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The Extraction of Pillars, largely by Trackless Mining Methods, at Randfontein Estates Gold Mining Company (Witwatersrand) Ltd

by

Clem Albert Foster Sweet

A Disertation submitted to the Faculty of Engineering,
University of the Witwatersrand for the degree
Master of Science in Engineering

July 1998

DECLARATION

I declare that this dissertation has not been submitted previously as a dissertation, or a thesis, for any degree at any university. The work was undertaken while I was employed as a Production Manager on Randfontein Estates Gold Mining Co (W) Ltd Cooke 1 Shaft. Randfontein Estates Gold Mining Co (W) Ltd is part of the Johannesburg Consolidated Investment Company Limited.



C A F SWEET

Dated this .....6.... day of ..July.....1998.

ABSTRACT

This dissertation covers the strategy applied in extracting Gold ore from geologically disturbed areas with dips ranging from flat through to ±60°. A further complicating factor was that the reef thickness varied from around 1 meter up to 3,5 meters wide.

This project covers the aspects from system planning and investigation of mining methods, the computer simulations of pillar dimensions and the predicted stress patterns. The project then proceeds to explain the mining methods applied and some reasons for having chosen these particular methods.

The results of these mining operations are evaluated and comments are made on relevant matters. Some of the problems encountered are explained and finally some of the lessons learned are enumerated so as to assist other persons who are faced with similar type of problems.

TO : KATHY

MY WIFE AND MY FRIEND

ACKNOW .EDGEMENTS

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- Mr. W A Nairn, M.D. Gold Division of J.C.I. and Mr. B R Fleetwood General Manager Randfontein Estates.

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CHAPTER 1

Introduction

According to the historians gold was first discovered on the Witwatersrand in March 1886 at Langlaagte in what was described as a "crumbly sediment containing large numbers of rounded pebbles of various colours". This became known as 'banket' which was named after a Boer confectionery of similar appearance which was very popular at the time.

Within a very short period of the initial gold find evidence of banket was traced over a distance of nearly 50 kilometres from east to west. The Western limit being traced to the farm Randfontein.

In 1886 Cecil John Rhodes, Hans Sauer and a Boer prospector by the name of Gericke visited the farm Randfontein to examine the reefs but Gericke was unsure of these reefs as they did not resemble the main reef or its distinctive red marker band. However, J B Robinson, who had acquired a large number of farms to the east of Langlaagte believed that the reefs to the west were an extension to those found on Langlaagte and set about buying up whatever farms he believed to have reef on them including Randfontein.

Early in 1889 the Randfontein Gold Farm's Company was floated with a capital of £2,000 000 of £1 shares and in March 1889 "The Randfontein Estates Gold Mining Company, Witwatersrand", Limited was registered in London.

Mining continued on Randfontein Estates almost unabated and in the late 1950's had 28 working shafts and winzes with around 21 000 employees. By the early 1960's the Uranium contract was nearing its end and the mine was winding down. At that time the mine was primarily a Uranium producer with gold as a by-product.

The search for gold continued and between 1962 and 1965, 31 boreholes were drilled on the farms Gemsbokfontein and Panvlakte, but only 10 holes were of economic value and interest in floating a new mine was extremely low, all the time the mine continued to wind down and by 1967 Randfontein Estates employed only 1120 people.

After continued drilling and numerous efforts to raise the funds to establish a new mine, in September 1970 the first sod was turned in what was to be the Cooke Section of Randfontein Estates. Trial stoping was scheduled to begin at the end of 1973 and in March 1974 Cooke 1

reached the full planned production of 70 000 tons of reef per month. By 1980 Cooke 1 Shaft, which had been designed to hoist 70 000 tons of reef and 25 000 tons of waste per month was now producing 125 000 tons of reef and 50 000 tons of waste per month.

In 1982 JCI at Western Area Gold Mine initiated the first trackless mining areas within the Group.

Trackless mining has been described as the replacement of track mounted locomotives, hoppers, scraper winches and hand held drilling equipment with rubber tyred load haul dump (L.H.D) machines, articulated dump trucks and with varying applications of mechanised drilling amongst which are the Electro Hydraulic drill rigs. Transport and technical support vehicles vary from modified four-wheel drive vehicles and tractors to purpose built transport equipment.

Trackless mining therefore is the conversion of a labour intensive mining system, to highly mechanised, highly technical system requiring less, but more highly skilled people. Two of J.C.I.'s major gold mines were facing declining grades but also had some very wide reefs, the mining of which was difficult and hazardous by conventional means. Trackless mining meant that lower grade ore's could be mined profitably.

Trackless Mechanised Mining commenced at Cooke 1 Shaft in August 1985 when the twin haulages to the new Doornkop shaft were temporarily stopped in bad ground and due to the intersection of water. These haulages were developed using trackless equipment loading into railbound hoppers. The K9a reef had been intersected a distance back from the working faces, and the decision was taken to do exploratory development on this previously unmined reef horizon while the support and cementation operations in the twin haulages were in progress.

This initial development led to the establishment of the K9a project with two other projects commencing soon thereafter. Mining in the window and bowl area which later was to constitute the remnant areas started in April 1986.

During the period of research, a detailed literature survey of a great many separate publications, papers and reports as well as Doctoral Thesis[20] and Dissertations on pillars, remnants and trackless mining was carried out by the author. It is of importance to note that although the Mine Managers' papers, Anglo American Group Mining Symposium papers, Johannesburg Consolidated Investment Company Technical papers and reports as well as the reports from

various mining conferences record various activities and operations relating to pillars, remnants and trackless mechanised mining, most were found to adequately cover the subject under review, few if any, detailed the process of determining a strategy and then following it through to fruition.

Not much research had yet been done on pillars and their designs particularly on wide reef mechanised mining layouts at that time understandably as the methods vary considerably. As one of the mining layouts was similar to practices in coal mines the design of these pillars (references [1,2,5,21,29]) were studied to evaluate this information. The effect of fracturing, strengths[6] of the material as well as deformation of the pillars (references [7,13,15,19,28]) the prediction of rock bursts[23] were all part of the initial study. The possibility of using mobile roof support[25] was considered where the hanging wall became friable and the values recovered could cover the additional costs. As it was, this option was not pursued).

In order to make the extraction as safe as possible and to maximise returns a number of mining methods and new developments [3,4] were reviewed including Trackless Mechanised Mining [8,9,10,20]. Over the years a number of papers had been published on up dip mining [17] and the extraction of shaft pillars [11,12] and whatever information, that was relevant to the situation, was cleared from these. One of the areas that we were all unsure of was the effect of stress [15] on critical excavations such as the workshops, haulages and workings. The determination of this stress would be a important part of the project.

The literature base is extremely wide and the information tends to be fragmented.

It is the intention of the author of this dissertation to prepare a text so that the student as well as practising Mining Engineers may refer to this as a practical method as well as a source of reference which will enable them to understand the initial problems, stimulate the necessary thinking and provide some options for addressing the problem of pillar and remnant extraction. Many more mines are facing similar situations as they near the end of their production. A brief review of some of the thinking and solution strategies are outlined. These are covered in greater detail within the text.

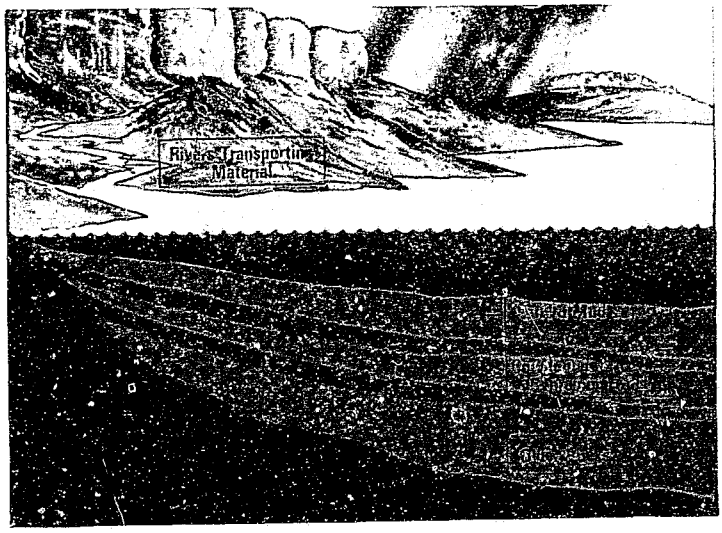


Plate 1.1
J.B. Robinson



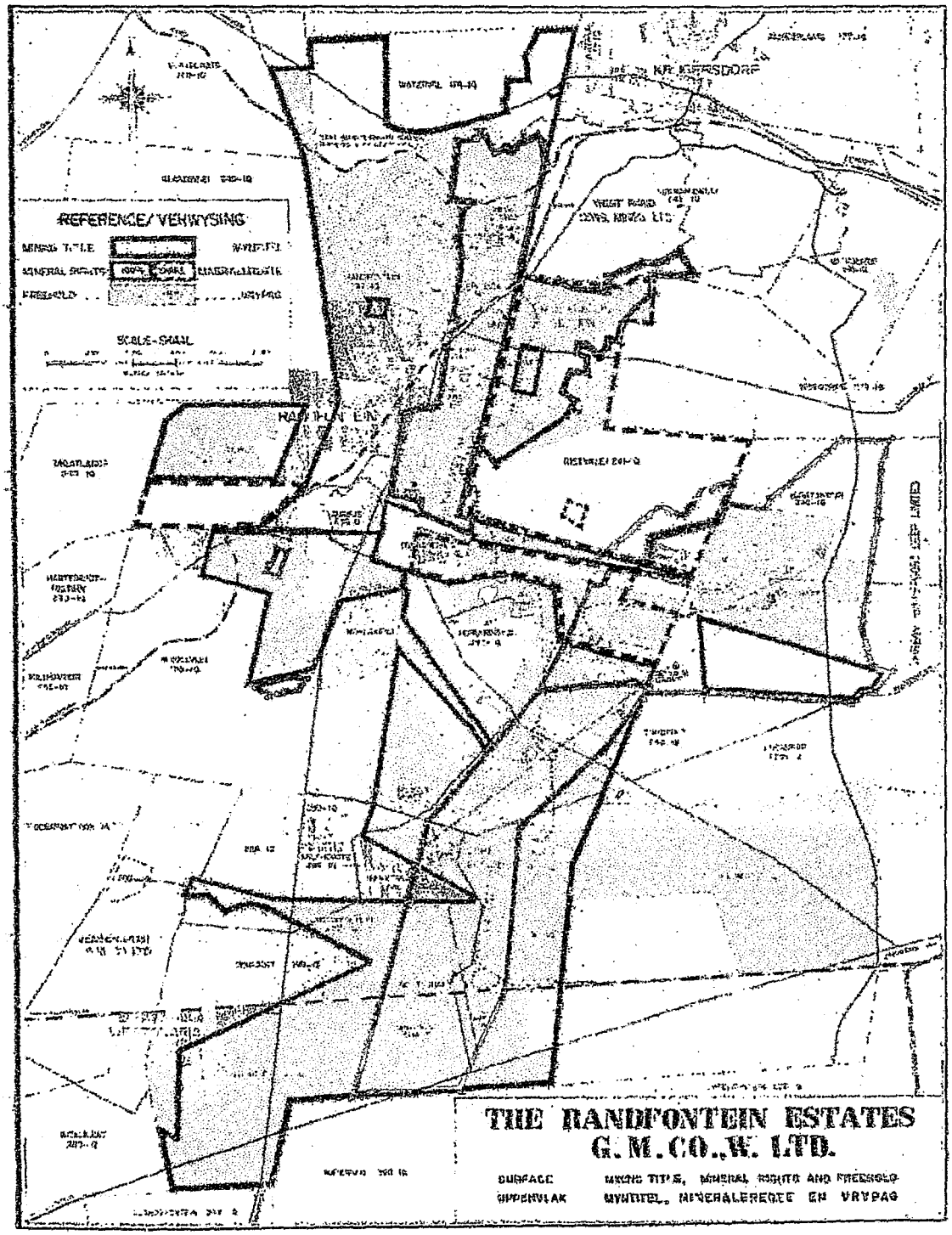
Plate 1.2
Debenture Certificate

Plate 1.3



Deposition of the gold Reefs

Plate 1.4 Surface lease and Mineral Rights plan



CHAPTER 2

Geology

2.1 Introduction

The geology of the two areas under review have many points in common but each had some specific local conditions which to a greater or lesser degree influenced the decisions taken on the mining layouts and methods of extraction. The geology is illustrated by an east-west section through Cooke 1 Shaft. The sub-outcrops of the various reef horizons with the younger overlying strata as well as the major dykes and faults are shown in figure 2.0.

2.2 Transvaal Sequence

The Cooke section is overlain by the Malmani Dolomites of the Chuniespoort Group. These vary in thickness from approximately 600m in the south to 100m to the north. The dolomites are significant for their water bearing properties with their potential for flooding a mine and their consequent association with sinkhole formation.

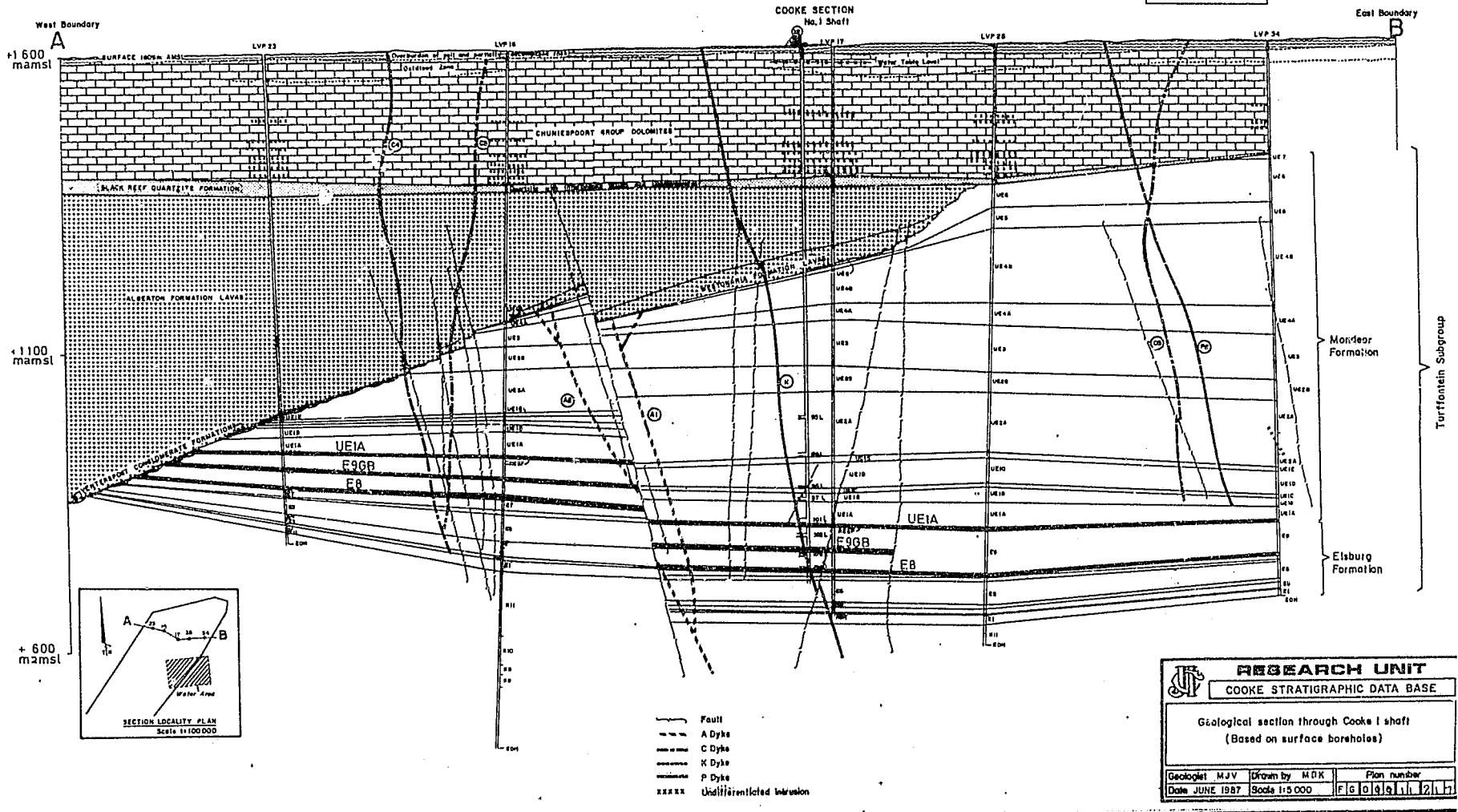
Conformably underlying the dolomites is the Black Reef formation comprising of grey quartzite, black shales and a conglomerate. The latter unit displays very patchy economic potential. Exploitation of this reef so close to the dolomites would, however, prove to be hazardous due to the presence of the water bearing horizon.

