Trackless Mining

In addition to the four papers already given in this issue, the following four contributions to the colloquium on the above topic were received for publication.

1. TRACKLESS MINING AT OAMITES

by K. E. MANTELL*

Oamites Mine, brought into production in late 1971, is located 50 km south of Windhoek, South West Africa, on the northern boundary of and just outside the Rehoboth Gebied. Oamites was known as a possible mining proposition at the turn of the century when Hanseatische Minen Gesellschaft was adit mining on the adjacent farm Kamzwas.

Rand Mines Limited carried out a small amount of prospecting in the early 1960s, Falconbridge Explorations Limited later taking over the prospect. Approximately 4600 m of diamond drilling was carried out between 1965 and 1967.

Feasibility studies indicated a viable proposition and, with a relatively low capitalization of R5 million and the participation of The Industrial Development Corporation of South Africa, Limited, the mine was brought into production with a target of 45 000 tonnes of ore per month at a head grade of 1,33 per cent copper and 23 g/t silver.

Sub-level stoping was chosen as the mining method because the orebody was ideally suited to this method and production cost was of prime importance. After careful study, a trackless mining method was developed for the following reasons:

(1) to break away from the traditional labour-intensive operations of Southern Africa in consideration of the inevitable rise in wages and possible labour shortage;
(2) the ramp system of opening up the mine was considered to be cheaper than conventional shaft sinking;
(3) the mine could be brought into production early since sub-level development could be carried out simultaneously with the sinking of the decline; and
(4) development lashing could be greatly speeded up.

The orebody outcrops on the surface and is located in a highly variable sequence of metamorphosed sedimentary rocks. The dip is consistent at 83 to 90 degrees, and, although the width varies between 3,5 and 23 m, it is fairly consistent at 14 to 16 m over the proven strike length of approximately 500 m. The orebody has been proven to a depth of 360 m. Exploration is continuing eastwards and westwards, and also down dip.

This tabular, steeply dipping orebody lends itself ideally to longitudinal sub-level stoping methods. The ore itself consists of conglomerates and quartzites, the principal minerals being bornite and chalcopyrite, with lesser quantities of pyrite and galena.

Labour

The total labour force at Oamites is 480, consisting of 50 Europeans, 80 Basters, and 350 Ovambos. Actual mining employs about half the labour force, the remainder being employed in the Concentrator, Engineering, Technical, and Administration departments. Local policy is to train as many of the local Baster population as possible up to supervisory positions; to date, six Basters have been awarded special Blasting Certificates.

Development and Mine Layout

The main hoisting decline was driven at an inclination of 14 degrees below horizontal on the hangingwall side of the orebody for a horizontal distance of 850 m at 5,5 m by 2,7 m, using ST5B Scooptrams for lashing. The decline was driven at this dimension in order to accommodate a 36 in conveyor-belt system and afford traffic facilities for Scooptrams, Landrovers, and other vehicles. (See Photo A.)

Between 8 level and 18 level (80 and 180 m below surface), crosscuts were driven from the shaft at vertical intervals of 25 m to intersect the orebody. From the crosscuts, sub-drives were driven eastwards and westwards in ore to the limits of the present proven strike length. At these limits, the sub-drives were connected by means of a ramping system to afford access when retreating from stoping faces.

Along the proven strike length, the orebody is divided into stoping panels of 50 m in length with 15 m rib pillars between every two panels. The rib pillars will be recovered at a later stage (Fig. 1).

The present extraction level is 18 level (Photo B), and a haulage

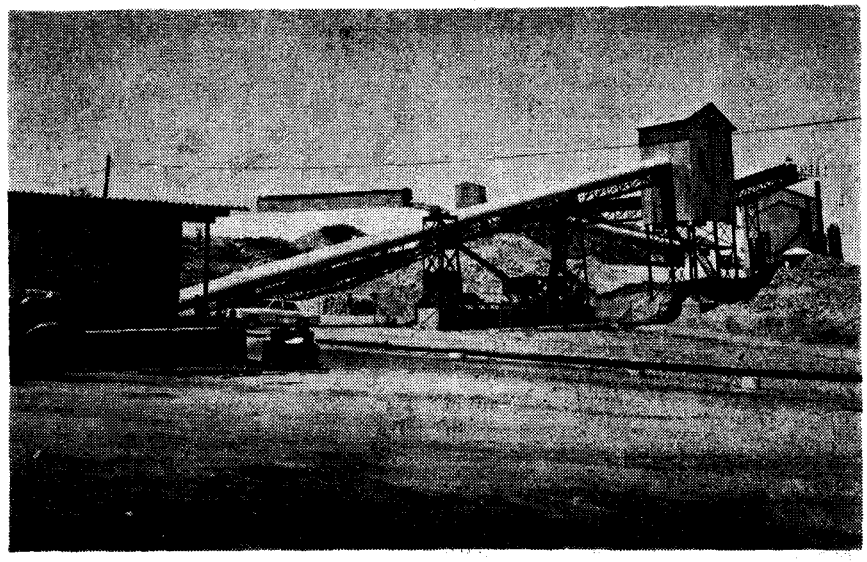


Photo A—Oamites Mine showing conveying system at the top of the Main Decline

*Manager, Oamites Mining Company (Pty) Limited

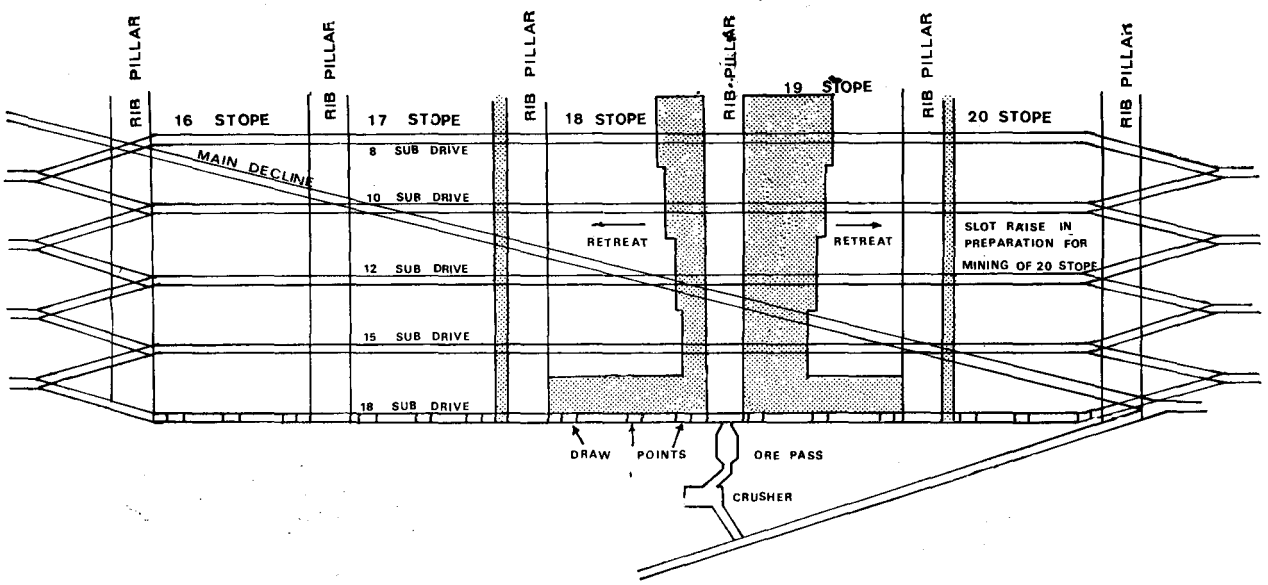


FIG. 1

TRANSVERSE SECTION SHOWING METHOD OF MINING

was driven in the hangingwall parallel to the 18 sub-drives and connected by drawpoint crosscuts excavated to accommodate Wagner ST5B Scooptrams (Fig. 2).

The primary jaw crusher is situated on 20 level and is connected to the tramming level by means of a finger raise orepass system.

The crusher discharges via a raise onto a 36 in conveyor running from 22 level to the surface.

Stoping

A typical stope is mined as follows.

From the sub-drives, and adjacent to the rib pillars, slot crosscuts are driven northwards and southwards to the hangingwall and footwall limits of the orebody. A slot raise is driven from 18 level to 10 m above 8 level to connect the crosscuts on the hangingwall side.

Longhole drilling then commences, utilizing 2¼ in diameter up and down holes. The length of up and down holes has been varied but it appears that the optimum is 15 m up and 10 m down. The slot is cut completely from 18 level to above 8 level by means of vertical parallel drilling breaking into the slot raise.

When the slot is complete, 18 level is completely undercut by means of conventional ring drilling. On this level, a half-cone is formed, leaving as far as practicable the lower-grade ore on the footwall side (Fig. 3).

When blasting of 18 level is complete, the stope is mined out by ring blasting (Fig. 1). Rings are drilled with a strike burden of 1,7 m and a maximum toe spacing of 2,4 m. Gardner Denver CH123 and CH99 drifters are in use with 1 m by 1 in hex. extension equipment and 2¼ in tungsten carbide cross-bits. Blasting has been carried out conventionally to date with 45 by 560 mm gelignite, initiated by Cordtex and Milli Second Electric detonators. However, Anfex is slowly being introduced into stoping and will be extended to development shortly.

Tramming with Trackless Equipment

Scooptrams are used for all tramming and development lashing. On 18 level extraction haulage, the average tramming distance is 200 m. There is a fleet of six Scooptrams and, of these, three are used on a three-shift basis to tram 45 000 tonnes of ore per month. However, with the recent changeover to concrete roadways, it is possible that a two-shift system may be introduced. Another Scooptram is fully employed on development lashing, one is continuously on major overhaul, and one is on standby.

