LOAD HAUL DUMP UNITS FOR SAFE, PRODUCTIVE STOPE CLEANING OPERATIONS

BY

JOHANNES LOUIS FOURIE

This is a project report submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg (in partial fulfilment of the requirements) for the degree of Master of Science in Engineering.

JOHANNESBURG

1988

DECLARATIONS

I declare that this project report is my own work except for being assisted by E. Bojé, Senior Technical Assistant and his staff, with the project observations and summaries. It has not been submitted before for any degree or examination at any other University. All information used in the report has been obtained while I was employed at The Randfontein Estates Gold Mining Company, Witwatersrand, Limited.

J Fourie

_______________________ Signature

26th day of February 1988.

ABSTRACT

This report describes a pilot investigation carried out in a stope of the Cooke 1 Shaft of The Randfontein Estates Gold Mining Company, Witwatersrand, Limited and some subsequent developments. The pilot investigation was initiated to assess the benefits of using load-haul-dump (L.H.D.) units for stope cleaning instead of face and strike gully winches and scrapers. A suitable L.H.D. unit cleaning method had first to be developed before final comparisons could be made.

The smallest L.H.D. units already being used successfully in wide and massive base metal, diamond and coal mines appeared to be suitable for adaptation to stope cleaning in the gold mines, without much alteration.

No major alterations to the L.H.D. units were required. The stope method was technically and safely adapted to an L.H.D. unit cleaning method. Conclusions reached during the pilot study showed that the use of L.H.D. units increased labour productivity by 29,1 percent but at a 108,1 percent greater cost per ton broken than that of winch cleaning methods. Therefore, for L.H.D. unit stope cleaning methods to become economically viable it would be necessary to reduce costs, in particular the shaft system and ore reserve layout costs.

ACKNOWLEDGEMENTS

The author wishes to thank the Technical Director, Gold and Uranium Division, Johannesburg Consolidated Investment Company, Limited, and the Consulting Engineer - Gold, for permission to submit this report. My colleagues in the Industrial Engineering Department, Geology Department and Rock Mechanics Department are thanked for their assistance in providing data.

CONTENTS

PAGE

PAGE

LIST OF FIGURES

LIST OF TABLES

LIST OF APPENDICES

1 INTRODUCTION

Management in the Mining Industry, as anywhere else, is always concerned about safety and health of people and the environmental conditions under which they work. Management, within the context of rising working costs, is therefore constantly seeking possible means of improving productivity and safety, in order that business objectives may be achieved.

Technological advancement has become a way of life. Such is the case with the cleaning function in a gold mine stope. Stope cleaning methods have changed from arduous manual hand lashing and hand tramming to simple mechanised face scraping and strike gully scraping. Scraper winches have improved in design with resultant increased power output and reduced physical dimensions. Further improvements are still possible. However, there is a natural interest to investigate the development of more productive mechanised stope cleaning methods.

To this end in 1974 the Chamber of Mines Research Organisation embarked on a project to investigate mechanized stope cleaning methods, but no method has been considered as yet for large scale introduction.

Since 1973 several gold mines also embarked on their own research programmes to varying degrees to find the best mechanized method to suit their own particular conditions. Companies such as Hubert Davies Heavy Equipment (Pty) Ltd, who were known to be marketing diesel and electrically powered mining machinery, were approached to determine the availability of equipment which could be used for stope cleaning in gold mines.

This report describes a pilot investigation carried out in 1975 in a stope of the Cooke 1 Shaft of The Randfontein Estates Gold Mining Company, Witwatersrand, Limited. A subsequent development whereby mechanized cleaning was introduced on a larger scale at the same shaft is summarised for comparison.

Load-haul-dump (L.H.D.) units as used in massive base metal mines were considered. Units of different sizes and capacities were available and appeared to have the best potential for adaptation to gold mine narrow, flat dipping ore body stope cleaning. L.H.D. units were being experimented with in some of the chrome mines in the Transvaal, but were mainly being used for trackless development such as the cleaning of winzes and inclined roadways.

The smallest L.H.D. unit available, with an $0.8m^3$ bucket capacity, had a small turning circle which could enable the manoeuvring of the unit between rows of permanent stope support. This was only feasible if some changes could be made to the support pattern and stope layout without jeopardizing the safe working of the stope.

The purpose of the L.H.D. unit investigation at Cooke 1 Shaft was to develop a stoping method to enable the economical application of L.H.D. units for the cleaning of broken ore instead of using scraper cleaning methods.

Two areas of investigation were concentrated on. The environment, which included reef width, type of reef, footwall conditions, type of support and support spacing formed one area, and the L.H.D. unit formed the second area.

In this latter area mechanical reliability, maintenance, fuel consumption, tyre life, manoeuvreability, safe operation and production efficiency was evaluated for comparison with the winch cleaning method.

A certain amount of preparation had to be done in the stope area to garage the L.H.D. units in safe positions.

Initial L.H.D. unit stope cleaning procedures were decided on with close supervision arranged on every shift. Sufficient data items were identified which needed to be recorded continuously in order that a proper analysis could be made at the completion of the experiment. The data would enable necessary information to be extracted periodically so that the method being applied could be evaluated and modified if required.

Work Study techniques were applied to determine the necessa requirements and L.H.D. unit operating objectives. It was expected that the experiment would be conducted for a period of six months, but actually was extended to ten months before removing the L.H.D. units from underground.

A suitable stope cleaning method was designed after some experimentation and a number of scraper winch panels and tipping points were converted to suit the L.H.D. units. The experiment was conducted in a well organized production stope where sufficient broken rock could be produced from stoping operations to test the production capacity of the loaders. Major problems proved to be mechanical reliability and tyre life.

Ongoing records[1] were also kept of the performance and working costs of conventional scraper cleaning methods per stoper. These results were extracted and summarised for the same period as the L.H.D. unit pilot investigation.

The final cost per ton broken for the L.H.D. unit cleaning method, during the experiment, was more expensive than that of the winch cleaning method at the time. So although the pilot investigation could be considered a technical success, it was not an economic success.

However, it was realised that in order for it to become an economical success, the shaft system and ore reserve layout would need to be re-designed to take advantage of the features of the L.H.D. equipment, and thereby reduce these costs sufficiently to offset the increased cost of operating and maintaining the L.H.D. equipment.

This led to large scale mechanised stoping being motivated in 1984 based on a theoretical study which indicated it to be economically viable with a reasonable chance of success. Changes in support methods made it possible to introduce larger trackless mining equipment to do stope cleaning which improved the production rates over those obtained during the 1975 tests. Changes were also made to the ore reserve layouts.

This was followed by a feasibility study report prepared in 1986 for the 90 level wide Reef trackless mining project. Details of this project and results for the past nine months are included in this report.

2 PROBLEM

2.1 Labour Cost

Since the early 1970's wage increases in the case of unskilled and semi skilled workers has proportionally far exceeded the increases of the skilled workers and is rapidly contributing to inflating the working costs.[2] Winch operator's wages escalated by 240 percent during the 12 years from 1975. Artisans wages escalated by 108 percent during the same period.

A means of counteracting this trend would be to determine the extent to which stope cleaning could be mechanised.

In this case mechanisation will result in a reduction of the number of unskilled workers required, and in an increase in the number of semi skilled and skilled workers. The overall effect will be that production might be maintained by a smaller number of people, thus making it possible to pay them more without adversely affecting the overall cost per ton milled.

2.2 Housing

The modern trend is to have less cramped housing with better facilities for single blacks and to provide married quarters to ensure a more stable work force with all its advantages. The high initial capital outlay for such housing projects and expenditure required to maintain the buildings in a satisfactory living condition needs to be offset by an improvement in efficiency.

2.3 Safety

Shifts lost due to injuries are costly. Apart from the treatment cost of a patient, there is also a production loss due to his absence. A reduced labour force will reduce the exposure to injuries while mechanised units could be designed to afford better protection to operators. An analysis of accidents[3] occuring in stopes showed that injuries concerning scraper cleaning and related work was an area which required attention for improvement.

2.4 Labour Efficiency

Stope cleaning methods in narrow, tabular, flat dipping reefs vary, depending on what is appropriate and preferential from hand lash cleaning to scraper cleaning with double drum winches.

Although an improved mechanised cleaning method, the scraper remains relatively inefficient due to lack of mobility of the winch, poor load carrying capacity of the scraper, excessive idle time, etc.

Labour efficiency may be improved by introducing some further technological advancement. The dissatisfaction of labour for working in adverse environmental conditions, and decreasing efficiencies might hasten the decision to introduce a change.

A substantial training and acclimatization centre, well staffed to cope with the migratory labour force has to be maintained. A smaller and more stable labour force would not require the same amount of training staff.

The greater the labour force the more difficult it becomes to exercise effective control, with subsequent loss of efficiency, which will necessitate a larger supervisory staff to ensure good productivity.

2.5 Total Mechanized Stoping

Mobile units have to be considered and experimented with in preparation for the time when fully mechanized stoping would be more profitable than labour intensive stoping. The purchase price of sophisticated mining machinery has unfortunately kept pace with increased labour costs, with the result that greater mechanization has remained an uneconomical proposition, when applied to the Witwatersrand gold mining situation in general.

The increasing price of gold and depleted reserves will encourage the mining of deposits at greater depth and in environmental conditions detrimental to manual labour, so that some form of mechanization may well become an economic proposition in the future.

3 CONSTRAINTS

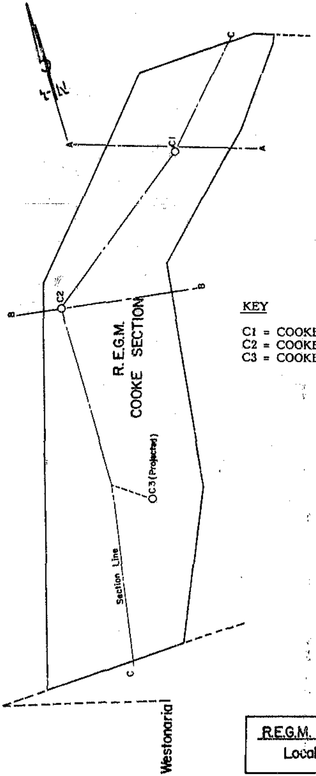
3.1 Geology of the Area

The reef being mined in the Cooke Section of The Randfontein Estates Gold Mining Company, Witwatersrand, Limited at the time when the 1975 experiment was conducted, was the UE1a reef of the Witwatersrand Super Group.[4] A locality plan **(Figure 3.1)** shows the Cooke 1 Shaft area where the L.H.D. cleaning experiment was initiated. The reef width varied from approximately 75 centimetres to 350 centimetres in the Cooke 1 Shaft area.

The smallest L.H.D. cleaning unit readily available could operate effectively and efficiently in a stope width of 180cms. The L.H.D. unit ideally had to clean both the broken rock in the stope face and the strike gully in order to replace conventional scraper cleaning methods.

At the time, the Cooke 2 Shaft area was being developed and the Cooke 3 Shaft was in the planning stage. The whole of Cooke Section is covered by Malmani Dolomite varying in thickness from approximately 600 metres to the South of Cooke 3 to some 100 metres at the Northern boundary of Cooke Section **(Figure 3.2)**. This dolomite is well known for rapid weathering when in contact with water and consequent associated with sinkhole formation. Water bearing dykes intersecting the dolomite were encountered at Cooke 1 Shaft. Fearing a major inrush of water during stoping operations, as had occurred at a sister mine, a system of pillars was designed for support along all major faults and dykes.

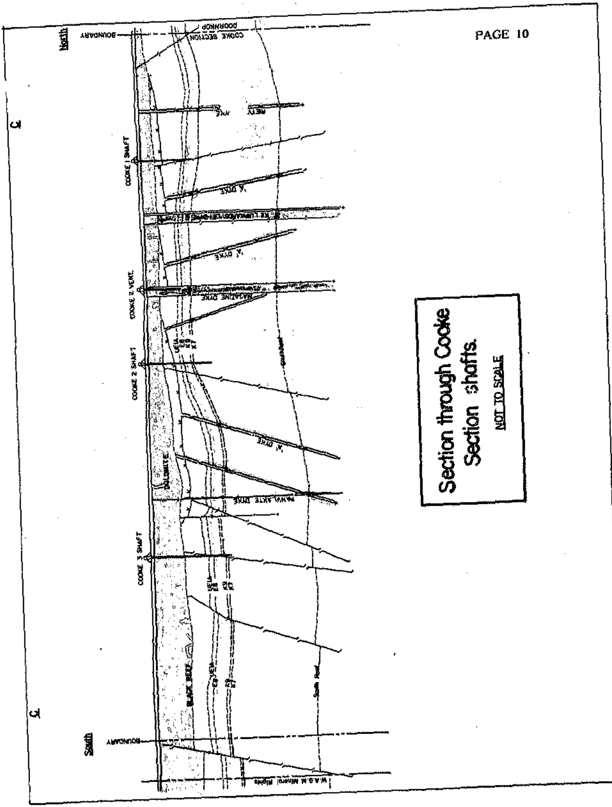
FIGURE 3.1



KEY

C1 = COOKE 1
C2 = COOKE 2
C3 = COOKE 3

R.E.G.M. Co.(W) Ltd.
Locality Plan


North
South
BOUNDARY
COOKE SECTION
DOORNKOP
COOKE 1 SHAFT
COOKE 2 VENT.
COOKE 2 SHAFT
COOKE 3 SHAFT
DOLOMITE
BLACK REEF
MAGAZINE DYKE
'A' DYKE
South Reef
W.A.G.M Mineral Rights
Section through Cooke
Section shafts.
NOT TO SCALE

Below the dolomite is a poorly developed Black Reef series usually represented by shales, black shaley quartzites and light grey orthoquartzites. The Ventersdorp Lava underlies the Black Reef and appears as a wedge shaped body truncating against the North-South striking Panvlakte fault immediately to the West of the lease area. The lava occurs along the entire length of Cooke Section and thins toward the East. Only isolated occurrences of payable Ventersdorp Contact Reef, which is situated at the base of the lava, have been found at Cooke Section.

A near complete stratigraphy of the Central Rand Group exists on Cooke Section. The reefs currently being mined (1987) are the UEla, E9Gd, E8 and Kimberley Reefs. The UEla and Elsburg Reef are the uppermost reefs of the "middle zone" and underly the entire area, having a regional dip of 15± to the East. The North-South anticlinal feature, which is more pronounced in the Cooke 3 area, results in westernly dips to the West of the anticlinal axis.

The UEla conglomerate unit diverges eastward from the paleo-topographical shore line, that is co-incident with the anticline in the Cooke 3 area, from a single reef to five conglomeratic bands. Some or all of them may be developed in one area.

Such areas are currently being mined by mechanized means using hydraulic drillrigs and L.H.D. units in stope widths varying from approximately 300 centimetres to 1 000 centimetres in width.

Excluding the A1 conglomerate there is an increase, from North to South, in the development of gold content of the remaining conglomerate bands. The result is that while only two of the conglomeratic bands can be recognised in the Cooke 1 area, all five are developed in the Cooke 3 area.

The middle Elsburg Reefs being mined in the Cooke Section are the E9Gd, E9Gb and E8 conglomerates. The E9Gd underlies the UE1a with a middling of 0 - 3 metres, while the E9Gb occurs some 20 metres below the E9Gd. The E8, situated approximately 75 metres below the UE1a, is a robust conglomerate unit with erratic values within defined payshoots. The E8 reef, well developed in the Cooke 2 Shaft area and originally mined by means of winch and scraper cleaning methods, isalso being mined by means of hydraulic drillrigs and L.H.D. cleaning units.

The Kimberley Reef packages with most consistent values are the K9 and K7 reefs. These reefs occur some 40 metres below the UE1a in the West to 250 metres in the eastern regions of the Cooke lease area and the so called "negative" area South of the Doornkop Fault. Value distribution on the K7 and K9 reefs is fairly erratic but occurs in reasonably well defined "payshoots".

Figure 3.3 is a diagrammatic section of the reef being mined by mechanised means at Cooke 1 Shaft.

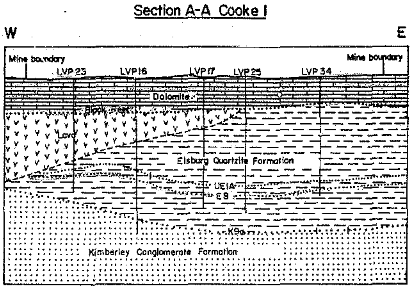


Figure 3.3 West-East Geological Section.

3.2 Rock Mechanics Considerations

The average reef depth at the Cooke Section is approximately 900 metres below surface, which is relatively shallow when compared with other mining depths in the Witwatersrand Basin. The overriding strata control problem, in the Cooke Section, is the control of subsidence of the rock below the overlying water bearing dolomites.

At the average mining depth in the Cooke Section, the primitive vertical stress varies between 16 and 27 MPa and the horizontal stress varies between 8 and 13,5 MPa. The strength of the reef varies between 150 to 175 MPa with the result that stress induced fracturing is usually generated in crush pillars and small mining zones adjacent to local disturbances.

The initial support system, designed and installed at Cooke 1 Shaft in 1975, was according to the grout base pack method which had proved to be successful in a deeper sister mine. The grout base packs, consisting of a four pointer, slabbed timber crib pinned at the corners, supporting a cement grout filled polypropylene bag, blocked to the hangingwall, were spaced at 2 metres skin to skin on dip and strike. Reef pillars 10m in width were left adjacent to major faults and dykes. Although the support pattern was not as dense as that of the sister mine, the smallest L.H.D. cleaning unit available was unable to turn in any direction between the packs, limiting its effectiveness.
It was later found that this "soft" grout base pack support allowed bed separation to take place before adequate resisting load could be generated by the grout.

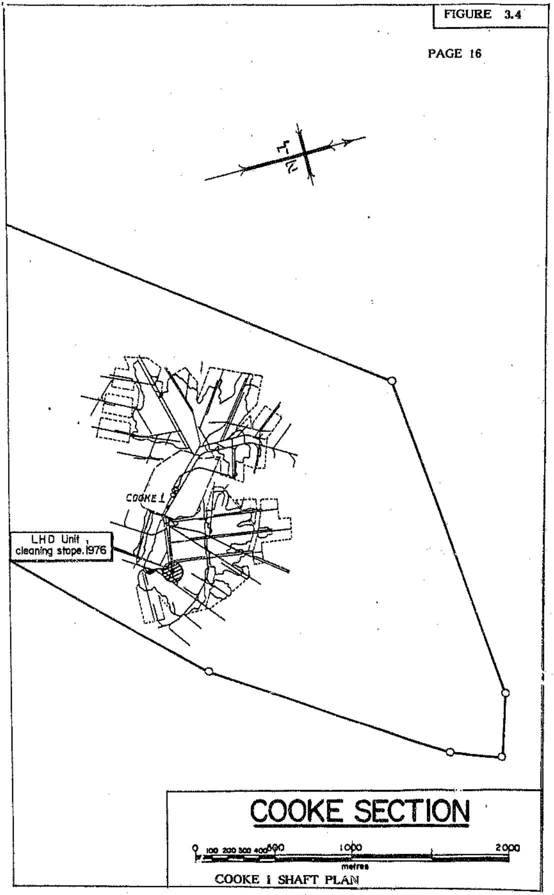
3.3 Environmental Conditions

Ventilation arrangements of the Cooke 1 Shaft consisted of a twin shaft system with a downcast main shaft and an upcast ventilation shaft. In 1975, at the commencement of the L.H.D. stope cleaning experiment, only a small area had been stoped out **(Figure 3.4).** Ventilation control was achieved by means of intake airways on the lower levels and up through the stopes. The first levels opened up were 95, 101 and 106 level with most extensive work being done on 95 level.

In the overall planning strategy the fresh air requirement per kiloton per month is ideally $2,5m^3/s$ in the case of conventional mining. The operation of diesel engine powered L.H.D. units would contaminate the air with exhaust fumes at a rate of approximately $0,06m^3/s$ per rated kW, and additional heat would be generated. Should large scale mechanised mining take place using diesel powered units then the fresh air requirement will have to be increased to $3,5m^3/s$ per kiloton per month.

The virgin rock temperature in the region of the stope where L.H.D. unit cleaning took place was 27 degrees with a wetbulb temperature of 28°C and a kata of 17,3. It was not necessary to make any changes to the ventilation of the stope to accommodate the L.H.D. unit cleaning experiment.

FIGURE 3.4



COOKE 1 SHAFT PLAN

3.4 L.H.D. Design Features

3.4.1 Freedom of Movement

Although the smallest L.H.D. unit available could operate in 180cm stope widths, the average stope width should not be less than 190cms to permit more freedom of movement in the stope face and back areas. Stope face of sufficient length and width had to be made available to set up an experiment comparing an L.H.D. unit cleaning method with a conventional winch and scraper cleaning method. In general L.H.D. units cannot operate effectively at dips in excess of seventeen degrees. The steeper the dip the greater the attention required to create a roadbed that would provide adequate traction for L.H.D. unit tyres.

3.4.2 Operational Stability

The narrow width of the L.H.D. units used in some hard rock mines compared with their height, makes the strike cleaning of broken rock in a steep stope panel a dangerous operation depending on the additional elevation of the updip side wheels caused by broken rock lying on the footwall of the stope. The L.H.D. units can overturn quite easily, endangering the life of the operator and anyone else in the immediate vicinity. In this case "steep stope" has been defined as being steeper than ten degrees.

3.4.3 Turning Circle

The strike gully packs have to be 2,5 metres skin to skin on dip to allow the smallest L.H.D. unit, with a turning circle of 4,5 metres outside radius and 2,4 metres inside radius, to travel between the dip rows of packs. There is a theoretical radius of curve for each size of L.H.D. unit, which takes into account the inner and outer dimensions of the machine (Figure 3.5).

