in an attempt to obtain more accurate estimates of the final slope angles that can be achieved in the waste cuts. The major areas of investigation are the detailed mapping of the joint planes and core drilling to establish the nature of passage beds that are known to exist between the ironstones and underlying dolomites.

Economic Open-pit Depth at Koffiefontein

The mining sequence at Koffiefontein was complicated by an already-existing excavation, and the slope angle of the excavation was, in effect, the failure angle of the country rock. Thus, the open-cast mining of Koffiefontein was planned in three distinct phases or cuts (Fig. 25).

The dimensions of the first cut were a minimum, commensurate with haul road requirements and the configuration of the original open pit to enable ore production to commence at the earliest possible

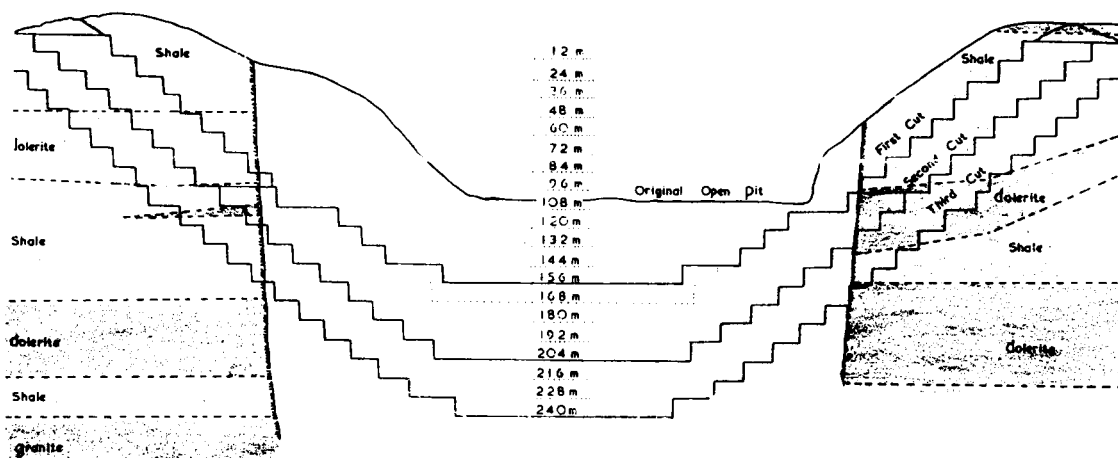


Fig. 25—Section showing pit in Koffiefontein Mine

date. The first waste cut was commenced in January 1970 and was completed in February 1972. The removal of 17 000 000 t of waste exposed 12 500 000 t of ore. The second waste cut, which dovetailed into the first cut, commenced in October 1971 and was due to be completed in May 1975. This cut was to remove 18 000 000 t of waste to expose a further 11 000 000 t of ore. The third waste cut, which was started in September 1974, will be completed in December 1977 and will remove 24 000 000 t of waste to expose 9 000 000 t of ore.

Work is currently being carried out to determine the critical slope angle for the Koffiefontein pit. The problem is not the normal slope-stability problem in that, in most stability calculations, the slope is allowed to fail when open-pit operations have ceased. At Koffiefontein, the slope must remain stable while the pipe is being mined underground. Thus, it could be required to stand for up to 30 years after the closure of the opencast operation.

The problem is further complicated by the fact that an unstable shale layer was detected during the sinking of the main rock shaft. According to present planning, this shale layer will be intersected in the near vertical side of the pipe, and slaking tests indicate that the shale will weather and fail. This failure could cause failure of the overlying and underlying granites, which would cause erosion of the rim of the pipe until the natural angle of repose of the country rock

is reached. This would cause massive dilution of the ore in the underground mine and the possible failure of underground drives due to the impact of rimrock falling onto the workings below.

A possible method of dealing with the shale band would be to seal it, but this would pose the problem of access to the band in the near vertical pipe. If current investigations indicate that the waste benches could be carried at a steeper angle than the present 45°, it would be possible to adjust the mining pattern below the 76 m level to allow the shale band to be carried in the deeper waste benches. This would also allow access to the adjacent granites, which might require anchoring. The determinant in this problem will be the economics of deepening the pit versus the cost of rim failure.

Future Developments in Open-pit Design

To date, all the calculations for pit design and pit bottom have been done manually. The results achieved for any set of parameters are adequate, but, owing to the complexity of the calculations, only two or three of the many variables are taken into account, and the exercise tends to be updated at infrequent intervals.