Routine maintenance and repairs are carried out in an underground workshop on the tramming level, Scooptrams coming to the surface

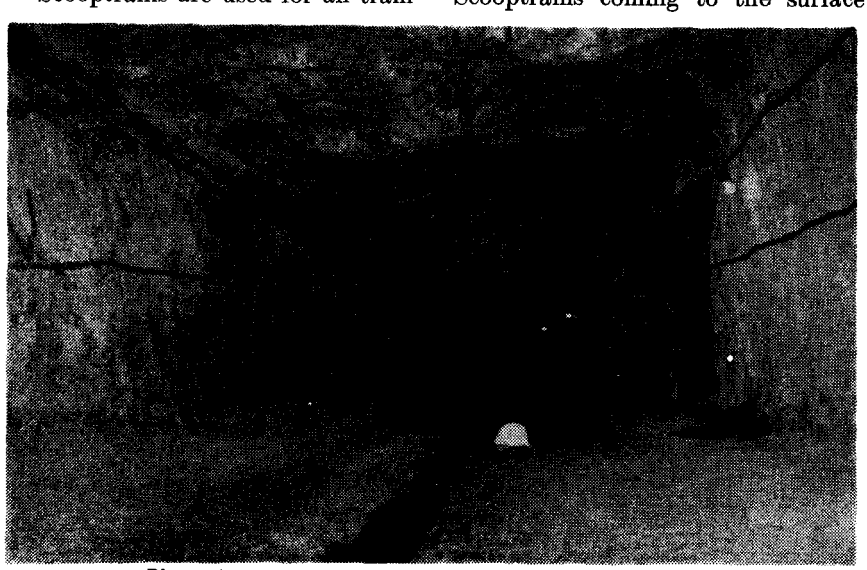


Photo B—Oamites Mine, concreted haulage at 18 level

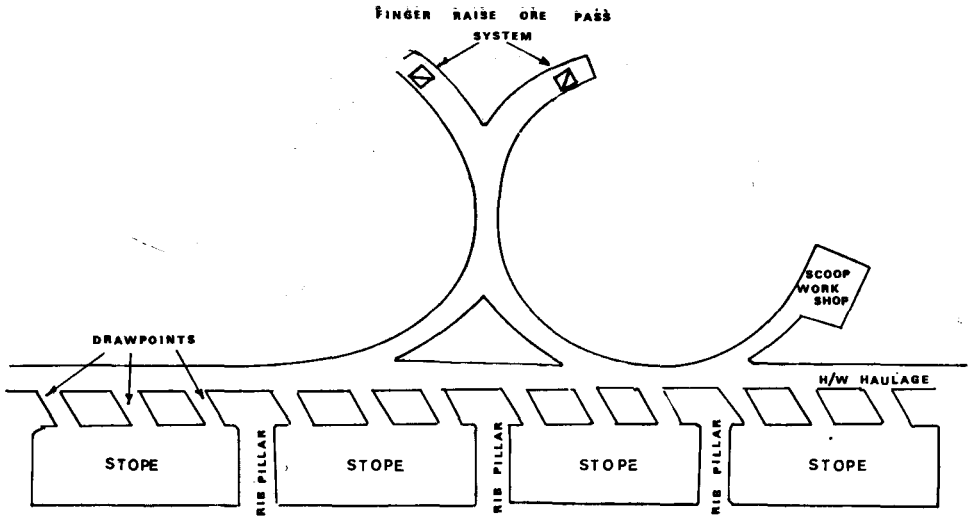


Fig. 2

only for major overhauls.

The Scooptrams used at Oamites are the Wagner Model ST-5B type of 5 yd³ capacity. A Scooptram is a dual-purpose machine designed to load in the manner of a front-end loader and to then transport or tram the load over a distance. The machine thus performs the two functions of a conventional loader and truck. The Wagner Scooptram is powered by a Deutz air-cooled engine noted for its relatively clean exhaust characteristics. The chassis is of the articulated type mounted on four pneumatic-tyred wheels. Drive is applied front and rear to all four wheels through Clark torque converter and full power-shift transmission. The design and geometry of the bucket actuation and control provides for a powerful crowding action into the load pile in conjunction with the drive from the wheels, together with full tilt into the horizontal position to facilitate trucking or tramming over a distance with a full bucket. The machines are purpose-designed for underground mining operations, and are exceptionally low and narrow in profile so that they can operate in conditions of restricted headroom and space.

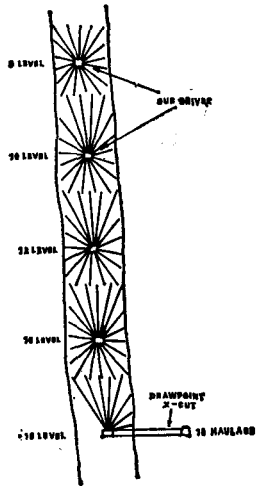


FIG 3.

TRANSVERSE SECTION THROUGH A TYPICAL STOPE.

Fig. 3

These machines are of very rugged construction and have performed satisfactorily. The two initial machines are still working and have both completed 25 000 hours. Two other Scooptrams have completed 15 000 hours. Some of the modifications carried out to the machines to enhance safety and improve operations are listed below.

(1) Initially the machines were equipped with water scrubbers. These were found to be troublesome, and all machines now employ either oxy-catalyst scrubbers or fume diluters.

(2) Transfer of batteries to old water-box area. (Batteries otherwise are directly under the oil-filler cap, and oil is spilt onto them.)

(3) All filters were removed from under the machine for easier access.

(4) Dry air cleaners were fitted in place of the oil type (far superior under operating conditions).

(5) The front cutting edges of the bucket were replaced with the type that will take bolt-on cutting edges (4 edges cost R280 as against R800, and can be replaced in two hours as against five days for the welding of the old type of cutting edge).

(6) The machines are fitted with dual-service brakes, an emergency brake system, and a parking brake.

(a) The service brake comprises an air over hydraulic system (Bendix-Westinghouse) applied front and rear (dual) to all four wheels.

(b) The emergency brake system provides for the automatic application of the service brakes and the parking brake should a failure in the air system occur. Such a failure results in a pressure drop in the auxiliary air receiver controlling the brakes, which are then immediately applied. Tests are being carried out on additional emergency systems to provide for the automatic application of the service brakes in the event of engine failure or overspeed.

(c) The parking brake is a disc type that is applied to the main driveline connecting the front and rear axles.

(7) The later-model machines are fitted with a non-spin differential on the front axle. This

system ensures that front and rear drive is maintained in a wheel spin as long as one wheel in the front is turning, in contrast with the conventional differential where, if two wheels front or rear spin, the drive to the other two is lost.

One of the primary considerations in trackless mining is the condition of the road surfaces. If the surface is uneven, tramming is slow and the maintenance cost is high. This should be borne in mind when developing, where lifters should be drilled to give an even grade. At Oamites, long development ends are surfaced with 1 in and 2 in gravel or crushed stone, this being maintained to within 15 m of the face and graded regularly. Gravel is not satisfactory for ore tramming from drawpoints, because of the continuous grading required, and in this case concrete is used. Tests are being carried out on the optimum mix of concrete and the method of placing. The concrete roads have improved Scooptyre life from an average of 900 hours to an average of 1600 hours, with a best-ever of 1835 hours. This life has been achieved with standard deeptread XT8 Goodyear tyres. The mine is in the process of changing the bucket end tyres to Dunlop Super Deep Smooth, the so-called 'semi-slick' or 5:1 ratio tyre. The design of this type of tyre is a compromise effort to achieve the obvious advantages of extra life from the greater proportion of rubber that is evident with the treadless or 'slick' type of tyre, and at the same time to retain some tread or lugs for the purposes of efficient traction and braking. A life of 2500 hours is expected. Tyres are filled with 80 per cent water and 20 per cent air by volume. This increases tyre life through improved traction due to the weight of the water. Another less important advantage of using water is that punctures soon become visibly apparent.

To accommodate Scooptrams, haulages are driven at 4 m by 3 m with arched backs, and sub-drives at 4 by 2,5 m with a rectangular section.

All ramps are driven downgrade since driving upgrade presents difficulties in lashing with Scooptrams. The weight of the vehicle has to be overcome, and bucket penetration into the muck pile is poor. Traction is also poor, causing excessive wheelspin and tyre wear. Ramps are driven at 1 in 4, and are used only for access and materials transport. Lately, the inclination has been reduced to 1 in 5.

All Scooptram drivers are Ovambos who undergo a two-month training course before qualifying. Standby drivers are being continually trained and travel on the Scooptrams during production. Turnover of Scoop drivers is very low because of high incentive pay and high job status, which gives them pride in their jobs.

The maintenance staff consist of a well-qualified Scooptram Foreman, with two European and three Baster mechanics. Preventive maintenance schedules are shown in Schedules I to III. Staff are encouraged to take part in practical courses to keep abreast of new Scooptram developments.

SCHEDULE I

DAILY MAINTENANCE OF WAGNER SCOOPTRAM 2-520

Frequency:	Daily
Target Time:	1 hour — *Carried out by:* Scoop Fitter
Oil:	Check engine oil level. Add if necessary
	Check engine oil pressure
	Check converter and governor oil levels. Add if necessary
	Check transmission oil pressure
Hydraulic:	Check hydraulic oil level. Add if necessary
	Check for hydraulic leaks. Repair if any
Brakes:	Check footbrake, handbrake, and emergency brake
	Check brake fluid in master cylinder
Tyres:	Check tyres for wear and damage
	Check tyre pressure
Aircleaner:	Clean aircleaner oil bath or blow out aircleaner element
Batteries:	Check battery water
	Check lights
	Check ammeter for charging
Air Receivers:	Drain oil from air receivers
	Check for air leaks. Repair if any
	Check alternator and compressor V-belt tension
	Check exhaust system for leaks. Repair if any
	Check bucket boom for possible cracks
Grease:	Flange bearing
	Steering cylinder pins
	Dump cylinder pins
	Boom to bogie pins
	Boom to bucket pins
	Hoist cylinder pins
	Stabilizer-arm bucket pins
	Hinge pins
	Seat suspension
	Oscillating axle

Signed:..... Date:.....
Scoop Fitter

Signed:.....
Foreman

SCHEDULE II
WEEKLY MAINTENANCE OF WAGNER SCOOPTRAM 2-520

Frequency: Weekly
Target Time: 10 hours *Carried out by:* Scoop Fitter

Clean scoop
Check all items on Daily Maintenance Chart
Change engine oil
Change engine filter (1)
Check oil level in injection pump and governor
Clean oil fine filterbody and change cartridge
Clean pre-cleaner on fuel filter
Change fuel filters (2)
Change airfilter in highpressure line
Check air pipes for leaks. Repair if any
Clean out transmission oil coolers (2)
Change transmission oil filters (2)
Check transmission pipes for scuffing. Replace if necessary
Change hydraulic oil filter
Check hydraulic hoses for scuffing. Replace if necessary
Check differential and planetary gears oil level
Check engine mounting bolts for tightness
Clean cooling fins
Remove engine covers, clean out cylinder fins
Grease universal joints
Tighten universal joint bolts
Check brake adjustment
Check bucket lip for wear

Signed:..... Date:.....
Scoop Fitter

Signed:.....
Foreman

SCHEDULE III
SIX-WEEKLY MAINTENANCE OF WAGNER SCOOPTRAM 2-520

Frequency: 6 Weekly
Target Time: 18 hours *Carried out by:* Scoop Fitter

Check all items on Weekly Maintenance Chart
Check valve clearance
After valve setting take compression test
Clean wire gauze in breather pipe
Clean radial-fin oil filter
Check injectors
Remove and clean starter
Remove and clean alternator
Clean brass filters
Clean magnetic filter in hydraulic tank
Change bucket wear caps

Signed:..... Date:.....
Scoop Fitter

Signed:.....
Foreman

At Oamites, great emphasis is placed on operation and maintenance of Scooptrams in an effort to reach optimum Scooptram performance. In addition to a preventive maintenance system, there is a good backup service from the agents in Windhoek, Messrs Hubert Davies, who, apart from normal after-sales service, carry out 15 000 hour overhauls in their Windhoek workshops.