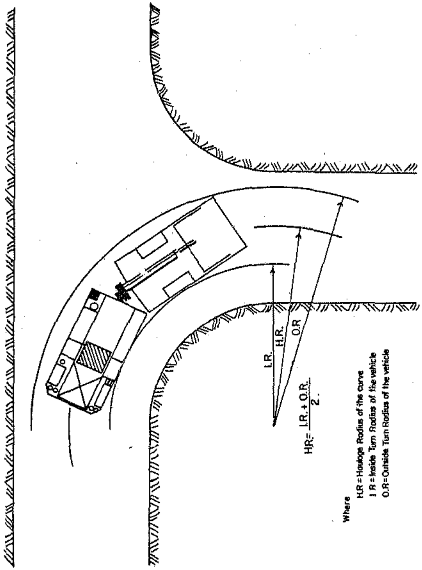


FIGURE 3.5 **Wagner Mining Equipment Company**

Theoretical Turning Clearance

3.4.4 Bucket Width

The smallest L.H.D. unit had a height of 1,6 metres which provided adequate operating clearance in a 1,8 metre wide stope. The bucket dimensions of the L.H.D. unit would vary in dimension depending on the density of the material being loaded. With pack spacings of 2,0 metres skin to skin the L.H.D. unit could not exceed a width of 1,6 metres, at most, to allow for operation error and sway of the unit over uneven surfaces. The bucket width of the 0,8m³ bucket is 1,2 metres. Although the next larger L.H.D. unit available, with a bucket capacity of 1,6m³, had a suitable bucket width, the turning circle was 40 percent greater making the machine unacceptable. In fact the buckets had been designed for use in the chrome mines where denser material was being loaded. A larger bucket would also have increased the length and width of the L.H.D. unit, thus adversely affecting the turning circle.

3.4.5 Operating Speed

The operating speed of the L.H.D. units had to be safe and yet fast enough to sustain a cleaning rate of at least [illegible] tons per hour. This would enable a stope face to be cleaned in two shifts (14 hours). The power plant for the L.H.D. unit had to be diesel powered rather than electrically driven, since it was considered that a trailing cable would be subjected to excessive drag due to the support pattern. As it was, at the time, the majority of the smaller L.H.D. units on the market were powered by Deutz air cooled engines, although other makes of diesel engines could be fitted. The general temperature of the air in the stopes at Cooke 1 Shaft, at 28°C, was suitable for the efficient functioning of the Deutz engines.

The next larger, more powerful engine which would have enabled a faster operating speed and loadability, could only have been fitted by increasing the length of the machine.

3.4.6 *Clearance*

The clearance of the L.H.D. unit from the footwall determined the extent to which breaking of the footwall during blasting operations had to be controlled. A minimum clearance of 250 millimetres was desirable since broken ore could drop off the L.H.D. unit bucket when travelling and damage the undercarriage of the L.H.D. unit **(Figure 3.6)**.

For the L.H.D. unit experiment to have the best chance of succeeding, the experimental stope needed to be tabular, with a dip of less than eight degrees and no, or only minor, faults to negotiate.

3.4.7 Operational

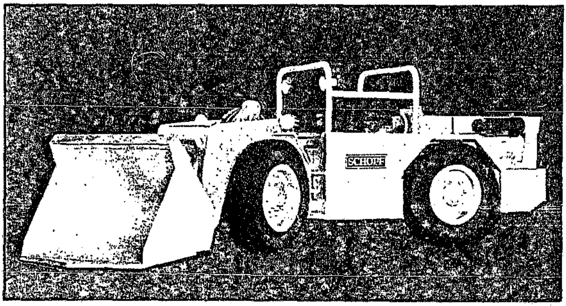
The operator seat should be positioned so that the operator could see equally well whilst travelling forwards and backwards. The operator should be protected from falling rocks during loading operations. The exhaust system had to dilute the gasses to permissable levels. It was preferable to use a catalytic converter with a fume diluter rather than a wet scrubber type of system. On mine experience with diesel engine driven locomotives showed that maintaining the water level of the scrubber box was often neglected causing a safety hazard.

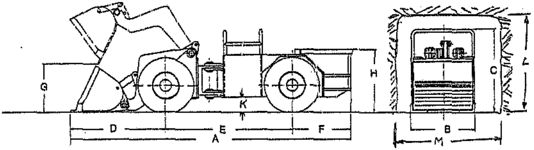
SCHOPF

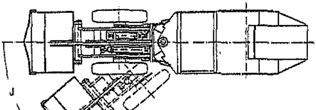
UNDERGROUND-LOADER

FIGURE 3.6

L62







	A	B	C	D	E	F	G	H	J	K	L	M
mm	5300	1240	1580	1760	2420	1120	1000	1200	40°	250	1900	224[illegible]

SCHOPF MASCHINENBAU GMBH STUTTGART

MY 2 0780 L 62 E

The differential should be of a non spin type in order to permit power transmission to the other wheel when one wheel loses traction. The L.H.D. unit should have four wheel drive to enable maximum use of the power available and should have articulated steering to permit the smallest turning circle possible, i.e. 3,5 metre radius.

The tyres should be able to withstand the ripping effect of sharp footwall quartzites and absorb some shock. The chassis should be of robust construction to absorb shock loads during loading operations and travelling with a loaded bucket. The air-cleaner had to effectively prevent any quartzite dust from entering the engine. If the L.H.D. unit were to be tilted, the dry cartridge type of air-cleaner would be more suitable than the oil bath type, since the latter could become less effective if the machine operated at a steep angle. The hydraulic cylinders of the bucket lift and steering should be protected to prevent abrasive dust from covering the shafts and entering the seals.

Spare parts should be readily available to avoid downtime without an abnormally high reserve stock of spares having to be held on the mine.

The L.H.D. units should be of robust construction and should be protected on the sides against damage as a result of accidental contact with the packs.

The suppliers should be able to deliver the L.H.D. units ordered within three months of placing the order, since it was expected that a stope could be prepared in that period and experimentation could commence.

3.4.8 Tyres

L.H.D. units are being used more frequently for station cleaning during shaft sinking operations. These L.H.D. units have smooth pneumatic rubber tyres which cause delays due to punctures. A variety of different tyre construction and materials are available, each with properties advantageous for safe driving, long tyre life and machine life e.g.

- Solid rubber tyres - moulded onto the wheel rim and used until the wheel diameter becomes too small.
- Solid rubber tyres with steel wire bristle - moulded onto the wheel rim, similar to a circular brush extending to a depth of 250 millimetres.
- Pneumatic rubber tyres with thick tread and distinctive coarse pattern.
- Pneumaric rubber tyres with thick smooth finished tread.
- Pneumatic rubber tyres, foam filled to prevent punctures.
- Pneumatic rubber tyres, chain clad in moulded rubber.
- Pneumatic rubber tyres chain clad only.

In order to keep production delays and operating costs to a minimum, the tyres should:

- Have the longest possible life at the lowest operating cost per ton.
- Have a co-efficient of friction permitting best power transmission under a variety of conditions.
- Be pliable enough to give driver comfort and machine protection.

- Be firm enough to prevent excessive sway during operations.
- Be tough enough to prevent shredding and rapid wear under conditions where tyres would be running over sharp quartzite footwall or broken rock, often under water.

Solid rubber tyres were initially selected due to their apparently having all the necessary properties. Pneumatic tyres were considered to be too prone to punctures by sharp rocks and broken jumpers protruding from the broken rock.

3.4.9 Experimental Stope

Bearing the foregoing constraints in mind, a stope was selected on 95 level Cooke 1 Shaft where two pay reef horizons of the upper Elsburg Series were so close together that they could be stoped together at an average stoping width of 2,5 metres. The average dip of the reef in the stoping area was seven degrees. There were no known faults in excess of 50 centimetres in the stope faces **(Figure 3.7).**

Face winches and strike gully winches rated at 37 kW and 56 kW respectively had been installed. A total of eight stope panels had been fully equipped. A winze below the bottom panel was being advanced to create more face. It was decided that two 0,8 cubic metre bucket capacity L.H.D. units would be purchased to operate in four of the eight stoping panels. Results of the L.H.D. unit cleaned panels could then be compared with the results of the four remaining panels which were cleaned by means of winches and scrapers.

FIGURE 3.7

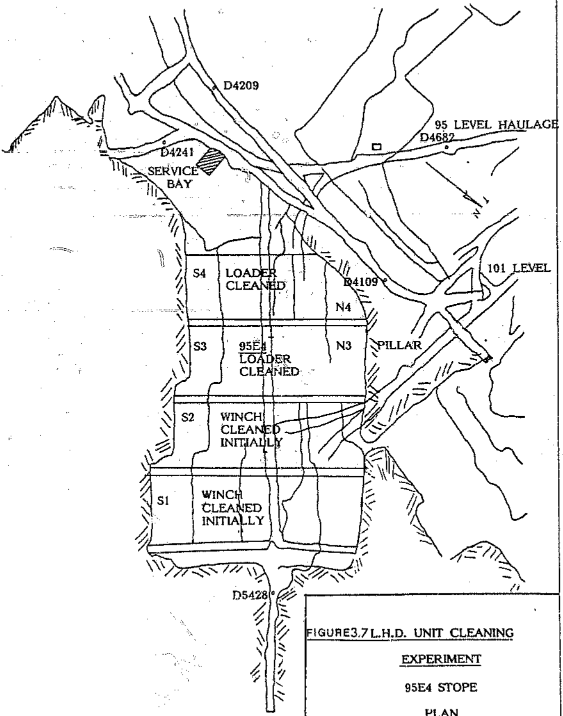


FIGURE3.7 L.H.D. UNIT CLEANING EXPERIMENT

95E4 STOPE

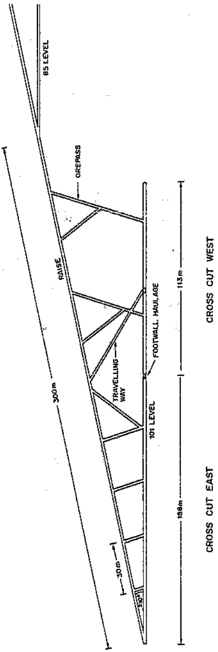
PLAN

NOT TO SCALE

Stope panels were approximately 30 metres in length and each panel was served by its own orepass. A diagrammatic section (**Figure 3.8**) indicates the amount of conventional development which was done along the crosscut line.

There was a nine month waiting list for two popular makes of L.H.D. units, which forced the selection to the next best L.H.D. unit available. Two Schopf L62 L.H.D. units were purchased with bucket capacities of 0,8 cubic metres, rated at 1,1 tons per load when loading quartzite. The units were powered by 50 kW Deutz air cooled engines. A spare bucket was purchased so that bucket repairs could be done without losing production. A spare set of tyres and rims was also purchased to reduce downtime. Delivery delays could not be determined with any degree of accuracy. All spare parts were imported from Germany. Technical details of the Schopf L62 L.H.D. unit are listed in **Table 3.1**.

FIGURE 3.8



<u>FIGURE 3.8</u> CONVENTIONAL DEVELOPMENT

CROSS-CUT LINE

SECTION

NOT TO SCALE

TABLE 3.1

SCHOPF L62 - TECHNICAL DATA

Length with lowered boom	5,5 m
Width	1,2 m
Height with roll bar	1,7 m
Track	0,9 m
Wheelbase	2,5 m
Lifting height	1,9 m
Maximum dumping height	1,0 m
Tipping clearance to front edge of tyres at maximum lifting height	1,0 m
Maximum dumping angle at maximum lifting height	59°
Maximum pull-in angle at lowered boom	48°
Turning radius interior	2,5 m
Turning radius exterior	4,2 m
Shovel width	1,2 m
Shovel capacity	0,8 m^3
Tyres (optional)	9.00 - 20/10.00 - 20
Engine performance	45 kW
Revolutions	2 500 rpm.
Maximum torque	23,5 mkg.
Fuel consumption	12,8 l/h
Tank capacity	90 Litres
Forward speec	18 km/h
Reverse speed	20 km/h
Lifting time	3 seconds
Lowering time	3 seconds
Tipping time	2 seconds
Maximum lifting force	6 200 kg.
Maximum break-out force	4 750 kg.
Maximum thrust	5 000 kg.
Shovel payload	2,4 tons
Weight	6,6 tons

4 POSSIBLE STOPE CLEANING METHODS

Since little information was then available concerning the use of L.H.D. units for stope cleaning in gold mines it was decided to experiment with three theoretical methods.

1) Strike cleaning where no specific strike gully would be provided and the L.H.D. unit would travel on strike between the support lines.

2) Updip cleaning where the L.H.D. unit would travel along a strike gully and then load updip in the stope face and between rows of support.

3) Downdip cleaning where the L.H.D. unit would travel along a strike gully and then load downdip.

4.1 Strike Cleaning Method

The strike cleaning method was designed so that the L.H.D. unit could travel on strike between the rows of support from the main dip roadway. The centre gully, dipping at approximately seven degrees would then form the main dip roadway. Sufficient width between the centre gully support would have to be allowed to permit the loaders to turn in on strike at any point in the stope.

No specific strike gully, would be used regularly. The L.H.D. unit would advance on strike to the broken ore **(Figure 4.1)**, fill the bucket and reverse to the main dip roadway and the orepass system.

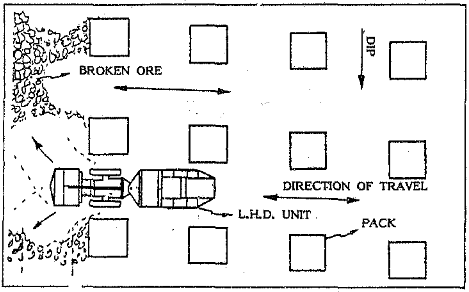


Figure 4.1 *A plan of an L.H.D. unit cleaning on strike.*

The main disadvantages are:-

- If the dip of the orebody should exceed a maximum of ten degrees, the L.H.D. unit could topple over if the updip wheels happen to strike a rock or rise on the footwall (**Figure 4.2**). The narrow loader width and high centre of gravity could cause difficulties on strike.
- The complete footwall in the back area would have to be levelled and attended to regularly, to avoid tilting of the loader.

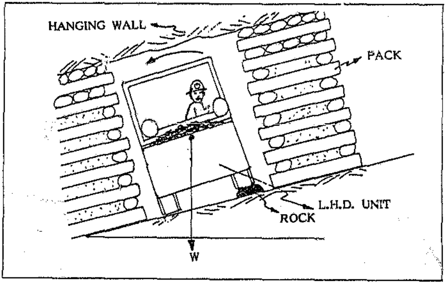


Figure 4.2 A section of an L.H.D. unit travelling over uneven footwall and strike.

- Hanging wall sag would limit the available roadway with a height of 2,5 metres. Since the sag is usually the greatest in the centre of the span of the stoped out area, the greatest restriction to travelling would start at the centre gully.

- A greater area would have to be made safe and examined regularly instead of merely barricading off the worked out area and maintaining only the strike gully areas as is the case in conventional scraper cleaning.

- The support patterns would have to be geometrically square and installed with great accuracy, otherwise all strike roadways would be blocked or reduced in width in "out of line" areas.

4.2 *Updip Cleaning Method*

In the updip cleaning method the L.H.D. unit would travel along main strike roadways from an orepass to the stope face, and start cleaning the panel from the bottom of the stope face upwards to the top of the stope face.

The L.H.D. units would also turn updip into the sweeping area and scoop up broken rock whilst travelling from the bottom to the top of the sweeping area.

Old dangerous areas could easily be avoided and barricaded off after sweeping.

The main disadvantages are:-

- With the force of gravity working against the traction on the wheels, the bucket could not always be filled properly, which could decrease the production rate **(Figure 4.3).**
- A much greater rate of tyre wear could be experienced due to uncontrolled slip of the wheels.

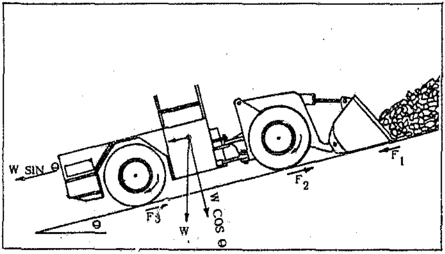


Figure 4.3 A section of an L.H.D. unit loading updip.

- The breakout force exerted by the bucket tip might be reduced (**Figure 4.4**) because of poor traction. Although the *bucket only has a capacity of 0,8m³ or 1,1 tons,* a force of approximately 4 tons could be applied to breakout uneven footwall if necessary when operating on a horizontal floor. Driver fatique would be increased *due to the additional* concentration required to fill the bucket properly and prevent wheel slip.

- It can be seen from **Figure 4.4** how a downward force of Wl, applied by the bucket lip, would cause the front drive wheels to lift off the footwall with resultant loss of traction. The friction component Fl, combined with the component W Sin Θ might equal the driving force at F3 *which would bring* the loader to a halt.

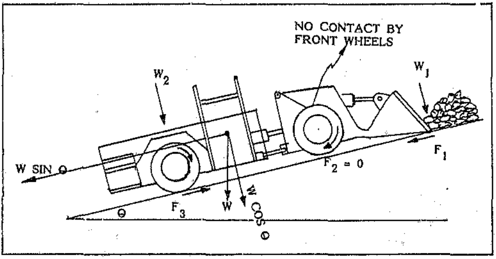


Figure 4.4 A section of an L.H.D. unit's front wheels losing contact

4.3 Downdip Cleaning Method

In the downdip cleaning method the L.H.D. unit would travel on strike along the main strike roadways between panels and start cleaning each face from the top downwards (Figure 4.5).

Sweepings would be cleaned in a similar manner. Old dangerous areas could be avoided and special attention need only be paid to the face, sweeping area and main strike roadways to protect the workmen from falls of hanging.

Full advantage could be taken of the mass of the loader during the bucket filling operation (Figure 4.6).

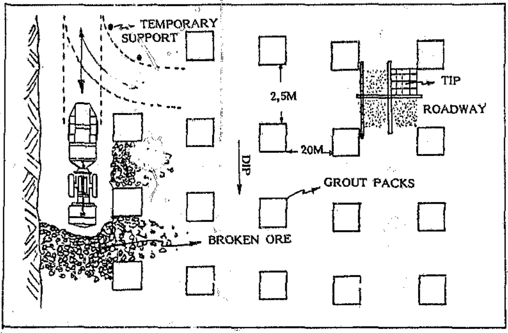


Figure 4.5 A plan of an L.H.D. unit loading downdip.

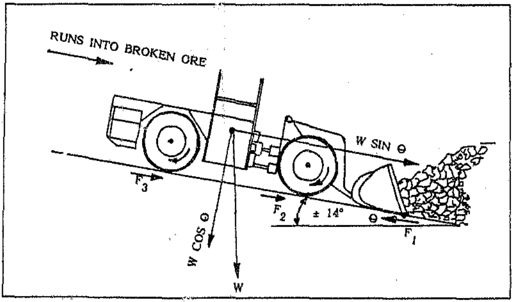


Figure 4.6 A section of an L.H.D. unit loading downdip.

The L.H.D. unit could operate under the most stable conditions at all times. Tyre wear would be reduced; fuel consumption could be improved; driver fatigue could be reduced.

Major barring of the working face could be successfully done during the cleaning operation using the outer edge of the bucket.

The bucket could also be used to dress down suspect slabs of rock from the hanging wall where the stoping width is narrow enough.

Theoretically none of the three cleaning methods was an automatic choice and experiments were to be conducted to see which worked best in practice.

5 INITIAL STOPE PREPARATION

5.1 General Description

The stope made available for the L.H.D. cleaning experiment was the 95E4 stope with a top entrance from the 95 level haulage. A travelling way from 101 level was the bottom entrance to the stope. Each stope panel was served by an orepass leading down to 101 level (**Figure 5.1**). A winze was being deepened to provide additional panels for stoping once it had holed with the workings below. The top four panels were set aside to be cleaned by the L.H.D. units. **Figure 5.1** shows the pack spacings, the general stope layout, and the position of the garage which was prepared for the service and repair of the L.H.D. units. Strike gullies were cut at an angle of 15° updip to facilitate the drainage of water and to allow for the negotiation of minor faults without getting waterlogged. Face winches were moved forward when they reached a distance of 25 metres from the face. Strike gully winches were positioned on both sides of the orepasses. Face winches pulled 2 ton scrapers with folding blades while strike winches pulled two by two ton hoe type scrapers with fixed blades. **Figure 3.8** is a section along the crosscut line indicating a conventional development layout.

Certain changes had to be made to the stope layout to accommodate the L.H.D. units. These changes are discussed below. Since it was a production stope with specific tonnages planned which had to be achieved, face and strike winches were left in position in the L.H.D. cleaned panels.

FIGURE 5.1

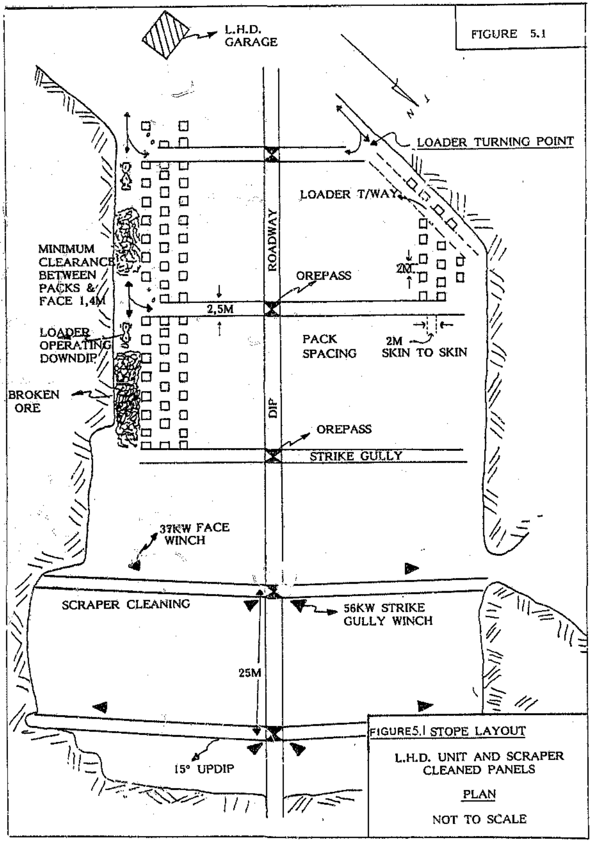


FIGURE5.1 STOPE LAYOUT

L.H.D. UNIT AND SCRAPER CLEANED PANELS

PLAN

NOT TO SCALE

Should any unforseen circumstances arise, causing a major disruption of production, the winches could be brought into service.