2.3 The Black Reef formation is underlain in part by a wedge of andesitic competent lava of the Klipriviersberg Group. The Black Reef formation is essentially flat, lying on an erosional contact with the lava to the west and directly on the Witwatersrand quartzite to the east. The lava is seen to dip at between 20 to 40 degrees to the west.

2.4 Venterspost Conglomerate Formation

The Venterpost conglomerate is found immediately below the lava of the Klipsriviersberg Group. It is a well packed conglomerate with poor mineralisation dipping between 20 and 40 degrees to the west and lies unconformably on the Witwatersrand quartzite.

Figure 2


West Boundary
A
COOKE SECTION
No. 1 Shaft
East Boundary
B
+1 600 mamsl
+1100 mamsl
+ 600 mamsl
LVP 23
LVP 18
LVP 17
LVP 28
LVP 34
CHUNIESPOORT GROUP DOLOMITES
BLACK REEF QUARTZITE FORMATION
ALBERTON FORMATION LAVAS
UE1A
E90B
E8
Mondeor Formation
Elsburg Formation
Turffontein Subgroup
SECTION LOCALITY PLAN
Scale 1:100 000
Water Area
Fault
A Dyke
C Dyke
K Dyke
P Dyke
Undifferentiated Intrusion
RESEARCH UNIT
COOKE STRATIGRAPHIC DATA BASE
Geological section through Cooke 1 shaft
(Based on surface boreholes)
Geologist MJV
Drawn by MDK
Date JUNE 1987
Scale 1:5 000
Plan number
F G 0 0 9 1 1 2 1 1

2.5 Witwatersrand Supergroup

The near complete stratigraphic column, Figure 2.1 illustrates the reefs currently being minded at Cooke 1 Shaft. The reefs are the Upper Elsburg UE1A, The Elsburg E9Gd, E9Gb, E8 and the Kimberly K9a reef.

The UEA1A is the lower most reef of the so called upper Elsburg zone and underlies the entire Cooke section with a regional dip of approximately 15 degrees to the east. The UE1A is underlain by the E9Gd, separated by a quartz middling varying in thickness from 10cm to 300cm. The E9Gb is 40m below the UE1A. The E8 is found between 35 and 80m below the UE1A reef band and eventually sub-outcrops against the Ventersdorp Conglomerate.

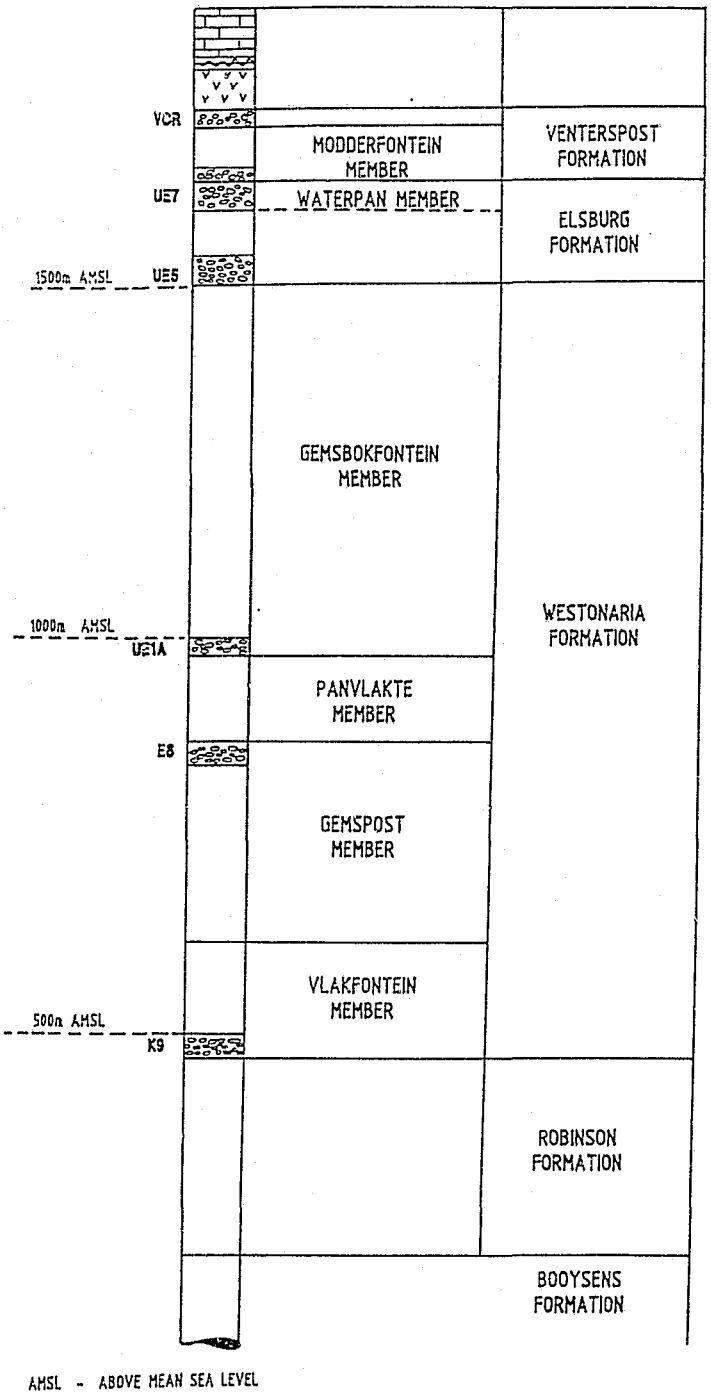
2.6 Structure

In general the regional dip of the UE1A reef is to the south, however, due to folding in two main directions the contours form a sinuous pattern. The UE1A reef therefore basically forms a basin with the synclinal axis striking eastwards. See figure 2.4 and 2.6 which shows this type of structure.

A major fault zone, parallel to the Panvlakte fault to the east is to be found at Cooke 1 with varying displacements. The total fault zone has a displacement of some 800 metres, the total details are still undefined but one of the faults traversed has a throw of 60m.

Four types of dykes are evident in the area, the "A" Dyke being the oldest. This is a dark fine-grained dyke, some 20 metres wide. Very little movement has taken place across this dyke which strikes in a north-north-east direction. The next period of dyke intrusion was the "C" dykes, which strike in an east-south-east direction. They are usually in the region of 5 metres thick, light grey in colour, fine-grained aphyric (glassy with small crystals) material. The "C" dykes generally caused displacement of the UE1A reef with displacements of up to 60 metres and at times with a scissor action found so preverently in the pillar areas. The P2 dyke is believed to be of the Pilanesberg age and comprise medium grained dark green material. Both the P2 and the C-dykes are known to be water bearing.

Figure 2.1



AMSL - ABOVE MEAN SEA LEVEL

GENERALIZED STRATIGRAPHIC COLUMN OF COOKE 1 SHAFT

The youngest phase of dykes are the "K' dykes presumed to be of Karoo age. They vary in thickness from 10 to 20 metres usually with little or no displacement. They generally strike east-south-east and are composed of dark fine-grained aphyric material. All the dykes mentioned are dolerite.

2.7 Local Geology

The 95 Haulage was developed on the UE1A and the E9Gd reef horizons. These two reef bands are found one on top of one another with no middling and are commonly referred to as the Composite Reef. The total reef width of the Composite Reef package varies between 1.5m and 2.5m.

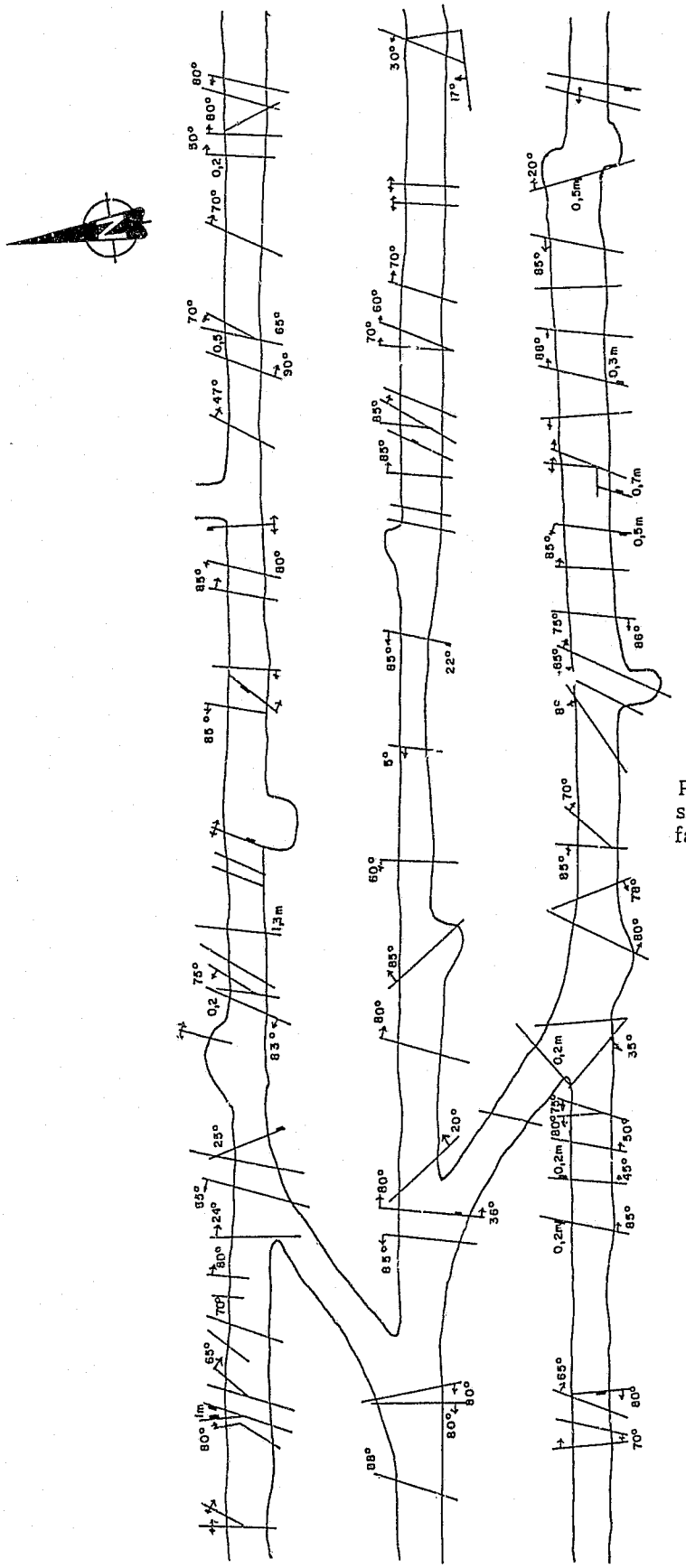
The UE1A reef band is generally a small to medium pebble supported, oligomictic quartz pebble conglomerate forming layers up to 20cm thick and often separated by thin quartzite partings. The UE1A is also, on rare occasions, represented by a rich carbon mud seam on its top contact and very high gold values are associated with these occurrences.

The E9Gd is generally more arenaceous than the overlying UE1A reef and often includes distinctive quartzitic middlings. The pebble size is similar to that of the UE1A, although large pebbles and cobbles may occur. Pyrite buckshot, up to 15mm in diameter is found in the composite reef package and this phenomena is generally associated with good gold and uranium values.

The general geology of the 95 East Pillar was well known as a result of previous mining activities in the late 1970's. There were numerous faults cutting across the pillar at right angles to the longitudinal axis of the pillar in a north-south direction displacing the reef up to 4.5m. Figures 2.2 and 2.3. The reef is shaded and the results of the faulting on the reef position is clearly visible. The result of this faulting will be seen under the section on Mining of the 96 pillar.

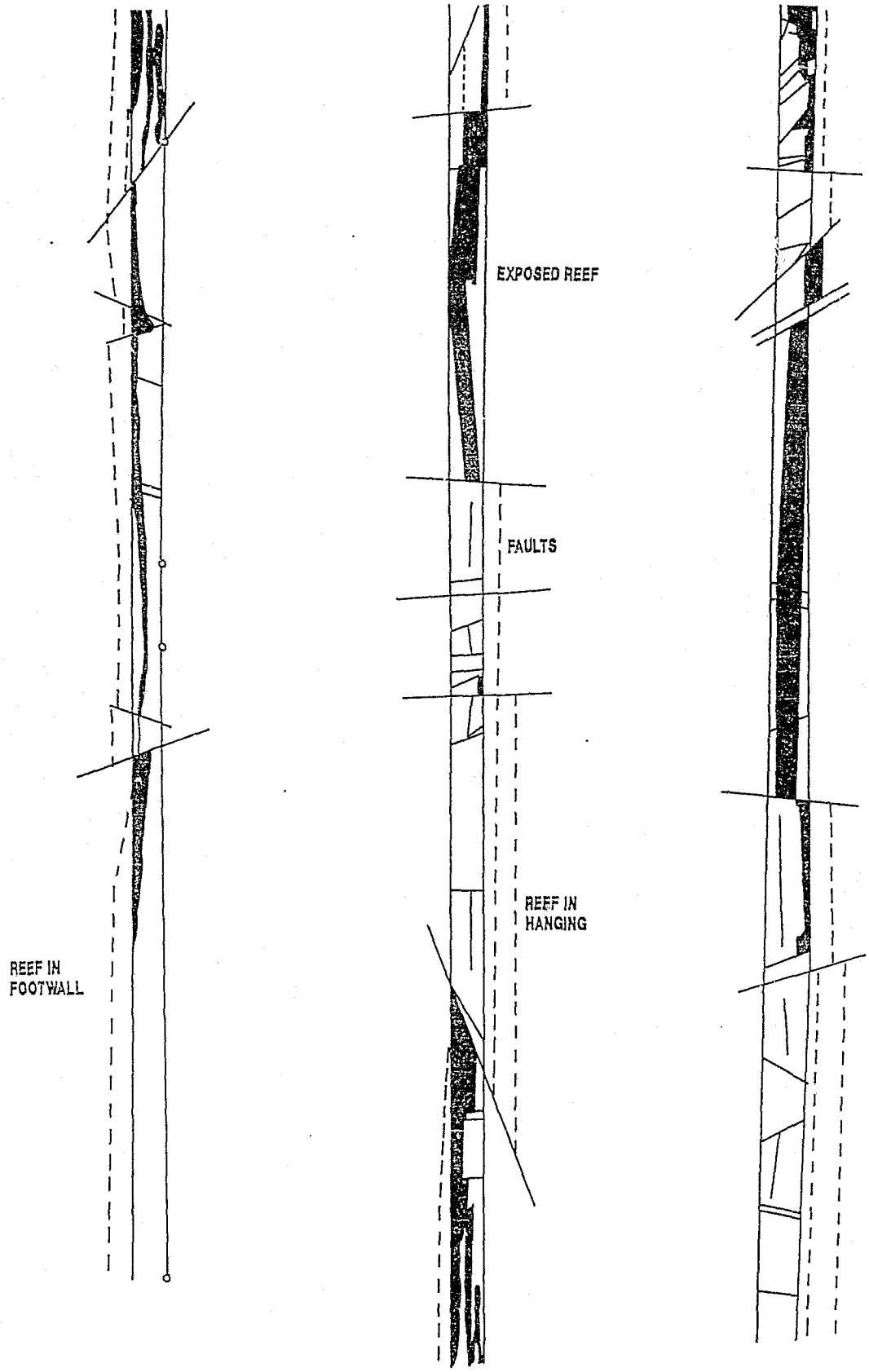
The 101 area was made up of two blocks of ground. The first, known locally as the Window area (Figure 2.5), consisted of a block of UE1A reef approximately 500m by 160m situated north-north east of the shaft. In this area the UE1A and the E9Gd reef

Figure 2.2



Portion of 95 pillar showing incidence of faulting.

