Improvements to the trackless-mining equipment and operation at Prieska Copper Mines (Proprietary) Limited

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SYNOPSIS

This paper gives an account of the changes that have taken place since the previous paper on trackless mining equipment at Prieska Copper Mines (August 1973). Details are given of the modifications to the equipment, the maintenance of equipment and roadways, the training of operators, and the operating costs.

SAMEVATTING

Hierdie referaat dek veranderings wat plaasgevind het sedert die vorige referaat van Augustus 1973. Sekere besonderhede van veranderings aan toerusting, die onderhoud van toerusting en masjinerie, opleiding van operateurs en aan kostes word verstrek.

Introduction

In 1973, Grobler and Constancon[1] gave a detailed description of mining methods, geology, and associated matters at Prieska Copper Mines. The present paper, which does not repeat those details, gives an account of the changes that have taken place since then in the operation and maintenance of the trackless equipment used.

Basically, there has been little change in the stoping methods employed, except that the strike length between the rib pillars has been reduced and the size of the pillars has been increased to cater for the greater depths and higher superincumbent loads (Fig. 1). The stope-cleaning operation is at present being done by 8 yd³ equipment, as opposed to the 5 yd³ equipment previously described, and many advantages have become evident from the use of the larger equipment.

Ventilation

As Grobler and Constancon indicated, adequate ventilating capacity is important to reduce engine wear due to overheating. Also important for the same reason is the prevention of dust on the engine block and cooling system.

The downcast air at a density of 1,2 kg/m³ totals 933,4 m³/s, including 17,5 m³/s of compressed air. The vitiated air is exhausted through four upcast shafts strategically located along the strike and in the footwall of the ore-body.

Initially, 2,4 m³/s was supplied per development end but, with increasing rock-breaking depths and virgin-rock temperatures (weighted average 27,4 °C), the volume per end has been increased to 9,0 m³/s. An exhaust/force overlap system is used. A force column of 570 mm diameter provides 4,0 m³/s, while 9,0 m³/s is exhausted through a 760 mm column.

The underground mobile equipment being used at present is rated at 9000 kW, resulting in an average volume of 0,10 m³/s/kW. This volume is considered adequate to cool the equipment and dilute the exhaust and other gases.

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Diesel-powered Trackless Equipment

The diesel-powered vehicles at present in use are listed in Table I.

Naturally, the equipment is not used simultaneously since, at any one time, units are undergoing maintenance or being held as reserve equipment. The importance of reserve or back-up equipment can be appreciated when it is considered that the total production of some 250 000 tons of ore per month is drawn from five or six operating stopes. Delays in cleaning can be costly, and, because oxidation causes metallurgical difficulties, the ore cannot be stored.

TABLE I

TRACKLESS EQUIPMENT IN USE IN 1978

Type of vehicle	Units in service
5 yd³ scoop trams	29
8 yd³ scoop trams	10
2 yd³ scoop trams	3
Front-end loaders	4
D4 bulldozers	3
Personnel carriers	17
Utility vehicles	28
Dump trucks	3

The total amount of rock handled averages some 325 000 tons per month.

Fig. 2 is a plan of a typical production level, and Fig. 3 gives details of a collection level. The wider entrance to the drawpoint crosscut allows for easier access of the equipment and reduces the risk of damage from collision with the sidewall. The installation of ventilation equipment, if necessary, is also facilitated.

Load-Haul-Drive Equipment

Engines

All the 5 yd³ equipment was originally fitted with Deutz FBL 714 air-cooled engines, and the 8 yd³ equipment with model F10 L 714 air-cooled engines.

From the start of mining operations, premature failure of the Deutz 714 engines was evident. Investigations revealed that the relative density of the ore (average 3,5) was greater than that for which the machines had been designed. The engines were clearly under-powered,