5.2 Garage

A safe, well supported area, large enough to house two L.H.D. units, was provided close to the working stope faces to park the L.H.D. units during the off shift **(Figure 5.2)**. This same area served as the maintenance and repair garage.

The environment, at 28°C, was relatively cool with sufficient volume of air passing up the stope to dissipate diesel engine exhaust fumes. The garage was on the return air side of the stope ventilation district so a timber barricade was erected to keep dust, created by blasting, from covering the garage work area. The position of the garage is indicated in **Figures 3.7** and **5.1**.

Maintenance equipment was purchased and installed in locked steel cabinets, conveniently situated for easy access. A small compressor was provided to pump pneumatic tyres to the correct pressure after the initially selected solid tyres had proven to be unreliable. The mine compressed air pressure was too low to satisfy tyre specifications.

Sufficient height was provided above the L.H.D. units to do major overhauls such as engine replacement and to jack up the loaders for transmission or differential repairs **(Figure 5.3)**.

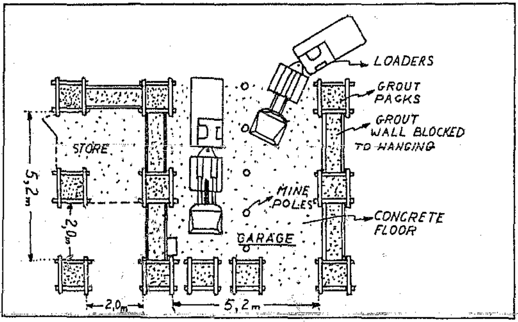


Figure 5.2 A plan of the garage and support spacing including additional support installed.

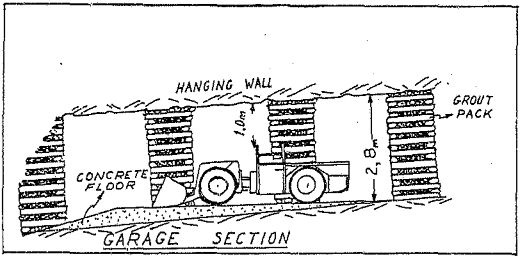


Figure 5.3 A section through the garage with clearances.

An impervious concrete floor was provided for easy cleaning of any oil or dieselene spillage.

5.3 Orepasses

The stope selected had been laid out with an orepass to serve each panel. The only alteration required was the strengthening of the grizzly at each orepass by the addition of girders in order to support a seven ton L.H.D. unit when it had to travel across the tipping point **(Figure 5.4).**

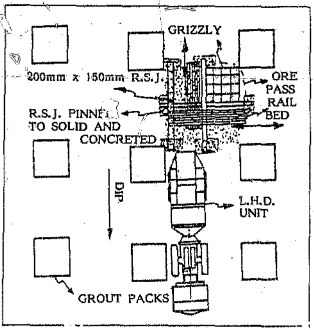


Figure 5.4 A plan indicating a two directional roadway tip construction.

The grizzly spacing was 35cm square to ensure that no large rocks could pass through and cause blockages in the box front feeding the trains. The maximum strike distance an L.H.D. would have to travel during cleaning operations would be 90 metres in one direction, since raise connections were spaced 180 metres apart on strike. Each raise was equipped with its own set of orepasses.

5.4 Roadways

The surfaces of all the roadways were filled in with fine broken rock from the stope face and levelled to ensure a fairly smooth ride and so that the driver would [illegible] control over the machine. Unnecessary shock loads to [illegible] chassis would also be avoided. One labourer was given the task of patching up the roadways and clearing any spillage away which could cause tyre damage.

The existing scraper winch strike gullies were also prepared by levelling and filling in hollows. The area was clear of faults resulting in evenly sloping strike gullies ideally suited for rubber tyred vehicles.

5.5 Support Spacing

Strike gully widths were increased by changing the pack spacing from two metre skin to skin to 2,5 metres skin to skin. This permitted an L.H.D. unit to turn into the stope face and the sweeping area (Figure 5.5). Grouted packs of 1,20 metre by 1,20 metre were used throughout the stope.

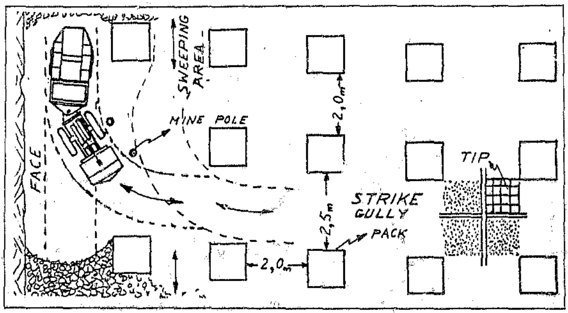


Figure 5.5 A plan view of support arrangements for L.H.D. unit turning points.

The grout pack support in the stope panel was spaced at a distance of two metres skin to skin on dip and strike. Two temporary mine poles were installed on the updip corner of the strike gully in the last row of packs in place of a grout pack which was left out to increase the space available for turning into the working face or the sweeping area. The maximum distance allowed between the last line of packs and the face was five metres. The pack line was constructed parallel to the face. A row of temporary support was only installed 1,8 metres from the face after the L.H.D. unit had completed cleaning operations, before drilling took place.

5.6 Stope Width

The stope width varied between 1,9 metres and 2,5 metres in height and was limited to a minimum of 1,9 metres for working clearance. The tabular reef seldom had any rolls and consisted of two payable bands evenly spaced with an average dip of 7°. The steepest portions of reef dipped at 15°.

5.7 Services

Air and water columns were supported above the normal travelling height of the L.H.D. unit since the units might sway over uneven roadbeds and could then damage the piping if it was left in the usual side position. Air and water columns measuring 150 millimetres and 100 millimetres in diameter respectively were installed from the haulage down the central roadway. Branch pipes 50 millimetre in diameter were installed in each strike gully.

All services were carried a standard minimum distance of 10 metres from the working face.

6 TRAINING

6.1 Artisan

A fitter was selected for his good maintenance record and sent to the L.H.D. unit suppliers to be familiarized with the different loader components during the assembly of the L.H.D. units.

He had the full time job of attending to the units, setting up a suitable garage in the stope, stating all requirements to effectively maintain the loaders according to a set planned maintenance programme worked out by the suppliers.

Scrupulous application of the planned maintenance programmes was emphasised. The fitter was assisted by two helpers on a part time basis in the case of a major breakdown.

The familiarization period with the L.H.D. units lasted approximately three months. It was important to be able to diagnose faults such as loss of power, failure of electrical or hydraulic systems.

6.2 *Operators*

It was planned to ultimately operate two L.H.D. units on three shifts. With this in mind, six operators were selected for training. Initially the L.H.D. units would be used on day shift only, then the cleaning operation would be expanded to include night shift cleaning. Three operators per loader would take turns at operating a L.H.D. unit during the day shift until they were competent.

Selection of the operators was based on mechanical ability and leadership qualities. These abilities were judged according to standard tests which were already being used for the selection of loco drivers and gang supervisors. It was preferable to select trainee operators who were licenced vehicle drivers with past experience on diesel locomotive maintenance. This however was not an essential criteria. During the past two years (1985 - 1987) greater emphasis has been placed on the selection of operators. The standard PUTCO transport test is now used. In addition the phsychological suitability of an operator is also tested. Perceptivity tests are conducted to determine a potential operator's understanding, patience, aggressiveness and mechanical dexterity. The results of these tests are examined by a phsychologist who recommends operators for further training.

6.3 Surface Training

An area was first selected on surface to get the trainees accustomed to the L.H.D. unit controls. Drums were placed in a specific pattern spaced to simulate the support pattern underground and limit the turning circle. The operators drove around the drums until they could confidently handle the L.H.D. unit controls.

Then as part of their training the operators assisted the fitter with the services and were responsible for greasing the L.H.D. units. The operators were taught the basis of operation of the hydraulic systems, the importance of using clean receptacles for filling the hydraulic oil tanks and of making regular checks.

After a three week training period on surface, during which time several mechanical problems were sorted out on the L.H.D. units, the loaders were taken underground to the experimental stope.

6.4 Underground Training

In the stope even the best operators had difficulty in adjusting to the confined space, uneven roadways and dip of the reef. The loaders bounced and swayed, causing over-reaction in steering with resultant collision with the support and the stope face. Constant supervision had to be exercised under slow speed operation for a further three weeks before any sort of confidence became apparent. The operators became more relaxed, permitting the driving period to be increased steadily from half hour to four hour sessions before being relieved.

The loading rate increased and results showed that the winches which were operating in some faces on a standby basis to ensure that the faces were cleaned, supported and drilled, could be stopped completely.

During the training period special attention was paid to the method of filling the L.H.D. unit bucket, power applied to lift the bucket and the forward motion necessary to avoid slip and excessive tyre wear.

Both L.H.D. units were then operated on two shifts, using three operators per shift, with one acting as a relief operator for either of the units.

7 PRODUCTION ESTIMATES

The specific gravity of gold bearing ore is generally accepted as 2,67 tons per cubic metre, although there is a variation depending on the metal and sulphides content. This factor was initially applied to all tonnage calculations in 1976. Some years later this factor was increased to 2,75 tons per cubic metre as being more applicable for this mine.

The swell factor was taken to be 1,67 as commonly used in the calculations of hopper capacities and tonnage duties. This gave a factor of 1,44 tons per cubic metre transported with a fully loaded bucket. (Struck capacity.) For these production estimates an average of 1,1 tons per bucket load was used taking into account a bucket fill factor of 78.6 percent.

7.1 Economical Operating Range - Downdip Loading With a Schopf L62 L.H.D. Unit

CYCLE TIME :	*Average for a day*	=	2,26 minutes
	Distance tip to face	=	30 metres
	Face Length	=	30 metres
	Average tipping time	=	0,4 minutes
	Average loading time	=	0,6 minutes
	Average travelling time	=	1,26 minutes
	Average speed	=	4,28 km per hour
	Average shift length	=	480 minutes (waiting place to waiting place).

7 PRODUCTION ESTIMATES

The specific gravity of gold bearing ore is generally accepted as 2,67 tons per cubic metre, although there is a variation depending on the metal and sulphides content. This factor was initially applied to all tonnage calculations in 1976. Some years later this factor was increased to 2,75 tons per cubic metre as being more applicable for this mine.

The swell factor was taken to be 1,67 as commonly used in the calculations of hopper capacities and tonnage duties. This gave a factor of 1,44 tons per cubic metre transported with a fully loaded bucket. (Struck capacity.) For these production estimates an average of 1,1 tons per bucket load was used taking into account a bucket fill factor of 78.6 percent.

7.1 Economical Operating Range - Downdip Loading With a Schopf L62 L.H.D. Unit

CYCLE TIME :	*Average for a day*	=	2,26 *minutes*
	Distance tip to face	=	30 metres
	Face Length	=	30 metres
	Average tipping time	=	0,4 minutes
	Average loading time	=	0,6 minutes
	Average travelling	=	1,26 minutes
	Average speed	=	4,28 km per hour
	Average shift length	=	480 minutes (waiting place to waiting place).

Expected average loading shift	=	384 minutes (At the face)
Expected average trips per shift	=	170 trips
Average tons per trip	=	1,1 tons
Expected tonnage per shift	=	187 tons

At the time the maximum one way distance was taken to be 90 metres, being the normal half way *limit between raise connections,* although 150 metres one way distance was still considered to be within the economical operating range of the L.H.D. units. It can be seen that for a one way distance of 100 metres, 18 tons per hour could be achieved according to the Wagner Mining Equipment production table below **(Table 7.1)**, at an average speed of 4 kilometres per hour.

Wagner Mining Equipment Co. Calculation:

TABLE 7.1 *HST-1A Production* rate in metric tons (50 minute hour) per hour.

One Way Distance Metres	4km/hour	6km/hour	8km/hour	10km/hour
50	30	38	44	
75	22	30	35	40
100	18	24	30	34
150	13	18	22	26
200	10	14	18	21
250	8	12	15	18

Therefore at 4 km per hour and a one way distance of 100 metres, the production for an eight hour shift would be 144 tons or 192 tons per shift at an average speed of 6 km per hour.

The average operating speed would be low during the ledging stage. The short distance between the loading point and the tipping point would not permit the operator to change gears into a higher range. At the extremity of the stope area, the travelling speed could possibly increase due to the longer strike travelling distance.

8 WORK STUDY TECHNIQUES

The work study team doing the complete study initially consisted of an Operation Research Engineer, a Research Mine Overseer, a Research Shiftboss and six Observers. The number of Observers was increased to take readings on additional shifts.

The Operations Research Mine Overseer and Shiftboss duplicated the readings taken by the Observers during the first three week period as a check and to ensure that no further training of Observers was necessary.

Readings taken by the Observers proved to be very reliable, so spot checks only were done from time to time. The cleaning efficiency for operating strike and face winches was taken by doing rated activity sampling for two weeks, determining actual working time and manpower requirement.

Rated time studies were done on the L.H.D. units and Operators, after the Operators had become proficient at their task, to compare results with winch cleaning results and efficiencies.

Daily records, such as trips, loads trammed, travelling time, tonnage cleaned, delay time, operating time, idle time, breakdowns, services, material used and tyre circumference measurements were taken as a routine to be summarized weekly and monthly. For consistency, tyre circumferences were measured after inflating the tyres to the same pressure.

Method study was done continually. Loading techniques and routines were changed on an ongoing basis after deficiencies were determined.

9 OBSERVATIONS

9.1 Operators

The L.H.D. unit operators became skilled in manoeuvring the L.H.D. units along narrow roadways and around sharp corners. Operator fatigue initially proved to be a problem, partially because of physical exertion, but mainly because of mental strain and nervous tension caused by the concentration required to judge widths, elevations, power and speed.

The judgement of the operators improved considerably with experience and this enabled them to fill the buckets in the least possible time with the minimum of damage to the machines and tyres.

9.2 Cleaning Operations

A few metres of working face can be cleaned updip of the strike gully without any real problem, provided the back wheels of the L.H.D. units are still on the level floor of the strike gully. The back wheels are less prone to spinning because of the compacted rock surface. On the updip ledge the broken rock on the smooth footwall caused a loss of traction.

The method of filling the bucket of the L.H.D. unit needed much practice to make full use of the power of the L.H.D. unit, and of gravity where possible.

Too great a downward force tended to lift the front wheels off the rock surface, decreasing traction progressively to zero on the front wheels which spun in the air, and reducing the driving force available to move the bucket into the broken ore.

The same effect occured when loading downdip. Just before the front wheels lifted clear of the surface, the applied tractive power caused the wheels to spin, increasing the wear rate.

The only downward force required was that amount necessary to prevent the bucket from being deflected upwards without taking in broken ore. The bucket had to be tilted forward slightly so that only the front lip was in contact with the floor, thus reducing the friction area and subsequent chance of wheel spin.

As the L.H.D. unit moved forward, the bucket, when in contact with broken ore, had to be lifted slightly in a scooping action and then forced down again slightly until the bucket had been filled. The lifting movement lifted rocks anchored deeper in the broken ore and then the bucket was forced under the rocks using less force than would have been necessary otherwise.

Large pieces of hanging wall which may have been dislodged during the blasting of the face, caused a problem, when buried under the broken ore. The bucket tended to deflect off the top of the rock, merely sliding over the rock. The loader had to be moved back first. The front edge of the rock was then exposed and the lip of the bucket could be forced under the rock.

The rock could then be lifted clear of the broken ore. Rapid wear of the bucket lip and bottom corners took place. A second spare bucket was purchased to facilitate timeous repair work.

All the leading edges of the repaired buckets were hard faced - using special welding rods and Benox steel - skid plates were welded onto the bottom surfaces to prolong bucket life. This method of protection is generally used with loading equipment.

The powerful hydraulic breakout force available at the bucket was sometimes used to lift large rocks out of the broken ore, and to break out uneven footwall which could cause a problem during loading operations. The units were also used to clear the hanging wall of loose rock and to bar the stope face.

Traction was improved over wet surfaces when the bucket of the L.H.D. unit was fully loaded, but occasionally dry coarse material would have to be placed on the roadway to assist the wheels to grip.

The loaders were occasionally run in reverse dragging the bucket on the floor, leaving it quite smooth and in this way assisting with tyre protection. After every trip to the tip during loading operations, it was essential to lower the bucket and clear the approach to the muck pile of loose rock. This action prevented damage to tyres and the bottom of the L.H.D. unit.

There was fortunately no clay material, talc schist or vermiculite present in the ore, which would have hampered loading operations. The above type of material causes wheel slippage with only slight application of power.

When loading down dip the momentum of the L.H.D. unit assisted with the rapid filling of the bucket giving an average loading time of approximately 0,6 minutes per bucket, although filling times of 5 seconds per bucket have been recorded. After the removal of the broken ore in the face, the bucket of the L.H.D. unit was run against the stope face to bring down loose rock before manual barring started (**Figure 9.1**). During the downdip loading operation the minimum amount of power was used for filling the bucket and driving the loader forward. The shortest bucket filling times were achieved using this method.

During the time that manual barring of the face was being done, the L.H.D. unit started clearing away the scattered rock behind the first row of packs from the face.

The lack of scatter barricades (**Figure 9.2**), resulted in an increase in cleaning time because of the widely spread broken rock. With the broken rock confined on the face by a sturdy scatter barricade, the loading time would have been reduced and the sweeping area would not have been contaminated. The erection of scatter barricades constructed of slab lagging and mine poles are labour intensive and time consuming, especially at wide stope widths. An alternate blasting barricade had to be developed but was not done by the time the Schopf loader experiment was stopped.

Figure 9.1 The bucket of the L.H.D. unit [illegible] against the stope face to assist with barric[illegible] of loose rock. The first pack on the corner was left out for turning purposes. (Stope width 2,5m)

Figure 9.2 Large fall of hanging being cleared away in the sweeping area.

After the stope face had been barred completely solid, the L.H.D. unit cleared away the loose material against the face for final face preparation to be done (**Figure 9.3**). The curved face shape was not ideal and was caused by shorter holes being drilled at the bottom and top of the face as a result of inclined drilling. Damage was caused to the tyres by the sloping bottom portion of the face. The bottom portion of the face should either be vertical or sloping away to improve tyre life.

The L.H.D. unit then moved into the next blasted face to commence cleaning operations while the recently cleaned stope face was again being watered down, with sockets cleaned and plugged as per standard.

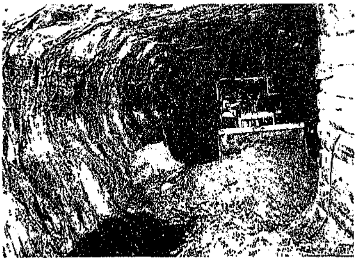


Figure 9.3 Final clearing of material after barring of the face. (Notice curvature of the face.)

9.3 Permanent Support

Due to the fact that all broken ore between the last row of permanent pack support and the face could be removed with relative ease, the next row of permanent support could be installed on a cleanly swept footwall. This prevented a lock-up of gold under the packs, also spillage of cement during the filling of the grout packs could not lock-up substantial layers of gold bearing broken rock, which does happen in scraper cleaned stopes. The standard is to sweep the footwall in the position where a grout pack has to be installed. Due to inadequately scraper cleaned faces, the pack positions were opened by hand lashing away broken rock. Sweeping the footwall in these positions proved to be a problem.

A further advantage was that the packs were installed at a faster rate, since it was not necessary to dig holes into the broken ore, left by the scrapers, to make space for the grout packs.

Pneumatic picks were used to get rid of cement spillage and expose solid rock. Grout leaking through the grout bags and spillage, caused by over filling a grout bag or moving from one pack to fill the next, penetrated the fine broken rock and scatter piles. The footwall was effectively, but undesirably, being concreted over.

9.4 Sweeping

After cleaning the stope faces the L.H.D. units moved into the sweeping area behind the first row of packs from the face. The finely broken rock, scattered in a thin layer over the footwall, was collected into heaps using shovels and bass brooms for loading by the L.H.D. units.

This reduced the loading rate of the loader to 15,4 tons per hour whilst loading the heaps of fines.

Final sweepings were cleared by stopping the L.H.D. unit with the bucket on the ground and then hand lashing the last few shovels full of fine material directly into the bucket and then transporting it to the orepass.

9.5 *General*

To ensure that sufficient broken ore was continually available for both loaders over the double shift, four 30 metre long stope panels with a stope width of 2,5 metres had to be blasted daily.

During the cleaning operations the relief operator constantly searched the broken ore for broken drill steel which could damage the tyres **(Figure 9.4)**. It can also be seen that the bucket was not filled properly with the first scoop and needed to be tilted back to allow space for a heaped load at the second scoop. The loading rate and bucket filling ability improved as the broken rock pile became higher.

The strike cleaning method was not attempted because there were too many packs out of line which blocked the roadways. The updip loading method initially had to be used to clean a top panel because there was no top access to the panel for the loaders.