To overcome some of these problems, a computerized open-pit design system is to be introduced. The system will be used to re-evaluate the work already done at Finsch and Koffiefontein, and to

update the exercises when significant changes in parameters are encountered.

Other areas of investigation at present being pursued are the development of an automated production reporting system that will include the maintenance of the earth-moving fleets, and the development of a computerized ore-reserve system.

UNDERGROUND MINING AT KOFFIEFONTEIN

Initial exercises indicated that the economic open-pit bottom at Koffiefontein will be the 240 m level, which will be reached in 1981. In mid-1973, initial planning for the change-over from open-pit to underground mining began, one of the first steps being the consideration of various mining methods for the Koffiefontein pipe.

Block caving and sub-level stoping, which are the main methods used in the Group, were considered first. Block caving is a relatively simple mining method and allows a moderate degree of mechanization, and its use at Koffiefontein would mean that any future refinements could be used in the Kimberley Division. Unfortunately, the relatively large area of the Koffiefontein pipe and the high daily tonnage call mean that the amount of pre-production construction would be immense. A total of 74 scraper drifts would be required, and, to utilize this number of winches, the pipe would have to be bisected with a central access tunnel. To improve loco-

motive utilization, the simple chinaman boxes would have to be replaced with ore passes in the kimberlite. A further problem would be that the ventilation requirement of 190 m³/s would need large-section tunnels that would be difficult to keep open under the pressures developed by a cave. Finally, rain falling in the open-pit catchment area, without any overburden to soak up this moisture as in the Kimberley underground mines, would percolate through the broken kimberlite and cause serious ground-handling problems.

The second method considered was the sub-level stoping method used with success at the Premier Mine, which lends itself to a large pipe and also to large production tonnages. However, the method is fairly labour-intensive and would tend to preclude a high degree of mechanization. In addition, the Koffiefontein kimberlite is much softer than at Premier, which would result in excessive wear at the brows of the grizzly levels and necessitate extensive support and maintenance of these brows.

A third mining method known as long-hole chambering has been practised at Wesselton Mine for about five years. The method is a refinement of the extremely labour-intensive chambering method, and has been brought to a greater degree of efficiency by the use of modern trackless-mining equipment.

The Long-hole Chambering Method

Production levels will be developed at 30 m vertical intervals, beginning just below the open-pit bottom. Each level will consist of an access tunnel driven around the rim of the pipe in the country rock (Fig. 26). From this access tunnel, 18 parallel drilling drives will be developed at 18 m centres across the pipe. At right angles to these drives, a slot will be developed across the other axis of the pipe. This slot will break through to the level above and will provide a free breaking face for the blasting of a fan pattern of long holes (Fig. 27). The fan is designed to form a continuous cone along the line of the drive that will tend to funnel the broken ground to the 'low point' to facilitate loading. It has been estimated that each fan blast will liberate approximately 3600 t of ore (25 per cent of one day's production).

The broken ground will be transported by rubber-tyred load-haul-dump units via the drive to the rim access tunnel and will be tipped into one of eight ore passes, which will be vertical raise-bored passes developed from the 52 level via all the production levels to the 28 level. Ore transport on the main haulage level will be by 20 t cars pulled by electric locomotives. The ore will be tipped into primary crushers sited near the main shaft and will gravitate into storage passes

of approximately 3000 t capacity. From these passes, short belts will feed the ore to the measuring flasks (Fig. 28).

Feasibility Test

Before the adoption of the long-hole chambering method to mine the Koffiefontein pipe, a feasibility test was set up in the Koffiefontein open pit to determine

- the viability of developing competent 3,4 m by 3,4 m drives in kimberlite,
- the viability of developing a breaking slot,
- the type and make of units best suited to the operation, and
- the timing of the various sections of the production cycle.

During the test a variety of equipment was used to drive a 3,4 by 3,4 m tunnel for a distance of 50 m under a 17 m lip. Thereafter, a slot was established at the end of the tunnel, and a series of fans was drilled and blasted into the slot from the tunnel, the broken ground being loaded and trammed away via the tunnel.

The long-hole chambering method was found to be acceptable because of the following.