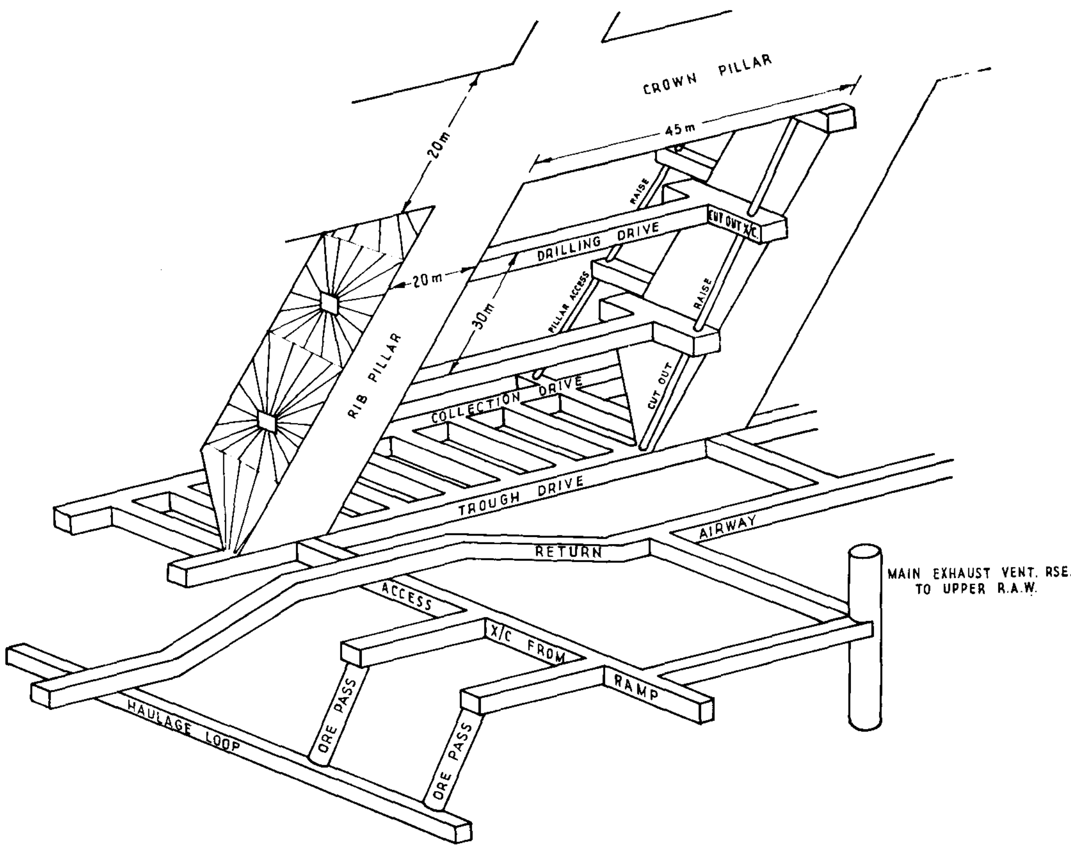

Fig. 1—A typical stope layout below 259 level

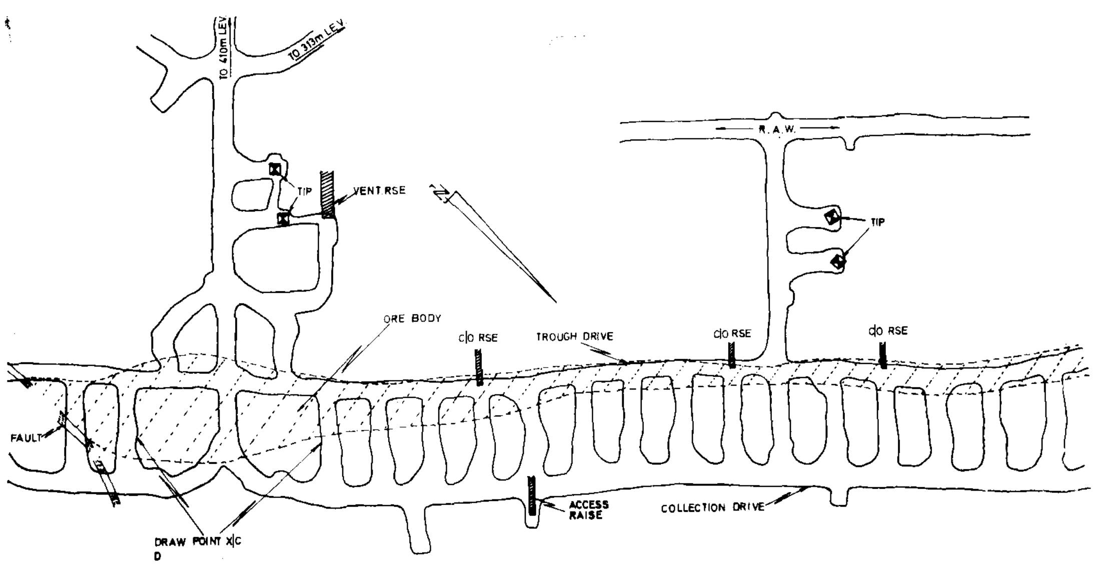

Fig. 2—A plan of 385 m production level (scale 1:1000)

and, to correct this deficiency, two Wagner 5 yd^3 machines were fitted with more powerful engines on an experimental basis.