Figure 2.3 Section 95 Pillar



bands are found to be contiguous with no middling and are commonly referred to as the Composite Reef. The E9Gd values were extremely erratic and its pay trends did not follow those of the UE1A. The unmined block of ground was heavily influenced by numerous faults and dykes. The area in general was heavily faulted with the dykes having displacements that apparently followed no pattern with many faults being scissor faults and / or splitting from other faults.

The second area, known locally as the Bowl Area, (Figure 2.4) because its structure resembled a bowl, had previously been left unmined because of its complicated structure, due to faulting and the presence of two major dykes.

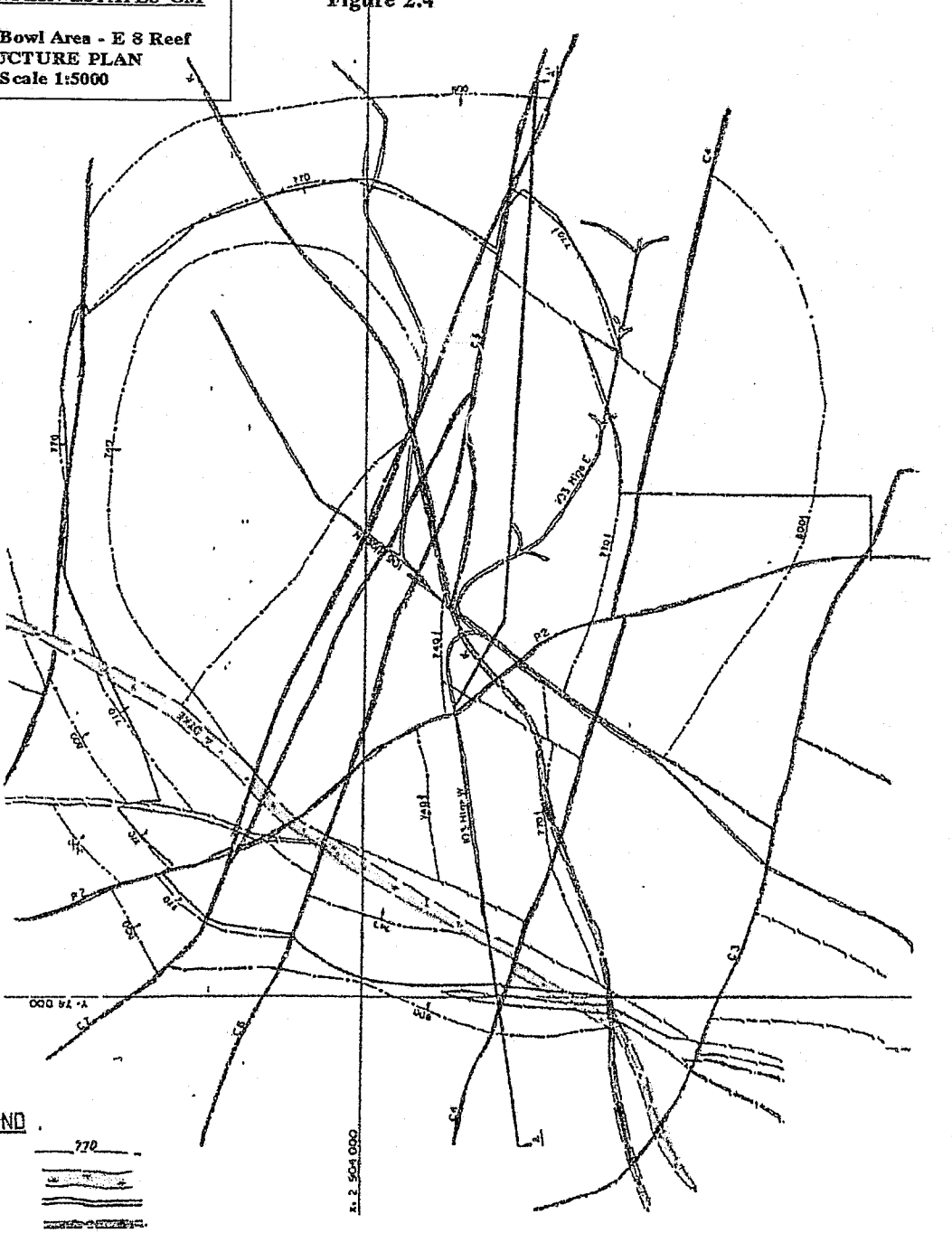
The first is the P2 dyke – known to be a water bearing dyke – and extreme care had to be exercised when approaching or traversing it; the second is the K2 Dyke which was known to have extensive secondary faulting associated with it. The Bowl Area is extensively on the UE1A and the E9Gd reef horizon (composite reef) with no or insignificant middlings in between.

The general geology of the two areas was believe to have been well mapped from the mining that had occurred as recently as 1986 although some areas were mined in the mid to late 1970's. During this project, a number of headings intersected faults that had split from other faults or the displacement was found to be very different from that projected. On a number of occasions faults were traversed where the displacement had reduced virtually to zero.

RANDFONTEIN ESTATES GM

Cooke 1 Bowl Area - E 8 Reef
STUCTURE PLAN
Scale 1:5000

Figure 2.4



Structure Plan - Bowl Area

Figure 2.5

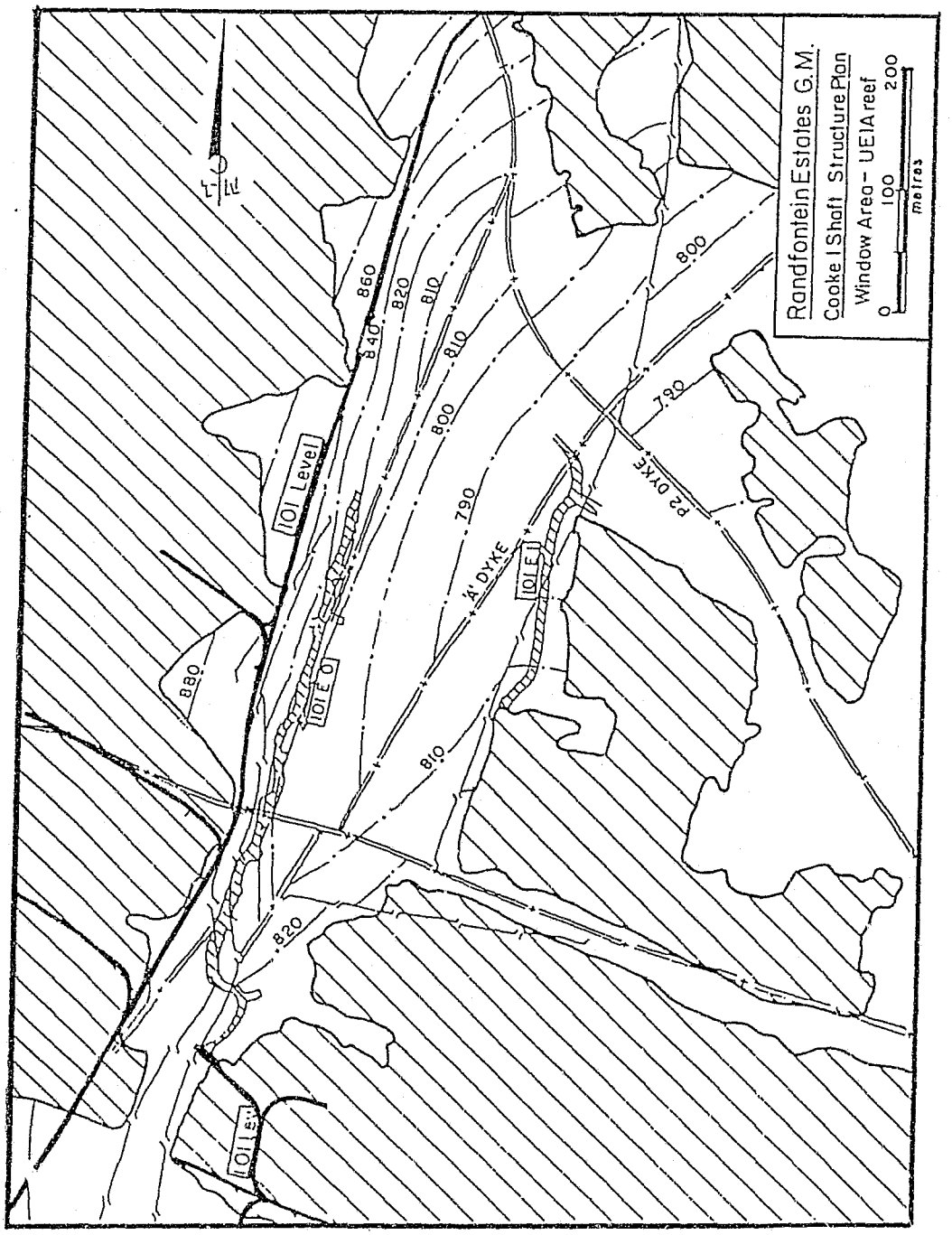
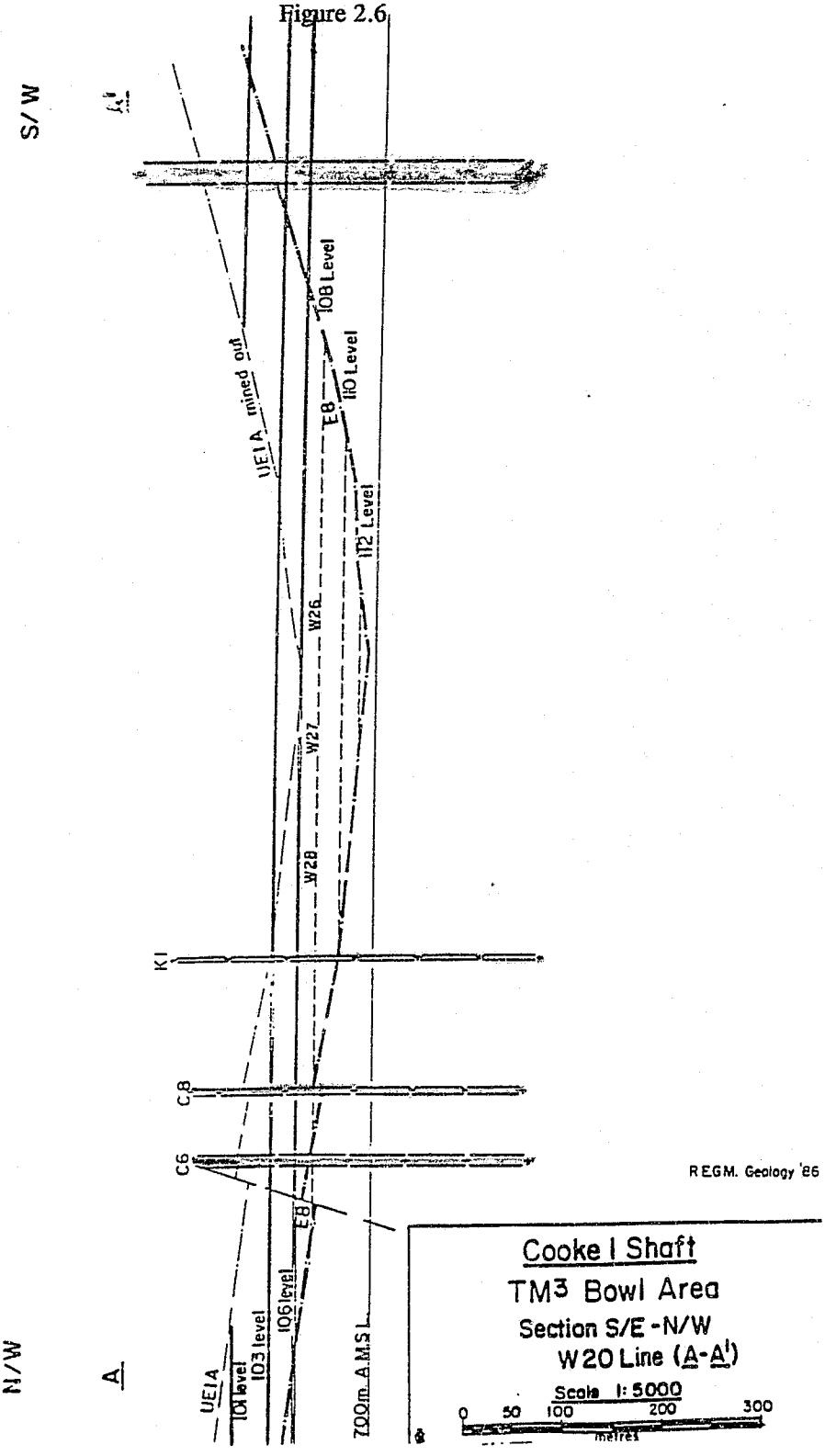

Randfontein Estates G.M.
Cooke I Shaft Structure Plan
Window Area - UE1A reef
0 100 200
metres
101 Level
101 E 01
101 E 1
A' DYKE
P2 DYKE
860
840
820
810
800
790
780
880

Figure 2.6


S/W
A'
108 Level
UE1A mined out
E8
110 Level
112 Level
W26
W27
W28
K1
C8
C6
E8
UE1A
101 level
103 level
106 level
700m A.M.S.L.
N/W
A
R.E.G.M. Geology '86
Cooke I Shaft
TM3 Bowl Area
Section S/E - N/W
W20 Line (A-A')
Scale 1:5000
0 50 100 200 300
metres

CHAPTER 3

History of the Areas

3.1 Introduction

Pillars are left for various reasons including regional support, haulage pillars to protect tramming routes, shaft pillars to protect the shaft or due to economics at the time or mining difficulties that made the extraction of the pillar unattractive. In the case of the pillars under review, 95 haulage pillar was left to protect part of the haulage and was heavily faulted. The 101 pillars were left, due to the Window area being heavily faulted and the development and re-development costs making it difficult and expensive to mine. Most of the previous working faces had stoped on faults or dykes with many having had serious support problems.