The average tramming distance on 18-level haulage is 200 m. The cost of tramming is 35 cents per tonne of ore, broken down as follows:

	Cents per tonne
Fuel and oil	7
Tyres	6
Labour (including maintenance)	11
Spares	11
Total cost	35

Crushing and Hoisting

The primary crusher is located on 20 level, approximately 20 m below the tramming level. Ore is crushed to minus 6 in and discharges via a vibratory feeder onto the conveyor system on 22-level horizon. The 36 in conveyor belt transports the ore to surface in three stages, having two intermediate transfer stations. A weightometer at the shaft bottom records tonnage and is used to control the feed rate onto the conveyor belt.

Conclusions

Once there is a territorial nucleus of trained operational and maintenance personnel for heavy L.H.D. equipment, it is suggested that a trackless decline system of opening up the majority of shallow and medium-depth prospects would be an optimum solution, giving good capital return and reducing infrastructure costs.

Trackless mining provides a very efficient mining method for certain types of orebody, and reduces labour costs and labour dependability.

2. TRACKLESS DRAW-POINT LOADING AT NCHANGA CONSOLIDATED COPPER MINES LIMITED, ROKANA DIVISION

by J. P. RAISIN*

Introduction

Rokana Division is the largest underground producer of the three Copperbelt mining operations within the N.C.C.M. Group. The mine township of Rokana is part of Kitwe, which is the second largest city in Zambia. Ore production from underground sources at Rokana averages about 440 000 tonnes per month at a mill head grade of about 1,70 per cent copper.

The three underground mines at Rokana are situated on the eastern limb of the Nkana Syncline. South Orebody Shaft and Central Shaft extract ore from the southern section of the orebody, which extends northwards from the outcrop of the syncline over a strike length of approximately 19 000 ft. The orebody continues north but is poorly mineralized for about 4000 ft. Mindola Mine (northern section of the orebody) extends northwards from this 'Barren Gap' for a strike length of about 14 000 ft (Fig. 1).

In general, the orebody ranges from 10 to 40 ft in thickness, and dips westwards at about 50 degrees. Large folding systems in the southern section of the orebody often produce extremely thick synclines and anticlines. The northern section is relatively tabular and ranges from about 20 ft true width at a dip of 40 degrees in the far north, to a thickness of 30 to 35 ft at 70 to 80 degrees in the south.

Stoping Method

The principal method of mining used at Rokana is sub-level stoping. Main haulages are mined at vertical intervals of 260 ft, and sub-level drilling drives are mined close to the hangingwall at vertical intervals of approximately 50 ft. Cut-out crosscuts, which delineate each stope block, are driven to the orebody footwall, and a cut-out raise (slot raise) intersects each of these crosscuts over the full dip length of the stope. Each stope has a strike length of 30 ft, and its rib pillar has the same dimension. Extraction and sizing of ore takes place in two grizzly crosscuts, which are mined from a drive situated approximately 30 ft true width from geological footwall and 70 ft above the footwall extraction haulage. Where poor ground conditions preclude the mining of orebody drill drives, sub-level development is carried approximately 30 ft true width above assay hangingwall.

Application of Trackless Draw-point Loading

Trackless draw-point loading as a means of production at Rokana has taken three basic forms.

(1) Draw-point loading from an intermediate level, in conjunction with conventional grizzlies on the lower extraction level. This splitback method aims at complementing grizzly production in order to improve recovery and is in operation at Mindola Shaft.

(2) Draw-point loading in fold areas. Severe folding in certain areas at Central Shaft and South Orebody Shaft leaves a relatively thick but flat dipping syncline between two conventional 'gravity' limbs. Immediate footwall ground conditions in the syncline area are extremely poor, and are unsuitable for grizzly, box-raise, and collection-haulage development. For this reason, diesel load-haul-dump machines are used to transport ore to the conventional limb raises and haulages that have been developed in the more competent, uncontorted footwall.

(3) Extraction of wide orebodies. Orebody thicknesses in excess of 100 ft are being encountered in the steeply dipping lower sections at South Orebody Shaft, and, in view of poor ground conditions in the orebody and hangingwall, it is apparent that a conventional sub-level stoping system as practised at Rokana has severe limitations.

Although this new system has not yet been implemented, it is intended to extract the orebody by means of a footwall stope and hangingwall stope, leaving an intermediate strike pillar that will eventually be wrecked into both these stopes. Extraction from the footwall stope will be through the conventional grizzlies, and loaders will be used to extract the hangingwall stope, carrying ore back to footwall haulage boxes. As opposed to the split-back system, which also utilizes both loader crosscuts and grizzlies, both extraction points in this method are on the same elevation.

Since the method is only now emerging from the design stage, it is not intended to describe it in this discussion, and the following description will thus be restricted to the draw-point loading system that is currently in operation at Mindola Shaft.

The Introduction of Split-back Stoping at Mindola Shaft

Mindola is the largest of the underground mines at Rokana, average production over recent years being about 270 000 tonnes per month at 1,9 per cent copper and 0,14 per cent cobalt. Production at present is from twelve stope faces, but approximately 60 per cent of the total output comes from the six faces in the sub-vertical shaft system, each of these producing between 25 000 and 30 000 tonnes per month.

Ground conditions vary considerably on strike, but, basically, faces on the southern retreat are stoped from orebody development, while on the northern retreat most sub-development is in the waste hangingwall. It was in these waste drives between the 3140 and 3920 ft levels that trackless-loader lashing of sub-level development was introduced in 1968/69.

Access to the sub-levels is made from a spiral connecting the upper and lower haulages. The first spiral was mined in the hangingwall waste, which was considered to be the most competent ground, but all subsequent spirals have been mined in the footwall since this has the added advantage of maintaining a permanent access between levels. Much experience has been gained at Mindola in the mining of inclined accesses. The first spiral was mined with long inclined straights and

*Underground Manager, Rokana Division

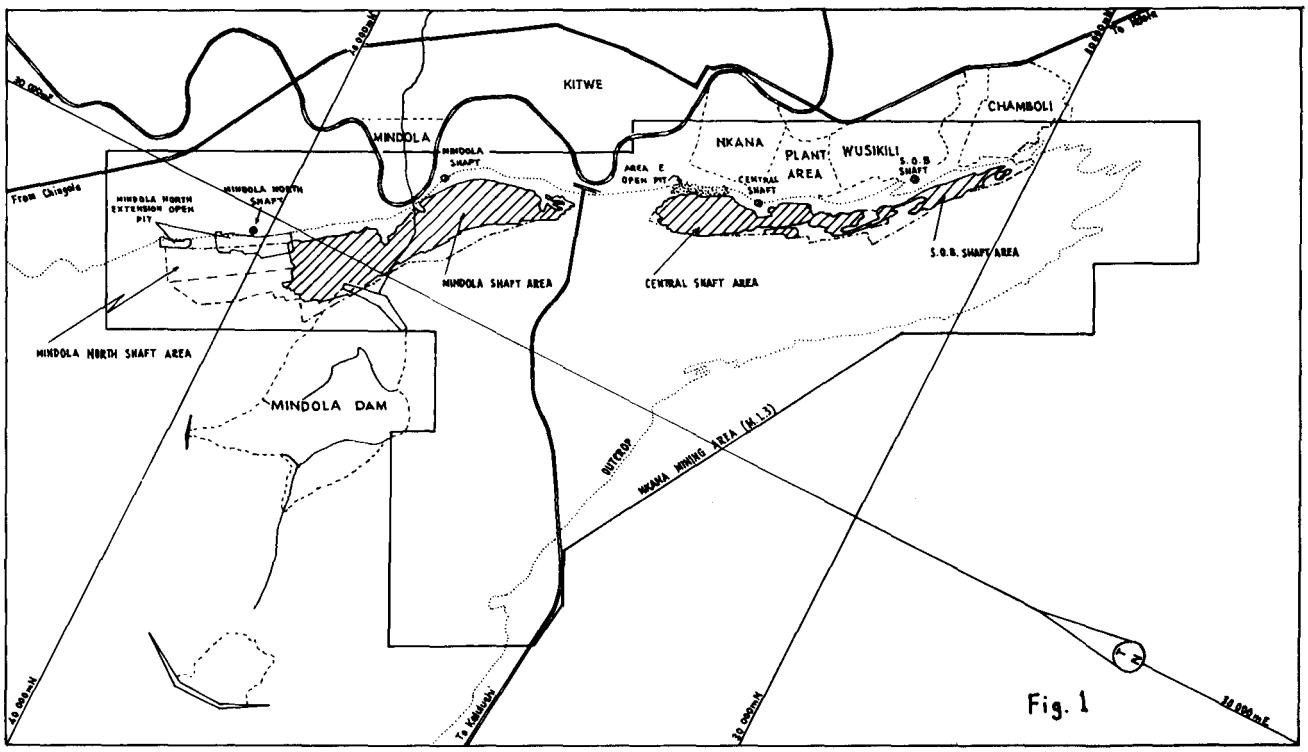


Fig. 1—The mines of the Rokana Division

flat curves in order to offset any survey and mining difficulties and to facilitate the easier breaking away of intermediate levels. Subsequent spirals of a similar design but with inclined curves were mined without difficulty, and this led to the present corkscrew spiral layout (Fig. 2). An interesting feature here is that the overall mining footage is a spiral, and its associated development has been reduced from about 3400 ft in the initial spiral to about 2500 ft in the present layouts.

The average ore tonnage contained in each stope and its associated pillars is about 50 000 tonnes.