Because of the drawbacks associated with air-cooled engines used underground, one machine was equipped with a 6V71 water-cooled Detroit diesel engine and the second with a Deutz F10 L 714 air-cooled engine. Tested under similar conditions, the air-cooled engine did not give a satisfactory performance. The cause was found to be an inadequate oil-filtration system, and the engine was recycling hot air into the air-intake duct and air cleaners.

After the air-cleaner inlet and air-intake duct had been redesigned and the exhaust silencer had been built into the frame of the machine, two machines equipped with F10 L 413 FW Deutz air-cooled engines gave improved performance. The advantages of this redesign of the Wagner ST 5 A front-end were also apparent with machines operating with the water-cooled engines, and these were modified in a similar way.

To improve the performance of the Wagner S.T.8 machines, they were converted to water-cooled 8V71 Detroit diesel engines with the front-end modification previously described.

As a result of the increased power output, the loading cycle is faster. In addition, the operating temperatures are lower. The life of the torque converter and of the hydraulic component is longer.

Buckets

Because the high relative density of the ore was

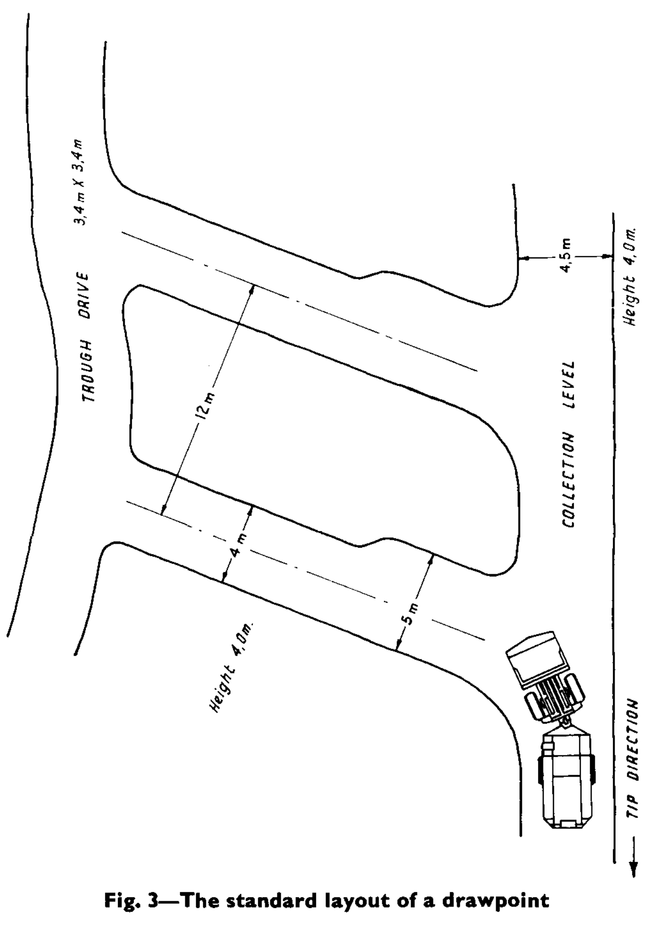

Fig. 3—The standard layout of a drawpoint

partially responsible for the engine failure, there was an obvious need for the capacity of the buckets to be reduced. A reduction of 1 m^3 was made to each bucket, and the effect on production and costs was immediately noticeable. Once the modifications had been made to the engines, the original bucket capacities were restored.

The side plates of the buckets were modified for improved penetration into the muck-piles, and the leading edges were fitted with removable manganese-steel wear caps.

Engine-shutdown and Brake Device

As an engine failure on an incline could have serious consequences, it was imperative that the safety devices should be absolutely reliable. Tests had raised doubts about the efficiency of the original design, and alternatives were therefore sought. As none of the available devices proved satisfactory, the mine designed and built its own.