3.2 **95 Haulage Pillar** area was last mined in the late 1970's. The pillar was highly faulted and contained high grade ore. The haulage had been extensively supported using wire mesh, lacing and shepherd crooks as well as steel arch sets in particularly bad areas generally associated with faults. In the intervening years a lot of obsolete mine equipment had been abandoned in the old haulages.

3.3 101 Area

The area was primarily left because of difficult mining conditions resulting from the faulting, and the shape of the deposit, i.e. the lower section of the bowl extended to below the lowest ore transport haulage. The bowl is not uniformly payable and has very distinct pay shoots and predicted barren areas. The reef thickness was known to vary from 60cm of high grade ore, to 4 metres with grades varying from marginal to highly payable.

CHAPTER 4

Mining Strategies

4.1 Introduction

Numerous factors influenced the overall mining strategy, some of which have been referred to earlier. These factors include the stress regime both due to depth as well as the induced stresses from the surrounding previously mined areas. In addition there were the extremes of geological complexity, the pressure on existing excavations, tonnage output requirements and the nature and complexity of the ore body.

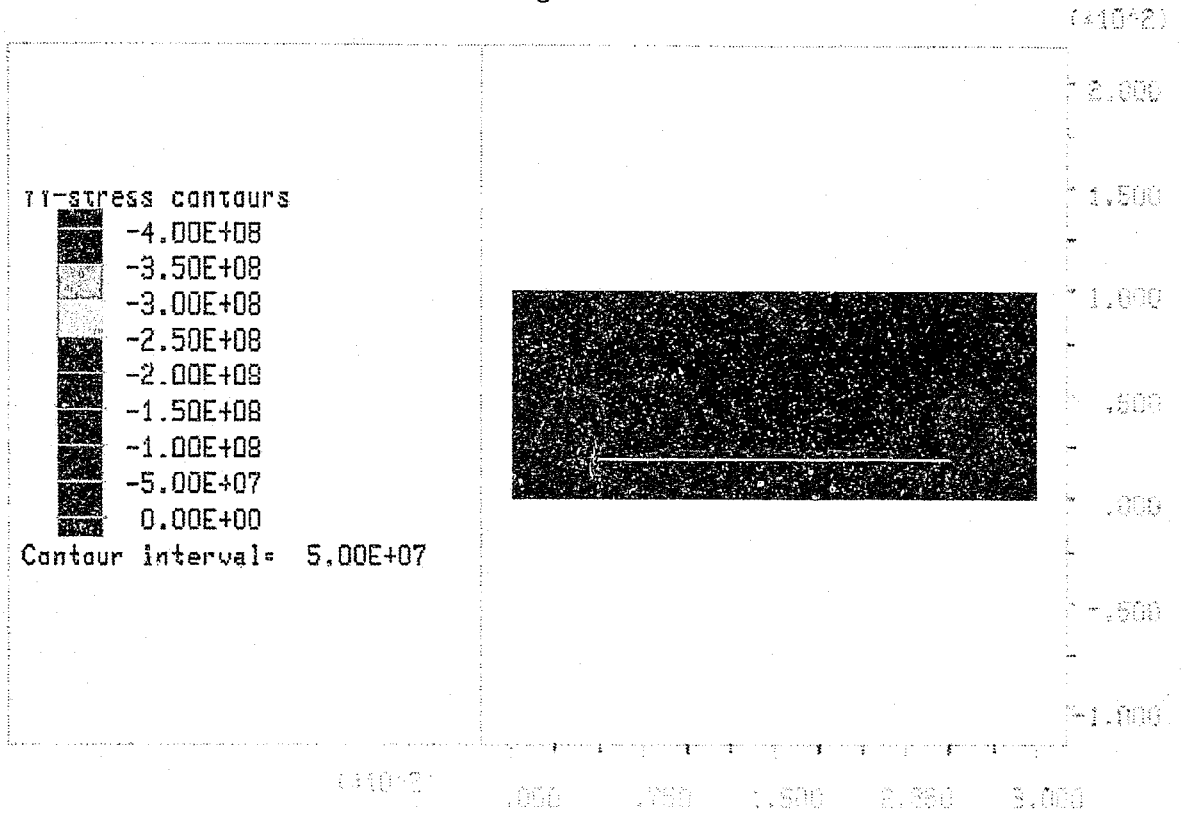
In the 1988 Guide to the Ameliorating of Rockbursts paragraph 1.1 on page 3 the following relevant statement appears: 'Mine design strategies to minimise the effects of rockfalls and rockbursts include the optimising of geometric layouts, the sequencing of extraction operations and the use of regional support systems". These had to be evaluated against various options and sequencing.

4.2 According to the 1988 guide to Methods of ameliorating the hazards of rockfalls and rockbursts the primary considerations in mining the relatively shallow areas are:

4.2.1 According to elastic theory extended volumes of rock in the hanging wall are subjected to tensile stresses which result in a large "Tensile Zone". Figure 4.1. In this tensile zone is a substantial volume of rock which is subjected to tensile stresses, permitting wide spaced tensile fractures to form. This creates the potential for large scale falls of ground.

4.2.2 The virtual absence of face fracturing which in deeper mines generates horizontal dilatational stresses that act to knit the hanging wall into a coherent beam indicated that potential problems existed. Reference 1988 guide paragraph 1.21 (ii) on page 3. Potential failure areas caused by the tensile zone are therefore indicated by relatively widely spaced tensile fractures or the opening up of geological discontinuities, the result of which would be large hanging wall beam failures. The designs were based on minimising the amount of tensile stress generated which, in this case, would subsequently be manifested by an increase in compressive stress as the pillars would be carrying more load. The

Figure 4.1



The Pink Area is the tensile zone in which the vertical stress is tensile.

Depth = 1000m
Span = 200m
k = 0.5
D/L = 5

CONTRAST BETWEEN LOW AND HIGH STRESS STRUCTURES

biggest danger was the potential for rockfalls resulting from the tensile hanging wall component.

4.3 Wide-Reef Mining Layouts

Wide reef mining is generally defined as where the stoping width is 2,5m or greater. At R.E.G.M this is the case where the trackless mining cut off generally occurs depending on various factors.

An important consideration in Bord and Pillar or Room and Pillar mining is the application of regional pillars to limit the size of the "in stope" pillars. In the computer modelling of the mining layouts the existence of unpay block and areas unlikely to be economical, formed part of the regional pillar regime. In the "Amelioration of hazards of Rockfalls and Rockbursts 1988 Edition D7(1) page 89 it is recommended that spans should not exceed 250 metres yet when the proposed mining was modelled on the computer, the stresses generated did not appear excessive.

The first span was mined and as it approached and subsequently exceeded the 250 metre span daily inspections were carried out by the Rock Engineering Department in conjunction with the mining supervision to establish if any stress patterns or problems were beginning to manifest themselves in contradiction to the computer model. At this stage, although the computer model had indicated that no undue stress regimes would exist, whereas the 1988 industry guide had indicated that the 250 mete span was a guideline.

According to the 1988 Industry Guide pillar design requirements for wide reef Bord and Pillar and Room and Pillar in hard rock have not been thoroughly investigated and no formulae have been determined, as is the case of coal mines. Consequently, stable geometries of these works for given depths, mining spans and mining heights are usually based on local experience and, in some cases, on back analysis. A literature search was done on the coal mining pillar designs in order to see if any of the techniques used could be effectively applied to the pillars under design. The analysis of the rock behaviour at Cooke 2 and Cooke 1 Shafts with the application of the Mining Stress Systems computer programme (Besol MS) was used for all modelling.

Despite the shortcomings in the Computer Modelling available at that time, and some scepticism of the validity of information on the rock types to be mined, it was believed that the computer modelling would be of assistance but cognisance had to be taken of the complex numerical and geometrical problems involved. Early versions of Besol MS assumed linearly elastic materials with deformation. The assumptions; linearly elastic, isotropic, and homogenous are obvious limitations which have to be considered when evaluating the results.

The Modulus and Poissons ratio had to be downgraded to a Rock Mass Modulus, which accounts for the influence of faulting, jointing and stratification of the rock mass. Dip variations and severely faulted reef are difficult to model, because and average reef dip is normally used. This implies that a number of grids with different dips and strikes as well as depths may be required. A further complication is the existence of multiple reefs. In all cases it is imperative to check the actual middlings to the haulage or excavation in question. Young's Modulus and Poissons Ratio were evaluated by means of laboratory tests, the problem of sample selection and the number of tests that were done (limited number) to obtain representative values was debatable. The computer simulation results would need to be approached circumspectively and the interpretation of these results and their application would need careful consideration.

4.4 Tonnage Requirements

The output requirements in order to supplement the shaft tonnages were calculated as part of the 5 year planning exercise. These tonnage, grade and gold output requirements would affect the overall strategy on the mining. Factors such as tonnage per level, output per connection and the tramming capabilities all influenced the final decisions.

CHAPTER 5

Evaluation of Alternate Mining Layouts

5.1 Introduction

With the dip of the reef varying from flat or rather bowl shaped through to 60° or more and with reef thicknesses varying from less than 1 metre though to in excess of 3 metres it is obvious that no single method or system could be applied.

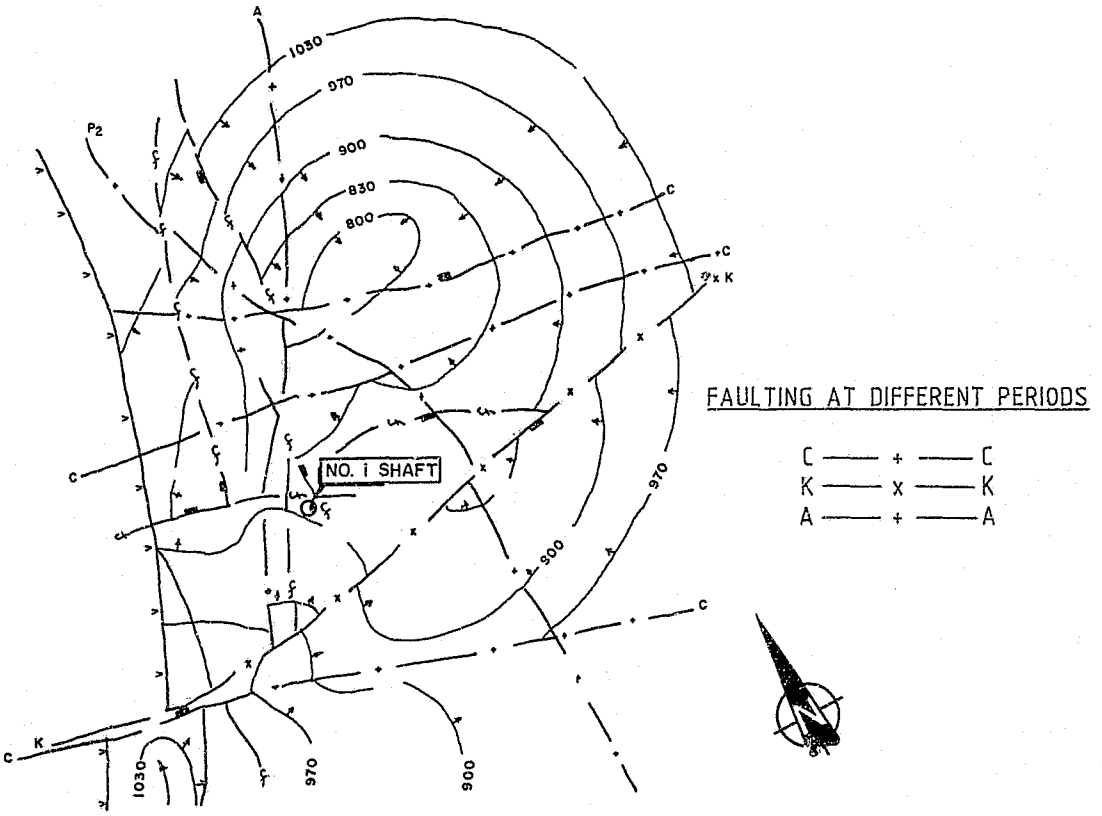
The geological plan and section shown in Figure 5.1 indicates the complexity of the area and its varying dip and strike directions due to faulting. The major faulting resulted in what was known as the stack zone. The stack zone refers to an area where numerous fault has resulted in the reef having been displaced in a series of steps. The mining of these areas would require careful consideration as not only were they broken up due to the major faulting and the resultant drag along the faults but numerous minor sympathetic faults existed in the areas adjacent to the major faults. The intensity of the faulting created major rock mechanics and mining problems.

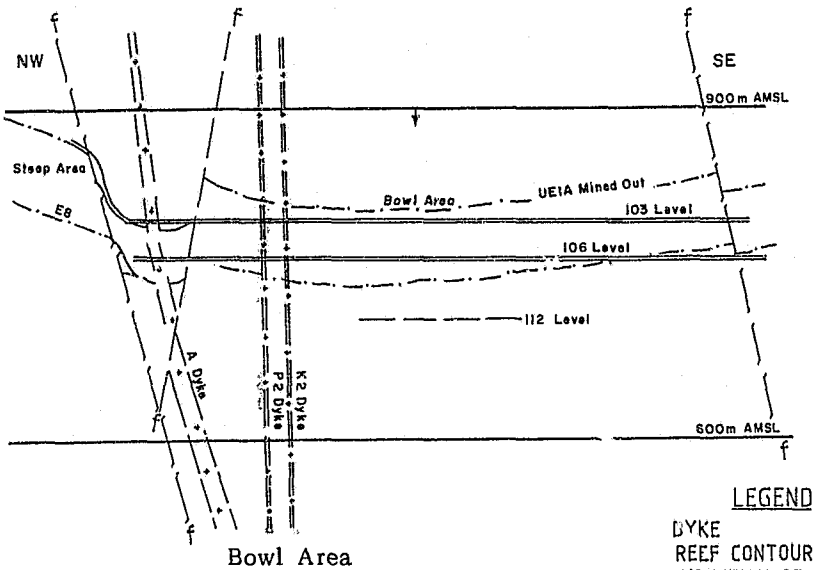
An overriding requirement for any mining layout is that it has to be as safe a method as possible. At no time would the safety of people be compromised. With this in mind obviously cognisance needed to be taken of the recommendation of the Rockburst Committee of 1962, the 1977 Guidelines and the much more recent publication on the amelioration of Rockfalls and Rockbursts 1988. These do not prescribe hard and fast rules but their recommendations were taken into account throughout.

The stoping and mining layouts, would to a large extent determine the stress levels, the energy release rate, the orientation and intensity of new stress fracturing which would interact with the stress fracturing already existing from the previous mining as well as geological fracturing associated with the dykes and faults.

Unpay areas were plotted on the plans as were blocks of ground that would not be extracted for whatever reason. These were used as the basis of regional pillars. Where these were insufficient, areas in sequence of problematic and/or lowest grade were then included into the regional pillars until the best stress profiles were obtained.

Figure 5.1





Bowl Area

Consideration had to be given to the existence of and positioning of future roadways cross cuts, workshops, travelling ways etc., in relation to proposed pillar siting and existing reef that was intended to be left in order to reduce the vulnerability to rock falls in these excavations.

Where the mining methods indicated high stress patterns in these excavations they were supported with wire mesh and rope lacing.