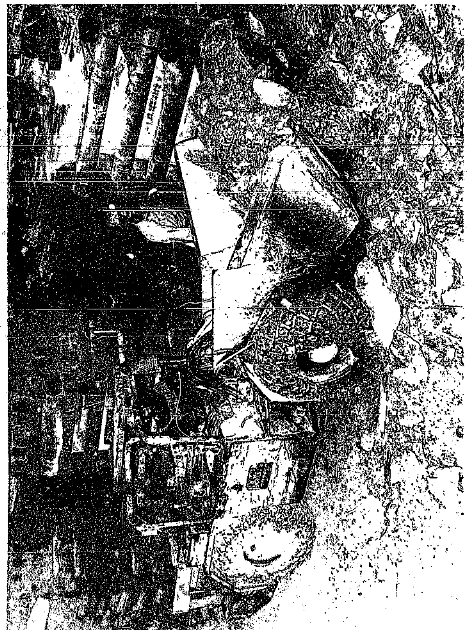


Figure 9.4 A Schopf L62 L.H.D. unit in operation with an operator checking for broken drill steel. The grout fill pack support (1,2m x 1,2m) can be seen in a stoping width of 2,5m.

The updip cleaning of the one panel continued for approximately two months. The loading rate, at 19 tons per hour, was poor and excessive wheel slippage occurred in spite of very careful attempts by the operators to control tractive power. Sometimes the L.H.D. units slipped downdip as the bucket was being filled. Uneven footwall due to irregular blasting accentuated the dip and had to be levelled to allow improved loading rates. The updip loading method was not used again.

The downdip loading method was certainly the most suitable method found during the weeks that followed. All travelling to the tipping point was done with the bucket leading. This had the advantage that the L.H.D. unit did not have to cross the tipping point or turn at the tipping point, where space was confined, to be able to tip the load. The disadvantage was that rock falling from an overloaded bucket could fall in the tyre path, causing damage to the tyres or causing the loader to be deflected against the packs. Care was taken in the stope face to shake the loaded bucket a few times while reversing up to the top strike roadway, getting rid of most of the loose unbalanced rock which could cause problems.

The turning point at the face was wide enough to enable a rapid change from reverse direction to forward direction without risk of hitting temporary support.

The only time the L.H.D. unit travelled to the face with the bucket leading was directly after the blast, when the turning point at the intersection of the strike roadway and the face had to be cleared.

9.6 Downdip Cleaning Ability

It became clear that the L.H.D. units could be used efficiently to clean out winzes or incline shafts if the dip did not exceed 17 degrees. A winze was being sunk at the bottom of the stoping area and only a metre advance was being achieved every second day due to scraper cleaning difficulties.

Broken rock in the winze had build-up to an average depth of 0,7 metres and the scraper was starting to damage the ventilation in places.

It was decided to use the L.H.D. unit to clean out the winze, starting at the entrance, to get down to the footwall. Once the excess broken rock had been cleared out, a metre advance per day was achieved in addition to cleaning the stope faces. Cleaning operations, however, were interrupted due to loader downtime.

The air and water services at the tipping points were lifted to provide tipping clearance and the packs spaced further apart to allow for a roadway to the opposite faces. Short lengths of hose were hung as warnings for people to prevent them from inadvertently stepping onto the grizzley **(Figure 9.5).**

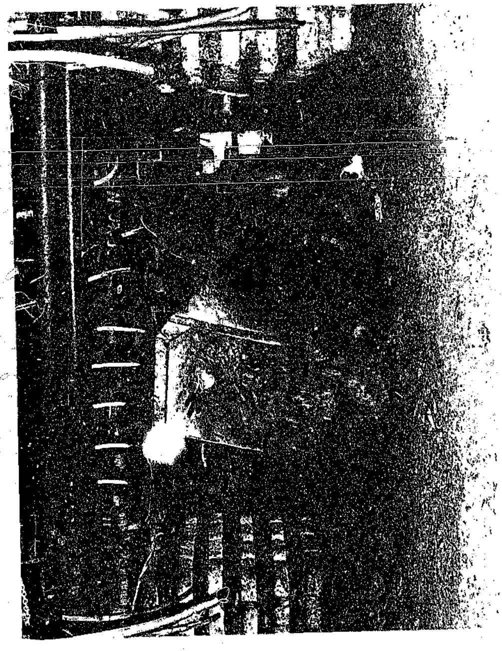


Figure 9.5 The L.H.D. unit in tipping position.

9.7 Tyres

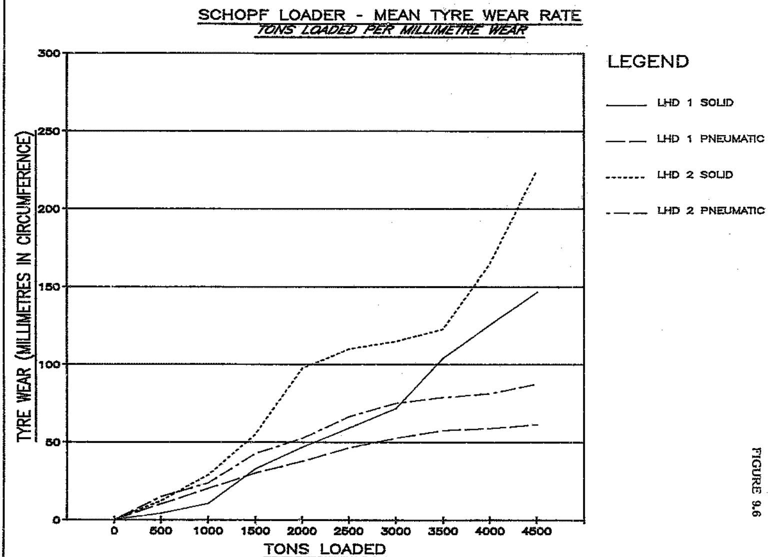
Solid rubber tyres were requested and supplied with the L.H.D. units because it was feared that punctures would be a major problem with pneumatic tyres. The possibility of chassis damage, because of the limited shock absorbing properties of the solid rubber tyres, was considered, but it was decided that the chassis was robustly constructed and would in fact withstand normal shock loads. An examination of the L.H.D. unit chassis at the end of the experiment showed no cracks developing.

Due to experience in other trackless mining operations in the country, loader operators were supplied with kidney belts to prevent health problems from developing.

The circumference of each tyre was measured when an L.H.D. unit was serviced. This means of indicating tyre wear was selected since a more accurate measurement could be taken of the tyre circumference than of either the radius or the diameter. The solid rubber tyres showed a good wear rate initially as can be seen in **Figure 9.6.** There was a steady reduction in tyre circumference until,in the case of L.H.D. unit No. 1, approximately 125mm in circumference had been worn away, after which a rapid deterioration of the tyres occurred. Large portions of the tyres were ripped off. After inspection it was found that the outer shell of the solid tyres had been properly cured, but the inner shell lacked bonding.

Another problem that manifested itself was that the differentials of the L.H.D. units started breaking down due to the unequal wheel radius caused by rapid tyre wear and long operating hours under these conditions.

FIGURE 9.6


SCHOPF LOADER - MEAN TYRE WEAR RATE
TONS LOADED PER MILLIMETRE WEAR
LEGEND
LHD 1 SOLID
LHD 1 PNEUMATIC
LHD 2 SOLID
LHD 2 PNEUMATIC
TYRE WEAR (MILLIMETRES IN CIRCUMFERENCE)
300
250
200
150
100
50
0
0
500
1000
1500
2000
2500
3000
3500
4000
4500
TONS LOADED

Care was subsequently taken to match the tyre circumferences either for both back wheels or for both front wheels. New tyres were replaced in pairs.

Down time due to tyres during the first five operating months amounted to 35,5 percent of all breakdown time. An average of 5 266 tons were loaded per set of four solid tyres before these were discarded at 141mm decrease in circumference.

It was decided to discard the solid rubber tyres and experiment with pneumatic tyres instead, but the pneumatic tyres would have to be protected against punctures in some way.

There were three main methods of protection, namely:-

- A polyurethane filler which would harden inside the tyre excluding all air, but still remain pliable.
- A rubber solution additive was available, which would remain fluid inside the tyre and seal off small holes.
- Chain tyre treads which were available in different forms, either as a loose fit or embedded in the tread rubber.

It was decided to have all the pneumatic tyres clad with a chain tread embedded in rubber to prevent movement of the chain tread and to thicken the rubber of the tread. This would also improve the wear rate of the tyres and provide an added protection against punctures.

The chain prevented cutting of the tyres by sharp quartzite slabs and avoided pieces of rubber being ripped off. The chain mesh was small enough to prevent punctures due to sharp rock, except for very narrow slivers of rock which had to enter the tread at right angles to be able to do damage. It was found that steel wire, jumpers and pinch bars caused some punctures in spite of the chain. An experiment was conducted using a solution inside the pneumatic tyres, with some success. In 1976 the initial cost of the tyre, chain and solution was high at R775,00 per wheel.

Flat mild steel radial ribs in the chain tread proved to be the weak links, and had to be redesigned to increase the life of the chain. The radial ribs in the chain protruded slightly above the other chain links and were consequently subjected to a greater load, which caused more rapid wear to the tyre as the ribs were being depressed into the moulded rubber. They also tended to be pushed off the vertical position by the force of the revolving wheels. The chain had to be tensioned daily to compensate for wear and bedding into the rubber.

Replacement of broken chain links was simple, as the complete chain could be removed without much trouble. Although embedded in the rubber the chain could be lifted out. A complete chain overhaul, in 1976, would cost R150,00 per wheel.

The thickness of the mild steel chain ribs was increased from 13mm to 16mm and the surface contact area was "hardfaced" to reduce the wear rate.

The repaired wheels cost an average of R747,00 per wheel, in 1976, but gave trouble free loading, only requiring some tensioning of the chains during the next three months. At this stage the experiment was discontinued, so that a complete costing of the alterations could not be done.

Delays caused by punctures, chain adjustments and tyre wear were summarized for the period 06-12-1975 to 14-05-1976, amounting to 6,61 percent of total breakdown time during the same period (Table 9.1) based on Tables 1 to 6 in Appendix A.

Table 9.1 Schopf Loader Breakdowns

	Breakdowns	Total Time (Minutes)	Percentage
1.	Transmission, differential	65 857	42,63
2.	Engine	41 559	26,90
3.	General (Body)	17 231	11,15
4.	Hydraulic	12 510	8,10
5.	Wheel repairs	10 214	6,61
6.	*Electrical*	7 128	4,61
	Total	**154 499**	**100,00**

The tyre chains were carefully examined to determine alterations required to increase the operating life. It was found that the ribs were wearing at an angle on the tread, and around the punched holes for links (Figure 9.7).

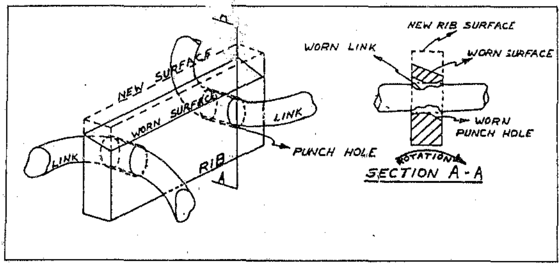


Figure 9.7 Rib chain link system.

The edges of the punch hole wore away the link, which eventually snapped. A better tyre chain was developed by another company and this has been working well on 5 cubic metre capacity L.H.D. units at Premier Diamond Mine. No chain clad tyres are being used in any of the most recent trackless mining projects.

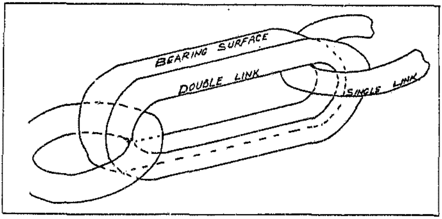


Figure 9.8 Sketch of tyre chain link.

The double link tread of the tyre chain wears evenly and there are no sharp edges where rap.d wear could take place, compared with the punched holes in ribs of the previous chain. The double link tyre chain could be turned inside out which could increase the useful life.

From **Figure 9.6** it can be seen that the wear rate of chain clad tyres was less than the wear rate of the solid tyres. A further reduction in wear rate took place after some 2 500 tons had been lashed.

The second reduction in wear rate appeared to take place after the protruding rib surfaces had worn down to the level of the rubber tread which reduced the load on the ribs.

Table 9.2 shows the erratic solid rubber tyre circumferential wear rate and **Table 9.3** shows the chain clad pneumatic tyre circumferential wear rate which was more predictable.

Table 9.2 Mean wear of solid rubber tyres related to tons loaded

	LOADER 75027	LOADER 75028
Tons Loaded	Mean Tyre Wear	Mean Tyre Wear
500	3,75mm	12,50mm
1 000	10,00mm	28,75mm
1 500	32,50mm	55,00mm
2 000	46,25mm	97,50mm
2 500	58,75mm	110,00mm
3 000	71,25mm	115,00mm
3 500	103,75mm	122,50mm
4 000	125,00mm	163,75mm
4 500	146,25mm	225,00mm

Table 9.3 Mean wear of chain clad pneumatic tyres related to tons loaded

Tons Loaded	LOADER 75027	LOADER 75028
	Mean Tyre Wear	Mean Tyre Wear
500	10,00mm	15,00mm
1 000	20,00mm	23,75mm
1 500	30,00mm	42,50mm
2 000	37,50mm	52,50mm
2 500	46,25mm	66,25mm
3 000	52,50mm	75,00mm
3 500	57,50mm	78,75mm
4 000	58,75mm	81,25mm
4 500	61,25mm	87,50mm

9.8 Breakdowns

Table 9.1 shows breakdowns grouped under six main headings while breakdowns listed in detail in diminishing order are shown in **Table 9.4.** Transmission, differential, drive shaft and related problems amounted to 42,63 percent of the total breakdown time, which in turn was 48,5 percent of the total operating time. Planned maintenance of the above units did not include the dismantling of the units, which would only be done when the whole loader was being overhauled. In most cases the delays caused by breakdowns would have been less if spare parts had been readily available.

Delivery delays of spare parts necessitated the purchase of a large buffer stock which could only be justified if the L.H.D. units were to be used on a permanent basis.

Table 9.4 Schopf Loader Breakdown Summary

	Breakdown	Total Time (Minutes)	Percentage
1.	Engine Repairs	26 979	17,46
2.	Gearbox repairs	24 843	16,08
3.	Universal joint repairs	18 230	11,80
4.	Oscillating bearing repairs	13 455	8,71
5.	Starter repairs	10 427	6,75
6.	Wheel repairs	10 214	6,61
7.	Brake repairs	9 000	5,83
8.	Drive shoft repairs	6 708	4,34
9.	Loader repairs	6 484	4,20
10.	*Burnt electrical couplings*	3 655	2,37
11.	Tightening bolts	3 517	2,28
12.	Hydraulic repairs	3 510	2,27
13.	Battery trouble	2 922	1,89
14.	Broken yoke	2 847	1,84
15.	Broken pump drive	2 826	1,83
16.	Differential repairs	1 670	1,08
17.	Steering repairs	1 280	0,83
18.	Canopy repairs	974	0,63
19.	Gear lever repairs	896	0,58
20.	General repairs	778	0,50
21.	Broken fan belt	626	0,41
22.	Alternator repairs	421	0,27
23.	*Fitting shovel*	378	0,24
24.	Exhaust repairs	362	0,23
25.	Replacing protection plate	346	0,22
26.	Bucket arm repairs	305	0,20
27.	Replacing fuel filter	300	0,19
28.	Fitting bucket pins	242	0,16
29.	Ignition switch repairs	130	0,08
30.	Bumper repairs	80	0,05
31.	Leaking transmission oil	55	0,04
32.	Broken water pipe	39	0,03
	Total	**154 499**	**100,00**

Gearbox breakdowns, amounting to 16,08 percent of the breakdown time, would normally not have occurred, but these were caused by the continuous slow speed operations in the stopes. Lubrication of all the parts in the gearbox depended upon oil being scooped up and spread over the gears. Due to the slow operating speed the revolutions of the gears was not high enough to lift the oil to the top of the gearbox above the gear clusters. Later models had pressurized oil feed to the gearbox bearings.

The above problem was alleviated by installing grease nipples in some areas and by increasing the level of oil in the gearboxes

The differential problems were caused by the unequal circumference of pairs of wheels when solid tyres were used, resulting in high stresses on the differential. Operating temperatures of the differentials increased above normal. A universal joint was unfortunately damaged by a rockdrill stem which was lifted by a front wheel and jammed into the universal joint.

The oscillating bearings seem to be a weakness, causing 8,71 percent of the breakdown time. These bearings were not easy to obtain, with consequent repair delays. Engine breakdowns amounted to 26,9 percent of the total breakdown time, while engine repairs alone accounted for 17,46 percent.

The air cooled, 4 cylinder, 4 stroke Deutz Diesel engine is commonly used as a power plant by most L.H.D. unit manufacturers, and is very reliable. However, rapid wear of pistons and piston rings was caused by poor intake air filtration and some lack of regular changing of the filter units. Fine quartzite dust must be prevented from getting into the engine due to the highly abrasive nature of the quartzite.

Special attention needs to be paid to the filtration of intake air and also to allaying dust by watering down. The air filter elements had to be cleaned out daily by blowing out the dust with compressed air, which may have caused some damage to the elements before replacement. Some channeling was observed in the filters before the elements were removed, and this would decrease the efficiency of the filters.

Most of the breakdowns in the "General" category were caused by the operators misjudging distances during turning and forcing the L.H.D. units through narrow passages. Some damage was caused by falls of hanging during loading operations.

No deterioration was observed in the chassis, although it was expected that the solid rubber tyres would cause some cracks to develop, since the chasis was unsprung.

A spare reconditioned bucket was kept available for quick exchange when the leading edge and corners were worn down.

Steering and hoisting cylinders needed dust covers to assist with keeping abrasive quartzite dust clear of the steering piston shafts, although wiper seals on the shafts were a standard item.

Brake repairs amounted to 5,83 percent of the breakdown time, mainly due to *leaking hydraulic pipes and cylinders.*

The hydraulic hoses were generally reliable, but still caused 2,27 percent of the breakdown time due to failure at the crimped ends. Great care was taken to keep the hydraulic fluid clean when refilling the hydraulic tank, in spite of the fact that 5 micron filters were fitted to the filler throat.

Burnt electric wire couplings caused 2,37 percent of the breakdown time and needed to be redesigned.

Battery trouble listed here was caused by defects in the batteries. This was excluded from delays caused by flat batteries as a result of the operator not switching off the lights with the engine stopped, or excessive cranking when the engine needed attention.

Table 9.5 lists the weekly working and travelling time, service time, *delay time and breakdown time for the period* 1 August 1975 to 14 May 1976.

Steering and hoisting cylinders needed dust covers to assist with keeping abrasive quartzite dust clear of the steering piston shafts, although wiper seals on the shafts were a standard item.

Brake repairs amounted to 5,83 percent of the breakdown time, mainly due to leaking hydraulic pipes and cylinders.

The hydraulic hoses were generally reliable, but still caused 2,27 percent of the breakdown time due to failure at the crimped ends. Great care was taken to keep the hydraulic fluid clean when refilling the hydraulic tank, in spite of the fact that 5 micron filters were fitted to the filler throat.

Burnt electric wire couplings caused 2,37 percent of the breakdown time and needed to be redesigned.

Battery trouble listed here was caused by defects in the batteries. This was excluded from delays caused by flat batteries as a result of the operator not switching off the lights with the engine stopped, or excessive cranking when the engine needed attention.

Table 9.5 lists the weekly working and travelling time, service time, delay time and breakdown time for the period 1 August 1975 to 14 May 1976.

Table 9.5 Schopf Loaders - Weekly Operating Summary

Week Commencing	Working and Travelling Time (Mins.)		Service Time (Mins.)		Delay Time (Mins.)		Breakdown Time (Mins.)		Total Time
	Time	Percent	Time	Percent	Time	Percent	Time	Percent	
01.08.75	2 516	50,1	251	5,8	356	8,2	1 211	27,9	4 334
08.08.75	3 295	83,9	59	1,5	126	3,2	446	11,4	3 926
15.08.75	4 076	79,3	311	6,1	339	6,6	411	8,0	5 137
22.08.75	2 945	76,0	156	4,0	158	4,1	615	15,9	3 874
2' 75	3 002	73,6	515	12,6	349	8,5	216	5,3	4 082
05.09.75	4 191	77,7	496	9,2	348	4,6	457	8,5	5 392
12.09.75	4 879	78,8	1 115	18,0	112	1,8	85	1,4	6 191
19.09.75	6 197	84,0	546	7,4	79	1,1	556	7,5	7 378
26.09.75	4 741	55,5	996	11,7	353	4,1	2 455	28,7	8 545
03.10.75	3 007	25,2	1 349	11,3	126	1,1	7 442	62,4	11 924
10.10.75	4 512	41,0	241	2,2	246	2,2	6 013	54,6	11 012
17.10.75	4 526	39,3	442	3,8	516	4,5	6 026	52,4	11 510
24.10.75	5 606	47,2	217	1,8	918	7,7	5 156	43,3	11 897
31.10.75	6 452	56,7	253	2,2	507	4,5	4 159	36,6	11 371
07.11.75	6 126	54,2	306	2,7	518	4,6	4 353	38,5	11 303
14.11.75	5 492	48,3	251	2,2	501	4,4	5 135	45,1	11 379
21.11.75	4 293	39,0	285	2,6	421	3,8	6 010	54,6	11 009
28.11.75	2 590	31,7	276	3,4	497	6,1	4 806	58,8	8 169
05.12.75	2 596	31,6	202	2,5	876	10,7	4 551	55,2	8 225
12.12.75	4 537	40,5	294	2,6	169	1,5	6 202	55,4	11 202
19.12.75	3 596	39,1	277	3,0	1 044	11,4	4 273	46,5	9 190
26.12.75	3 284	37,7	185	2,1	868	10,0	4 378	50,2	8 715
02.01.76	4 461	39,8	486	4,3	3 100	27,6	3 178	28,3	11 225
09.01.76	4 630	40,5	1 275	11,2	2 528	22,1	2 998	26,2	11 431
16.01.76	5 230	45,9	992	8,7	1 520	13,3	3 662	32,1	11 404
23.01.76	5 887	51,5	542	4,7	1 852	16,3	3 146	27,5	11 427
30.01.76	5 529	48,6	927	8,2	800	7,0	4 115	36,2	11 371
06.02.76	4 936	42,3	1 031	8,8	677	7,5	4 836	41,4	11 680
13.02.76	4 583	39,8	542	4,7	1 193	10,4	5 201	45,1	11 519

Week Commencing	Working and Travelling Time (Mins.)		Service Time (Mins.)		Delay Time (Mins.)		Breakdown Time (Mins.)		Total Time
	Time	Percent	Time	Percent	Time	Percent	Time	Percent	
20.02.76	6 648	58,2	845	7,4	816	7,1	3 125	27,3	11 434
27.02.76	3 872	33,9	648	5,7	281	2,5	6 623	57,9	11 434
05.03.76	5 357	46,6	818	7,1	517	4,5	4 813	41,8	11 505
12.03.76	7 258	55,5	1 244	9,5	1 340	10,3	3 225	24,7	13 067
19.03.76	5 567	37,3	1 064	7,2	1 241	8,3	7 037	47,2	14 909
26.03.76	5 146	34,3	906	6,0	362	2,4	8 621	57,3	15 035
02.04.76	3 457	23,0	419	2,8	337	2,2	10 815	72,0	15 028
09.04.76	1 447	9,6	159	1,1	14	0,1	13 360	89,2	15 000
16.04.76	1 867	15,0	205	1,6	92	0,7	10 280	82,7	12 444
23.04.76	3 736	24,5	485	3,2	121	0,8	10 926	71,5	15 268
30.04.76	4 011	26,6	468	3,1	65	0,4	10 512	69,9	15 056
07.05.76	2 864	19,1	508	3,4	70	0,5	11 589	77,0	15 031
14.05.76	1 255	8,4	409	2,7	-	-	13 327	88,9	14 991
Total	208 200	40,4	22 996	5,2	26 463	5,9	216 365	48,5	446 024

NOTE : The *L.H.D.* units were not operated *consistently on three* shifts due to an occasional shortage of operators, resulting in a variation in total time.