The device consists of a contact oil-pressure switch (which is normally in the open position), an oil-temperature switch (which is normally in the closed position), an air-operated solenoid valve, a starter switch that incorporates an override switch, and an air-operated slave cylinder. When either the oil pressure is low, or the oil or water temperature is high, the circuit to the solenoid valve is broken, causing the valve to shut off the air supply to the slave cylinder. This cylinder, which is directly connected to the governor of the fuel pump, shuts off the engine. In conjunction with this, the air supply to the park/emergency brake valve is closed, causing the brakes to be activated and the machine to be brought to an immediate stop.

Steering System

A further cause of heat build-up in the hydraulic system was found to be in the steering system.

The standard orbital control valve in the steering mechanism was found to be sluggish and to be placing a strain on the operators. The original orbital valves required twelve full revolutions of the steering wheel to turn the vehicle from full-right to full-left steering lock. This sluggishness was caused by the pump delivering more oil to the unit than was necessary, and creating a back pressure of 6,33 Mpa. The original pump supplied 100 l/min, and the orbital valve was designed to cope with half this quantity.

A modification to the system incorporating an orbital valve with a capacity of 135 l/min served by a pump with a capacity of 140 l/min at 2300 r/min reduced the sluggishness of the system. With the orbital valve delivering 1,18 litres per revolution, it now takes only 4 revolutions of the steering wheel from lock to lock both at idling speed and at its maximum number of engine revolutions.

Compressors

The original compressors fitted to the load–haul–dump (LHD) units were air-cooled. All the air-cooled compressors have now been replaced by either water-cooled or oil-cooled units. The liquid-cooled units are 50 per cent cheaper than the original air-cooled units, and they have a life of between 4000 and 5000 hours, whereas the original units had a life of between 500 and 800 hours.

Electric System

The original LHD units were fitted with a 12 to 24 V system via a series/parallel switch. The entire electrical system has been modified to a simpler 24 V system.

Maintenance

In addition to routine maintenance, diagnostic tools are important to indicate the conditions under which the equipment operates. Analysis of the oil has assisted in pinpointing dusty operating conditions and poor air-filter maintenance, as well as diagnosing the condition of the engine.

Samples of engine oil are taken after every 50 hours of operation, and are analysed in a laboratory on the mine. The following limits have been set and, if any element exceeds the limit when the oil is analysed, the engine is examined:

Viscosity	98 to 126 cm/s	Cu	15 p.p.m.
Flashpoint	195 to 245 °C	Al	12 p.p.m.
Fe	250 p.p.m.	Si	15 p.p.m.
Pb	15 p.p.m.	T.B.N.	4 to 8

Operators are trained to complete a check list before the start of each shift, and, before they start the engine, they must make the following checks.

(1) Engine oil levels must be checked. Operators are taught to read the dipstick and to add the required amount of oil.

(2) Fuel levels are checked, and the tank is topped up if necessary. Drivers are taught the importance of preventing water and dirt from entering the fuel system.

(3) Air pre-cleaners must be cleared of all dust and dirt.

After the engine has been started, the following main items must be checked.

(a) The brakes are applied and tested against full engine power in second gear.

(b) The driver is taught to listen to the engine and, if he hears any peculiar noises or roughness, to switch off and report the fault to the maintenance staff.

(c) The exhaust system, gas purifiers, manifolds, and piping are examined visually for cracks and leaks.

(d) The fuel system is also examined visually, and any leakages are reported to the maintenance staff.

All the diesel equipment is scheduled for a daily inspection and service. The LHD units receive a 4-hour daily service, and a weekly service that lasts 8 hours. This schedule is rigidly adhered to. Maintenance personnel perform these services and inspections according to a daily and weekly service sheet. Owing to the large number of diesel-powered units operating underground, underground workshops were established on major haulage levels adjacent to the main hoisting and service shaft.

Four underground workshops service the LHD units, and a fifth workshop is used solely for service vehicles. A standard workshop layout is depicted in Fig. 4. Underground workshops are manned on a three-shift basis.

A large surface workshop has also been established to do major repairs and overhauls. LHD units receive major overhauls after 10 000 hours of service underground. At these major overhauls, the scoops are completely rebuilt. The surface workshop is manned only on the day shift.