5.2 95 Level Haulage Pillar

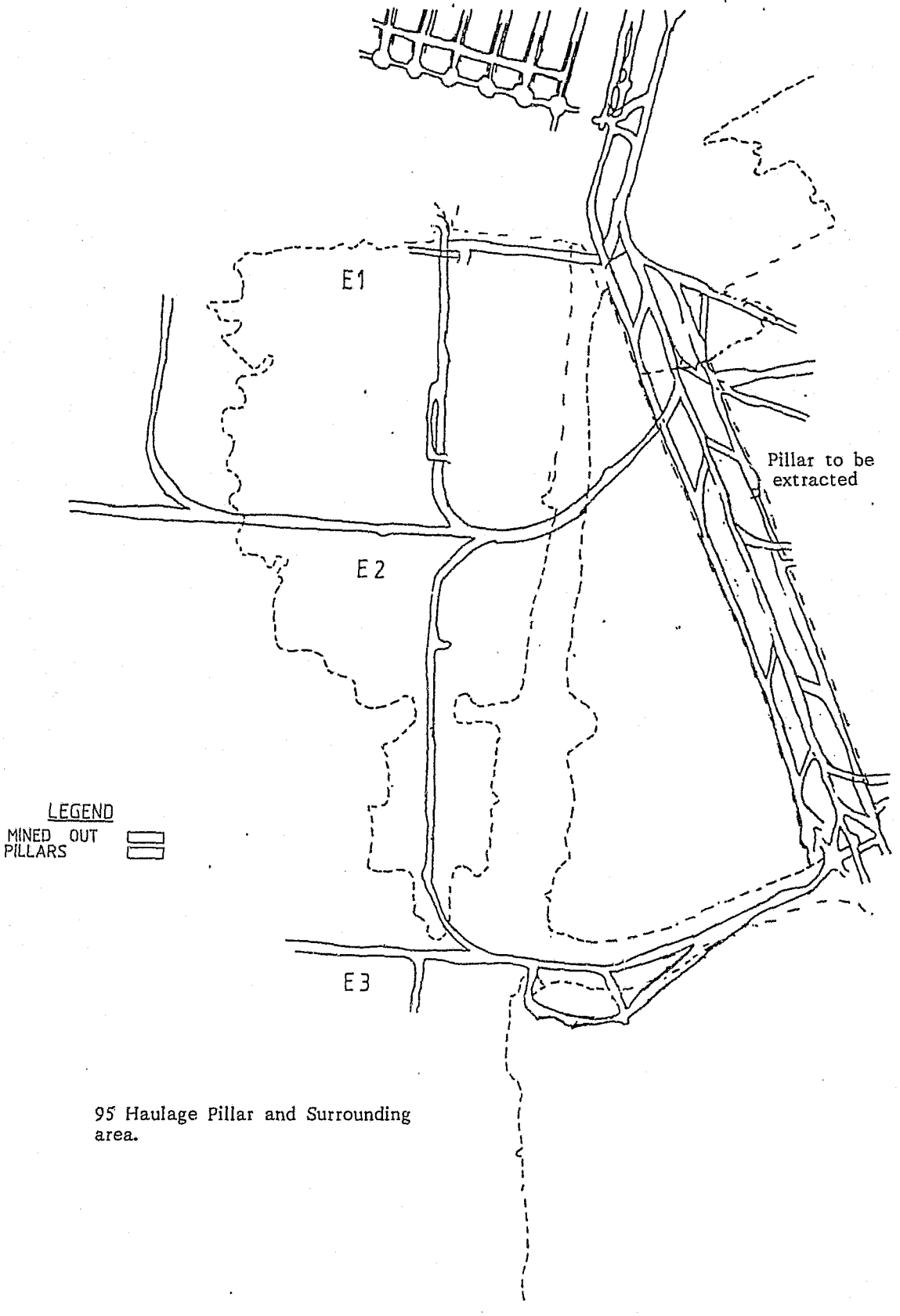
The configuration of the 95 haulage pillar is depicted on Figure 5.2. This was a high grade (with grades in excess of 20g/ton) pillar which would not yield high tonnages but would contribute significantly to the gold output of the shaft.

On inspection of the pillar it was found to have scaled very badly and from the plates 5.1, 5.2, 5.3 visually it would appear that it was going to be extremely difficult to extract. Further inspections in the back areas indicated that some major hanging wall collapses had occurred. In general the area had been well supported during previous mining operations and good hanging wall conditions were evident. The areas had been supported using sticks and packs as well as concrete packs.

Plate 5.4 and 5.5 are typical of highly stressed tunnels. The footwall heave is primarily associated with the relaxation of highly stressed footwall (bulking).
Indications of high stress are:
Sidewall slabbing and spalling
Footwall heave
Crushing in the corners
Cable anchors pulling into the holes
The shape of the excavation (over break and stress orientation can be inferred)

The hangingwall condition along the 95 haulage was found to still be competent except in the vicinity of the faults, where greater movement had occurred. This was especially evident where steel sets had previously been installed as these were found to be extensively damaged and deformed.

Figure 5.2



95 Haulage Pillar and Surrounding area.

SCALE 1:2500

Plate 5.1



Scaling from sidewalls

Plate 5.2

Severe sidewall
Dilation and
Footwall heave



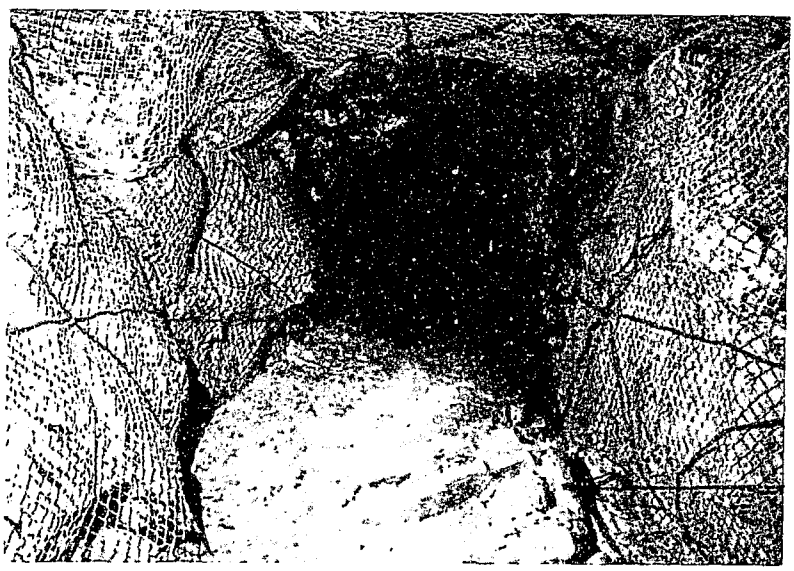
Initial pillar Inspection

Plate 5.3



Haulage and x-cut breakaway

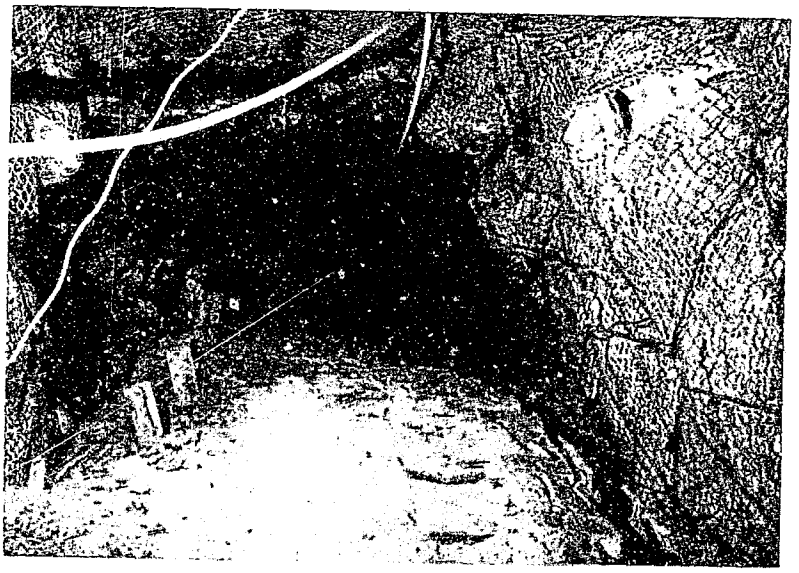
Plate 5.4



Approximately $^1/_3$ of the way into the haulage.

Footwall Heave and Dilation of Pillars

Plate 5.5



An old waiting place at the Stope entrance



Footwall Heave

Approximately in the centre of the pillar in the haulage

As a point of interest regarding the sets installed, people should not regard sets as a panacea for all hanging wall problems. In a 3m x 3m tunnel, the uprights are made of 150mm x 75mm R.S.J. steel 2,5m long. Each is capable of withstanding concentric endloading of 70kN before buckling. Provided that the bending load is distributed uniformly on the cap, and that the uprights are not displaced by sidewall movements, each set will have support capacity of about 140kN. It is interesting to note that two rock bolts can provide a greater support capacity than the much more massive sets. (Budavari 1983). The pillar was situated relatively close to the shaft (Figure 5.3) but was far enough away not to be of any influence on the shaft. The pillar was part of a haulage stabilising pillar in the period when the area was mined. No further mining was envisaged to be done at this site and therefore no travelling ways or haulage ways required further protection. However, a portion of the return air from the 90 south 3 project was upcast through the old areas.

Previously 97 and 101 footwall haulages were used to transport the ore from the areas. These were evaluated as to the condition and the costs involved in opening up and re-equipping of one or both of these long abandoned footwall haulages. The cost of re-equipping and operating of these haulages would have an impact on the mining method selected, particularly with respect to conventional mining.

With each evaluation a basic SWOT analysis (i.e. strengths, weaknesses, opportunities and threats) was done in order to eliminate least likely options in order not to waste time on too many options.

There were basically two mining options that required evaluation.

5.2.1 **Conventional Mining** (Table 5.1)

Figure 5.4 indicates a section of the area and as can be seen the pillar is on the same elevation as 95 level. As the area was mined by conventional mining methods it was possible to remove the balance of the pillar by similar means.

When considering all the aspects it was found that the broken ore would have to be scraped relatively long distances through worked out areas. The box fronts had been removed and would need replacing. The stability of the back areas was

Figure 5.3

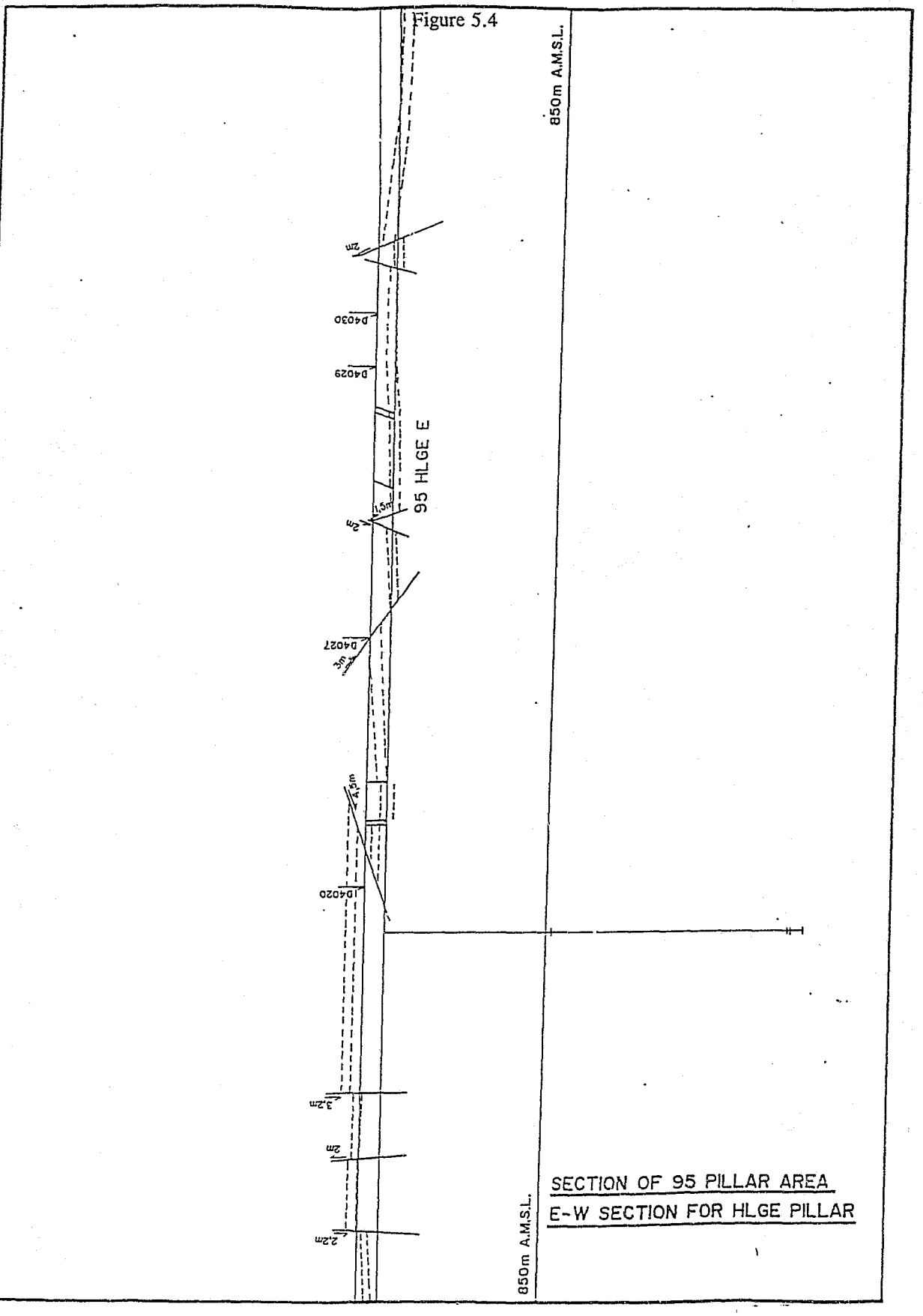


LEGEND

MINED OUT
UNPAY
PILLAR TO BE MINED

95 Haulage Pillar in relation to
Shaft Pillar on UE1A Reef Horizon

Figure 5.4



SECTION OF 95 PILLAR AREA

E-W SECTION FOR HLGE PILLAR

a major question particularly when considering that the pillar that was to be extracted was now acting as a regional stability pillar.

The S.W.O.T. analysis is done by listing:

1. All the potential strengths
2. All the favourable situations/conditions applicable
3. All the unfavourable situations/conditions prevailing
4. The weaknesses which would seriously affect the effective performance of the project be it resources, skills, conditions or costs etc.

The amount of faulting (Figure 5.4) would make conventional extraction very difficult.

On physical examination of the long abandoned 97 and 101 haulages it soon became apparent that the opening of either of these sections would be a long and tedious job. When the costs of opening up and re-equipping were evaluated and all analysed using the haulages looked less favourable. The costs of having to man and maintain a haulage that would be in some remote areas were also unattractive.

The term S.W.O.T. being an acronym for strength, weakness, opportunity and treat analysis. An example of this is shown in Appendix 1.

The S.W.O.T. is not a definitive system but a tool for putting down the point and being men able to analyse these and arrive at a conclusion.

Information on the S.W.O.T. analysis is available in a number of publications among others in Global Strategy pages 177 to 195.

5.2.2 **Trackless Mechanised Mining** (Table 5.2)

The reef thickness of ±2,5 metres, the multiple faulting, the fact that the pillar was to all intents and purposes on the 95 level haulage, the ease of access from the haulage, the tramming distance to the 95 level station tips all weighed heavily in

favour of using a ($2m^3$) LHD with hand held drilling. A drill jumbo was not applied due to its size and the utilisation of such a unit in a low tonnage operation. For the layout refer to Section 7 figures 7.2 and 7.3.