9.9 Loader Operating Summary

The total times reflect the variation in shifts worked by the two loaders, starting off with a single shift operation, going over to a double shift in October 1975 and then to a treble shift operation in March 1976.

On average, both loaders worked for 40,4 percent of the total time of the test period.

The main reason for the short working time was the excessive breakdown period and delays.

From the operating percentage per week indicated in **Figure 9.9**, it can be seen that the breakdown time directly affected the operating time.

Initial working time losses were caused by the unexpected failure of solid tyres and the changeover to pneumatic tyres in the ninth week of operation. This was followed by engine breakdowns in the eighteenth week and gearbox breakdowns in the nineteenth week, as can be seen from the daily analysis of the Schopf Loaders **(Appendix A)**.

The engine of the L.H.D. units started failing completely after the thirtyfifth week of operation, with progressive loss of power reducing the working time. The loss of power was mainly due to the high wear rate c. .he rings and cylinders caused by the abrasive dust being sucked in past the filters. This emphasised the need to pay special attention to the condition of air filters during maintenance periods.

The loading rates varied from 14,4 tons per hour to 21,6 tons per hour on average for both loaders **(Figure 9.10)**.

FIGURE 9.9

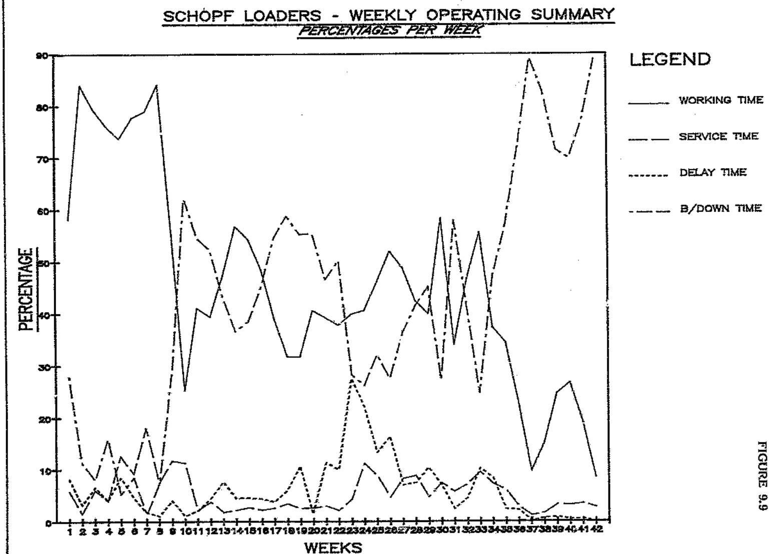
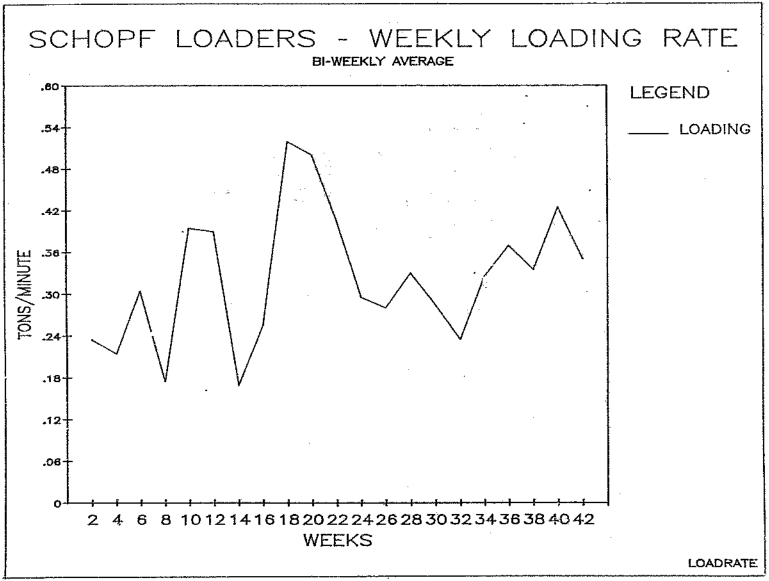

SCHÖPF LOADERS - WEEKLY OPERATING SUMMARY
PERCENTAGES PER WEEK
LEGEND
WORKING TIME
SERVICE TIME
DELAY TIME
B/DOWN TIME
PERCENTAGE
WEEKS

FIGURE 9.10


SCHOPF LOADERS - WEEKLY LOADING RATE
BI-WEEKLY AVERAGE
LEGEND
LOADING
TONS/MINUTE
.60
.54
.48
.42
.36
.30
.24
.18
.12
.06
0
2 4 6 8 10 12 14 16 18 20 22 24 26 28 30 32 34 36 38 40 42
WEEKS
LOADRATE

It was found that a daily service at the start of the working shift according to a set check list was most important and helped to reduce downtime by early recognition of problems which were developing. An alert operator could avoid emergency breakdown repairs by prompt recognition and reporting of mechanical problems.

The daily records and check lists showed that the services were completed as specified. Loader drivers assisted with the greasing of the loaders which also reduced the service time. Services done at the end of each shift were found to be incomplete. Some faults were overlooked as operators and artisans tended to rush the services to get out earlier from underground.

The lack of broken rock to be cleaned out (Table 9.6), caused 62,89 percent of the delay time. More stope faces had to be blasted to keep ahead of the loader cleaning rate.

Limited working face available to the loaders prevented full utilization. Breakdowns were so high that it did not seem worthwhile to increase the number of working faces. Breakdowns amounting to 48,5 percent of the total time was considered to be excessive.

The poor availability of the L.H.D. units was most disappointing.

Table 9.6 Schopf Loader Delays Summary

Delay	Time (Minutes)	Percent
1. Idle (no rock)	12 048	62,69
2. Flat Battery	2 226	11,58
3. Removing scrapers	1 642	8,54
4. Fetching diesel	1 157	6,02
5. Towing other loader	739	3,85
6. Loader stuck	392	2,04
7. Barring	262	1,36
8. Cleaning panel	205	1,07
9. Pulled out pack	165	0,86
10. Cleaning tip	124	0,65
11. Tips full	95	0,49
12. Falls of ground	60	0,31
13. Cleaning bucket	32	0,17
14. Teaching driver	32	0,17
15. Cleaning travelling way	15	0,08
Total	**19 217**	**100,00**

9.10 Labour Productivity

The planning of black labour strengths for conventional winch cleaned faces was done according to standard minute allowances. These allowances were established for the stoping method and stope width used in the experimental stope. **Appendix B** gives the standard labour allocations.

A standard factor of 7,2 tons broken per black was used to allocate labour for the winch cleaned and L.H.D. unit cleaned faces.

It was decided not to reduce the labour complement for the duration of the experiment and preferably to increase the tonnage broken as much as possible.

From **Table 9.7** it can be seen that black labour productivity for the L.H.D. unit cleaning method was 29,1 percent beter than the winch cleaning method. The winch cleaning method had deliberately been maintained in four panels of the experimental stope for control purposes. The erratic labour productivity for the L.H.D. unit cleaned panels was mainly due to the L.H.D. unit breakdowns. It should be noted that the standard factor of 7,2 tons br~~k~~en per black for winch cleaned panels was not achieved once in nine months.

Table 9.7 Labour Productivity - Tons per Black per Shift

Month	Panels Cleaned By	
	L.H.D. Units	Winches
	Tons/Black	Tons/Black
September 1975	6,4	4,5
October 1975	7,6	4,6
November 1975	7,8	6,3
December 1975	6,65	5,0
January 1976	7,8	5,2
February 1976	6,8	5,6
March 1976	9,4	5,9
April 1976	10,0	6,9
May 1976	1,5	5,4
Average	**7,1**	**5,5**

9.11 Production Results

The monthly tonnage broken for both the L.H.D. unit and winch cleaned panels is listed in **Table 9.8.** The tons broken in the L.H.D. unit panels increased as the operators became accustomed to the L.H.D. units. The tonnage broken in the same panels decreased during *the last three months* of operation due to increased breakdown time.

Table 9.8 Production Results

Month	Panels Cleaned By		
	L.H.D. units (t)	Winches (t)	Total Tons
September 1975	5 230	5 814	11 044
October 1975	6 217	5 925	12 142
November 1975	6 437	8 050	14 487
December 1975	5 467	6 478	11 945
January 1976	6 392	6 702	13 094
February 1976	5 565	7 261	12 826
March 1976	7 738	7 649	15 387
April 1976	8 235	8 873	17 108
May 1976	1 241	6 925	8 166
Average	**5 836**	**7 075**	**12 911**

The combined 17 108 tons roken during the April 1976 measuring month set a new stoping record for the shaft for a single stoper. Similarly the combined nine month average of 12 911 tons broken also established a new stoping record on the shaft.

10 OPERATING COSTS

A calculation done in 1976 showed that, on average, the operating cost for stope winches was R0,60 per ton broken (Table 10.1). These operating costs excluded the cost of the remainder of the stope crew as allocated according to the standard factor. The same labour categories were excluded in the cost structure for the L.H.D. unit cleaning method.

Table 10.1 Winch Operating Cost per ton Broken

Category	Rands per Ton	Percent
Power	0,03	5,0
Stores	0,23	38,33
Driver's wages	0,32	53,33
Artisan's wages	0,02	3,33
Cost per Ton (Rands)	0,60	99,99

Winch driver's wages formed 53,3 percent of the operating cost. Stores and power amounted to 43,33 percent of the operating costs.

The stores costs included all items used for face and strike gully cleaning and the engineering stores necessary to repair the winches.

Owning cost for winches amounted to R0,14 per ton broken, giving a total owning and operating cost of R0,74 per ton broken (Table 10.2).

Table 10.2 Winch Owning Cost per Ton Broken

Depreciation cost	R0,12
Interest on Investment	R0,02
Total Owning Cost	R0,14

The operating costs for the two Schopf loaders were summarized monthly from September 1975 to May 1976 inclusive, and averaged for the period as can be seen in **Table 10.3.** The average operating cost per ton was R1,38 calculated over nine months.

In spite of the fact that down time due to tyres only formed 6,61 percent of the breakdown time, the operating cost for tyres amounted to 49,28 percent of the total operating cost, emphasizing the need to find suitable tyres to load quartzites.

An abnormal expenditure occurred in November 1975, when two complete spare sets of tyres and wheel rims were purchased to avoid delays. This purchase should have been made as part of the initial expenditure when the loaders were bought. The tyre cost per ton loaded was then reduced to some R0,40 per ton and tyres were still in good condition when the experiment was stopped, so that it can be expected that the average operating tyre costs would be reduced over a longer period **(Table 10.3).**

Mechanical failures which formed 69,53 percent of the breakdowns amounted to 28,26 percent of the operating costs, although service stores were also included in the above figure **(Table 10.4).** Poor air filtration, gearbox design and some inexperience on the part of the fitter in detecting early signs of mechanical problems, were the main reasons for the high breakdown rate.

Table 10.3 Schopf Loader Operating Cost – Monthly Summary

Category	September 1975			October 1975			November 1975			December 1975			January 1975		
	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton
Tyres	–	2 771,90	0,53	–	3 295,01	0,53	–	17 379,90	2,70	–	164,01	0,03	–	319,60	0,05
Fuel	–	83,68	0,016	–	118,12	0,019	–	128,74	0,02	–	202,28	0,037	–	287,64	0,045
Engineering Stores	–	99,37	0,019	–	404,11	0,065	–	251,04	0,039	–	1 104,33	0,202	–	1 763,37	0,279
Driver Wages	–	203,97	0,039	–	820,64	0,132	–	971,99	0,151	–	874,72	0,160	–	1 195,30	0,187
Artisan Wages	–	104,60	0,02	–	124,34	0,02	–	302,54	0,047	–	185,88	0,034	–	357,95	0,056
Total	5 230	3 263,52	0,624	6 217	4 762, 22	0,766	6 437	19 034,21	2,957	5 467	2 531,22	0,463	6 392	3 943,86	0,617

Category	February 1976			March 1976			April 1976			May 1976			Average		
	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton	Tons	Cost	Cost/Ton
Tyres	–	389,55	0,07	–	7 660,62	0,99	–	2 964,60	0,36	–	483,99	0,39	–	35 429,18	0,68
Fuel	–	294,95	0,053	–	340,47	0,044	–	485,87	0,059	–	67,01	0,054	–	2 008,76	0,04
Engineering Stores	–	2 704,59	0,486	–	2 120,21	0,274	–	9 980,82	1,212	–	1 976,91	1,593	–	20 424,75	0,39
Driver Wages	–	784,67	0,141	–	1 416,05	0,183	–	1 152,90	0,140	–	210,97	0,170	–	7 631,21	0,15
Artisan Wages	–	339,47	0,061	–	3 559,48	0,46	–	1 095,26	0,133	–	173,74	0,140	–	6 243,26	0,12
Total	5 565	4 513,23	0,811	7 730	15 096,83	1,951	8 235	15 679,45	1,904	1 241	2 912,62	2,347	52 522	71 737,16	1,38

Table 10.4 Average Loading Cost per Ton

Category Cost Base	*Rand/ton* Actual	*Percent* Distribution	*Rand/ton* (Possible)
	1975	1975	1975
Tyres	0,68	49,28	0,40
Fuel	0,04	2,90	0,03
Engineering Stores	0,39	28,26	0,30
Operator's Wages	0,15	10,87	0,15
Artisan's Wages	0,12	8,70	0,12
Totals (Rand per Ton)	**1,38**	**100,00**	**1,00**

Operator's wages amounted to 10,87 percent of the total cost and the combined cost per ton for drivers and artisans was R0,27 per ton, compared with R0,34 per ton for the same combination in the case of winches.

From Table 10.4 the labour cost per ton broken for the L.H.D. units was 20,59 percent cheaper than that of winches and would have been lower if the artisan had more units under his control. The L.H.D. unit operator's cost per ton broken alone was 53,13 percent beter than the cost for winch drivers in the experiment.

The loader owning costs in 1975 amounted to R0,16 per ton loaded (Table 10.5).

Table 10.5 Loader Owning Cost per Ton

Loader Owning Cost per Ton	
Depreciation cost	R0,13
Interest on investment	R0,03
Total Owning Cost	R0,16

Table 10.6 Cost Comparison - Rands per Ton

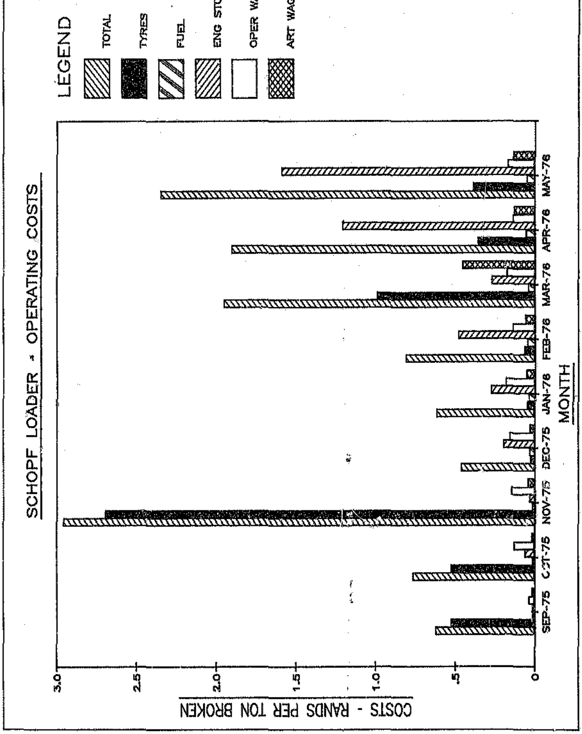
Category	Loaders	Winches
Working	R1,38	R0,60
Owning	R0,16	R0,14
Total Cost	**R1,54**	**R0,74**

The loader/winch comparison shows that the L.H.D. units cost R1,54 per ton compared with R0,74 per ton for winches, which is 108,1 percent more than winch operating costs **(Table 10.6)**.

The Schopf loader operating costs were graphed in Figure 10.1, reflecting the initial high tyre cost in November, and the progressively increasing engineering stores' cost, also from November onwards. Fuel costs increased because of increased travelling distance, engine deterioration and increased fuel price.

The fact that a tip had been provided for each panel and that the travelling distance between the loading point and tipping point was relatively short, assisted in keeping the operating cost reasonably low. To be able to make the L.H.D. unit cleaning method more cost effective than a winch cleaning method, two further avenues should be considered. The first is that a larger L.H.D. unit, still operating in the stope face could prove to be more economical in the long term, since one L.H.D. unit could clean more stope panels. The second is that, since the stope cleaning experiment was conducted in an existing stope designed for conventional winch cleaning methods, unnecessary development was done.

FIGURE 10.1


SCHOPF LOADER - OPERATING COSTS
LEGEND
TOTAL
TYRES
FUEL
ENG STORES
OPER WAGES
ART WAGES
COSTS - RANDS PER TON BROKEN
3.0
2.5
2.0
1.5
1.0
.5
0
SEP-75
NOV-75
DEC-75
JAN-76
FEB-76
MAR-76
APR-76
MAY-76
MONTH

Should the main development layout be altered to suit L.H.D. unit cleaning, the cost of development could decrease and so also the overall shaft head working cost.

11 PILOT INVESTIGATION SUMMARY

Of the two L.H.D. unit cleaning methods observed, the down dip cleaning method was the most efficient and effective. The L.H.D. unit bucket was filled in an average of 6 minutes. The loading rate increased from 14,4 to 21,6 tons per hour for the updip and the downdip cleaning methods respectively. A maximum average loading rate of 31,2 tons per hour was achieved during a two week period using the downdip cleaning method.

Pneumatic chain clad tyres showed 59,93 percent less wear rate than solid tyres at 4 500 tons loaded. The cleaning experiment was unfortunately discontinued before the pneumatic tyres could be tested to destruction.

It was easier and more efficient to use L.H.D. units to clean the sweeping area when compared with conventional winches. Large rocks were removed without delay.

The construction of timber packs was faster since less broken rock was left behind after the L.H.D. units had completed cleaning operations.

The L.H.D. units were effectively used to bar the stope faces breaking away large rocks.

Labour productivity increased by 29,1 percent. More work could be done by the same labour complement or the complement could be reduced.

The L.H.D. unit stope cleaning operating costs were 108,1 percent greater than winch operating costs.

B... downs amounting to 48,5 percent of the shift time were observed and this was considered to be excessive, interrupting the cleaning cycles at unexpected and unplanned for times. Early signs of mec... or electrical failure had to be recognised and corrected during planned maintenance periods.

The selection of the correct combination of clea...ng equipment was most important to the profitability of trackless mining.

12 PILOT INVESTIGATION CONCLUSIONS

Improvements and cost reductions in stope cleaning operations alone would not be sufficient to make L.H.D. unit cleaning methods more economical than conventional winch cleaning methods. The whole stoping and development layout would have to be redesigned eliminating all unnecessary development.

Special attention should be paid to the cause of engine, differential, transmission and drive shaft breakdowns. The improved availability of essential spares related to the breakdowns above would reduce the downtime.

The improvement in labour productivity was encouraging. The use of L.H.D. units for stope cleaning purposes could, if the above conclusions were implemented, be more economical than conventional winch cleaning methods and it would be worthwhile to persevere with large scale production experiments.

The Schopf L82 L.H.D. units were too small for the best productivity in the 2,5 metre stope width. However a major change in the support method would be necessary to make use of larger L.H.D. units for stope cleaning.

13 COOKE 2 SHAFT 90 LEVEL WIDE REEF PROJECT

13.1 Introduction

Major changes were made to support methods in 1982 to be able to mine newly identified wide reef horizons. The grout pack support method was used in a maximum stope width of 3,0 metres. Industrial Engineering Department studies showed that the pack support construction rate decreased as the height of the grout packs increased. The basic materials used for grout pack construction were expensive.