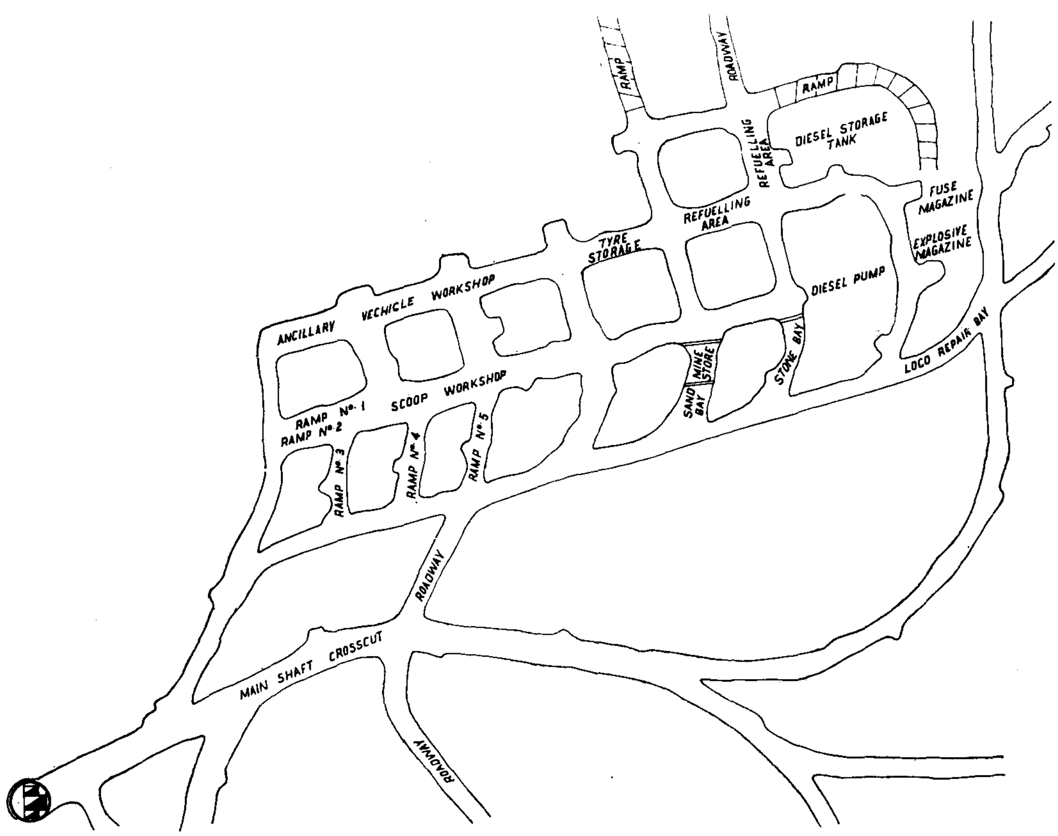

Fig. 4—The standard layout of an underground workshop (scale 1:1000)

Tyres

In the early stages of the mine's life, tyre wear on the LHD units was found to be excessive. As experience was gained, however, numerous improvements were made.

Various types of tyres and tread patterns were tested, and the treadless or 'slick' type of tyre was found to give the best life. Tyre chains were tested extensively but yielded disappointing results owing to the abrasiveness of the rock and its ability to cut rubber that has lodged under the chain.

Tyre pressures are strictly controlled by both the maintenance and the operating staff. Shift bosses fill in a tyre check list daily for each LHD unit operating in their sections. These check lists are scrutinized by the service personnel, and the details are entered in a master control file.

An analysis of the tyre details over a period of six months to June 1978 is given in Table II.

TABLE II

TYRE LIFE AND COSTS

	Stoping	Development
Average hours per new tyre	718	635
Average hours per retread* tyre	683	639
Cost per hour†, new tyre	R4,03	R2,11
Cost per hour†, retread tyre	R1,51	R0,96
Discard tyres:	%	%
Wear	3	6
Spin cuts	12	7
Impact fracture	1	3
Other	12	8

*An average of 4 retreads per casing is obtained and the life of a casing approximates 2500 hours.

†As S.T.8 machines are used on stoping and 5 yd^3 equipment on development, a comparison of the hourly costs of tyres is not valid.

Maintenance of Roadways

Trackless mining requires good roadways if damage to tyres is to be prevented. Wheel spin can cause havoc to tyres in stope drawpoints and can cause cavities in the natural rock footwall, resulting in inefficient loading.