- The construction period required for initial production falls well within the economic life of the open pit.
- A very rapid build-up to full production can be achieved.
- The method is highly mechan-

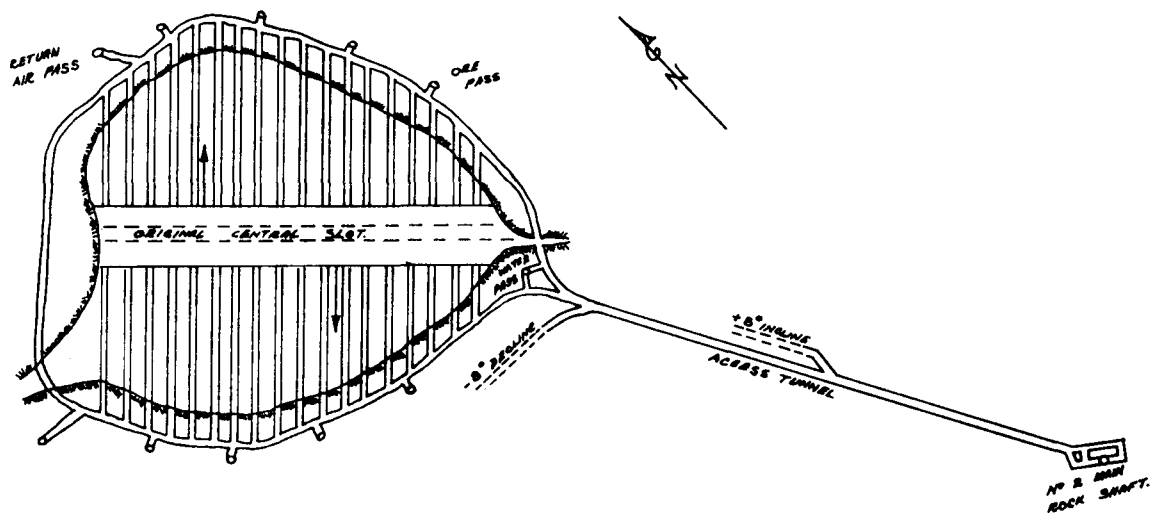


Fig. 26—Plan of a production level—Koffiefontein proposed underground-mining method