Geological boreholes drilled down from the UE1a reef horizon identified wide areas of E8 reef. This reef width varied from nil to approximately 10 metres.

The standard grout pack support method was not suitable to mine the full reef width. It was then decided to establish a conventionally winch cleaned room and pillar stope on the E8 reef horizon in areas where the stope width was greater than 2,5 metres. The total pillar support method cost was less than any timber support method, partly because of the relatively low grade of the E8 reef. Gold left behind in the pillars amounted to approximately 10,5 percent of the area being mined.

While this area was being mined, another payable area of E8 reef was identified. The potential to mine this area by a trackless mechanised mining method became apparent.

A feasibility study was completed in 1984 where a comparison between a trackless mechanised mining and a conventional winch cleaned stoping method was done.[6] On the basis of findings of the feasibility study a decision was taken in December 1984 to mine the area by trackless mechanised mining methods. The feasibility study report was used as introductory information and background to production results for the past nine months.[7]

Production results were compared with the feasibility study planned results. Due to primary development, preparatory work, ordering and supply of trackless mining equipment, it took approximately two years to raise the production rate of the project to 30 000 tons broken per month (January 1986).

During the past year production at The Randfontein Estates Gold Mining Company, Witwatersrand, Limited was affected by labour strikes. The trackless mechanised projects were influenced to a lesser extent since striking operators were replaced by trained personnel from other disciplines.

Synthesised working costs were developed for trackless mining whereas actual costs were used for conventional mining.

14.2 Geology

The 90 level E8 reef lies between two major faults and is invaded by the two prominent C15 and C17 dykes (Figure 13.1).

FIGURE 13.1

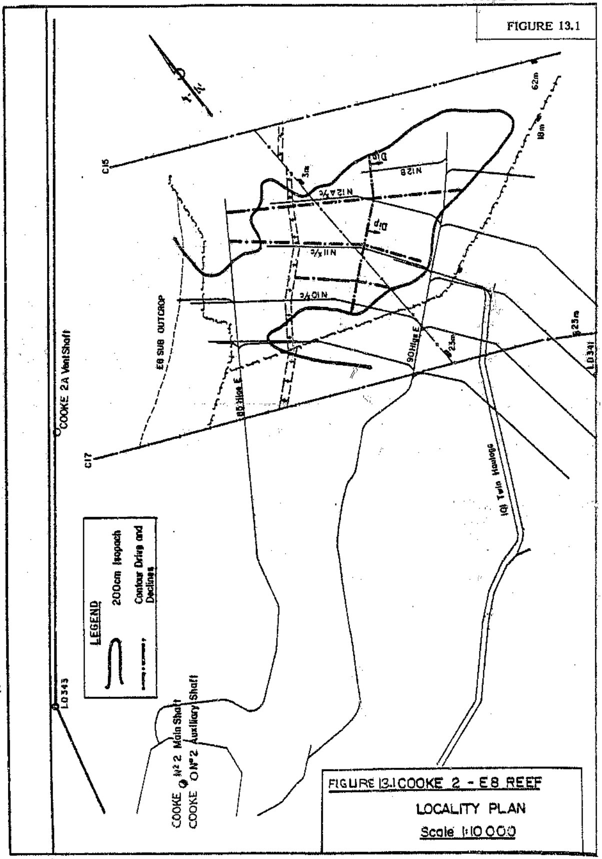


FIGURE 13.1 COOKE 2 - E8 REEF

LOCALITY PLAN

Scale 1:10 000

The reef is relatively undisturbed and occurs as a fan shaped conglomerate deposit dipping from 2 degrees to 10 degrees eastward.

The reef attains a thickness in excess of 4 metres in the centre of the area and thins gradually to the edges of the fan shaped body to less than one metre.

The area comprises the largest block of payable E8 reef at Cooke 2 Shaft. **Figure 13.1** shows the 200 centimetre isopach and major faults and dykes. A transverse section through the N11 stoping line is shown in **Figure 13.2.**

13.3 Paylimit and Working Cost

Whilst the actual data is classified as confidential, paylimit and working cost estimates were done for both the trackless and conventional mining methods. The paylimit for trackless mining methods was 19,05 percent less than the conventional mining method *paylimit as a result of anticipated lower working costs,* due to the reduced need for off-reef development. The trackless mining working cost per ton milled was in fact 21,06 percent less than that of conventional mining methods over a six month period.

As a result of the lower paylimit the reserves for trackless mining were 25,2 percent greater than that of conventional mining.

13.4 Development Layout (Trackless)

Primary development of the area commenced on 90 level elevation. Initially contour reef drives were developed from 90 N11 crosscut and from these drives, decline winzes or access ramps were developed on true dip at approximately 150 metre centres.

FIGURE 13.2

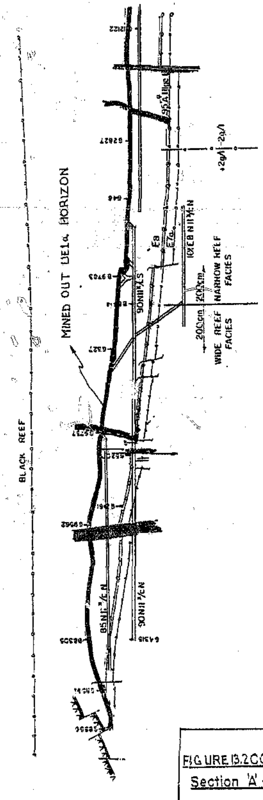


FIGURE 13.2 COOKE 2 SHAFT

Section 'A' – 'A'1 Through N11 Line

Not to scale

As a decline holed into a bottom access airway/travelling way, stoping commenced. The general development layout showing contour drives and winzes can be seen in **Figure 13.1.** Dimensions of the drives and declines were 8 metres wide by 4 metres high.

13.5 *Stoping Layout (Trackless)*

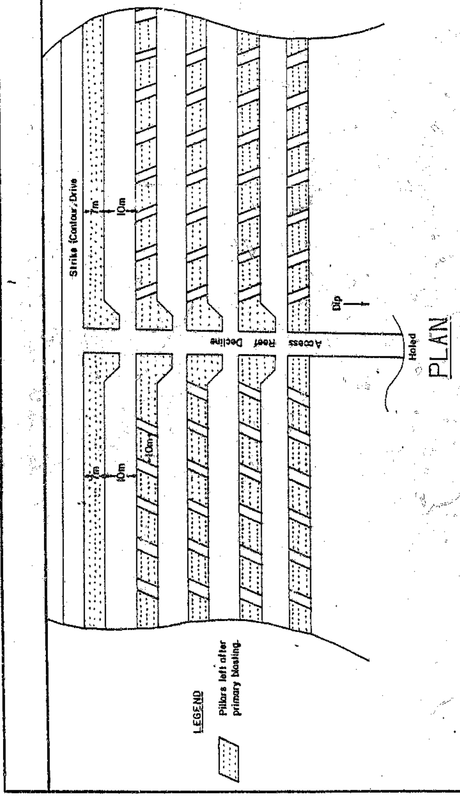
The stepped room and pillar method of primary stoping was selected for this operation. The general stoping configuration has been shown in **Figure's 13.3 and 13.4** with a detailed layout of rooms and pillars shown in **Figure 13.5.** Roo[illegible] developed 3 degrees off true strike so as to contain water o[illegible]nd off the roadways.

Secondary extraction would be carried out when primary stoping in any winze/ramp connection has been completed. During the secondary extraction operations partial extraction of pillars would take place on retreat, pillars being reduced in size in stages to minimum dimensions. The stages in the partial extraction of pillars have been shown in **Figure 13.6.** Final pillar dimensions would be 5 metre by 5 metre spaced 12 metres apart on dip and 9 metres on strike from pillar to pillar.

13.6 Development Layout (Conventional)

Had this area been mined conventionally, development would have commenced on both 90 level as well as 101 level. Some 200 metres of flat development and 300 metres of box holes would have been required on 90 level while 10 039 metres of development would have been required on 101 level.

FIGURE 13.3



PLAN

FIGURE 13.3 TRACKLESS ROOM AND PILLAR MINING METHOD

(GENERAL STOPING CONFIGURATION)

Scale 1:1000

FIGURE 13.4

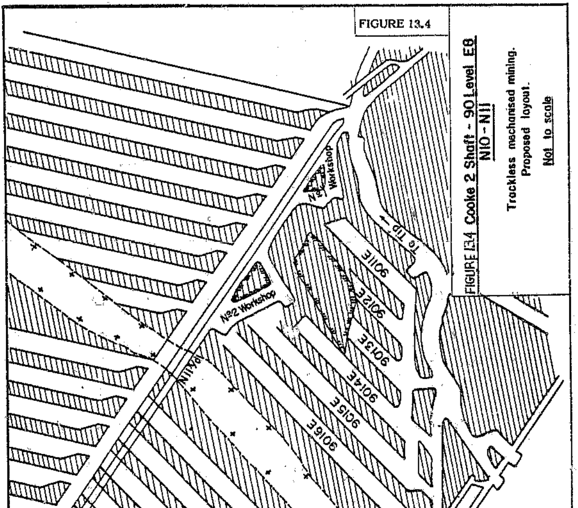


FIGURE 13.4 Cooke 2 Shaft - 90 Level E6 NIO - NII

Trackless mechanised mining. Proposed layout.

Not to scale

FIGURE 13.5

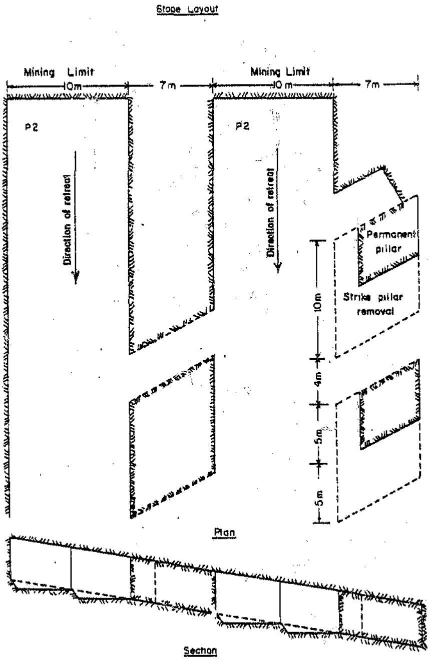


FIGURE 13.5 STEPPED ROOM AND PILLAR METHOD DETAILED PANEL LAYOUT

Scale 1:200

FIGURE 13.6

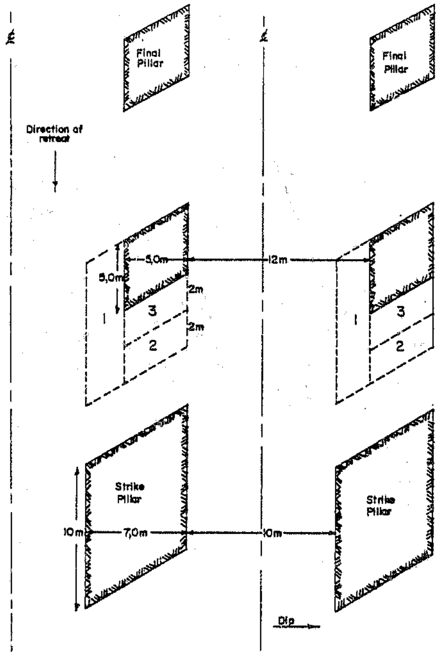


FIGURE 13.6 STAGES IN THE PARTIAL EXTRACTION OF PILLARS

Not to scale

This development would have consisted of 700 metres of haulage, 2 800 metres of crosscut, 5 700 metres of ore passing and 800 metres of travelling ways.

In order to produce 40 000 tons per month conventionally, a 2 year production build-up would be required, the tonnage eventually being produced from 4 stopes, 2 each on the N10 and N11 lines (**Figure 13.7**).

13.7 Stoping Layout (Conventional)

Primary stoping layouts would be similar to that of trackless mining except that all strike gullies would be mined 10° updip of strike.

Secondary extraction for the conventional system would be similar to the trackless mining secondary extraction indicated in **13.5**.

13.8 Support

Support is easier and safer to install in mechanised operations due to the higher reach of trackless rockbolters compared with hand held jackhammers as per **Figure 13.8**. The support standard remains the same in both trackless and conventional stoping. Full column resin grouted rebars 2,7 metres long and 25mm in diameter would be installed two metres apart on dip and strike. Areas showing signs of deterioration would be supported by additional six metre long wire rope anchors.

The same support would be installed in the conventional room and pillar stoping method.

FIGURE 13.7

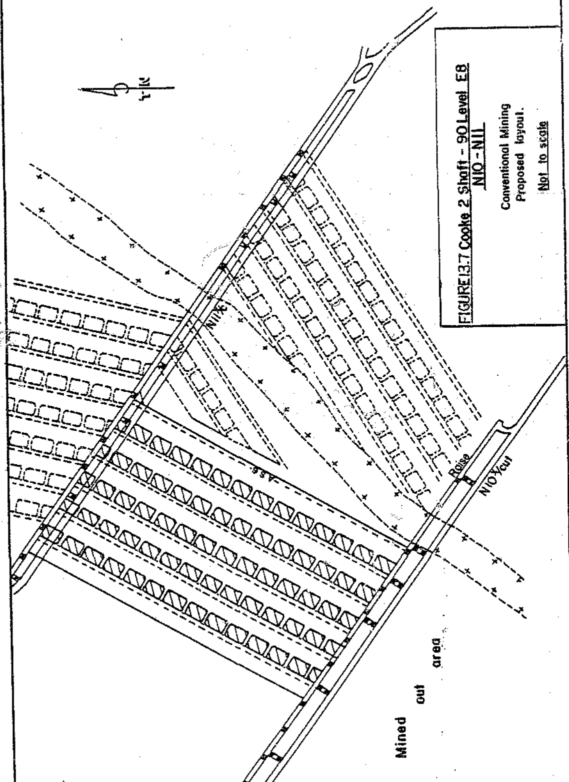


FIGURE13.7 Cooke 2 Shaft - 90 Level E8
N10 - N11
Conventional Mining
Proposed layout.
Not to scale
Raise
N10 X/cut
Mined out area

FIGURE 13.8

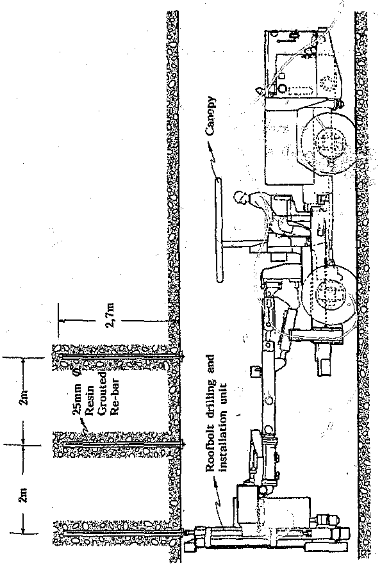


Figure 13.8 **Trackless Roofbolt Installation.**

13.9 Equipment

Trackless mining equipment selected for this project is listed in **Table 13.1.** A comparison of the capital cost of purchasing a suite of trackless mining equipment and conventional mining equipment was done. The initial capital outlay for a suite of trackless mining equipment would be 52,41 percent more than a similar set of conventional mining equipment. Costs were calculated for a production rate of 40 000 tons milled per month.

Table 13.1 ***Trackless Mining Equipment required to produce 40 000*** **tons milled per month.**

Category	Make	Quantity
L.H.D. unit	Toro 350D (7,3 t capacity)	3
Drillrig	Tamrock (Twin boom)	3
Bar down/Scaler	Tamrock (Single boom)	1
Utility Vehicle	Charger (Normet)	0
	Scissor (Normet)	1
	Carrier (+4 cassettes)	1
Truck	Volvo 18T	2
	Toro 35D	1
Roofbolter	Tamrock (Single boom)	2
Utility vehicle - Boom & Basket	Normet PK 1000	2
Transport	Toyota Landcruiser	3
Grader	Rocor Rand	1
Impact breaker		1

The equipment listed above was ordered and is currently in use in the 90 level E8 wide reef project.

13.10 Labour

Labour comparisons were also done for both the trackless mining method and conventional mining method. A summary of the labour comparison can be seen in **Table 13.2 and 13.3.**

Table 13.2 White Labour - 90 Level E8 Wide Reef Project

	Conventional	Trackless Mining
In Stope	6	2
Development	1	0
Commons	2	0
Supervision	7	5
Engineering	9	8
Services	5	3
Total Labour	**30**	**18**

Table 13.3 Black Labour - 90 Level E8 Wide Reef Project

	Conventional	Trackless Mining
In Stope	185	51
Development	30	0
Commons	132	18
Supervision	6	0
Engineering	12	2
Services	8	6
Total Labour	**373**	**77**

Planned labour productivity has been listed in **Table 13.4**. The table shows stope production labour productivity and total labour productivity which includes engineering, stoping, development, services, supervision and general labour.

Table 13.4 Labour Productivity Summary (Planned)

Year	Conventional Tons Broken				Trackless Tons Broken			
	Per Shift		Per Shift		Per Shift		Per Shift	
	Prod. Blacks	Prod. Whites	Total Blacks	Total Whites	Prod. Blacks	Prod. Whites	Total Blacks	Total Whites
1984/85	6,1	64,9	2,2	22,7	16,5	116,3	7,9	43,1
1985/86	6,9	98,5	3,4	44,8	28,2	250,8	19,5	80,2
1986/87	8,0	137,2	4,1	52,3	32,7	833,3	21,6	92,6
1987/88	9,1	277,7	4,5	55,6	32,7	633,3	21,9	92,6

The above labour productivity was calculated for a 24 shift month. According to the planning the stope black labour productivity for trackless mining was 251,7 percent better than the conventional stope labour productivity. The labour allocated was for morning shift blasting and double shift cleaning, supporting and drilling.

13.11 Planned Production Rates

The planned reef production build-up comparison indicated that it would take eight months longer, by conventional means, to achieve the planned maximum of 40 000 reef tons broken per month, than by the trackless mining method Table 13.5.

It sho ld be noted that 40 000 tons of waste had been broken, for the project, by March 1986 mainly due to sliping of haulages, crosscuts, tipping points and the preparation of workshops.

No further waste development would be required, in the 90 Level E8 wide reef project, while using the trackless mining method, which was not the case with the conventional mining methods.

Table 13.5 Planned Production Rates

	Conventional Tons Broken		Trackless Tons Broken	
	Reef Tons (1000's)	Waste Tons (1000's)	Reef Tons (1000's)	Waste Tons (1000's)
To Date i.e. March 1986	194,9	55	259,9	40
April	24,5	1,3	40,0	-
May	24,5	1,3	40,0	-
June	29,3	1,3	40,0	-
July	29,3	1,3	40,0	-
August	29,3	1,3	40,0	-
September	34,3	1,3	40,0	-
October	34,3	1,3	40,0	-
November	34,3	1,3	40,0	-
December	40,0	1,3	40,0	-
Jan. - June 1987	240,0	7,6	240,0	-
1987/88	480,0	15,1	480,0	-

13.12 Production Results

Production results of the 90 level E8 wide reef project have been summarised for the past ten months and were compared with the feasibility study parameters. Reef and waste tons broken were combined to give the total tons broken.

An average of 45 014 tons broken per month was achieved with a maximum of 55 081 tons broken in June 1987 (**Table 13.6**). The low tonnage broken in September was as a result of a labour strike. The trackless mining projects were the least affected since trained white operators were able to man essential equipment such as L.H.D. units, hydraulic drillrigs and utility vehicles. On average the tonnage broken was 12,5 percent better than the Feasibility Study Plan.

Table 13.6 Tons Broken per Month - 90 Level E8 Wide Reef

Month	Feasibility Study Tons/month	Actual Tons/month
January 1987	40 000	29 890
February	40 000	41 336
March	40 000	51 424
April	40 000	57 511
May	40 000	48 966
June	40 000	55 081
July	40 000	45 969
August	40 000	44 538
September	40 000	32 199
October 1987	40 000	49 223
Average	**40 000**	**45 014**

Waste dilution control proved to be a problem. The planned 4,6 percent dilution was never achieved in the ten month period (**Table 13.7**). The average of 15,73 percent waste dilution was 241,96 percent greater than planned. The main cause of the dilution was the intersection of areas with reduced channel width within the wide reef orebody.

Table 13.7 Percentage Waste Dilution per Month - 90 Level E8 Wide Reef

Month	Feasibility Study % Dilution	Actual % Dilution
January 1987	4,6	8,7
February	4,6	8,0
March	4,6	12,0
April	4,6	10,6
May	4,6	10,0
June	4,6	11,0
July	4,6	18,0
August	4,6	32,0
September	4,6	28,0
October 1987	4,6	19,0
Average	**4,6**	**15,73**

Channel and stope widths for September 1987 have been listed in Table 13.8 as an example. Thirty eight percent of the panels being mined were below the 250 centimetre wide reef lower limit.

Table 13.8 Stoping Widt/Channel Width Comparison

Panel Number	September 1987	
	Channel Width (cm)	Stoping Width (cm)
20W	208	347
9E	375	395
8E	375	395
7E	375	395
20E	* 230	383
West 22W	* 217	366
21E	* 193	369
22E	286	377
12E	* 192	399

Panel Number	September 1987	
	Channel Width (cm)	*Stoping Width* (cm)
West 23W	340	345
Ventilation Raise	340	345
19E	* 196	392
East Decline	* 230	409
W 19W	* 196	317
E Raise	* 233	400
3W	352	391
Decline 1	322	416
Return Way	300	388
12E	300	388
7W	261	400
Decline 2	261	400
7E	261	400
Decline 3	* 242	411
10E	373	387
Average	**277**	**384**

* Channel width narrower than the wide reef lower stoping limit of 250cms.