In the development sections, one Wagner ST2B, one Caterpillar D4 bulldozer, and one low-profile Gallion grader 503 are used to maintain the roadways. Approximately 7 km of roadway are maintained in the development area. When lashing development headings, the drivers are taught to pay particular attention to the removal of 'fly dirt', which accumulates against the side walls of the tunnels.

The standard development roadway has a 0,2 m foundation layer of waste rock, followed by a 10 cm layer of graded 17 mm aggregate, and the topping is a 10 cm layer of crusher sand. Old engine oil is sprayed on the roadways to allay the dust.

In the stoping sections, two Michigan 75Bs, two Wagner ST2Bs, and two Caterpillar D4 bulldozers are used to maintain the roadways. The low-profile Gallion 503 road grader is also used occasionally on the collection levels. In the stoping areas, the roadway fill is identical to that used in the development areas.

The footwalls of all the stope drawpoints on the mine were checked, and extensive tests were carried out on the following types of lining.

(1) The drawpoint was lined with a 0,4 m thick concrete pad. This method of lining allowed good crowding traction but, unfortunately, did not have a very high wear resistance. This method of lining is still used in areas where the ore-body is very narrow (about 4 m) and where very rapid face advance can be expected, i.e. where a drawpoint's working life is about two weeks.

(2) Heavy box-section RSJs of 13 mm steel were placed longitudinally in the drawpoints to form steel 'tracks' for the tyres. This method of lining is relatively expensive, R3500 per drawpoint, but the ease of installation is strongly in its favour. It was found, however, that lack of frictional resistance provided very little crowding traction. Once the tracks have been lifted accidently or damaged, it is almost impossible to replace them. This method of lining is no longer in use.

(3) Various combinations of systems (1) and (2) were tested, but the steel runners, which were designed to prevent abrasive wear on the concrete, were invariably damaged and had to be removed. These composite drawpoints cost approximately R2000 to install.

(4) The method of drawpoint lining currently in use is shown in Figs. 5 and 6. With this method, the drawpoint is lined with a 'Deckwerk' concrete block, which has a relatively high resistance to wear. The floor of the drawpoint is levelled with a base layer, roughly 0,2 m thick, of waste rock. A cushioning layer of crusher sand, 0,1 m thick, is spread on top of this waste. The drawpoint is then decked with the interlocking Deckwerk concrete blocks. A concrete sill, 1 m thick, is poured on the trough-drive side of the drawpoint, and a 2 m sloping sill is poured on the collection-drive side of the drawpoint. All the drawpoints are lined for a total length of 15 m from the hangingwall contact of the ore-body. The cost of paving one of these drawpoints varies from R2700 to R3000. To date this method of drawpoint lining has proved very successful.

Training of Operators

The training of operators naturally plays a large part in the safe and economic running of a mechanized underground vehicular fleet. Operators normally graduate from lighter to heavier and more-complicated vehicles. With the large turnover of the labour force and the high cost of training, it is essential to ensure the maximum use of the trained operator before he completes his employment contract, and, if possible, to ensure his return to the mine on successive contracts.

Apart from being physically fit and having good visual and hearing acuity, the potential operator must display a mechanical ability, which is determined by an aptitude test. Furthermore, all the candidates must pass an optic-motor co-ordination test, be able to read and write English, and be between the ages of 20 and 40 years.

The operator's training manual takes him through the activities of identification of components, pre-start checking, starting and post-start checks to operating under