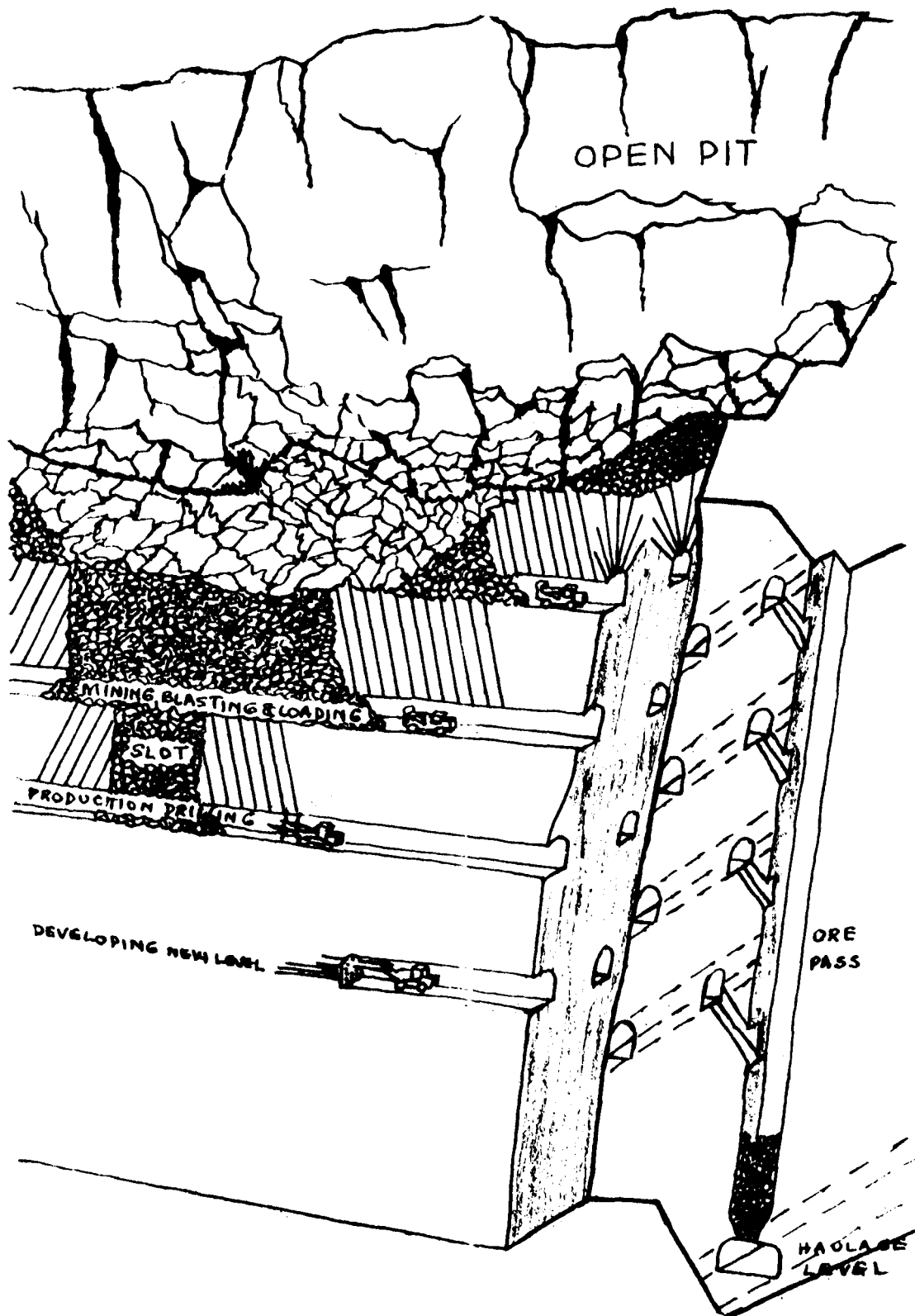


Fig. 27—Diagrammatic view of production drilling and blasting—Koffiefontein proposed underground-mining method

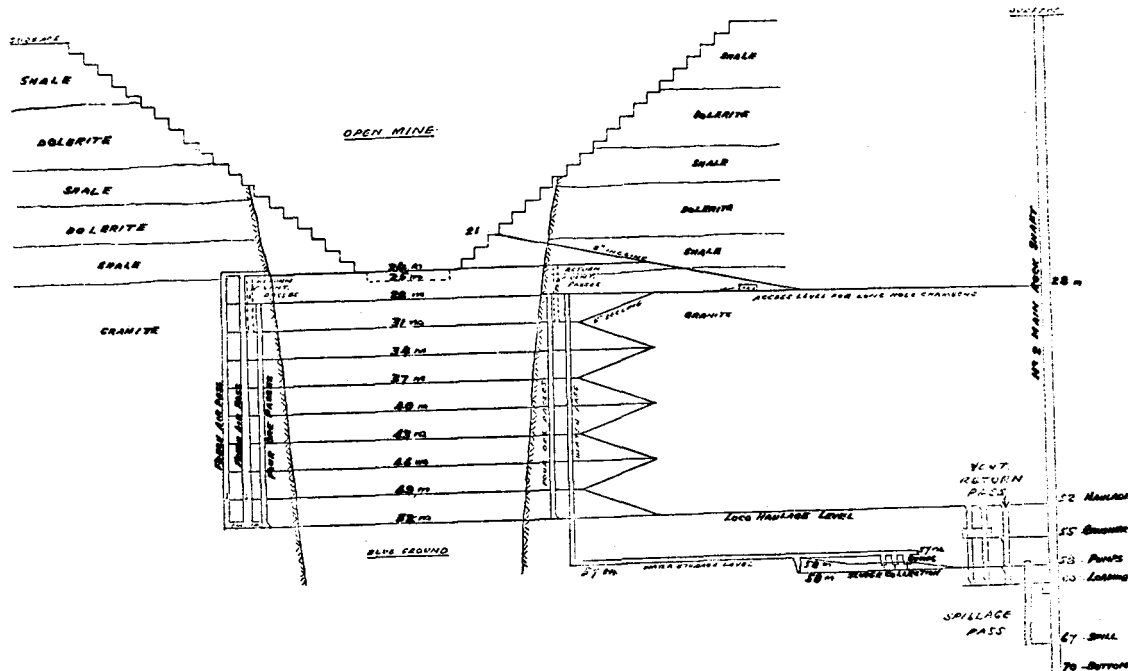


Fig. 28—Section through the mine—Koffiefontein proposed underground-mining method

- (d) The method is flexible. If production experience indicates that the vertical interval between loading levels is too small, the level spacing can be increased without any disruption to the production effort.
- (e) No return ventilation shaft is required, all the return air being exhausted into the open pit.
- (f) The effect of rain can be minimized by leaving the wet broken ground, and drilling and blasting fresh dry ground. (This is a major consideration, and possibly the overriding one.)
- (g) Pressure problems on drives and tunnels should be minimized as the deeper tunnels lie in the country granites and there should never be more than 90 m of ground overlying the drives in kimberlite.