A method of reducing the waste dilution in these areas would be to reduce the access roadways to the minimum dimensions to suit the trackless mining equipment until the narrow channel width section has been passed. The larger pillars remaining could then be mined on retreat only taking the full channel width. Drilling the pillars would not be a problem for the hydraulic drillrigs using extension drill steel.

Another solution would be to stop mining by the room and pillar method, then combine two panels to form one narrow reef panel and mine it using winch cleaning methods in the stope face and smaller L.H.D. units for access stope drive cleaning. It is important to have a flexible stoping method to cope with changing channel widths.

White labour productivity averaging 1 303 tons broken per man was 41,4 percent less than planned **(Table 13.9)**. Supervisory and maintenance staff had been increased to ensure the success of the trackless mining project and for the training of personnel for other trackless mining projects. The white labour force would be decreased in the future. The productivity per ton broken per [illegible] for trackless mining, was 2,4 percent less than the [illegible] conventional labour productivity per white (refer again to Table 13.4).

Table 13.9 *Total Trackless Tons Broken per Total White - 90 Level* E8 Wide Reef

Month	**Feasibility Study Tons per White**	**Actual Tons per White**
January 1987	2 222	1 272
February	2 222	1 074
March	2 222	1 390
April	2 222	1 493
May	2 222	1 399
June	2 222	1 574
July	2 222	1 332
August	2 222	1 255
September	2 222	920
October 1987	2 222	1 330
Average	2 222	* 1 303

Black labour productivity averaging 309 tons broken per man was 40,6 percent less than planned (Table 13.10). More labour was used to ensure a high standard of *roadway maintenance, trackless* equipment maintenance and construction. Operators and maintenance staff would be reduced and used in new trackless mining projects in the future. The black labour productivity for trackless mining methods was still 186,1 percent better than the planned labour productivity for conventional mining methods (refer again to *Table 13.4*).

Table 13.10 Total Trackless Tons Broken per Total Black - 90 Level E8 Wide Reef

Month	Feasibility Study Tons per Black	Actual Tons per Black
January 1987	520	356
February	520	254
March	520	279
April	520	277
May	520	320
June	520	358
July	520	321
August	520	309
September	520	325
October	520	335
Average	520	* 309

* Weighted Average.

13.13 Operating Costs

The average operating cost per ton broken was 33,16 percent more than planned (Table 13.11). This was mainly due to excess white and black labour which would be reduced in future. It is also expected that the operating cost would reduce when secondary extraction of the pillars commences. Minimal additional permanent support would be used and reclamation of piping and power cables would also be initiated. Sliping of the pillars would also be a relatively cheap mining operation. The costs planned in the Feasibility Study report did not differentiate been primary or secondary extraction and is an average planned cost for the project.

It was established from Company records that the conventional mining wide reef working cost for the same period was 26,60 percent greater than the trackless mining wide reef working cost. Labour disturbances leading up to the September 1987 labour strike adversely affected both conventional mining and trackless mining production and costs.

Table 13.11 Operating Cost per Ton Broken - 90 Level E8 Wide Reef

Month	Feasibility Study Rand/ton/milled	Actual (In Stope Rand/ton/milled
January 1987	13,01	29,56
February	13,01	21,74
March	13,18	17,66
April	13,44	19,10
May	13,44	18,65
June	14,17	13,68
July	14,37	14,17
August	14,37	14,37
September	14,37	14,45
October 1987	14,37	21,50
Average	**13,57**	*** 18,07**

* Weighted Average.

13.14 90 Level E8 Wide Reef Project Summary

The 90 level E8 wide reef trackless mining project had a 12,5 percent greater average production rate than planned. At times of labour unrest alternate arrangements could be made to operate essential equipment to reduce production losses.

Waste dilution was 241,96 percent greater than planned. This was mainly caused by the reduction of reef channel width within the wide reef zone.

Waste dilution could be reduced and minimised by making use of a versatile stoping method and selecting smaller trackless mining equipment to operate in the narrower areas of the E8 wide reef project.

White labour productivity per ton broken was 41,4 percent less than planned. Excess personnel, deliberately carried overstrength for training purposes, would be deployed to other trackless mining projects in the future.

Black labour productivity per ton broken was 40,6 percent less than planned. Excess personnel, also deliberately carried overstrength for training purposes, would be moved to other trackless mining projects. In spite of the reduced black labour productivity it was still 186,1 percent better than the planned conventional wide reef labour productivity. The training of personnel, to ensure the achievement of the planned tonnage build-up of future trackless mining projects, was considered to be of the utmost importance.

Operating costs per ton broken was 33,16 percent more than planned, mainly due to excess labour and the primary reef extraction method. The trackless mining method remained 21,7 percent cheaper than conventional wide reef mining methods. The decrease in waste development and improvement in black labour productivity were the main reasons for the trackless mining method being more economical.

13.15 90 Level E8 Wide Reef Project Conclusions

The wide reef trackless mining project results proved that a high production rate could be maintained using trackless equipment.

Operating costs could be improved upon but the trackless mining method was still more economical than conventional wide *reef mining* methods.

The selection of trackless mining equipment, matching units to suit the width of the orebody, was most important. L.H.D. units and trucks being used in the narrow portions of *the 90 level E8 wide reef project* need to be dimensionally smaller. The profitability of the project could be improved by reducing the amount of waste dilution sent to the reduction plant and replacing it with reef tons.

14 CONCLUSIONS AND RECOMMENDATIONS

14.1 Summary and Conclusions

The pilot trackless mining stope cleaning project of 1975 showed that the change in the stope cleaning method alone was not sufficient to make trackless mining more profitable than conventional mining. It was necessary to change the mining layout thereby reducing waste development.

This could also be seen in the 90 level E8 wide reef project producing at an average of 45 000 tons per month. Major changes were made in the development layouts. Stope layout changes permitted free access by trackless mining equipment to all stope faces.

The question of selection and matching of L.H.D. cleaning units with trucks to suit particular stope widths was again prominent. The profitability of a trackless mining project could be adversely affected should incorrectly sized equipment be used. It is not simply a matter of using smaller or larger equipment. Trackless cleaning equipment should be carefully selected for the reef width and mining method planned.

Flexibility to change from a trackless wide reef mining method to a trackless narrow reef mining method should be part of trackless equipment design criteria.

L.H.D. units should preferably be operated down dip during cleaning operations to achieve the fastest loading rate.

Trackless mechanised mining methods being used on The Randfontein Estates Gold Mining Company, Witwatersrand, Limited were proving to be successful and economically viable.

14.2 Recommendations For Future Work

Trackless mechanised mining equipment planned for various projects needs to be selected with the total profitability of the project in mind. All factors affecting revenue from a trackless mining project related to trackless mining equipment need to be investigated.

A future investigation should be the selection of trackless mining equipment for different reef widths to optimise the return on investment.

TABLE A1 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	06.12.1975	08.12.1975	09.12.1975	10.12.1975	11.12.1975	12.12.1975	WEEK ENDING 12.12.1975	13.12.1975	15.12.1975	16.12.1975	17.12.1975	18.12.1975	19.12.1975	WEEK ENDING 19.12.1975
WORKING TIME	567	628	820	740	719	835	4308	685	1277		355	539	552	3408
TRAVELLING TIME	37	21	29	33	59	50	229	33	28		19	66	42	188
SERVICE TIME	41	39	30	30	44	110	294	65	69		60	53	30	277
TOTAL DELAY TIME	149	–	20	–	–	–	169	142	68		697	44	93	1044
TOTAL BREAKDOWN TIME ...	960	975	1031	1078	1191	967	6202	925	72		796	1277	1203	4273
TOTALS	1754	1663	1930	1881	2012	1962	11202	1850	1514		1927	1979	1920	9190
DELAYS														
NO ROCK – IDLE	130	–	20	–	–	–	150	–	–		–	–	–	–
WATERING DOWN	15	–	–	–	–	–	15	–	–		–	–	–	–
		–	–	–	–	–		–	–		–	–	–	–
IDLE	–	–	–	–	–	–	–	117	68		60	44	93	182
FETCHING DIESOLENE	4	–	–	–	–	–	4	–	–		162	–	–	162
FLAT BATTERY	–	–	–	–	–	–	–	25	–		475	–	–	500
LOADER STUCK	–	–	–	–	–	–	–	–	–		–	–	–	–
PULLED OUT PACK	–	–	–	–	–	–	–	–	–		–	–	–	–
TOTAL DELAYS	149	–	20	–	–	–	169	142	68		697	44	93	1044
BREAKDOWNS														
CANOPY REPAIRS	–	15	–	–	65	165	245	–	–		–	–	–	–
FLAT WHEELS	–	–	13	118	–	–	131	–	–	HOLIDAY	–	–	–	–
TIGHTENING WHEEL CHAINS	–	–	58	–	83	–	141	–	–		30	302	116	448
GENERAL REPAIRS	–	–	–	–	95	–	95	–	72		101	–	–	173
BROKEN BUCKET ARM	–	–	–	–	18	–	18	–	–		–	–	112	112
BRAKES O.O.O.	–	–	–	–	–	65	65	–	–		–	–	–	–
TOWING OTHER LOADER	–	–	–	–	–	17	17	–	–	MINE	–	–	–	–
LOADER O.O.O.	960	960	960	960	930	720	5490	–	–		–	–	–	–
GEARBOX O.O.O.	–	–	–	–	–	–	–	925	–		215	975	975	3090
BATTERY TROUBLE	–	–	–	–	–	–	–	–	–		450	–	–	450
BROKEN HYDRAULIC HOSE ..	–	–	–	–	–	–	–	–	–		–	–	–	–
CHANGING WHEELS	–	–	–	–	–	–	–	–	–		–	–	–	–
STARTING TROUBLE	–	–	–	–	–	–	–	–	–		–	–	–	–
STEERING O.O.O.	–	–	–	–	–	–	–	–	–		–	–	–	–
TOTAL BREAKDOWNS ...	960	975	1031	1078	1191	967	6207	925	72		796	1277	1203	4273
TONNAGE							2080							1730

	20.12.1975	22.12.1975	23.12.1975	24.12.1975	25.12.1975	26.12.1975	WEEK ENDING 26.12.1975	27.12.1975	29.12.1975	30.12.1975	31.12.1975	01.01.1976	02.01.1976	WEEK ENDING 02.01.1976
WORKING TIME	125	645	1121	584		612	3087	303	399	1141	986	860	492	4181
TRAVELLING TIME	5	53	100	14		25	197	42	34	36	42	74	50	280
SERVICE TIME	–	46	33	46		58	185	49	62	45	117	108	105	486
TOTAL DELAY TIME	–	284	247	247		90	868	1311	1115	63	279	192	120	3100
TOTAL BREAKDOWN TIME ...	1670	596	460	822		830	4378	105	–	699	567	576	1231	3178
TOTALS	1800	1626	1961	1713		1615	8715	1810	1610	1986	1991	1810	1998	11225
DELAYS														
NO ROCK – IDLE	–	–	–	–		–	–	1311	1050	–	105	–	–	2486
WATERING DOWN	–	–	–	–		–	–	–	–	–	–	–	–	–
	–	–	–	–		–	–	–	–	–	–	–	–	–
IDLE	–	187	114	113		55	467	–	65	63	102	152	55	437
FETCHING DIESOLENE	–	97	133	136		35	401	–	–	–	–	–	–	–
FLAT BATTERY	–	–	–	–		–	–	–	–	–	–	–	–	–
LOADER STUCK	–	–	–	–		–	–	–	–	–	–	40	65	105
PULLED OUT PACK	–	–	–	–		–	–	–	–	–	72	–	–	72
TOTAL DELAYS	–	284	247	247		90	868	1311	1115	63	279	192	120	3100
BREAKDOWNS														
CANOPY REPAIRS	–	–	–	–		–	–	–	–	–	–	–	–	–
FLAT WHEELS	–	–	–	120	HOLIDAY	–	120	–	–	59	117	35	1126	1337
TIGHTENING WHEEL CHAINS	–	–	–	–		–	–	–	–	–	–	–	–	–
GENERAL REPAIRS	–	215	105	–		50	370	–	–	–	–	–	–	–
BROKEN BUCKET ARM	900	–	–	–		–	90	–	–	–	–	85	–	85
BRAKES O.O.O.	–	–	–	–		200	200	105	–	585	450	450	–	1590
TOWING OTHER LOADER	–	–	–	–	MINE	–	–	–	–	–	–	–	–	–
LOADER O.O.O.	–	–	–	–		–	–	–	–	–	–	–	–	–
GEARBOX O.O.O.	720	–	–	–		–	720	–	–	–	–	–	–	–
BATTERY TROUBLE	–	–	–	–		–	–	–	–	–	–	–	–	–
BROKEN HYDRAULIC HOSE ..	–	105	110	648		580	1443	–	–	–	–	–	105	105
CHANGING WHEELS	50	96	–	54		–	200	–	–	–	–	–	–	–
STARTING TROUBLE	–	180	245	–		–	425	–	–	–	–	–	–	–
STEERING O.O.O.	–	–	–	–		–	–	–	–	55	–	6	–	61
TOTAL BREAKDOWNS ...	1670	596	460	822		830	4378	105	–	699	567	576	1231	3178
TONNAGE							1070							1470

TABLE A2 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	03.01.1976	05.01.1976	06.01.1976	07.01.1976	08.01.1976	09.01.1976	WEEK ENDING 09.01.1976	10.01.1976	12.01.1976	13.01.1976	14.01.1976	15.01.1976	16.01.1976	WEEK ENDING 16.01.1976	17.01.1976	19.01.1976	20.01.1976	21.01.1976	22.01.1976	23.01.1976	WEEK ENDING 23.01.1976	24.01.1976	26.01.1976	27.01.1976	28.01.1976	29.01.1976	30.01.1976	WEEK ENDING 30.01.1976
WORKING TIME	431	185	964	1187	839	748	4354	716	527	781	1259	977	621	4881	743	1206	1198	765	686	976	5574	856	439	677	832	1413	930	5147
TRAVELLING TIME	30	6	54	77	50	57	276	32	32	47	111	45	82	349	48	61	74	20	45	65	313	64	43	41	44	114	76	382
SERVICE TIME	142	125	407	162	195	244	1275	87	52	77	322	160	294	992	75	115	150	34	87	81	542	114	127	197	60	285	144	927
TOTAL DELAY TIME	82	-	577	263	862	744	2528	288	113	90	260	161	608	1520	922	280	133	115	236	166	1852	200	177	65	47	137	174	800
TOTAL BREAKDOWN TIME ..	1115	1299	-	350	-	234	2998	689	909	990	46	643	385	3662	14	-	440	1060	934	698	3146	591	834	1012	978	39	661	4115
TOTALS	1800	1617	2002	2039	1946	2027	11431	1812	1633	1985	1998	1886	1990	11404	1802	1662	1995	1994	1988	1986	11427	1825	1620	1992	1962	1988	1985	11371
DELAYS:-																												
IDLE	82	-	577	159	747	468	2033	239	83	70	193	161	588	1354	912	223	133	97	236	166	1767	101	146	50	47	114	88	546
PULLED OUT PACK	-	-	-	-	23	70	93	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BARRING	-	-	-	10	-	17	27	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15	-	-	-	-	-	15
FETCHING DIESEL	-	-	-	-	92	159	251	49	30	20	-	-	-	99	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CLEANING TIP	-	-	-	94	-	30	124	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REMOVING SCRAPERS	-	-	-	-	-	-	-	-	-	-	67	-	20	87	10	57	-	-	-	-	67	25	31	15	-	-	86	157
TOWING OTHER LOADER ...	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	18	-	-	18	-	-	-	-	-	-	-
CLEANING TRAVELLING WAY	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	23	-	23
LOADER STUCK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	59	-	-	-	-	-	59
TOTAL DELAYS	82	-	577	263	862	744	2528	288	113	90	260	161	608	1520	922	280	133	115	236	166	1852	200	177	65	47	137	174	800
BREAKDOWNS:-																												
FITTING SHOVEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	55	-	55	-	-	-	-	-	-	-
BROKEN OSCILATING BEARING	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	370	990	335	-	1695	-	-	-	62	-	-	62
GENERAL REPAIRS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14	-	-	-	20	92	126	-	-	-	-	-	-	-
STARTER O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	535	535	-	-	-	-	-	-	-
BROKEN WHEEL CHAIN	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	70	70	-	-	140	-	-	-	-	-	-	-
REPAIRING EXHAUST	-	-	-	-	-	-	-	-	-	-	-	19	-	19	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FIXING BUCKET PINS	-	-	-	-	-	-	-	-	-	-	11	30	-	41	-	-	-	-	-	71	71	-	-	-	-	-	-	-
FAULTY TRANSMISSION ...	-	-	-	-	-	-	-	-	349	990	-	-	-	1339	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TIGHTENING BOLTS	-	-	-	-	-	-	-	-	-	-	-	-	42	42	-	-	-	-	-	-	-		-	-	-	-	-	-
FAULTY IGNITION SWITCH	-	-	-	-	-	-	-	-	-	-	-	-	130	130	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING SMOKE BOX ...	-	-	-	40	-	21	61	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TIGHTENING WHEEL CHAIN	-	-	-	-	-	38	38	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BRAKE O.O.O.	-	-	-	-	-	-	-	310	-	-	-	-	-	310	-	-	-	-	-	-	-	-	-	-	-	-	-	-
HYDRAULIC OIL LEAKS ..	310	772	-	310	-	-	1392	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
STEERING O.O.O.	-	-	-	-	-	-	-	-	-	-	35	164	-	199	-	-	-	-	-	-	-	-	-	-	247	-	470	717
FLAT WHEELS	805	527	-	-	-	175	1507	379	560	-	-	430	213	1582	-	-	-	-	524	-	524	-	-	-	-	-	191	191
GEARBOX O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	431	810	993	609	-	-	3063
BROKEN WATER PIPE	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	39	-	39
STARTING PROBLEMS																							24	19				43

TABLE A3 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	31.01.1976	02.02.1976	03.02.1976	04.02.1976	05.02.1976	06.02.1976	WEEK ENDING 06.02.1976	07.02.1976	09.02.1976	10.02.1976	11.02.1976	12.02.1976	13.02.1976	WEEK ENDING 13.02.1976
WORKING TIME	602	441	765	899	1253	644	4604	409	399	801	855	794	1062	4320
TRAVELLING TIME	43	26	48	69	45	101	332	12	46	84	40	44	37	263
SERVICE TIME	44	74	60	295	424	134	1031	60	38	178	95	97	74	542
TOTAL DELAY TIME	65	211	141	121	169	170	877	198	520	141	225	55	54	1193
TOTAL BREAKDOWN TIME	1106	868	966	596	56	1244	4836	1301	617	776	765	990	752	5201
TOTALS	1860	1620	1980	2030	1947	2245	11680	1980	1620	1980	1980	1980	1979	11519
DELAYS:-														
IDLE	65	167	87	100	130	129	676	60	115	90	90	55	37	447
REMOVING SCRAPERS	–	44	30	21	39	41	175	60	–	51	75	–	17	203
BARRING	–	–	–	–	–	–	–	–	–	–	40	–	–	40
LOADER STUCK	–	–	24	–	–	–	24	24	95	–	20	–	–	139
FLAT BATTERY	–	–	–	–	–	–	–	54	310	–	–	–	–	364
FETCHING DIESEL	–	–	–	–	–	–	–	–	–	–	–	–	–	–
TOTAL DELAYS	65	211	141	121	169	170	877	198	520	141	225	55	54	1193
BREAKDOWNS:-														
REPAIRING EXHAUST	–	–	–	–	–	–	–	–	–	–	–	–	–	–
BROKEN YOKE	–	–	–	–	–	–	–	–	–	–	–	–	–	–
BROKEN OSCILATING BEARING	–	–	–	–	–	–	–	–	–	–	655	990	752	2397
LEAKING TRANSMISSION OIL	–	–	–	–	–	–	–	–	–	–	55	–	–	55
REPAIR CANOPY	–	63	–	–	23	313	399	–	–	–	–	–	–	–
PUMPING WHEELS	–	–	21	–	–	303	324	–	–	–	–	–	–	–
CHANGING WHEELS	–	–	–	–	–	48	48	–	–	–	–	–	–	–
LOADER O.O.O.	81	–	–	–	–	–	81	–	–	–	–	–	–	–
BRAKES O.O.O.	930	805	945	596	–	580	3856	625	280	–	–	–	–	905
GEARBOX O.O.O.	–	–	–	–	–	–	–	–	–	446	–	–	–	446
FLAT WHEELS	95	–	–	–	33	–	128	676	337	297	55	–	–	1365
STEERING O.O.O.	–	–	–	–	–	–	–	–	–	33	–	–	–	33
LOOSE BUCKET PIN	–	–	–	–	–	–	–	–	–	–	–	–	–	–
TOTAL BREAKDOWNS	1106	868	966	596	56	1244	4836	1301	617	776	765	990	752	5201
TONNAGE							1737							1627