The haulage passing through the pillar had deteriorated appreciably (Plate 5.6, 5.7) and the use of an LHD in cleaning this up was not only simpler but also safer.

From the safety aspect the use of the LHD meant that the broken rock or discarded material could be loaded mechanically thereby removing people from the potentially dangerous areas.

The scaling was broken up and loaded into hoppers and trammed directly to the station orepass system.

A facility, that existed as a cross cut previously, was converted at very little cost into a workshop and maintenance facility for the proposed LHD and scaler.

The ore from mining operations was then loaded at the face and trammed by the LHD to the hopper loading point. This system would not only minimise the handling of the rock but also minimise gold losses and eliminate the eventual vamping of long scraper paths that would have been required through the mined out areas to the old ore pass system. As this was a high grade pillar significant gold would have been lost in the fractured footwall. After all the relevant information had been considered the decision taken was to apply a limited trackless mining system to extract these pillars.

By a limited trackless operation is meant where some trackless equipment, in this case, one LHD and one scaler is applied and all other activities being fulfilled by track bound ore transport and manual labour.

5.3 101 Area

The 101 area was (table 5.3) very different from the 95 level pillar and each operation had to be evaluated on its merits. The area in question applied to at least two reef bands with the possibility that a third, in close proximity to the

Plate 5.6

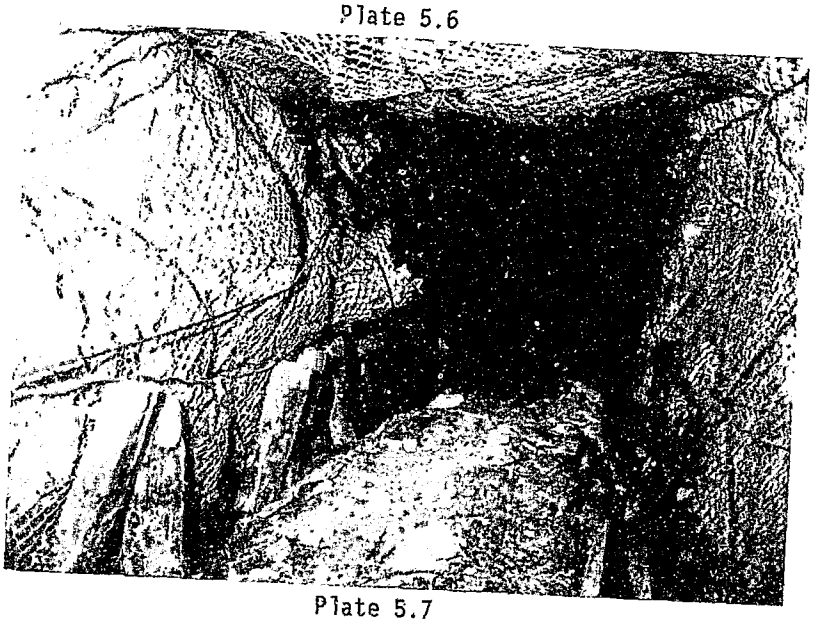


Plate 5.7



Haulage Condition showing sidewall dilation and footwall heave

UE1A band, would be payable in isolated pay shoots. Where the E9Gd proved payable it could be mined simultaneously with the UE1A. The possibility had also been expressed that these pay shoots could be followed into the previously mined areas, provided the system applied could cater for the mining of these on an economical basis.

The volume of the UE1A reef remaining in the 101 area amounted to some 4,2 million tons and extended from above 101 level to below 106 level. (figure 5.5).

The reef dip and thickness showed considerable variations and the area was heavily faulted.

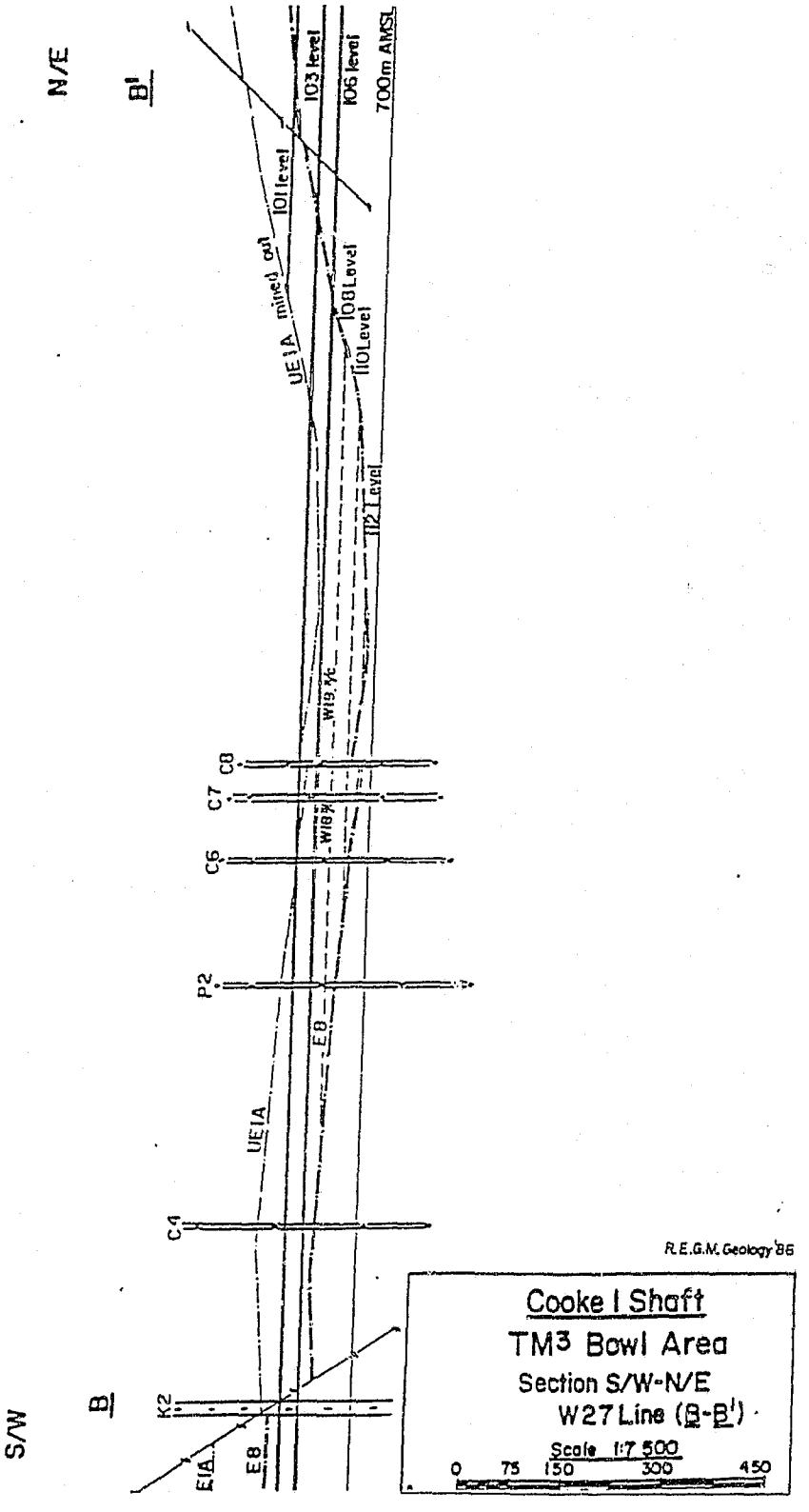
5.3.1 Conventional Mining

Initial investigation centred on mining these pillars by conventional means as the Mine had largely been mined out using these methods and the crews were well experienced. The support patterns, mine standards and procedures were well understood by all. Because of the inflexibility of these methods the present pillars had been left.

In preparing tentative mining layouts to analyse the advantages and disadvantages of breast and/or updip mining it became obvious that although considerable development had previously been done a large amount of additional development would be required for either of the conventional layouts (Table 5.4).

It was found, from survey data as well as underground inspections, that the majority of stope faces had stoped on fault contacts either partially or as a whole. Some of the faults were known to be scissor faults and the displacement of the faults at various places was unknown. On some of the major faults the displacement was such that the block would require extensive re-development in order to mine on either side of the fault. Material handling, particularly timber, into these extremely faulted areas

Figure 5.5


N/E
B'
103 level
106 level
700m AMSL
101 level
UE1A mined out
108 Level
110 Level
112 Level
W19 N/c
C8
C7
W18 N/c
C6
P2
E8
UE1A
C1
S/W
B
K2
E8
E1A
R.E.G.M. Geology '86
Cooke I Shaft
TM3 Bowl Area
Section S/W-N/E
W27 Line (B-B')
Scale 1:7 500
0 75 150 300 450

was going to be labour intensive. Broken rock would require re-handling in a number of situations. The proximity of the reef horizons to the lowest haulage level and also the lowest level of the shaft was perceived to have major production problems as the ore would have to be scraped up dip in places, or at times loaded directly into hoppers on the 106 level.

5.3.1.1 **Steep Mining Layouts** (Figure 5.6)

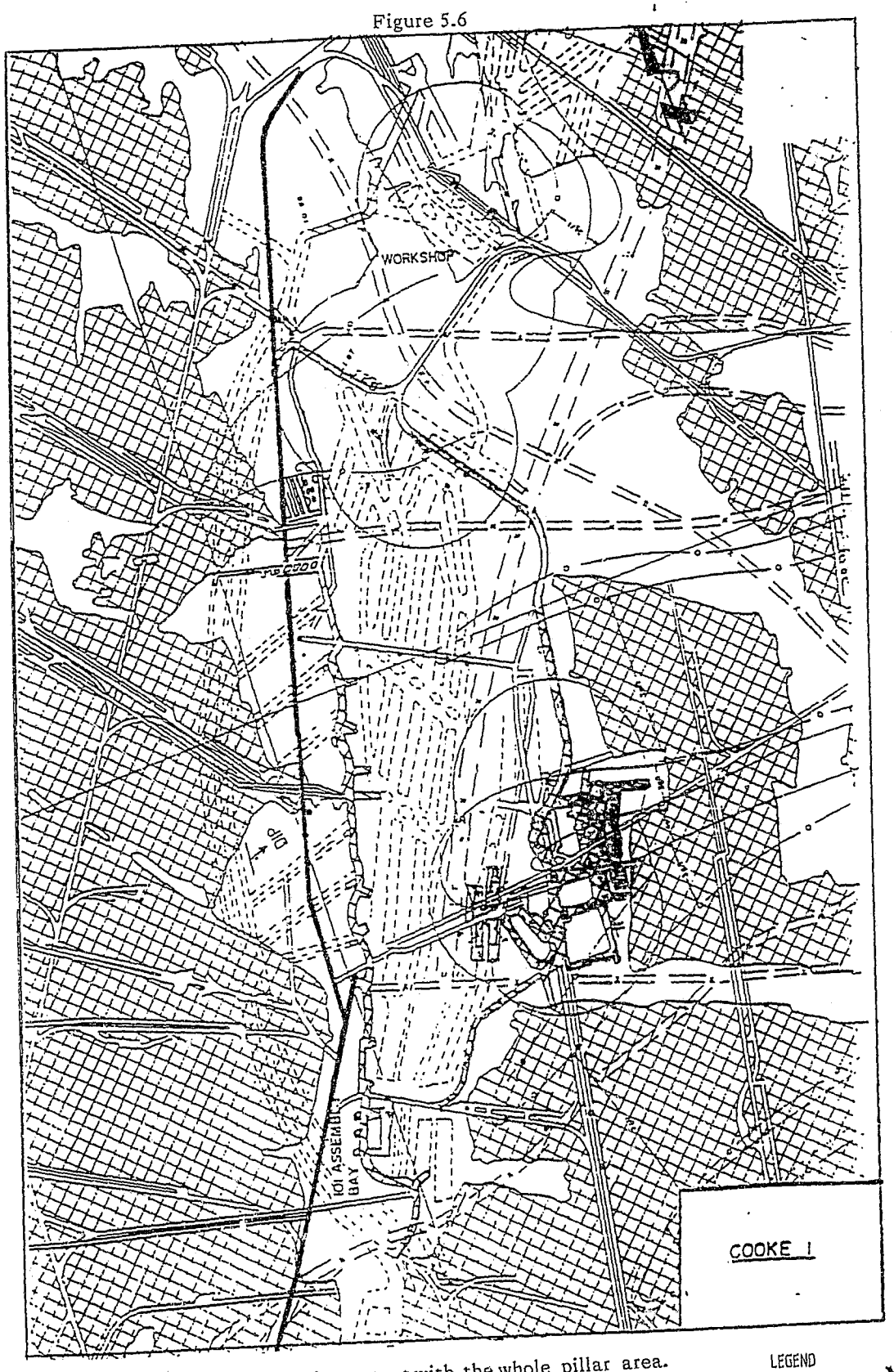
Workings are considered to be steep when the dip exceeds 35°, (Minerals act 50 of 1991 Chapter 1 page 5 define 32) the steep dip of the reef imposes practical constraints on layouts and their options, particularly when dealing with narrow reef situations.The stopes could be mined breast, overhand, underhand or updip. As these form part of the pillar to be extracted and much of the literature[12] indicated that updip was the preferred method from experience and safety in the operations, this was the method initially evaluated.

The stopes could either be serviced by footwall haulages with x-cuts to reef or reef drives (Figure 5.7) provided the necessary protective measures were taken for travelling below the stopes. New methods of rock handling could be applied[13].
Steep mining had previously been practised at Randfontein so this was not new to the system. From this point of view methods similar to those which had been used previously were the second option examined.

These are discussed in a number of publications including C Biccard Jeppe where underhand and overhand is covered on pages 838 through 852 – Gold mining on the Witwatersrand. A brief evaluation was made of both underhand and overhand methods and their advantages discussed with the mining supervision that would be involved. It was decided to model the

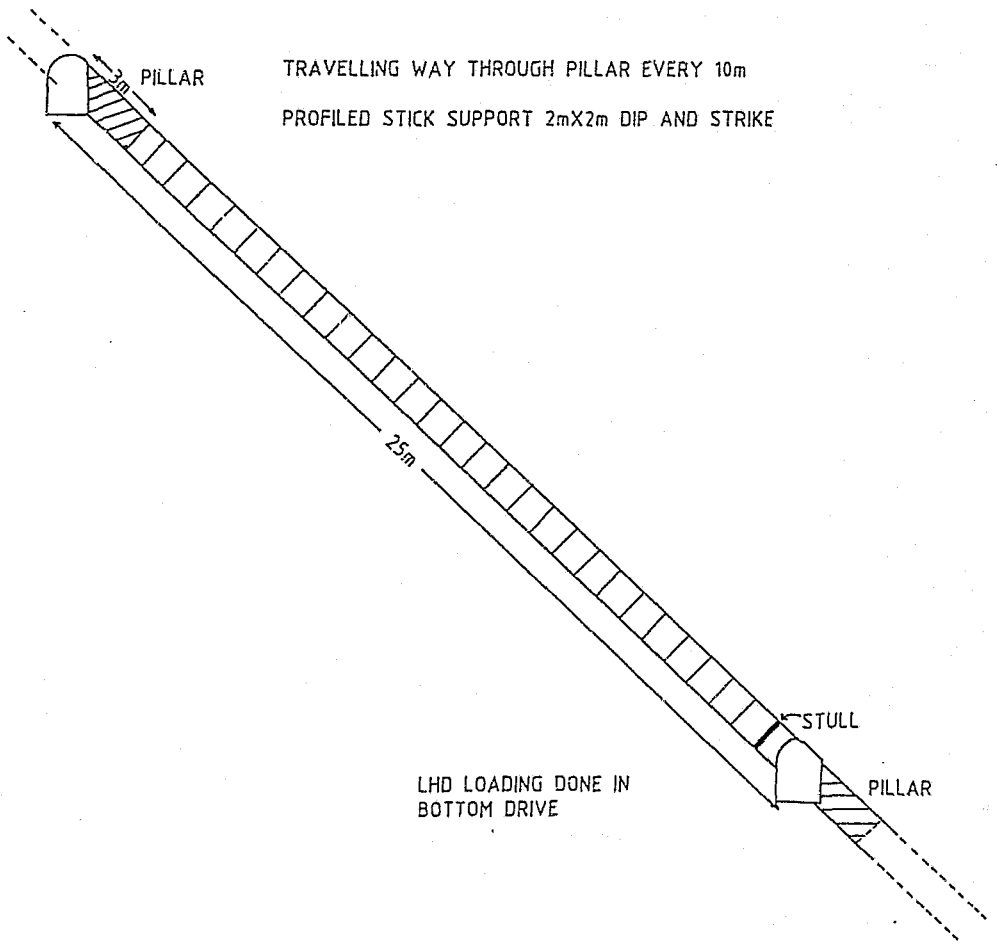
updip extraction[14] method (Figure 5.8) and the breast method on the computer and should difficulties be foreseen only then would more work be done on the underhand and/or the overhand mining layouts.