Recovery* from the high-production sub-vertical area has been about 65 to 70 per cent, the main

*Extracted tonnes of copper / Block tonnes of copper × 100%

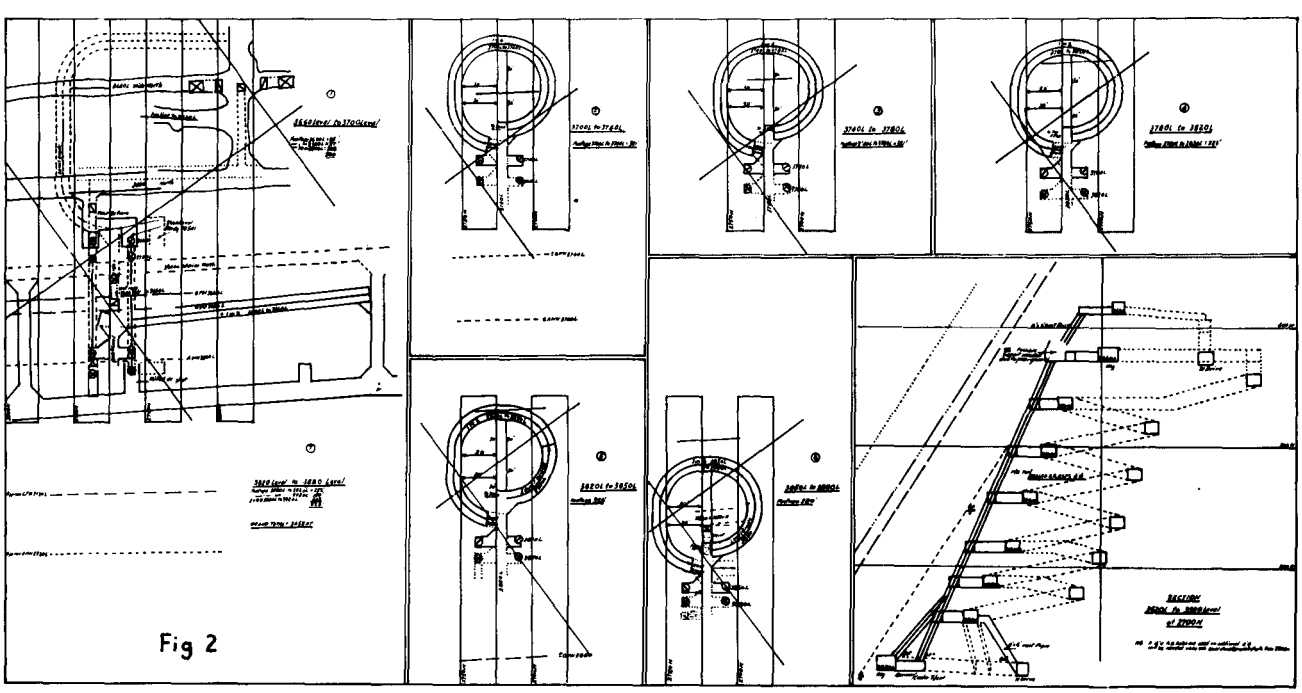


Fig. 2—Inclined accesses at Mindola Mine

reasons for dilution being as follows:

(a) failure of rib pillars,
(b) failure of hangingwall,
(c) loss of rib-pillar remnants, and
(d) over-riding of waste to the grizzly level soon after the blasting of the crown pillar (this over-riding of waste increases as dips decrease from the vertical).

In an effort to improve recoveries, it was decided in 1969 to introduce a form of split-back stoping using trackless equipment loading from footwall draw-points that were mined from an intermediate level. This would allow recovery of ore from the footwall of the stope, which had previously been 'lost' as soon as crown-pillar waste over-rode to the grizzly level (Fig. 3). In addition to this, it was hoped that the effects of hangingwall and rib-pillar deterioration would be minimized by the speeding-up of extraction of each block.

Because the development faces in the sub-vertical sections at Mindola are usually carried between eighteen months and two years ahead of the production face, all attempts so far at the introduction of draw-point loading production have had the advantage of being compatible with, but not affecting, conventional sub-level stoping methods.

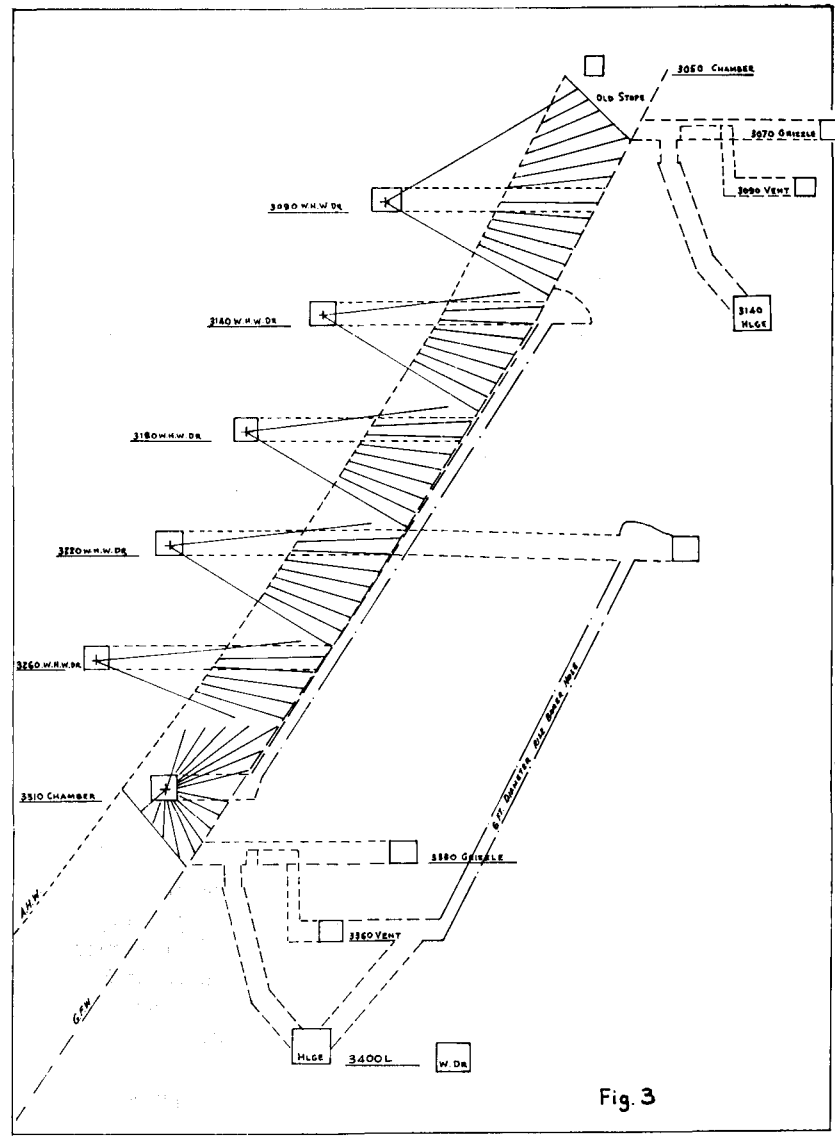


Fig. 3—Typical split-back section showing loader drive, Mindola Mine

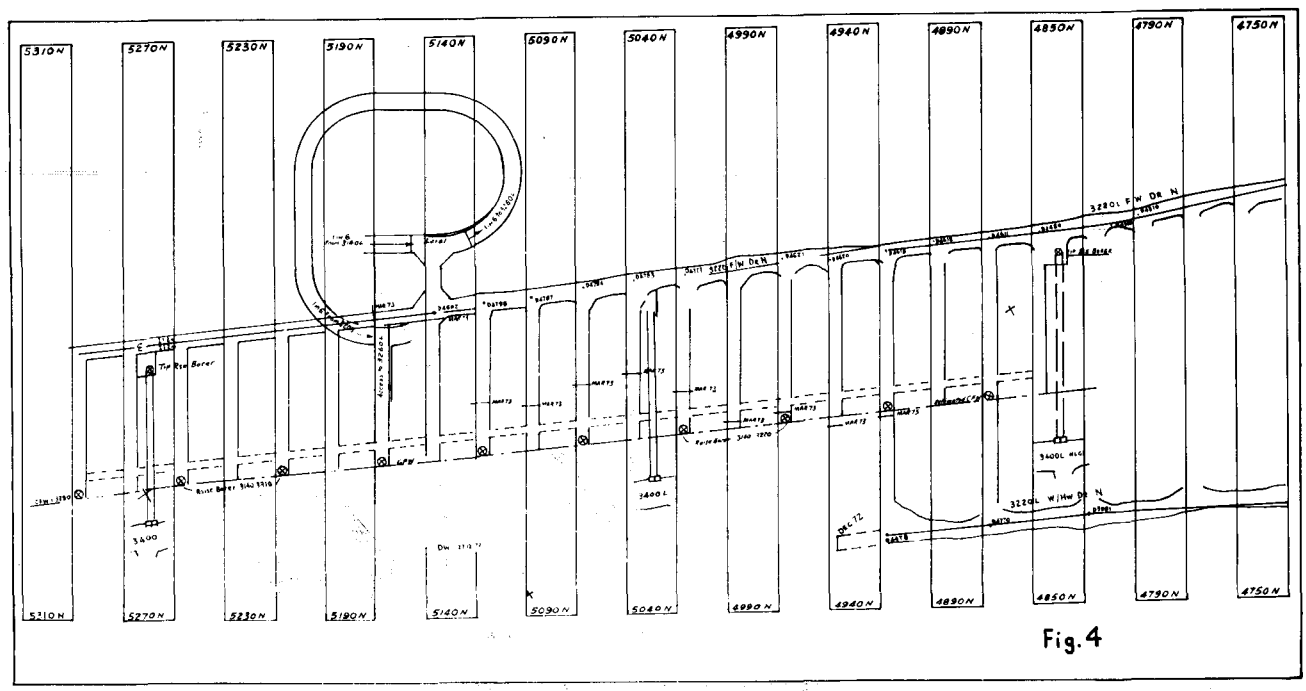


Fig. 4—A loader drive connecting the access crosscuts, Mindola Mine

Initial attempts at split-back production had only limited success. The reasons for this were as follows:

(1) Poor ground conditions. The footwall loader drive was located too close to the orebody, and, although conditions were good at first, rapid deterioration of the hangingwall side of the drive occurred about 200 ft ahead of the stope face, which necessitated heavy support of the drive and loader crosscuts.

(2) Poor ventilation. Many of the draw-point crosscuts holed the existing cut-out crosscuts, and use was made of this to connect the draw-point ventilation system with the conventional section ventilation. This had adverse effects on both systems, and the experiment was eventually discontinued. However, experience gained here was used in the development of the present method.

The Present Application of Split-back Stoping at Mindola

Access crosscuts are driven from the intermediate draw-point level into the footwall at 250 ft intervals on strike. At a position approximately 80 ft from footwall (this position being determined by selection of the most suitable mining ground from visual inspection of the crosscuts), a loader drive connecting the access crosscuts is mined in the footwall argillite (Fig. 4). Raises with a diameter of 6 ft are bored from the loader drive to box positions in the lower haulage at 250 ft intervals on strike. The raises are also connected into the lower water drive (ventilation return drive). Three raises are in use at any one time as intake, tip, and return. Draw-point crosscuts are mined at 30 ft strike interval from the loader drive to the orebody footwall, and a ventilation drive is mined about 20 ft from footwall connecting each draw-point crosscut.

During the initial experiment, a chamber drive was mined in the orebody above the draw-point elevation to define an intermediate crown pillar and to facilitate possible independent extraction of the upper and lower sections of the split-back stope. This was unsuccessful, and was discontinued for the following reasons.

(1) The draw-point connecting the loader crosscuts to the intermediate chamber drive sloughed badly along strike, causing the collapse of the chamber drive into draw-points.

(2) Extraction of the lower 'mini' stope through two conventional grizzlies was much quicker than the draw-point extraction of the upper stope. This initially created a remnant area in the upper section, and the excessive pressures that were evident in the intermediate crown pillar resulted in closure of holes and difficult drilling conditions.

In view of this, no development is carried into the orebody from the present loader crosscuts, and thus they are non-productive until such time as conventional stope blasting passes above their elevation. This has not been a problem from the production point of view, since at Mindola two adjacent stopes are always on draw and a state is always reached where production is continuous from successive draw-points.