	14.02.1976	16.02.1976	17.02.1976	18.02.1976	19.02.1976	20.02.1976	WEEK ENDING 20.02.1976	21.02.1976	23.02.1976	24.02.1976	25.02.1976	26.02.1976	27.02.1976	WEEK ENDING 27.02.1976
WORKING TIME	1027	230	1085	940	1100	1357	6739	902	202	824	100	802	854	3684
TRAVELLING TIME	85	30	125	71	97	101	509	30	43	27	18	33	37	188
SERVICE TIME	120	35	253	183	97	157	845	107	216	95	27	103	100	648
TOTAL DELAY TIME	65	45	121	92	365	128	816	81	80	47	15	53	15	291
TOTAL BREAKDOWN TIME	566	1280	–	696	329	254	3125	750	1079	990	1820	1004	990	6623
TOTALS	1863	1620	1984	1982	1988	1997	11434	1860	1620	1983	1980	1995	1396	11434
DELAYS:-														
IDLE	65	35	121	92	74	86	473	81	40	47	15	30	15	228
REMOVING SCRAPERS	–	10	–	–	175	42	227	–	40	–	–	23	–	63
BARRING	–	–	–	–	–	–	–	–	–	–	–	–	–	–
LOADER STUCK	–	–	–	–	–	–	–	–	–	–	–	–	–	–
FLAT BATTERY	–	–	–	–	–	–	–	–	–	–	–	–	–	–
FETCHING DIESEL	–	–	–	–	116	–	116	–	–	–	–	–	–	–
TOTAL DELAYS	65	45	121	92	365	128	816	81	80	47	15	53	15	291
BREAKDOWNS:-														
REPAIRING EXHAUST	–	–	–	–	167	–	167	–	–	–	–	–	–	–
BROKEN YOKE	480	1280	–	696	162	224	2847	–	–	–	–	–	–	–
BROKEN OSCILATING BEARING	–	–	–	–	–	–	–	695	810	990	990	990	990	5465
LEAKING TRANSMISSION OIL	–	–	–	–	–	–	–	–	–	–	–	–	–	–
REPAIR CANOPY	86	–	–	–	–	–	86	–	–	–	–	–	–	–
PUMPING WHEELS	–	–	–	–	–	–	–	–	–	–	–	–	–	–
CHANGING WHEELS	–	–	–	–	–	–	–	–	–	–	–	–	–	–
LOADER O.O.O.	–	–	–	–	–	–	–	–	–	–	–	–	–	–
BRAKES O.O.O.	–	–	–	–	–	–	–	–	269	–	–	–	–	269
GEARBOX O.O.O.	–	–	–	–	–	–	–	–	–	–	830	–	–	830
FLAT WHEELS	–	–	–	–	–	–	–	–	–	–	–	–	–	–
STEERING O.O.O.	–	–	–	–	–	25	25	45	–	–	–	–	–	45
LOOSE BUCKET PIN	–	–	–	–	–	–	–	–	–	–	–	14	–	14
TOTAL BREAKDOWNS	566	1280	–	696	329	254	3125	740	1079	990	1820	1004	990	6623
TONNAGE							1373							1054

TABLE A4 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	28.02.1976	01.03.1976	02.03.1976	03.03.1976	04.03.1976	05.03.1976	WEEK ENDING 05.03.1976	06.03.1976	08.03.1976	09.03.1976	10.03.1976	11.03.1976	12.03.1976	WEEK ENDING 12.03.1976
WORKING TIME	770	612	727	728	1312	846	4995	765	213	1288	1616	1438	1531	6851
TRAVELLING TIME	40	45	36	91	89	61	362	45	10	87	106	67	92	407
SERVICE TIME	45	102	95	180	227	169	818	78	22	453	257	191	243	1244
TOTAL DELAY TIME	75	58	146	75	81	82	517	45	270	110	275	450	190	1340
TOTAL BREAKDOWN TIME	930	810	990	923	320	840	4813	943	905	269	471	467	170	3225
TOTALS	1860	1627	1994	1997	2029	1998	11505	1876	1420	2207	2725	2613	2226	13067
DELAYS:-														
IDLE	47	45	70	10	20	10	202	9	-	30	102	41	24	206
REMOVING SCRAPERS	28	13	44	35	36	37	193	-	-	-	58	25	-	83
TEACHING DRIVER	-	-	32	-	-	-	32	-	-	-	-	-	-	-
LOADER STUCK	-	-	-	30	-	35	65	-	-	-	-	-	-	-
CLEANING PANEL	-	-	-	-	25	-	25	-	-	65	115	-	-	180
FETCHING DIESEL	-	-	-	-	-	-	-	36	-	-	-	-	-	36
FLAT BATTERY	-	-	-	-	-	-	-	-	270	-	-	270	115	655
BARRING	-	-	-	-	-	-	-	-	-	15	-	114	51	140
TOWING OTHER LOADER	-	-	-	-	-	-	-	-	-	-	-	-	-	-
F.O.G.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CLEANING BUCKET	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL DELAYS	75	58	146	75	81	82	517	45	270	110	275	450	190	1340
BREAKDOWNS														
REPLACE BATTERY	-	-	-	-	-	-	-	37	30		-	-	45	112
REPAIRING CANOPY	-	-	-	-	-	-	-	-	-	-	-	117	-	117
REPAIRING BUMPER	-	-	-	-	-	-	-	-	-	-	-	80	-	80
REPLACING FUEL FILTER	-	-	-	-	-	-	-	-	65	-	-	-	-	65
REPAIRING GEAR LEVER	-	-	-	-	-	-	-	-	-	-	440	300	-	740
REPAIRING ENGINE	-	-	-	-	-	-	-	280	-	-	-	-	-	280
REPAIRING STARTER	-	-	-	-	-	-	-	-	260	-	-	-	-	260
STARTER O.O.O.	-	-	-	-	-	-	-	-	45	-	-	-	-	45
BATTERY O.O.O.	-	-	-	-	-	-	-	-	-	-	-	270	125	395
LOADER O.O.O.	-	-	-	-	-	-	-	626	270	-	-	-	-	896
FILLING UP WITH BREAK FLUID	-	-	-	44	-	-	44	-	-	-	-	-	-	-
FLAT WHEELS	-	-	-	22	-	-	22	-	-	-	-	-	-	-
TENSIONING WHEEL CHAINS	-	-	-	-	78	-	78	-	-	-	-	-	-	-
BROKEN OSCILATING BEARING	930	810	990	857	-	-	3587	-	-	-	-	-	-	-
BRAKES O.O.O.	-	-	-	-	242	840	1082	-	235	-	-	-	-	235
BROKEN MOUNTING BOLTS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CHANGING WHEELS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REP. PROTECTION PLATE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPLACING SHOVEL PIN	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BROKEN PUMP DRIVE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BROKEN FAN BELT	-	-	-	-	-	-	-	-	-	-	-	-	-	-
GEARBOX O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BURNT ELEC. COUPLINGS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING LIGHTS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING BATTERY COVER	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING EXHAUST	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL BREAKDOWNS	930	810	990	923	320	840	4813	943	905	-	440	767	170	3225
TONNAGE							1054							1770

	13.03.1976	15.03.1976	16.03.1976	17.03.1976	18.03.1976	19.03.1976	WEEK ENDING 19.03.1976	20.03.1976	22.03.1976	23.02.1976	24.03.1976	25.03.1976	26.03.1976	WEEK ENDING 26.03.1976
WORKING TIME	1110	358	729	709	1005	1322	5223	1616	672	882	781	947	143	4841
TRAVELLING TIME	85	48	49	31	36	85	334	96	65	43	48	48	5	305
SERVICE TIME	272	50	129	250	181	182	1664	188	189	208	169	105	47	906
TOTAL DELAY TIME	30	549	193	212	38	219	1241	118	47	47	95	55	-	362
TOTAL BREAKDOWN TIME	972	1130	1491	1385	1290	769	7037	644	1359	1290	1516	1427	2385	8621
TOTALS	2469	2135	2591	2587	2550	2577	14909	2462	2332	2470	2609	2582	2580	15035
DELAYS:-														
IDLE	30	10	25	-	14	10	89	60	-	-	-	-	-	60
REMOVING SCRAPERS	-	-	-	-	24	117	141	-	47	30	10	-	-	87
TEACHING DRIVER	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LOADER STUCK	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CLEANING PANEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FETCHING DIESEL	-	-	-	-	-	-	-	58	-	-	-	-	-	58
FLAT BATTERY	-	539	168	-	-	-	707	-	-	-	-	-	-	-
BARRING	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOWING OTHER LOADER	-	-	-	212	-	92	304	-	-	17	25	47	-	89
F.O.G.	-	-	-	-	-	-	-	-	-	-	60	-	-	60
CLEANING BUCKET	-	-	-	-	-	-	-	-	-	-	-	8	-	8
TOTAL DELAYS	30	549	193	212	38	219	1241	118	47	47	95	55	-	362
BREAKDOWNS														
REPLACE BATTERY	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING CANOPY	-	-	47	-	-	-	47	-	80	-	-	-	-	80
REPAIRING BUMPER	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPLACING FUEL FILTER	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING GEAR LEVER	-	-	-	-	-	-	-	-	125	-	-	-	-	125
REPAIRING ENGINE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
REPAIRING STARTER	-	-	-	-	-	-	-	-	-	-	141	-	-	141
STARTER O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BATTERY O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LOADER O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FILLING UP WITH BREAK FLUID	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FLAT WHEELS	47	-	-	-	-	-	47	-	125	-	20	-	-	145
TENSIONING WHEEL CHAINS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BROKEN OSCILATING BEARING	-	-	154	95	-	-	249	-	-	-	-	-	-	-
BRAKES O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
BROKEN MOUNTING BOLTS	625	1110	1290	450	-	-	3475	-	-	-	-	-	-	-
CHANGING WHEELS	-	-	-	-	-	73	73	112	-	-	-	35	-	147
REP. PROTECTION PLATE	300	-	-	-	-	-	300	-	-	-	-	-	-	-
REPLACING SHOVEL PIN	-	20	-	-	-	-	20	-	-	-	-	-	-	-
BROKEN PUMP DRIVE	-	-	-	840	1290	696	2826	-	-	-	-	-	-	-
BROKEN FAN BELT	-	-	-	-	-	-	-	482	-	-	-	-	-	482
GEARBOX O.O.O.	-	-	-	-	-	-	-	-	-	-	-	1290	2385	3675
BURNT ELEC. COUPLINGS	-	-	-	-	-	-	-	-	1029	1290	1290	-	-	3609
REPAIRING LIGHTS	-	-	-	-	-	-	-	-	-	-	-	56	-	56
REPAIRING BATTERY COVER	-	-	-	-	-	-	-	-	-	-	-	46	-	46
REPAIRING EXHAUST	-	-	-	-	-	-	-	50	-	-	65	-	-	115
TOTAL BREAKDOWNS	972	1130	1491	1385	1290	769	7073	644	1359	1290	1516	1427	2385	8621
TONNAGE							2283							2118

TABLE A5 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	27.03.1976	29.03.1976	30.03.1976	31.02.1976	01.04.1976	02.04.1976	WEEK ENDING 02.04.1976	03.04.1976	05.04.1976	06.04.1976	07.04.1976	08.04.1976	09.04.1976	WEEK ENDING 09.04.1976	10.04.1976	12.04.1976	13.04.1976	14.04.1976	15.04.1976	16.04.1976	WEEK ENDING 16.04.1976	17.04.1976	19.04.1976	20.04.1976	21.04.1976	22.04.1976	23.04.1976	WEEK ENDING 23.04.1976
WORKING TIME	-	853	929	760	383	346	3271	982	391	-	-	-	-	1373	-	286	656	568	254	-	1764	396	588	534	677	560	784	3539
TRAVELLING TIME	-	25	55	47	34	25	186	54	20	-	-	-	-	74	-	9	52	28	14		103	36	29	33	40	30	29	197
SERVICE TIME	-	88	82	85	69	95	419	99	60	-	-	-	-	159	-	42	94	47	22		205	46	111	41	140	87	60	485
TOTAL DELAY TIME	-	135	39	20	91	52	337	-	14	-	-	-	-	14	-	56	21	-	15		92	13	-	44	16	30	18	121
TOTAL BREAKDOWN TIME ..	2460	1144	1478	1673	1998	2062	10825	1325	1735	2580	2580	2580	2580	13380	2460	1827	1776	1937	2280		10280	2014	1499	1930	1880	1910	1693	10926
TOTALS	2460	2245	2583	2585	2575	2590	15028	2460	2220	2580	2580	2580	2580	15090	2460	2220	2599	2580	2585		12444	2505	2227	2582	2753	2617	2584	15268
DELAYS:-																												
TOWING OTHER LOADER ...	-	129	-	-	91	52	272	-	-	-	-	-	-	-	-	56	-	-	-		56	-	-	-	-	-	-	-
REMOVING SCRAPERS	-	-	-	-	-	-	-	-	14	-	-	-	-	14	-	-	21	-	15		36	13	-	22	16	-	[illegible]	9
IDLE	-	6	15	20	-	-	41	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	22	-	-		22
CLEANING BUCKET	-	-	24	-	-	-	24	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-			-
TIPS FULL	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	[illegible]	-	30
TOTAL DELAYS	-	135	39	20	91	52	337	-	14	-	-	-	-	14	-	56	21	-	15		92	13	-	44	16	30	18	121
BREAKDOWNS:-																												
BROKEN OIL PIPE	-	-	-	-	-	107	107	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
REPLACING FUEL FILTER .	-	-	-	-	235	-	235	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
REPLACING BUCKET	-	-	-	-	323	-	323	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
REPAIRING BRAKES	-	-	-	-	40	-	40	-	-	-	-	-	-	-	-	-	-	-	-		-	144	-	-	-	260	-	404
REPAIRING STARTER	-	-	-	-	75	-	75	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
REPLACING BUCKET PINS ..	-	-	-	25	35	-	60	-	-	-	-	-	-	-	-	-	-	-	-	MINE HOLIDAY	-	-	-	-	-	-	-	-
REPAIRING WHEEL CHAINS .	-	-	188	58	-	-	246	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	103	103
FLAT WHEELS	-	34	-	-	-	-	34	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
GEARBOX O.O.O.	2460	1110	1290	1290	-	665	6815	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-
BROKEN FAN BELT	-	-	-	-	-	-	-	64	80	-	-	-	-	144	-	-	-	-	-		-	-	-	-	-	-	-	-
BROKEN GEAR LEVER	-	-	-	-	-	-	-	31	-	-	-	-	-	31	-	-	-	-	-		-	-	-	-	-	-	-	-
BROKEN DRIVE SHAFT	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	647	1290		1937	1230	1110	1290	1290	-	-	4920
ALTERNATOR O.O.O.	-	-	-	-	-	-	-	-	145	-	-	-	-	145	-	-	276	-	-		276	-	-	-	-	-	-	-
BATTERY O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	210	-	-		210	-	-	-	-	160	-	160
BROKEN UNIVERSAL JOINT .	-	-	-	300	1290	1290	2860	1230	1510	2580	1290	1290	1290	9190	1230	380	1290	1290	990		6180	-	-	-	-	-	-	-
STARTER O.O.O.	-	-	-	-	-	-	-	-	-	-	1290	1290	1290	3870	1230	447	-	-	-		1677	640	-	-	-	-	300	940
ENGINE SMOKING EXCESSIVELY	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		-	-	189	640	590	1290	1290	4199
STEERING O.O.O.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	-	200	-	200
TOTAL BREAKDOWNS ..	2460	1144	1479	1673	1998	2062	10815	1325	1735	2580	2580	2580	2580	13380	2460	1827	1776	1937	2280		10280	2014	1499	1930	1880	1910	1693	10926
TONNAGE							1130							479							643							1589

TABLE A6 DAILY ANALYSIS OF SCHOPF LOADERS

DAILY ANALYSIS OF SCHOPF LOADERS

	24.04.1976	26.04.1976	27.04.1976	28.04.1976	29.04.1976	30.04.1976	WEEK ENDING 30.04.1976	01.05.1976	03.05.1976	04.05.1976	05.05.1976	06.05.1976	07.05.1976	WEEK ENDING 07.05.1976	08.05.1976	10.05.1976	11.05.1976	12.05.1976	13.05.1976	14.05.1976	WEEK ENDING 14.05.1976
WORKING TIME	448	-	129	1139	1024	1085	3829	200	250	169	782	642	633	2670	683	238	261	-	-	-	1182
TRAVELLING TIME	26	-	14	32	54	56	182	12	5	22	69	32	48	188	48	11	14	-	-	-	73
SERVICE TIME	40	-	26	65	166	171	468	15	15	25	293	70	90	508	89	300	20	-	-	-	409
TOTAL DELAY TIME	-	-	-	-	65	-	65	30	30	-	-	-	10	70	-	-	-	-	-	-	-
TOTAL BREAKDOWN TIME	1946	2220	2411	1355	1290	1290	10512	2208	1920	2364	1450	1845	1802	11569	1646	1656	2285	2580	2580	2580	13327
TOTALS	2460	2220	2580	2591	2599	2606	15056	2465	2220	2580	2594	2589	2587	15031	2466	2205	2580	2580	2580	2580	14991
DELAYS																					
REMOVING SCRAPERS	-	-	-	-	30	-	30	-	-	-	-	-	10	10	-	-	-	-	-	-	-
FETCHING DIESEL	-	-	-	-	-	-	-	-	30	-	-	-	-	30	-	-	-	-	-	-	-
TIPS FULL	-	-	-	-	35	-	35	30	-	-	-	-	-	30	-	-	-	-	-	-	-
TOTAL DELAY TIME	-	-	-	-	65	-	65	10	30	-	-	-	10	70	-	-	-	-	-	-	-
BREAKDOWNS																					
DIFFERENTIAL O.O.O.	110	1110	450	-	-	-	1670	-	-	-	-	-	-	-	-	-	-	-	-	-	-
STAR O.O.O.	806	-	671	65	-	-	1542	-	-	1074	-	-	-	1074	-	-	-	-	-	-	-
BROKEN DRIVE SHAFT	-	-	-	-	-	-	-	978	810	-	-	-	-	1788	-	-	-	-	-	-	-
BROKEN PISTON RINGS	1230	1110	1290	1290	1290	1290	7500	1230	1110	1290	1290	1290	1290	7500	1230	1110	1290	1290	1290	1290	7500
BROKEN GEARBOX AND ENGINE	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	995	1290	1290	1290	4865
REPLACING SHOVEL PINS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	36	-	-	-	-	36
LOOSE WHEEL CHAINS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	95	-	-	-	-	-	95
BATTERY O.O.O.	-	-	-	-	-	-	-	-	-	-	160	555	150	865	220	510	-	-	-	-	730
LEAKING HYDRAULIC HOSES	-	-	-	-	-	-	-	-	-	-	-	-	362	362	101	-	-	-	-	-	101
TOTAL BREAKDOWN TIME	1946	2220	2411	1355	1290	1290	10512	2208	1920	2364	1450	1845	1802	11569	1646	1656	2285	2580	2580	2580	13327
TONNAGE							1704							1030							432

APPENDIX B

STANDARD LABOUR ALLOCATION

GROUT PACK SUPPORT METHOD

LABOUR CATEGORY

Stope Width (cms)	190	210	230	250
Day Shift				
Team Leader	1	1	1	1
Gang Supervisor	3	3	3	3
Winch Driver	4	4	4	4
Rockdrill Operator	8	8	8	8
Rockdrill Assistant	5	5	5	5
Miner's Assistant	4	4	4	4
Teambar	3	3	3	3
Timber	4	4	5	6
Transport/Mono winch	2	5	5	6
Lashing	5	4	4	4
Total Day Shift	39	41	42	44
Night Shift				
Team Leader	1	1	1	1
Gang Supervisor	2	2	2	2
Bar/Lash	4	4	4	4
Winch Drivers	4	4	4	4
Total Night Shift	11	11	11	11
Grand Total	50	52	53	55
Tons per Black per Shift	6,0	6,4	6,8	7,2
Tons Broken per Shift	300	333	360	396

REFERENCES

1. Planning Department. Monthly Productivity and Efficiency Summaries. **Company Records.** 1975/1976.

2. Administration Department. Monthly *Financial Statistics.* Company Records. 1975 to 1987.

3. Loss Control Department. Monthly Statistical Analysis. *Company* **Records.** 1975 to 1987.

4. Geology Department. Feasibility Study Reports. **Company Records.** 1986 and 1987.

5. Industrial Engineering Department. Standard labour allocation. **Company Records.** 1975.

6. Rooker, D.J. Mechanised Gold Mining - A layout for using Load-Haul-Dump *units in deep level mines.* A *project* report submitted to the Faculty of Engineering, University of Witwatersrand, *Johannesburg, for* the degree of Master of Science in Engineering, 1987.

7. Rhodes, K.A. 90 Level E8 Reef Trackless Mechanised Mining Project. **Company Feasibility Study.** 1984.

8. Webbstock, D.S. 90 Level E8 Reef Trackless Mechanised Mining Project. **Feasibility Study Report.** 1986.

BIBLIOGRAPHY

Rhodes, K.A. Wide Reef Mechanised Room and Pillar Operations at Cooke 2 Shaft. The Randfontein Estates Gold Mining Company, Witwatersrand, Limited. **A.M.M. Circular.** 1986.

Keating, J.M. "Rails vs Trackless Study" Abstract. Plaserville ~~California, Lake Shore INC. Iron Mountain.~~ December 1970.

Rutherford, J.G. Environment Control at Inco underground diesel operations April 1971. The International Nickel Company of Canada, Limited Copper Cliff, Ontario.

Rao, K.V. Coal Productive Potential of Diesel Load-Haul-Dump Equipment and Systems. Virginia Polytechnic Institute and State University.

Hardman, D.R. Productivity Potential of Load-Haul-Dump machines. Chamber of Mines of South Africa.

Fauman, H.C. Engineering and Material factors resulting in increased availability for L.H.D.'s. Eimco Tunneling and Mining Machinery Division, Envirotech Corporation.

Hews, C.F. Trackless Mining at the International Nickel Company of Canada, Limited. August 1973.

Wagner Mining Equipment Company, Equipment features and application data - Catalog 150A, **Technical Manual.**

Hughes. F.H.R. Case histories of Dome Embedded Tyres. Dome Tyre (Pty.) Ltd. 1976.

Kirchner, M.H. Trackless Mining at Thabazimbi Mine. August 1973.

Grobler, M.S. Trackless Mining at Prieska Copper Mines (Pty.) Ltd. August 1973.

Espach, T.M. Improvements to the Trackless Mining Equipment and Operation at Prieska Copper Mines (Pty.) Ltd. April 1979.

Williams, A.W. and Diering, D.H. Trackless Mining - The Western Deeps View. 1985.

Ferguson, B.A., Vezina, R.O., Boissonneault, R.R. and Gandet, M. Innovative Management Techniques at Kiena Gold Mines Limited. January 1987.