Figure 5.6



Steep mining layout shown in context with the whole pillar area.

FIGURE 5.7



SECTION STEEP MINING LAYOUT

NOT TO SCALE

In each case, the alternative layouts were modelled on the computer and the stress determined. In most of the cases when the breast method was compared to the generally recommended updip method for remnant or pillar extraction no significant advantage was observed in the computer simulations.

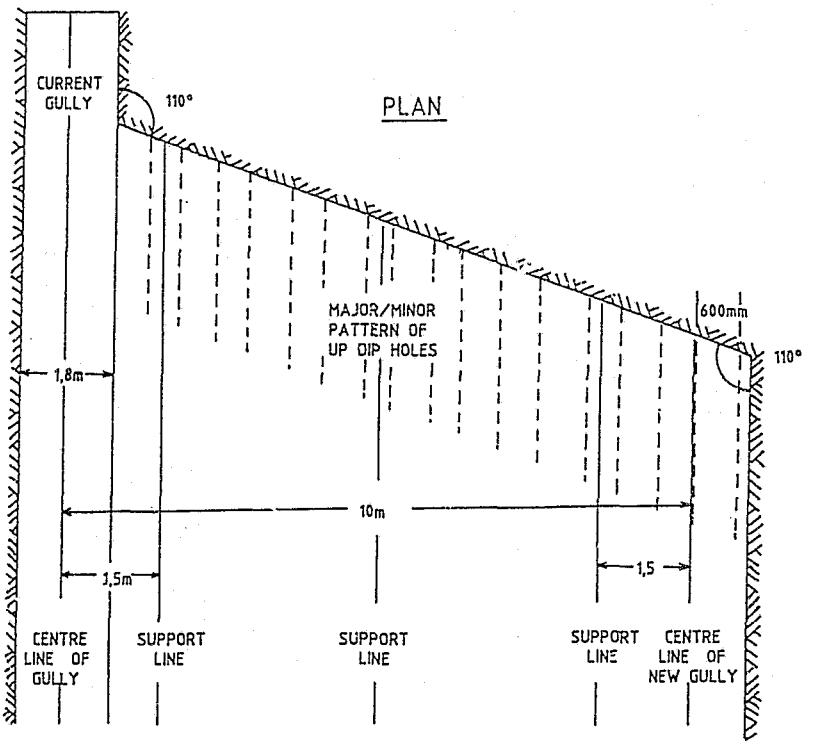
The method decided upon was to put up raises and stope east and west from these raises. Cleaning in the majority of cases was by gravity directly into LHD roadways. In certain cases where it was possible, a short boxhole was put up to the centre raise and the blasted ore scraped along a scraper path to a grizzley. The orepass was equipped with a box and pneumatically controlled door to facilitate loading directly into dump trucks (Figure 5.9).

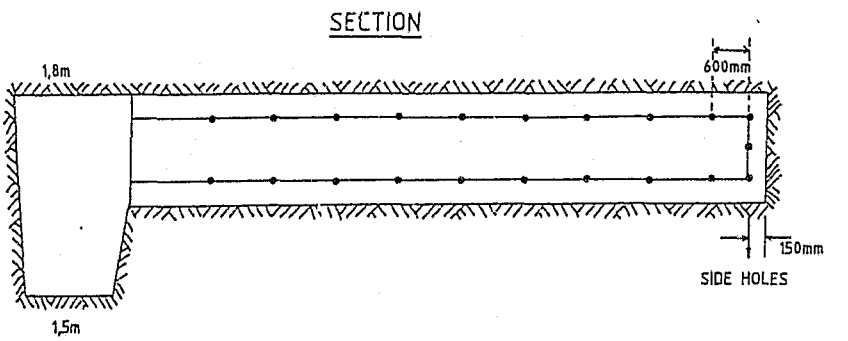
5.3.1.2 **Stack Zone**

This is a highly complex and faulted zone where the reef becomes stacked as a result of numerous step faults. The extreme complexity of the area meant that very few attempts had been made to mine the zone previously. Some of the problems experienced in earlier mining attempts were

a) Serious dilution problems occurred, often making the mining thereof unpay.
b) Extreme difficulty of establishing a raise on reef.
c) The almost impossible task of removing the broken ore after the blast.
d) Ventilating the area was extremely difficult particularly maintaining ventilation equipment. Because of the frequent changes to accommodate the faults the scrapers frequently pull out or damage ventilation columns.

FIGURE 5.8

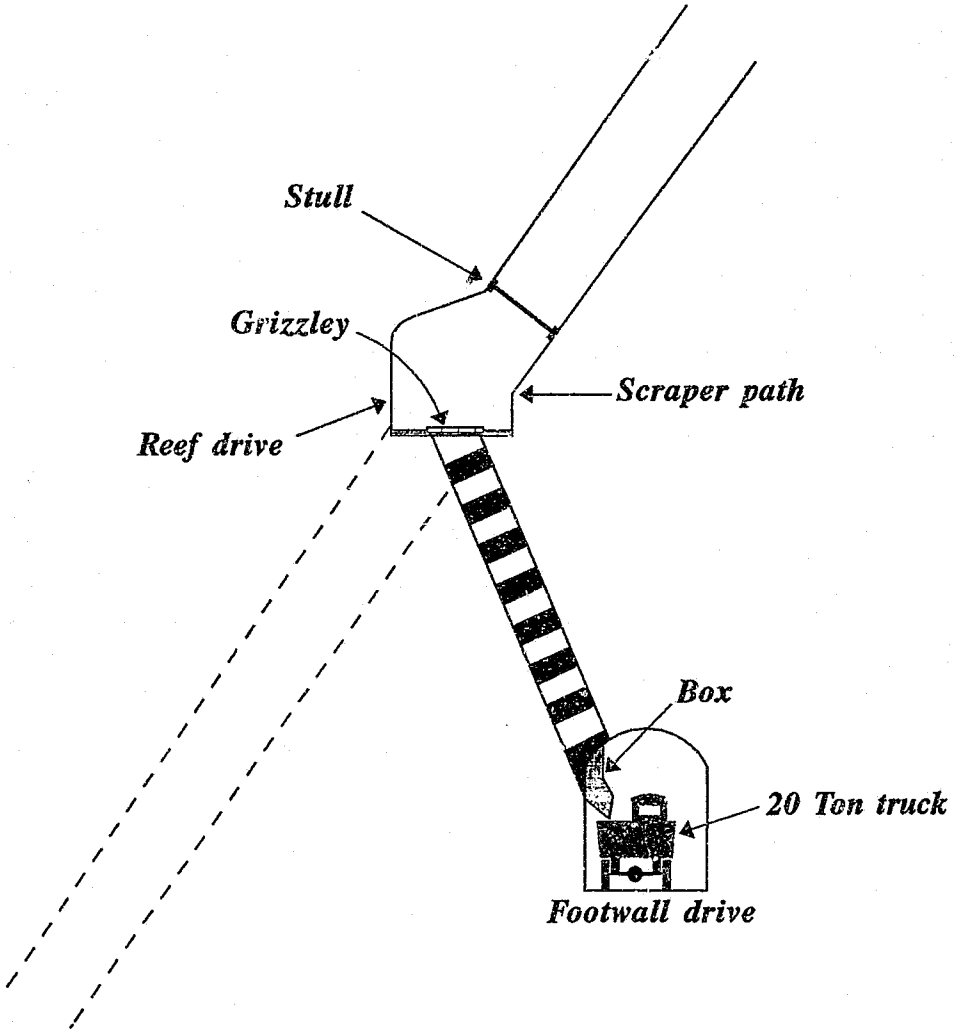




UP DIP STOPE PANEL LAYOUT

PLAN AND SECTION OF DRILL HOLE PATTERN FOR STOPING WIDTHS 120cms OR LESS

Figure 5.9



SECTION OF STEEP AREA WHERE FOOTWALL DRIVE WAS UTILISED FOR ORE COLLECTION AND TRANSPORT.

e) As the result of the faulting, supporting the area was a major problem.

The final layout is depicted in Figure 5.10 and 5.11. To date only trackless mining method has the potential to make the mining of this zone feasible.

5.3.1.3 **Relatively Flat Areas**

Conventional bord mast mining layouts were prepared using different starting points.

5.3.2 **Trackless Mechanised Mining Methods** (TM^3)

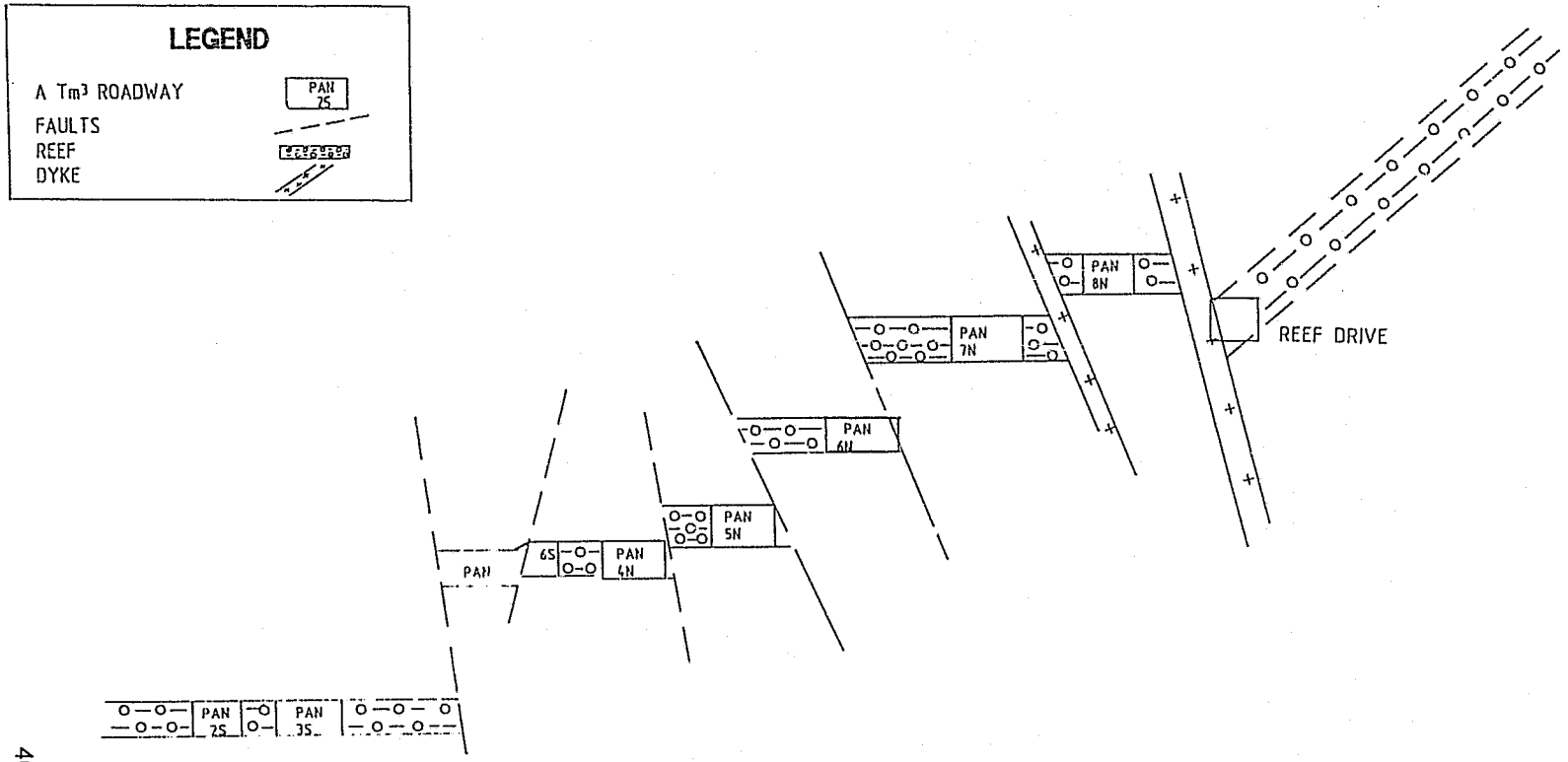
Trackless mining had been applied earlier at Randfontein Estates and Western Areas Gold Mine[15] extremely successfully.The advantages of trackless mining as derived from the analysis are amongst others:

5.3.2.1 The ability to cost effectively mine wide reef horizons.

5.3.2.2 The ability to mine the areas which were on or near to the lowest haulage level or even below the haulage horizon.

5.3.2.3 The computer simulations showed that the proposed Bord and Pillar mining or Room and Pillar[16] and its modifications, had by far the lowest stress levels. Therefore it was by all evaluations safer.

5.3.2.4 On the safety aspect it was obvious that if the labour could be kept away from the faces during the loading and especially the drilling[17] operations, the potential for as well as the severity of injuries would be greatly reduced.