To date, extraction from the draw-points has been of the order of 12 000 to 15 000 tonnes per month, and the recoveries from stopes have increased by between 10 and 15 per cent.

Production from the draw-points is by means of 2 yd^3 loaders (Wagner ST2B), the norm being two operating loaders and one back-up loader per production area. Wagner ST5A loaders (5 yd^3 capacity) were tried at one stage, but the size of excavation required for these loaders, particularly at junctions of crosscuts, caused many support problems. In the initial experiment, what appeared to be reasonably strong argillite deteriorated rapidly along cross-joint planes, and eventually, intensive steel support was required in the loader drive and at the crosscut junctions.

Moving the drive further into the footwall and reducing its size largely eliminated these problems.

Within the limits of its design, the present method of split-back stoping appears to be working reasonably well, but the full economic appraisal, which offsets the benefits of increased recovery against the cost of the entire intermediate-level operation, can be undertaken only when several stopes have been extracted by this method. Only then can a firm decision be made on the viability of the method.

Distribution, Maintenance, and Operation of Loaders

Wagner ST5A loaders were initially introduced at Mindola as large-capacity machines to be used in a cut-and-fill experiment. A comparison with other makes of loaders indicated that the Wagner was simpler to operate and maintain, and was more suited to working conditions at Mindola.

The subsequent failure of the cut-and-fill experiment left the ST5A's available for draw-point production, but, after tests, it was decided that the size of excavation required for successful operation of these loaders was too large. A limited use has been found for these machines in some of the footwall drawpoints at South Orebody Shaft, but it is probable that they will be replaced with 2B loaders as they become uneconomic to operate. Machines of 2 yd^3 capacity were later introduced at Mindola for sub-development lashing, and, as with the 5 yd^3 loaders, it was felt that the Wagners were mechanically simpler and more robust than the other types of loaders tested, and had a profile that was better suited to Mindola conditions.

In an attempt to reduce development sizes, a 1 yd^3 loader (Wagner H.S.T.1) was purchased, but many problems were experienced with the transmission and hydraulic systems, and the tramming speed of the loader was limited by its high centre of gravity. The need for a 1 yd^3 loader still exists, but it is felt at Rokana that a suitable low-profile machine of this capacity has not yet been developed for use underground.

The present diesel loader fleet at Rokana is distributed as follows:

	Wagner 2B	*2A*	*5A*	*Service vehicles*
Mindola Shaft	19	3	1	8
Central Shaft	14	0	0	2
South Orebody Shaft	8	0	2	2

The term 'service vehicle' in this table is a misnomer, since the vehicles referred to are Benford transporters. These are used mainly for transporting explosives and small mining equipment, and do not specifically enter into any function concerned with maintenance of the diesel-loader fleet.

The trackless transport of rock or personnel over long distances is not practised at Rokana, since all mechanized sections have been mined from long-established haulages that are furnished with standard rail-track equipment. Thus, once a loader is delivered to a section, it operates within the geographical limits of that section until it is required to be brought to surface for major overhaul or replacement.

Every area where loaders operate at Rokana has its own workshop, and all maintenance on loaders is carried out underground as follows:

- Daily service (duration about 2 hours)
- 100-hour service (duration about 3 to 4 hours)
- 200-hour service (duration about 4 to 6 hours)
- 600-hour service (duration about 8 to 10 hours)
- 1200-hour service (usually undertaken over a week-end)
- 2400-hour service (usually undertaken over a week-end).

After 6000 hours, all units are sent to surface for overhaul, and it is anticipated that units will be written off after 15 000 hours.

No overhauls of components are carried out underground. All defective or suspect components are changed and sent to the Central Mechanization Workshop, where there are comprehensive facilities for the overhaul of all loader parts except electrical components, which are repaired under contract.

The total labour force employed to maintain the loader fleet averages 2,9 men per machine (excluding service vehicles). This includes all categories from Cleaner to the Shaft Mechanization Engineer. Skilled expatriate Heavy-vehicle Artisans are generally recruited from Britain and account for approximately 20 per cent of the labour force.

Suitably qualified Zambians are selected after induction for training as Mechanics. After a three-month period in the Mechanic Training School (which includes both theoretical and bench work), the trainees return to operating sections for practical experience. Further training depends entirely on an 'on the job' assessment from their Artisan Foreman. It is possible under this scheme, by a series of alternate practical and theoretical training periods, for a Zambian to rise to Leading Mechanic level; the minimum period is 186 weeks. If they have attained suitable academic qualifications, Leading Mechanics can proceed to Artisan level and thence to Foreman level through the normal channels of progression.

For higher-educated Zambians, a more-advanced training scheme exists through the Government-sponsored Trades Training Institutes. This is an accelerated apprenticeship course, and its objective is to produce a qualified artisan in the minimum period of 3½ years. This includes a diagnostic period of 6 months (to determine the candidate's suitable trade), followed by 2 years of concentrated practical and theoretical trade training, and a minimum of 1 year's 'on the job' experience. These men can also progress to Foreman level.

Loader drivers are trained in the Training School at South Orebody Shaft. The course covers all types of loaders in use at Rokana and lasts for two months. It is now a prerequisite that employees starting the course have a blasting licence.

At Central and South Orebody Shafts, loaders are widely distributed, with the result that some loaders in isolated areas cannot be used to their full capacity and provision of adequate back-up loaders is not always possible. This situation will obviously improve as development in these sections expands and production is started. The loader sections at Mindola are localized and have the added advantage of being interconnected by footwall spirals. Because of this, close supervision is possible, and effective distribution can provide each loader with a regular, even work load. The back-up requirement is also considerably reduced in this case.

Ventilation Aspects

Zambian Mining Regulation 913 states:

> The Manager of any mine in which any self propelled diesel unit runs, shall by systematic sampling ensure that no such unit runs underground
> (a) if the exhaust gases of the engine are found to contain more than 0.2 % by volume of carbon monoxide or 0.1% by volume of oxides of nitrogen.
> (b) if the engine has any defect which may cause danger to persons.

For general purposes, Mining Regulation 902 (2) describes the maximum allowable concentrations of noxious gases in the general body of the air as follows:

Gas	*Parts per Million*
Carbon Dioxide	7500
Carbon Monoxide	100
Nitrous Fumes	10
Sulphur Dioxide	20
Hydrogen Sulphide	20

At Rokana, samples are taken monthly in all places where loaders operate, and these concentrations have not been exceeded in more than 1 per cent of all the samples taken. For this reason, more attention is paid to combating the problems associated with heat in underground diesel-loader operation at Rokana.

Rokana has an unusually high geothermic gradient (about 1°F per 103 ft of depth), and virgin-rock temperatures (VRT) on the 3920 ft level at Mindola have been measured at 114°F (about 45°C). Most of the loaders at the mine operate in areas where the VRT is about 110°F (43,3°C).

In addition to this, rock-skin temperatures can rise by a further 6 or 7°F (up to 4°C), even in well-ventilated ends or draw-points where loaders operate. In order to define the upper limit of working conditions, it is now stated at Rokana that work must stop in any area where the wet-bulb temperature exceeds 87,5°F. Because temperatures below this are difficult and expensive to maintain by conventional ventilation and refriger-

ation methods, efforts at Rokana have been directed towards personal, or micro-climate, cooling, which will enable loader drivers to operate effectively in higher wet-bulb temperatures.

Over recent years, the following three aspects of micro-climate cooling have been considered.

(1) Refrigerated Cabs. This method was rejected because of the practical difficulties involved in providing a durable sealed cab that would give the driver 360° vision.

(2) Vortex-tube cooled jackets. The air supply for this type of cooling was taken from the compressor of the loader. However, it was found that the compressor was too small to serve a dual purpose, and that a larger oil-free unit would be required. The expense involved in this conversion proved prohibitive.

(3) Frozen jackets. Research undertaken by the South African Chamber of Mines Human Sciences Laboratory indicates that this method of micro-climate cooling might prove to be the most effective and cheapest, and a refrigerated room has been installed on the 3660 ft level at Mindola. The jackets are made with small pockets that are designed to spread about 10 lb of ice evenly about the body and are worn underneath an insulating jacket. It is anticipated that the jackets will be effective for about 4 hours. Since testing of this method of cooling is taking place only now, results will not be available for some time, but the success of the experiment could have far-reaching implications, since loader drivers at Mindola will be required to operate in increasingly adverse temperature conditions.

The Future Draw-point Loading at Rokana

The benefits of the application of draw-point loading at Mindola Shaft have yet to be fully assessed. There must, however, be some doubts about the potential success of draw-point loading in any relatively narrow, steeply dipping area from which a high level of production is expected. This point of view is further emphasized by the fact that the system as introduced at Mindola must operate within the constraints of an already developed and operating method of production.

The transition from hand lashing of sub-level development to loader lashing in the lower sections at Mindola has already been made, and the increase in size of development required to bring about this change has permitted the introduction of larger, independent rotation drifters for stope drilling. To date, mechanized sub-level development rigs at Rokana have been tested but have not yet been established successfully. This will undoubtedly be the next step in the mechanization process. It is therefore possible that the future utilization of trackless loaders at Mindola will be restricted to development lashing in fully mechanized sections.

The application of draw-point loading in the lower fold at Central Shaft can certainly be regarded as reasonably successful, and, whilst not necessarily being cheaper than conventional methods, development of the system could lead to production from several of the fold blocks that have previously been left because application of conventional layouts has been considered to be impracticable or uneconomic.

Similarly, at South Orebody Shaft it is thought that good recoveries from the thick lower orebody sections will be difficult to achieve without some form of draw-point loading.

Acknowledgements

The author wishes to thank R. G. Crisp (Divisional Engineer, Rokana Division, N.C.C.M. Ltd) and A. R. Bell (Ventilation Engineer, Rokana Division, N.C.C.M. Ltd) for assistance in the preparation of this paper, and the Managing Director, N.C.C.M. Ltd, for permitting the presentation of the paper.

3. TRIALS TO ASSESS THE POTENTIAL OF TRACKLESS MINING IN THE GOLD MINES OF THE UNION CORPORATION GROUP OF COMPANIES

by G. MAUDE*

Introduction

It has long been realized that the labour intensiveness of present gold-mining methods causes the industry to be vulnerable both to rapidly increasing unit costs as wages increase, and to fluctuation in production if foreign labour becomes unwilling to continue to work in this country. It is essential that mechanized methods be sought that will decrease the demand for unskilled labour, particularly in the more arduous occupations, and thus enable cost inflation to be contained.

Gold mining poses particularly difficult mechanization problems, which stem from the hard rock, thin seams, and variable payability of the deposits. Considering only rock-handling, small tonnages of rock per unit area are broken daily over very large areas and the mechanization of loading and transporting the broken ore is a very different problem from that encountered in the working of massive open-cast deposits.