Operation

All production and development loading, hauling, and drilling, except the hauling duty on the main haulage level, will be effected by self-propelled rubber-tyred units. For the initial development phase, four electrically powered rigs equipped with twin hydraulic booms and

hydraulic drifters have been ordered. Hydraulic drifters were chosen in preference to pneumatic machines because of their superior efficiency and to improve environmental conditions at the drilling face. They will also avoid expensive and inefficient compressed-air reticulation systems. The rigs will be able to drill an elliptically shaped tunnel 28,5 m² in area from a single set up. The booms are equipped with a parallel holding device, and the rig has the ability to drill fans and roof-bolt holes. Six diesel-powered load-haul-dump units fitted with 5 yd³ buckets have been ordered to handle the loading and hauling duties during the development phase.

Production fan drilling will be carried out by means of electric drill rigs carrying hydraulic drifters. On the top level it will be necessary to blast up to nine fans per day owing to a low initial recovery of the blasted ground. This requirement will drop to four or five fans per day on underlying levels. To facilitate inter-level movement, all levels including the main haulage level will be connected by a figure-8 ramp, which will also serve as a major intake airway.

Ventilation calculations have been based on the provision of 0,05 m³/s

of air per rated horsepower of the production units. With the provision of a 30 per cent safety margin, the total volume of air required will be 425 m³/s. The intake air will be split into two main streams: approximately 140 m³/s will be brought into the 28 level, which will be the top production level connected to the shaft, and the remaining 285 m³/s will enter the mine via a special airway on the 52 haulage level and will then be carried up two 3,7m diameter raise-bored airways to the 28 production level and subsequently to the underlying levels as they come into production. Each drilling drive on the production levels will be equipped with a plug-in fan so that a quantity of 8,5 m³/s can be exhausted from each operating end. The return air will be exhausted to the open mine via raise-bored ventilation passes and worked out areas.

All the water that falls into the open pit will run into the drives developed on the upper production levels. It is intended to concrete the footwall of these drives, and the water will be led from the drives to the rim tunnel and via a water pass that connects with all levels to the main sumps on the 58 level. The sumps have been designed with

a total capacity of 25 million litres, and the main pumps will be capable of delivering 1 500 000 litres per hour. The system will be able to cope with a 100 mm cloudburst over the open-pit catchment area.

Thus, the method will be highly mechanized, electric power and diesel power being dominant. Compressed air, for a change, will play a very minor role, being used only for main-shaft chute operation and other minor work, for which it will be supplied by small individual compressors.

MANAGEMENT AND LABOUR

Improved methods of mining and higher productivity in general call for higher standards of labour, supervision, and management, and it is considered to be of value to record the steps being taken in this direction.

A management training programme was instituted in 1973, starting with top management, and this has now progressed as far as second-line supervisors (i.e., miners and artisans). Upon completion at this level, the first-line supervisors (i.e., team leaders) will be trained. The value of using the same course material for all levels of management is already apparent in that all levels have clear objectives and understanding of the company policy. Managers now feel free to innovate within the policy parameters, making productivity schemes more readily accepted and applied. For senior staff, a more intensive and advanced management training scheme is now being introduced that specifically deals with objectives, organization, and motivation. This training will also be given to more junior managers. It is also planned to parallel this programme with training in social skills and interpersonal relationships so that more value will be obtained from the group management sessions.

Great benefit is being obtained in Kimberley from the use of local Black labour rather than migratory labour. At present 1150 out of the 2550 Black workers are local men who live in the Kimberley townships, and these men are conveyed by the Company to and from work each day. They take normal leave

each year (approximately three weeks depending on rank) and are therefore a permanent and stable force, allowing the Company to spend far more time and money on their training and development. Supervisors can become acquainted with their workers as people, knowing their names, their family background, and their problems. This is assisted by their knowledge of English or Afrikaans, which in most instances is fluent.