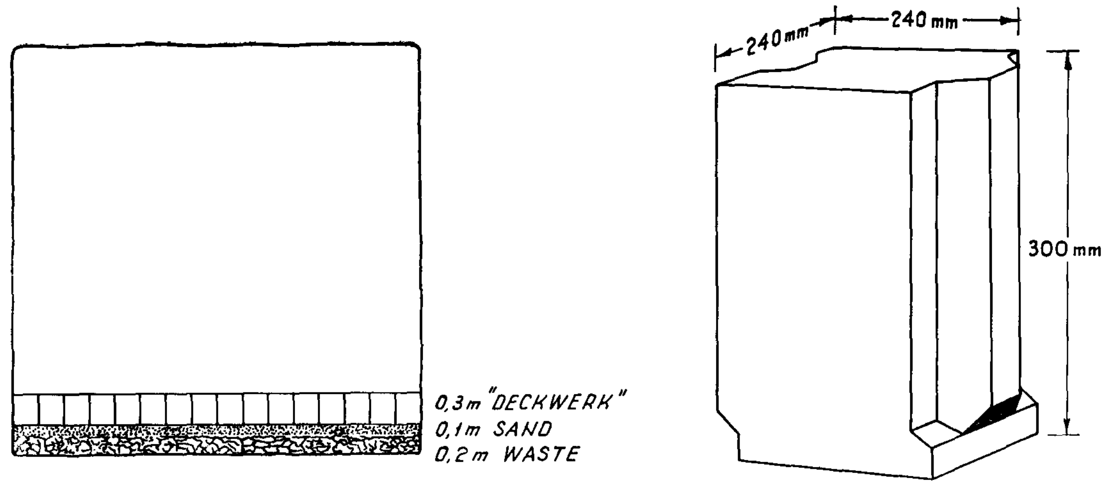

Fig. 5—Left: Section through a drawpoint **Right: Detail of a Deckwerk block**

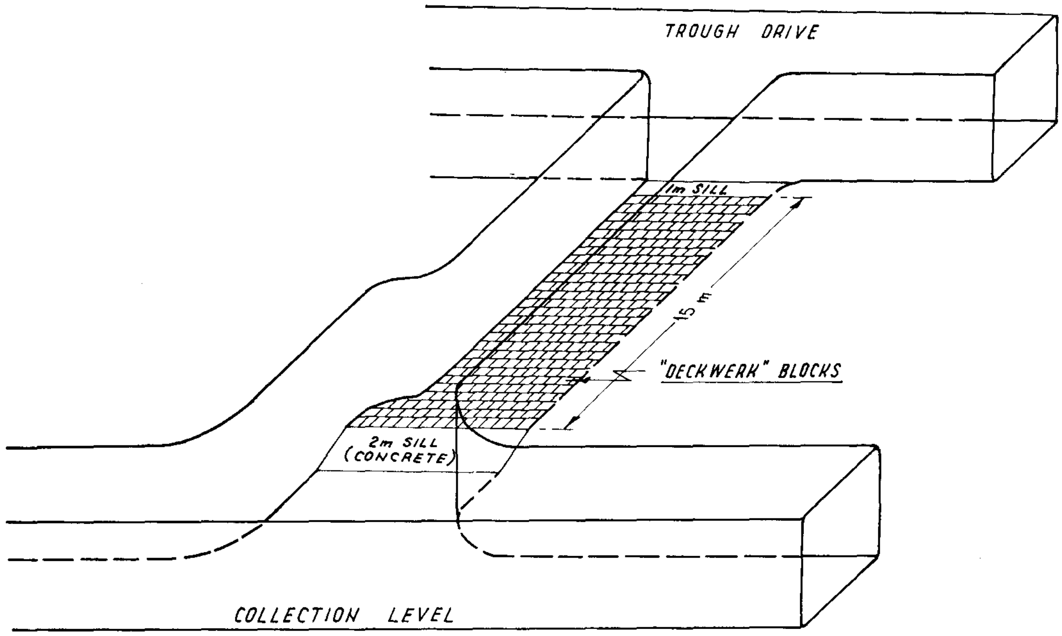

Fig. 6—The standard paving arrangement in a drawpoint

various conditions. A pass mark of 80 per cent in an oral examination is required.

Practical instruction is given both on surface and underground, and, before graduating as LHD operators, the trainees must be able to load and dump ore at a rate of at least 60 t/h in normal underground conditions and to complete a prescribed check list within 15 minutes.

Operating Costs

The average operating costs of the LHD units for 1978 were as shown in Table III.

As shown by Table III, the major costs are for spares and tyres, and it is in this direction that efforts have been made, and will continue to be made, to effect economies in the operation of these machines.

Continuous attention to the training and discipline of the operators, and to the analysis of the working lives of all the components of the machines, is necessary if the working costs are to be improved. No clear picture has yet emerged of the optimum operating life of the equipment; this has to some extent been obscured by the redesigning that has taken place.

TABLE III

AVERAGE OPERATING COSTS OF LHD UNITS IN 1978

	Percentage of total costs		
Element	Wagner ST8	Wagner ST5	Eimco 913A
Labour	10	10	12
Stores	52	54	50
Fuel and oil	12	13	13
Tyres	18	16	16
Other costs	8	7	9
	100	100	100

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