The appearance on the market of compact 1 yd^3 rubber-tyred load-haul-dump machines aroused interest in that these appeared more closely suited to the needs of gold mines than the larger machines previously available. In October 1970, one was purchased and put on trial at Winkelhaak Mines, Limited, and, in April 1972, the results of the trial were closely assessed, leading to the conclusions expressed here.

Objectives of the Trial

It was essential to specify closely the objectives of the trial, since the absence of experience with these machines in gold mining meant that the trial conditions would probably be very different from the conditions under which the machine would work if introduced on a large scale. The objectives can be summarized as follows:

*Union Corporation, Limited

(a) to determine whether small load-haul-dump machines offer any advantages over conventional gold-mining methods,
(b) to determine the probable cost per ton of using L.H.D. machines on a large scale,
(c) to determine at what level of Bantu wages the replacement of Bantu by L.H.D. machines would be profitable,
(d) to determine the most suitable stope layout for these machines, and
(e) to gain experience in the maintenance and running problems associated with L.H.D. machines under gold-mining conditions.

Results of the Trial

The trial provided a good basis on which to assess the potential of L.H.D. machines in gold mines. The following are the more important conclusions drawn.

(1) When compared with the Union Corporation strike-track system of mining, the introduction of L.H.D. machines could reduce the underground Bantu complement by about 7 per cent, or by 350 Bantu at a mine having an underground strength of 5000.
(2) The most probable running cost per ton for loading, hauling, and dumping would be 30 cents excluding the cost of the driver.
(3) The replacement of Bantu by L.H.D. machines would be profitable at a Bantu wage and compound cost in excess of R4 per shift.
(4) The design of stope layouts and ramps is important, and the L.H.D. machine would perform most economically in a mine or part of a mine specifically designed for it, rather than in an established area modified to suit it.
(5) From the maintenance and running point of view, it was learnt that one spare machine is required for every three machines working. Convenient and clean workshops must be provided underground, and particular attention must be paid to tyres if economic results are to be obtained.

General

No more than a very brief account of the trial and its results can be given here. Much detail has been amassed and, if any aspect of this contribution arouses particular interest, it could be dealt with more fully by correspondence. Here, only brief items of interest arising from the trial can be mentioned.

Stope layout

The L.H.D. test was run in a typical strike-track stope. The only modifications made were the construction of a dip L.H.D.-travelling way running parallel to the centre gully, bridges laid across the centre gully for tipping, and strike cuttings (carried without tracks) 2 m wide and 1,9 m high at the normal 10 m spacing.

After blasting, about one-third of the ore blasted is scattered in the face and two-thirds lie in the cutting. The third of the blast lying in the face had to be lashed to the cutting, and this adversely affected the efficiency of the method. A small winch, for face scraping this scattering, was tried, but no really adequate solution was found.

Tyres

At the start of the trial, the sharp broken quartzite, on which the front wheels skid during loading, caused pneumatic-tyre life to be well under 100 hours. Much attention was given to this problem, and a life of 500 hours, from a solid wire-reinforced smooth tyre (including one rebuild), can now be expected as a matter of course.

Tyre chains were tried and found to be completely unsuitable.

Drivers

Bantu drivers were used throughout, and after training performed well. Part of the reduction of tyre costs was due to the better treatment of the tyres as the drivers became more proficient.

Evaluation

The results of the trial were evaluated by a Monte Carlo technique, as this was considered to be the only way of using all the experience gained to influence the assessment. Probability curves of each independent cost factor were drawn and discussed with the operators, manufacturers, and tyre suppliers. The figures used in this contribution were derived from the 50 per cent probability estimates of the Monte Carlo curves.

Acknowledgements

The trial was carried out in a normal producing stope at Winkelhaak Mines, Limited, and it was only because of the interest and enthusiasm of the General Manager and his staff that the trial was able to produce a valid assessment of the use of an L.H.D. machine under gold-mining conditions.

Thanks are also due to the Senior Consulting Engineer of Union Corporation, Limited, for permission to make this contribution.

---

4. TRACKLESS MINING AT O'OKIEP COPPER COMPANY LIMITED

by J. B. NANGLE*, B.Sc., F.I.M.M. (Member)

Some 4,5 million tons have been mined by trackless-mining methods by the O'okiep Copper Company Limited since the introduction of load-haul-dump (L.H.D.) equipment in January 1970.

In total, O'okiep is currently milling 255 000 tons per month, of which about 65 per cent is obtained from trackless-mining sections. There are eight producing mines and another five deposits in various stages of preparation. All are underground operations, and the sub-level open stoping method is standard.

The ore reserves of individual mines vary considerably from 0,25 to 8,0 million tons. Orebodies are contained within steeply dipping, mainly dyke-like, bodies of mafic rock emplaced within granitic gneiss. There is considerable variation in depth, from deposits mineable through adits to ore depths of up to 1200 metres below surface.

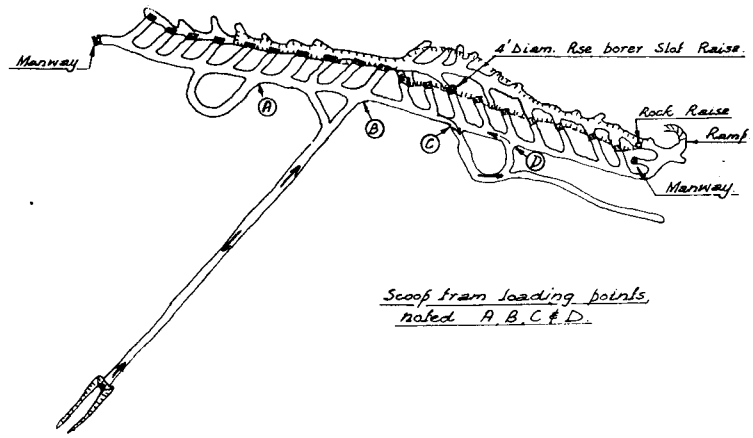
In this short contribution, comments are confined to some particular applications of trackless-mining methods at O'okiep and to a summary of operational features, costs, and statistics for L.H.D. equipment.

Three methods are discussed below.

The Direct Trucking Method

Three adit mines have been de-

*General Superintendent (Mining Operations), O'okiep Copper Company Limited



KOPERBERG CENTRAL MINE
PLAN OF 700 LEVEL ADIT
SCALE 1:2000

FIG. 1.

veloped for direct trucking from the stope drawpoints to central crushing plants. The largest of these, Koperberg Mine, had an original ore reserve of 1,2 million tons and is situated 3,4 km from the Carolusberg Mine complex. In this range of reserve tonnages it has proved more economical to truck uncrushed ore to existing crushing plants, than to install additional crushers at the outside mines.

Exploitation

Koperberg Mine was developed using one Wagner ST-5B and one ST-2B Scooptram. The first Main Adit development round was blasted on 1st March, 1970, and production commenced in December 1970.

Figs. 1 and 2 show the sequence of mine exploitation in plan and vertical projection. Development was completed in the following general sequence.

(1) The Main Adit crosscut was developed to the footwall of the ore zone.
(2) The central undercut drive was developed eastwards in ore, crosscutting to the hangingwall and footwall of the orebody to check the orebody outline.
(3) The footwall extraction haulage and Scooptram drawpoints were developed.
(4) The central slot raise and eastern rock raise were raisebored from the upper to the Main Adit levels, concurrent with Alimak development of the eastern manway raise.
(5) An ST-2B incline ramp was developed to 635 level to hole the central slot raise.
(6) The western end of the orebody was developed by means of adit sub-levels on 525 and 635 elevations.

Stoping of the eastern and then the western ends of the orebody followed.

(a) The 700 level was undercut completely, east of the central slot.
(b) The slot was mined to 580 level, drilling down and up from 635 level.
(c) The stope face was retreated eastwards, undercutting the eastern block at 580 elevation.
(d) The upper part of the eastern block was mined.
(e) The western block was retreated in a single-face operation.

Complete exploitation of the mine required 4482 m of development. The tonnage stoped per metre developed was 268.

Production System

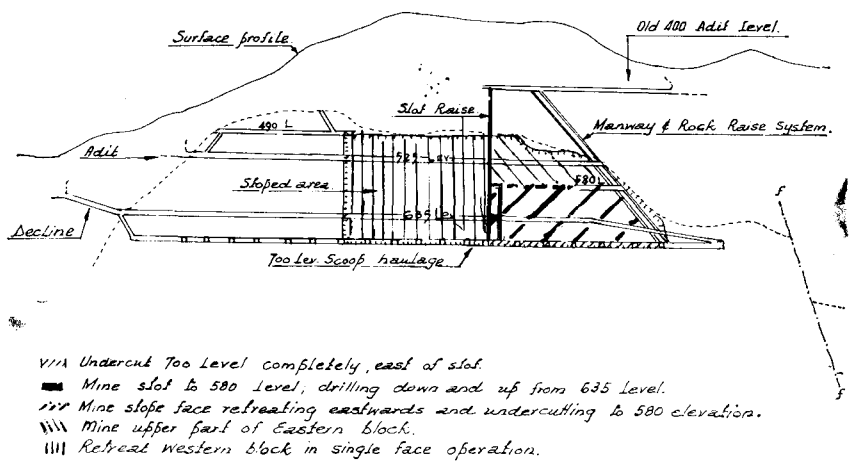
Fig. 1 is a plan of the 700 Main Adit level showing the east and west split of the production haulage in the footwall of the orebody.

The ore is hauled from the drawpoints by a Wagner ST-5B Scooptram and tipped into Bedford 10-ton tractor-trailer units. The main considerations in the choice of the 10-ton Bedford units, rather than larger trucks, were as follows.

(1) Under the terms of labour agreements, any truck with a payload of more than 10 tons must be driven by a White person. The Bedford units have Coloured drivers.
(2) The local dealer service is excellent.
(3) The unit is widely used throughout the Transport Department.

The truck turning loops developed on the south side of the haulage give short Scooptram hauls from the drawpoints to the truck-loading points, the average haul length being about 30 m. The truck routes underground are indicated on Fig. 1.

The movement of the trucks into and within the mine is controlled by a series of robot lights operated manually by the truck drivers. A series of daylight to darkness lights



KOPERBERG CENTRAL MINE
VERTICAL PROJECTION
SCALE 1:2000

FIG. 2.

is installed inside the adit portal to accustom the drivers to the change in light intensity.

Various types of roadbed material for the Main Adit and production haulages have been tried. The best results have been obtained with minus 5 cm crushed aggregate. The roadbeds are maintained with a road grader adapted for underground use.

The length of the return haul from the underground loading points to the central tip at Carolusberg Mine (Photo A) is 6,8 km.

A hydraulic pump unit is mounted at the ore tip, obviating the need for truck-mounted hydraulics.

The average monthly production rate is 30 000 tons, working two eight-hour shifts. Five trucks per shift are used, one truck being available as a standby. The production crew per shift consists of:

1 Miner	
1 Scooptram operator	White
5 Truck drivers	Coloured
2 Tip attendants	Bantu
1 Labourer	Bantu

There is one spare Scooptram operator/truck driver on the day-shift. In an average month of 26 working days, at a production rate of 30 000 tons, an efficiency of 55 tons per manshift is obtained.