It is intended to extend the concept of local labour at Kimberley, and a start has been made at Koffiefontein, where a township exists. At Finsch, where there is no township, the problem is greater, but consideration is being given to conveying local labour from a township 20 km away, or building a new township on the mine once approval from the Bantu Administration can be obtained.

Localization, of course, creates new problems. Housing in the townships very often is not up to a reasonable standard, and it has been found necessary to start a building programme. De Beers is shortly to build 250 houses, which have 3 bedrooms, a full bathroom, and all facilities. Since they have to be built in the township, private ownership is not at present allowed; they have to be given to the Bantu Administration, but it is possible to reserve occupation for Company employees only. It is envisaged that this building programme will have to be extended and that houses at Koffiefontein will also be required.

Localization emphasizes communication problems in that the accepted hostel machinery falls away. Two main steps have been taken to overcome this problem. Blacks who have graduated in the personnel field have been trained to take senior positions in which they deal with their own people, including training, selection and manning, welfare, and assistant personnel officer positions. The division now employs eight graduates and, although they are essentially a part of management, they provide an extremely effective and useful line of communication with the Black employees.

In addition, works committees are being formed to be followed by a co-ordinating works committee. The first committee has been elected, and the representatives have been trained to undertake their duties. Works committees, as opposed to liaison committees, were chosen because more relaxed discussion will take place when the representatives discuss matters within an all-Black body, and the resolutions that are presented to management are more likely to be their true feelings. It is too early to expect these men to discuss their problems openly while sitting in awe of senior management.

To improve communication 'down the line' to all employees, a system of briefing groups is to be established. Starting at the senior manager, each man briefs the next level below him on matters of importance and interest at meetings held once a month. The intention is that, if employees are given correct and motivating information, 'grape vine' sources will be avoided together with the disputes and problems that arise from misunderstanding.

It is now accepted in industrialized countries that communications up and down the line cannot be achieved satisfactorily by the same system, and that representatives of the union or works committee, either directly or via a consultative committee, provide the essential 'up the line' communication while a separate system such as the briefing group concept should provide the 'down the line' communication.

Other problems encountered during localization were absenteeism and job expectation.

The former was a short-lived although serious problem. Intensive counselling was required to instill a sense of responsibility into those who felt that taking a day off for one of many reasons (drinking being a particular problem) was not serious. It was here that the Black graduates were particularly good at improving the attitudes of the absentees. After 18 months of localization, the avoidable absentee rate is less than 1,0 per cent, a remarkably low figure.

The local men are in general, relatively well educated, and they expect more job satisfaction, not being

happy in menial or purely manual work. Consequently, re-organization of work and mechanization become essential. They are also more aware of differences in rates of pay and the evaluated level of work, and it was necessary to question the pay rate for certain jobs. An added complication is that the Company is in a competitive market and has to compete with the South African Railways and other industry and business in Kimberley. Of course, this is not an unhealthy situation provided the men are allowed to be productive.

The training of workers to do more than one job has significantly improved productivity. In the block caves, for example, a machine operator who drilled hang-ups and large rocks in the drawpoints was also trained to drive the scraper winch. This put him into a higher category

of pay and allowed a 30 per cent reduction in the complement. The introduction of rigs and larger loaders into development with a crew trained to do all the jobs also increased productivity. This multi-training of the workers is proving a great success in all areas of the mines and plants. So far, 440 men out of a trained total of 1000 have received training in more than one job.

To date 70 artisan aides have been trained and introduced into the work situation. Acceptance of the aide is improving, and some aides are doing really productive work, allowing the artisan to concentrate on the more highly skilled jobs. There is no doubt that the engineering field offers great scope for increased productivity, and, with the present shortage of artisans and the large number of new mining and industrial projects in the pipeline,

it seems to be essential that more non-Whites must be trained.

CONCLUSION

The mining industry will undoubtedly present very great challenges in the future. Working costs are increasing at an astronomical pace, and commodity prices, in general, are not rising as fast and in some cases are falling. Better methods, and increased efficiency and productivity are the only answers to this problem, and this in most cases means more mechanization and better utilization of the labour resources available.

In conclusion, thanks are extended to De Beers Consolidated Mines Limited for permission to publish this paper. Thanks are also due to Messrs Vertue, Clements, and Michie of De Beers Consolidated Mines Limited who assisted in its compilation.

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The following members have been admitted to the Institute as Company Affiliates.

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