The tractor/trailer combination used is shown in Photo B. The tractor is a Bedford KHA 70 of 100 kW and tows a 10-ton articulated trailer manufactured to O'okiep design. The tractor has a wheelbase of 2,45 m (8 ft), and the trailer is 4,34 m (14 ft 3 in) long. The width of the unit is 2,18 m (7 ft 2 in), and the height 2,51 m (8 ft 3 in). The height of the trailer is 1,9 m (6 ft 3 in).

The low profile of the trailer obviates the necessity for large excavations in the hangingwall for loading and also minimizes the possibility of fouling of the trailer body by the Scooptram bucket. The dump cylinder assembly of L.H.D. equipment has been found to be vulnerable to such jarring.

A modification has been made to the original layout at Jan Coetzee South West Mine, where a trucking loop completely separate from the Scooptram haulage has been developed. Difference in elevations and right-angle loading make for a more efficient operation (Photo C). It also allows the use of standard Bedford 10-ton rear-dump trucks, which have higher-profile bodies than the O'okiep trailers.

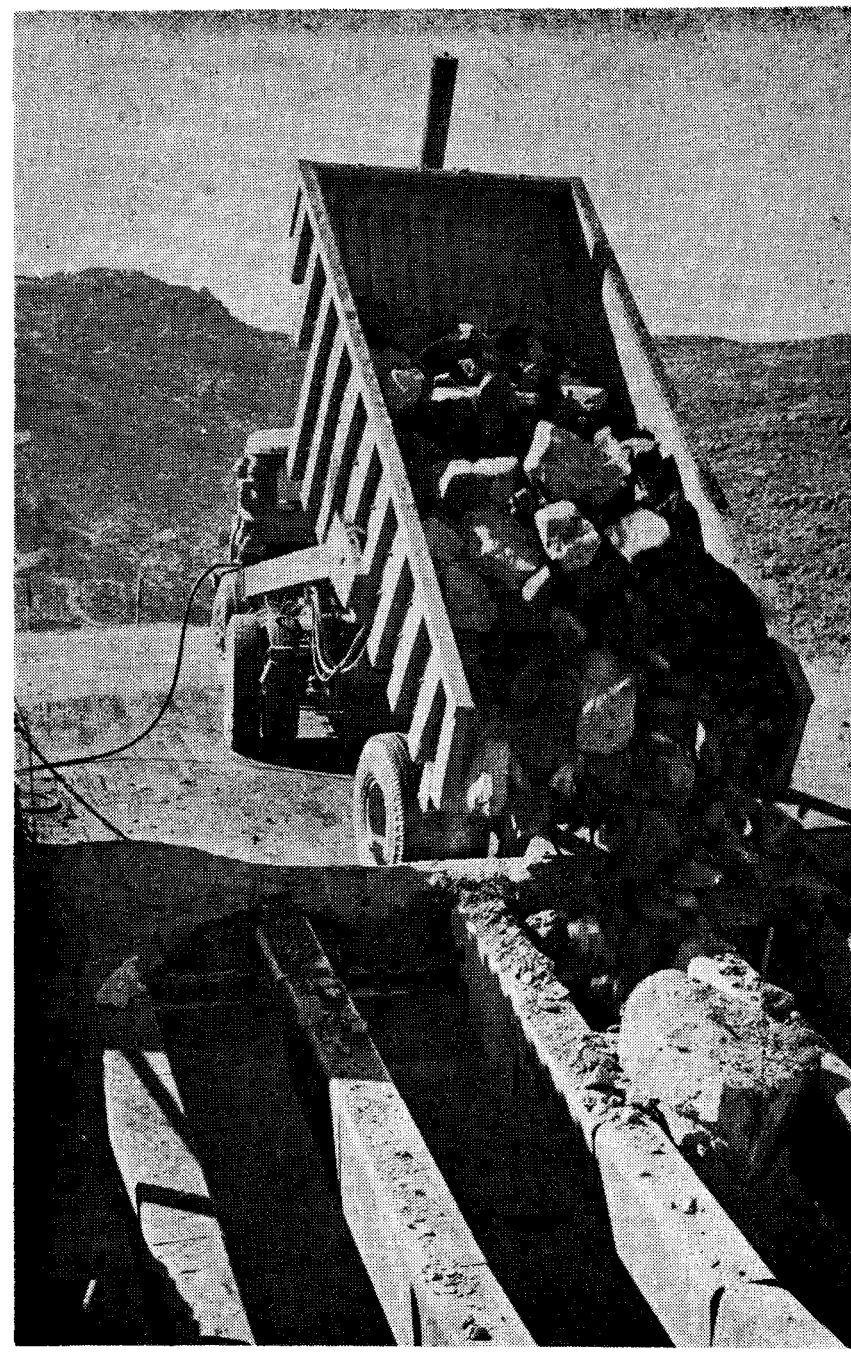


Photo A—Central ore tip at Carolusberg Mine

The Conveyor Decline Method

Two alternatives existed for the mining of a 273 000-ton orebody situated below the bottom main level of East O'okiep Sub-Vertical Shaft.

Deepening of the shaft while production from the upper levels continued was discarded in favour of developing a Conveyor and Access Decline Shaft from the existing crusher level, coupled with the installation of an underground crusher under the orebody. Crushed rock is conveyed to the existing shaft-loading system.

Exploitation

A decline 4,9 m wide and 2,7 m high was developed using a Wagner ST-5B Scooptram, at a dip of 1 in 4 from the existing crusher level of

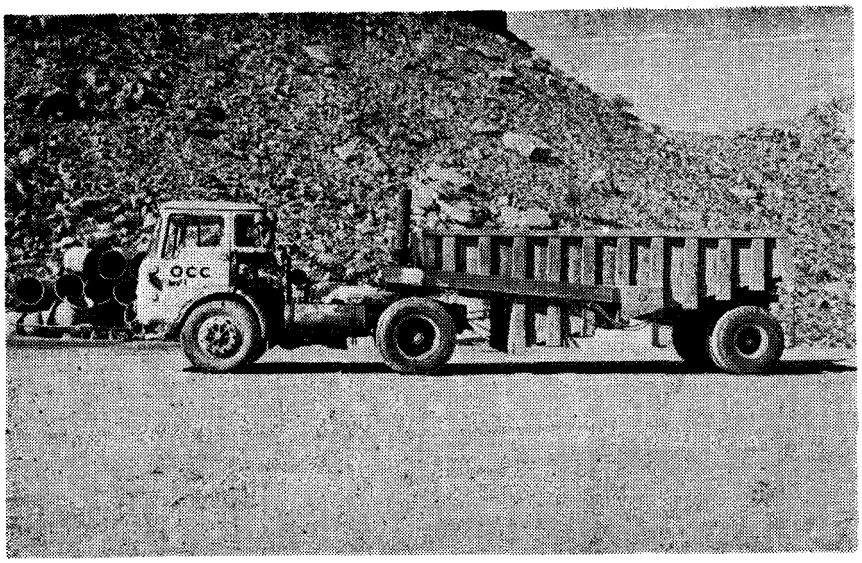


Photo B—Bedford KHA 70 truck

the mine for a total dip length of 300 m. The arrangement of the development is shown in Fig. 3.

The extraction haulage, all drawpoints, the Crusher Chamber, and Pump Station were developed from the Decline by means of an ST-5B. The sub-levels were developed conventionally from a twin Alimak rock and manway raise system. A total of 2374 m of development was required for the project.

During the development stage, while the crusher and the conveyor were being installed, the development rock was hauled up the decline by the Scooptram. When the crusher and conveyor installation was completed, all rock was handled through the crusher.

Production System

The ST-5B Scooptram on production, on an average haul of 40 m, hauls the ore from the stope drawpoints into a small surge bin above the 91 cm-by-61 cm Hadfield crusher, which has a capacity of 150 tons per hour. The crushed ore is fed direct onto the decline conveyor by means of a vibratory feeder. The 0,76 m-wide conveyor has a length of 270 m and a capacity of 250 tons per hour. A short transfer conveyor at the head of the decline conveyor delivers the ore into the sub-vertical shaft crushed-ore bin.

The production target rate is 1000 tons per day on a three-shift basis. The production crew per shift consists of:

1 Miner	White
1 Scooptram operator	Coloured
1 Crusher operator	Coloured
3 Labourers	Bantu
2 Conveyor attendants	Bantu

In a month of 26 working days, at the target production rate, 42 tons are produced per manshift.

The planned installation of closed-circuit television on the crusher/conveyor system will enable the Coloured crusher operator to look after the whole system, thereby eliminating the need for conveyor attendants.

Ore Spiral Method

The Rietberg Mine of O'okiep has an ore reserve of 4,0 million tons at a grade of 1,44 per cent copper. The general position and shape of the orebody make the mine eminently suited to exploitation by trackless methods.

The orebody is contained in the Rietberg, which rises to a height of about 300 m above the general level of the surface. This configuration permits exploitation by means of horizontal and declined adits, and complete trackless mechanization of development and production.

Ground conditions in both ore and country rock are good, and stope width is about 35 m. Strike length of mineable blocks within the separate ore lenses is approximately 130 m.

Access to a considerable part of the lower section of the orebody was gained by means of a main ramp developed in footwall waste, from which mechanized development of sub-levels was possible, and from which access could be gained during the whole of the stoping operation. However, such conditions were not applicable to the upper part of the orebody. Ramps developed in the footwall would have required large footages of waste development for relatively small tonnages of ore. Further, Rietberg Mine is situated in the Steinkopf Coloured Reserve, and Coloured persons make up the complete hourly-paid work force of the Mining Department. Such labour is, in total, considerably more expensive than the traditional White/Bantu combination, and harbours a strong resistance to the physical

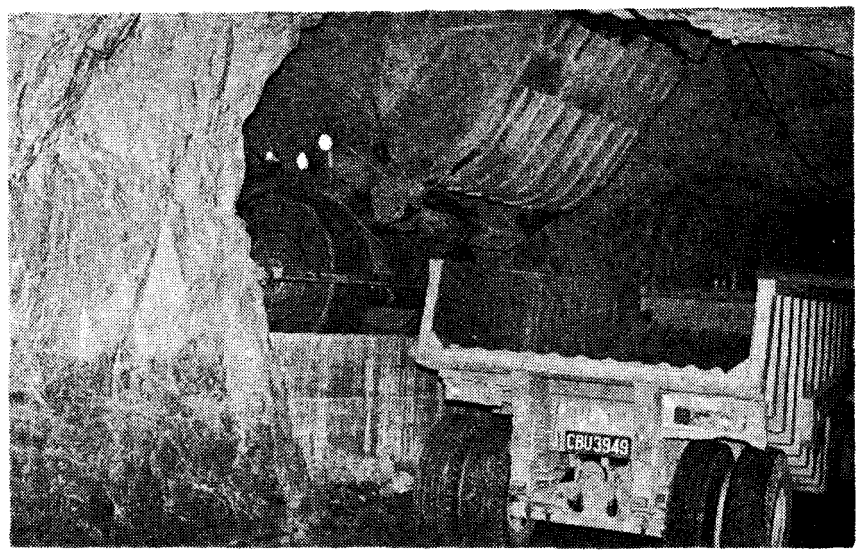


Photo C—Loading at Jan Coetzee South West Mine

effort involved in traditional sub-level development methods.

The Ore Spiral Method resulted from these considerations, and is an attempt to mechanize development while at the same time avoiding the prohibitive cost of waste ramp development in areas of small ore reserve tonnages. It is illustrated in Fig. 4.

Development was by means of an HST-1 and an ST-2B Scooptram, with sub-levels at 15 m intervals. The sub-level development end spirals downwards on the ore contacts, holing alternately on each sub-level elevation, to a 1,22m-diameter raise-bored slot raise, or a 1,52 m-diameter raise-bored manway.

Drifting in the direction of the slot is done at an upgrade of 1,5 per cent. Drifting away from the slot, towards the manway raise, is at a downgrade of 25 per cent (14°), while slot crosscutting is on level grade. Sub-levels declined away from the slot obviate the possibility of water entering the stope during drilling operations and of persons inadvertently slipping towards the open stope.

The method has application in sub-level stopes wider than about 15 m, where sub-level drives on each wall are required for efficient long-hole drilling. In narrower stopes, where one drift on either contact suffices, use of the method results in unnecessary duplication of development.

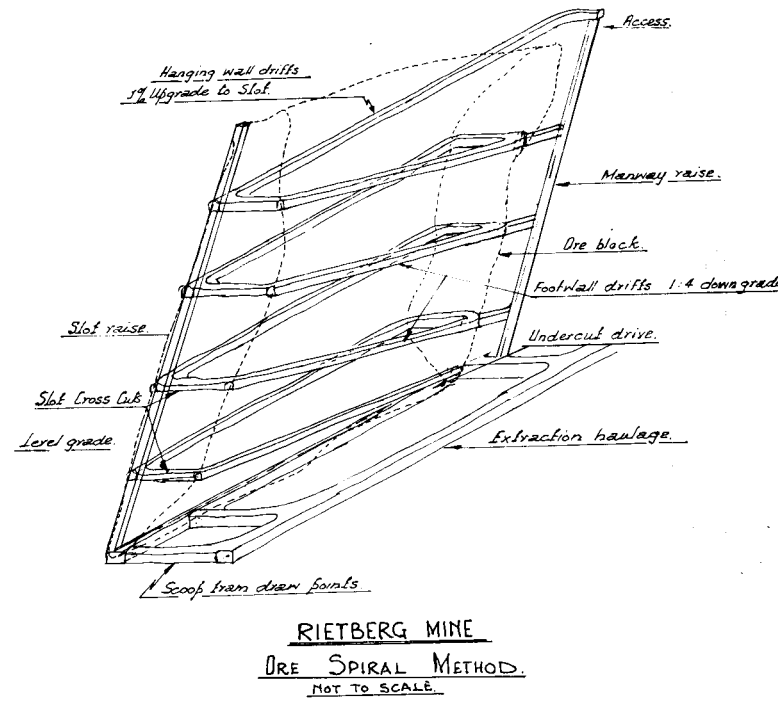


Fig. 4

The 25 per cent inclination of the ramp drives would, it is thought, make the use of mechanized long-hole rigs difficult. Therefore, all long-hole drilling in the two blocks that have used the ore spiral method has been done with Gardner Denver CF-99 drifters on bar H-rigs. Rigging problems were expected, but crews soon adapted themselves to working on ramps.

The following disadvantages must be weighed against the benefit of reduced waste development resulting from this method.

(1) As soon as slotting commences, access down the spiral is cut off. Stope service must then be through the manway raise.
(2) Use of the method in narrow orebodies involves duplication of development.
(3) Ramps of 25 per cent grade are more difficult to clean after a stope blast than level stope drives.
(4) There is no possibility of multi-end development, sub-level development being confined to a single face throughout development of the stope block.

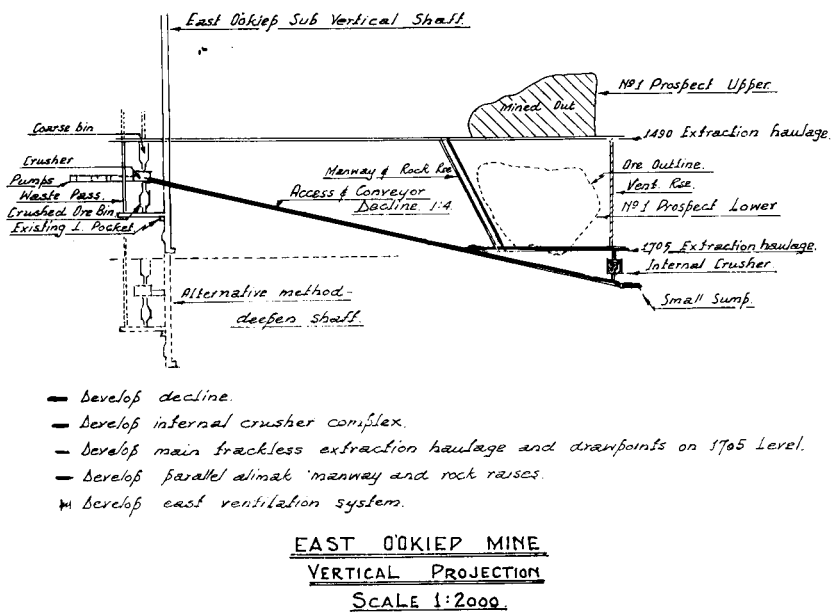


Fig. 3

Cost Analysis

The costs given below are the total direct costs for trucks and Scooptrams and are made up as follows:

Operating Costs

1. Truck-driver labour
2. Fuel, lubricating oils, greases, and tyres

Maintenance Costs

1. Labour for maintenance and major overhauls

2. Supplies

The percentage monthly utilization in a 25-day month is defined as:

$$\frac{\text{Actual engine-running hours}}{600} \times 100$$

ST-5B Scooptrams Costs, Koperberg Mine

Cost per hour:

1973 to date	R7
1972	R9
From inception	R9

Cost per ton:

1973 to date	16 cents
1972	16 cents
From inception	15 cents

The Koperberg ST-5B cost per hour and cost per ton are very close to the Company average costs for ST-5B operation on production and development.

Bedford KHA 70 Trucking Costs, Koperberg Mine

	Cost per ton	*Cost per ton km*
1973 to date	23	10
1972	24	11
From inception	22	9

ST-5B Costs, East O'okiep Mine

Cost per hour—1973 to date: R8

Cost per ton—1973 to date: 12c

Utilization of ST-2B and ST-5B

1972	39,6 %
1973 to date	43,8 %

Tyre Life

Tyre life has increased gradually since the start of the mechanization owing to greater care in road-bed construction and maintenance, and better operator training.

Some statistics are as follows:

Scrapped-tyre life, including retreads (hours):

	ST-2B		*ST-5B*	
	1972	1973	1972	1973
Average	423	—	1192	933
Best	1167	—	2501	—

The best life of tyres still in service, including retread life, is 2866 and 3658 hours for the ST-2B and ST-5B models respectively. Considerable improvements in total tyre life have been realized through improved retreading, the total avoidance of water in road beds, and the use of the virgin-rock footwall as drawpoint roadbeds.

List of Equipment

Wagner ST-5B	13
Wagner ST-2B	5
Wagner HST-1	1
Landrover material carriers	4
Road grader	2

Further equipment is in the process of delivery.

Safety Devices

The following safety devices, developed at O'okiep, are fitted to Scooptrams engaged in inclined working:

(1) a device that automatically applies the brakes in the event of engine failure,

(2) an overspeed device designed to cause automatic brake application at speeds in excess of 10 km/h, and

(3) a first-gear transmission lock.

The devices are illustrated schematically in Fig. 5.

A T-piece is fitted on the air-cooler oil pipe, A, which is mounted on the right-hand side of the engine. The T-piece is connected to an adjustable pressure switch, B, set to close its contacts at 280 kPa (normal engine oil pressure is 400 kPa).

The overspeed switch, C, is mounted on the propeller shaft and driven by means of pulleys and a V-belt. It is set to close at a propeller-shaft speed equivalent to a speed of 10 km/h. The solenoid, D, is fitted underneath the emergency brake valve, E. When the solenoid is energized, it moves the emergency brake valve to the 'ON' position, which in turn actuates the brakes.

The overspeed device can be switched out when speeds higher than 10 km/h are safe. This is done by means of the switch F. To use the overspeed device, a machine must be locked in first gear. A light, G, is installed on the instrument panel to indicate whether the overspeed device is switched in or out.

Acknowledgements

The author wishes to thank the staff of the O'okiep Copper Company Limited for their assistance in the preparation of this contribution, and the General Manager for his permission to present these comments.

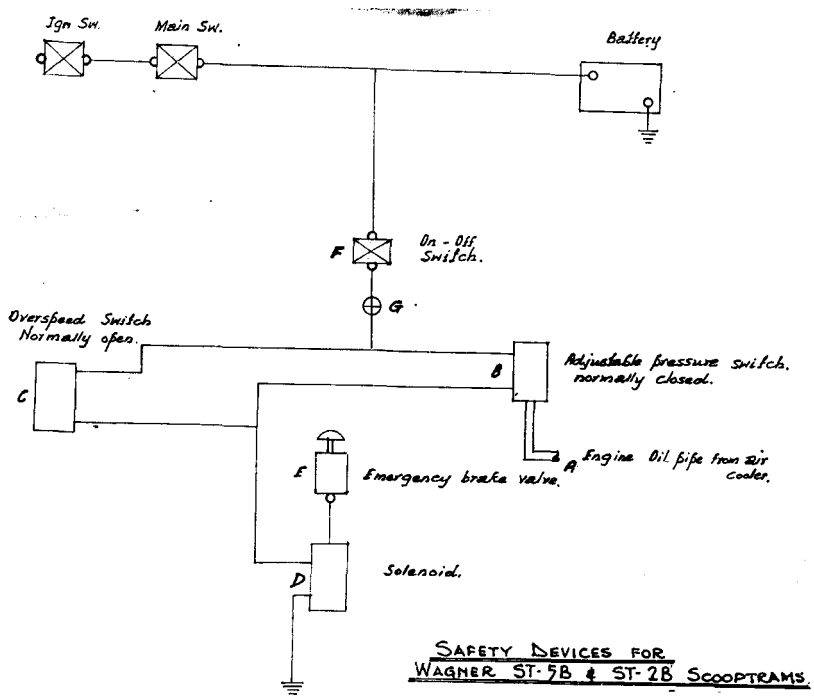


Fig. 5