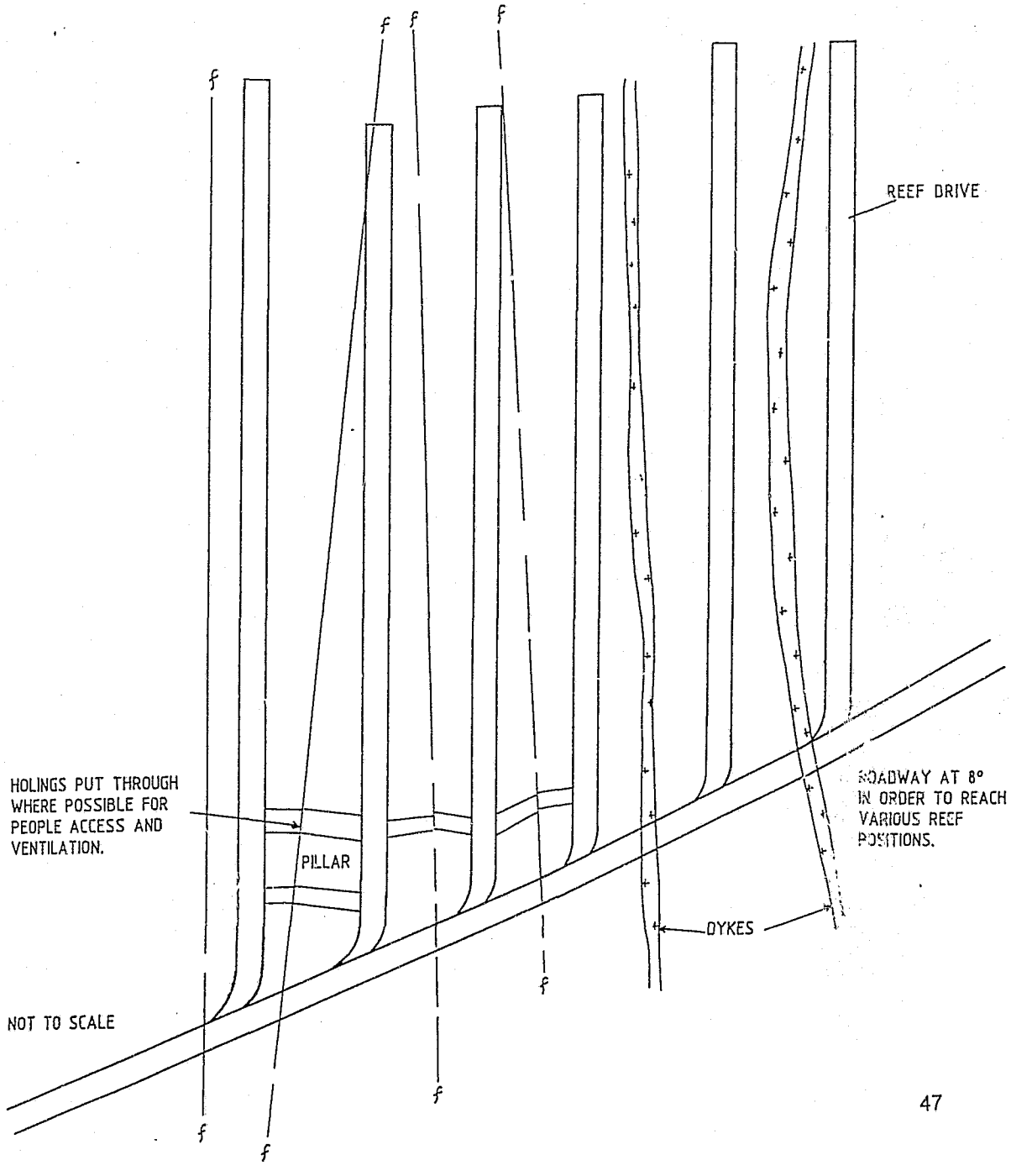


DIAGRAMATIC SECTION THROUGH STACK ZONE
NOT TO SCALE

FIGURE 5.10

FIGURE 5.11

DIAGRAMATIC LAYOUT OF PANNELS THAT WOULD BE MINED IN THE STACK ZONE



5.3.2.5 Productivity would be significantly enhanced.

5.3.2.6 The ability of rubber tyred equipment to negotiate the faults and dykes and accommodate the changes in dip was certainly one of the major considerations.

5.3.2.7 the equipment was available on REGM and had previously shown its benefits, particularly as regards costs and productivity. For detail on trackless mining refer to the Association of Mine Managers. Trackless Mining symposium February 1988.

Narrow reef areas are in general not suited to trackless mining as the amount of dilution created in developing the access ways as well as the loading points makes or can make the operation unpay. The objective is to send clean products to the mill.

After evaluating all the options the decision was to mine the majority of reef by TM^3 means[17].

Table 5.1

ADVANTAGES AND DISADVANTAGS OF CONVENTIONAL MINING OF 95 HAULAGE PILLAR

1. **Advantages**

1.1 Mining was originally done conventionally therefore the layouts were in place.

1.2 Development of haulage was already completed.

1.3 It was a well-used method at Cooke 1.

2. **Disadvantages**

2.1 Extensive faulting existed with reef below the 95 haulage in places.

2.2 97 and 101 Haulages had been stripped and would required total re-equipping.

2.3 Whichever haulage was used would require re-equipping of several kilometres of haulage and then maintaining and operating the haulage system for a very low tonnage.

2.4 Falls of ground had occurred in the mined out areas through which the ore would have to be scraped.

2.5 The footwall of the old scraper paths was found to have scaled and had fractures visible which together would result in dilution and gold losses.

2.6 The reef was in the order of 2.5m wide.

2.7 Low labour efficiencies

2.8 Location of pillars would result in slow mining.

2.9 Proximity of people to working faces while establishing and mining.

Table 5.2

Advantages and Disadvantages of Trackless Mining of 95 Haulage pillar

1. Advantages
   - 1.1 Mechanised equipment meant that people were physically further away from the working face.
   - 1.2 The faulting could be negotiated more easily.
   - 1.3 the reef below the footwall elevation was not problem.
   - 1.4 No long scraping was required as the ore could be loaded directly into hoppers.
   - 1.5 Much shorter hauling distance of the ore.
   - 1.6 Simple and easy material transport using the LHD.
   - 1.7 Easy negotiation of pillars – flexibility.
   - 1.8 Scaler could be used to bar face and hanging wall as well as break up large rocks.
   - 1.9 LHD could be used as a working platform for roofbolt and wire wire mesh installation.
   - 1.10 Minimum handling of ore should result in minimum reef/gold l losses.
   - 1.11 No scraper ropes in areas where machines are drilling.
   - 1.12 Less labour requirements.
   - 1.13 Unnecessary to re-equip old haulages.
   - 1.14 Scaling easily loaded with LHD.
   - 1.15 Stoping width of 2.5m suited the equippment.
   - 1.16 The most cost effective method.

2. Disadvantages
   - 2.1 Can only be used in wide reef areas.
   - 2.2 Ventilation could be a problem.
   - 2.3 Skilled operators and maintenance still required.
   - 2.4 Establishment of service bay required.
   - 2.5 Transport and storage of diesel fuel.
   - 2.6 Tied up expensive equipment in a low tonnage operation.

Table 5.3

DIFFERENCE BETWEEN 101 AND 95 PILLAR

1. **101 Pillars were:**
   - 1.1 Far greater in area.
   - 1.2 Covered a much wider area.
   - 1.3 Possibility of mining 2 of 3 reefbands.
   - 1.4 Effect of pay shoots and un pay areas.
   - 1.5 Trackless mining was already used in the area and a trackless workshop existed.
   - 1.6 Previously mined areas would have to be traversed.
   - 1.7 Orepass system would have to be established.
   - 1.8 Far higher tonnage.
   - 1.9 Bigger equipment could and would be used.
   - 1.10 Wider reef in the 101 area as compared to the 95 pillar.
   - 1.11 Geologically, the 101 area was different from the amount of dykes and faults present.
   - 1.12 Some dykes were water bearing.
   - 1.13 Faulting had a greater displacement.
   - 1.14 The bowl area was in part below 106 level which was the lowest tramming level.
   - 1.15 101 was a lot further from the shaft.

2. **94 Haulage pillar**
   - 2.1 was limited to a section of 95 haulage only.
   - 2.2 The grade was a lot higher than the 101 pillars.
   - 2.3 Whereas the reef dip on the 95 pillar was to all intents and purposes, over the very short span, flat whereas in the 101 area, it would range from steep to flat and also have a bowl shaped area.

2.4 Although the 95 pillar had numerous faults, the displacements were such that they were more easily managed.

The wide reef mining methods used at Government Gold Mining areas[9] and methods used in the extraction of shaft pillars[10,11] were considered and evaluated.

Comparison of development requirements for a conventional mining layout and a trackless mining layout for the 101 area.

FINANCIAL YEAR	CONVENTIONAL					TM³				
	Reef Tons	Recovery Grade g/t	Reef Metres	Waste Metres	Total Metres	Reef Tons	Recovery Grade g/t	Reef Metres	Waste Metres	Total Metres
1986 - 1986				580	580	58 000	3,74	370	140	510
1986 - 1987	202 850	3,84	690	6 338	7 028	379 000	3,75	2 306	1 590	3 896
1987 - 1988	376 000	3,85	1 936	6 912	8 848	600 000	3,76	6 912	848	7 760
1988 - 1989	400 000	3,86	2 304	8 064	10 368	600 000	3,76	1 842		1 842
1989 - 1990	590 000	3,86	3 456	4 384	7 840	600 000	3,76			
1990 - 1991	600 000	3,88	3 456		3 456	600 000	3,78			
1991 - 1992	600 000	3,88	1 720		1 720	600 000	3,78			
1992 - 1993	600 000	3,85				600 000	3,75			
1993 - 1994	229 083	3,85				102 211	3,75			
TOTAL	3 597 933	3,88	13 562	25 278	39 840	4 139 211	3,78	11 430	2 578	14 008

Table 5.4

APPENDIX 1

S.W.O.T. ANALYSIS

CONVENTIONAL MINING OF 95 HAULAGE PILLAR

STRENGTHS	WEAKNESSES
1. Known system.	1. Old haulage stripped.
2. A lot of development exists.	2. Requires funding for re-equipping for a low tonnage.
3. People skilled in operation.	3. Ties up tramming equipment on another level.
4. Conventional equipment available.	4. Spread the supervision over to wide an area.
5. Standard systems were available for the mining and support.	5. Amount of faulting and displacement of reef would make:
6. Conventional mining was taking place in the adjacent south 3 area.	a. Mining difficult
7. If area was pre-developed more geological information would be	b. Create high dilution in attempting to negotiate faults.
	6. Panels would have to be short thereby tying up a large amount of scraper winches.
	7. Faulting would necessitate mining below the present footwall.
	8. The area would need to be pre-developed in order to best negotiate and mine, with the faulting.
	9. Scraper ways would have to be maintained in previously worked out areas with poor hanging wall conditions.
	10. Cleaning up and getting into the existing tunnels would be dangerous and time consuming.

S.W.O.T ANALYSIS

CONVENTIONAL MINING OF 95 HLGE PILLAR (CONTINUED)

THREATS	OPPORTUNITIES
1. Working in old areas always has a greater element of risk even with replacing support along working ways.	1. A large portion of the gold currently tied up in the pillar would be mined.
2. Gold losses would occur in the fractured footwall.	2. The project would add to the shaft and mines revenue.
3. Conventional mining would limit the ore available due to the faulting.	3. The life of the shaft can be extended.
4. The additional dilution would add to gold loosed in the plant.	
5. The ground conditions may deteriorate to such extent that a lot of ore would be lost.	
6. The ground may become so disturbed that no other method would be successful.	
7. The pillar may already be so disturbed as to be economically and safety wise-unminable.	

Table 5.2

Advantages and Disadvantages of Trackless Mining of 95 Haulage pillar

1. Advantages
   - 1.1 Mechanised equipment meant that people were physically further away from the working face.
   - 1.2 The faulting could be negotiated more easily.
   - 1.3 the reef below the footwall elevation was not problem.
   - 1.4 No long scraping was required as the ore could be loaded directly into hoppers.
   - 1.5 Much shorter hauling distance of the ore.
   - 1.6 Simple and easy material transport using the LHD.
   - 1.7 Easy negotiation of pillars – flexibility.
   - 1.8 Scaler could be used to bar face and hanging wall as well as break up large rocks.
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   - 1.16 The most cost effective method.

2. Disadvantages
   - 2.1 Can only be used in wide reef areas.
   - 2.2 Ventilation could be a problem.
   - 2.3 Skilled operators and maintenance still required.
   - 2.4 Establishment of service bay required.
   - 2.5 Transport and storage of diesel fuel.
   - 2.6 Tied up expensive equipment in a low tonnage operation.

The amount of faulting (Figure 5.4) would make conventional extraction very difficult.

On physical examination of the long abandoned 97 and 101 haulages it soon became apparent that the opening of either of these sections would be a long and tedious job. When the costs of opening up and re-equipping were evaluated and all analysed using the haulages looked less favourable. The costs of having to man and maintain a haulage that would be in some remote areas was also unattractive.

CHAPTER 6

Rock Mechanics Analysis and Evaluation

6.1 95 Level

6.1.1 Discussion

This area had been mined some 10 years prior to mining recommencing. The mined out area was extensive, thus anticipated stresses were high. In order to optimize design the program Besol MS was used to conduct a sensitivity analysis to ensure that an acceptable factor of safety existed in the pillar which were to be left adjacent to 95 Haulage East. From this evaluation the design was optimized with 10,0m x 10,0m pillars spaced every 22,0m along the length of the haulage. Crush pillars 3,0 m wide were also left adjacent to the 10,0m x 10,0m pillars to isolate the mined out areas, see Figure 7.1.

Normal modelling procedures were followed with the windowing option used in the identification of the required area. The following parameters were used in the numerical models.

a. Elastic Parameters of Rockmass:-

Young's Modulus	-	70 Gpa
Poissons's Ratio	-	0,2

b. Elastic Parameters of Seam:-

Young's Modulus	-	64 Gpa
Shear Stiffness	-	24 Gpa

A normal stress gradient, applicable to Witwatersrand rocks, was also assumed.

6.1.2 Pillar strength

The empirical pillar design formula of Hedley as modified by Stacey and Page was used in the design process. This formula is as follows: -

$$\text{Pillar Strength } (P_s) = \frac{133}{H^{0,75}} \frac{W^{0,50}}{}$$

Where: W = Pillar width is meters.

H = Pillar height is metres.

P_s = Pillar strength is Mpa.

The value of 133 Mpa is well below the intact strength of the rock and can be related to the design rock mass strength, DRMS, which takes into account the rock mass quality, unfavourable joint orientation and the excavation method.

$$(P_s) = \frac{133}{H^{0,75}} \frac{W^{0,50}}{}$$

$$= \frac{133}{2,5^{0,75}} \frac{W^{0,50}}{}$$

$$= \frac{133}{2,0} \frac{3.2}{}$$

$$= 212,8 \text{ Mpa}$$

Say 213,0 Mpa

This calculation assumes an isolated 10,0m x 10,0m pillar and does not include the additional stabilizing effect of the 3,0 m wide rib pillar.

6.1.3 **Pillar stress**

The average Pillar Stress (APS) as computed was a maximum of 150Mpa. The Factor of Safety (FOS) is therefore:-

$$\text{FOS} = \frac{\text{Pillar Strength}}{\text{Pillar Stress}}$$

$$= \frac{213\ \text{Mpa}}{150\ \text{Mpa}}$$

$$= 1{,}42$$

Say 1,5

The pillars therefore were considered to be stable. In practice no pillar failures were reported confirming the design.

6.1.4 **Additional support**

Local support was provided in the form of:-

- 200mmØ profile at a density of 2,25m^2/unit. Low total closure was experienced thus no premature failures of the units occurred.
- 2,4m x 25mm diameter rebar on a 2,0m c 2,0 m square grid. The rebar was installed in the face area to protect personnel in the face area during the loading cycle.

6.2 **101 Area**

6.2.1 **Discussion**

The area in question comprised a block of reef some 500,0m x 160,0m and located some 700,0m from the shaft. The area was transversed by numerous faults and dykes and was also folded. Previously it was not possible to profitably mine the area but with the advent of trackless mining methods it's was now possible to mine this area.

6.2.2 **Pillar strength**

A similar exercise was conducted as previously, where a sensitivity analysis was conducted to optimize the pillar dimensions. From this design, work pillars with dimensions 7,0m x 10,0m with 7,0 bords were cut.

The pillar strength was again calculated using the modified formular of Hedley.

$$(P_s) = 133 \frac{W^{0,50}}{H^{0,75}}$$

However, due to the rectangular nature of the pillars, the effective pillar width was calculated based on the hydraulic radius. The formula used to calculate the effective width was:

$$We_{ff} = 4 \frac{A_n}{R}$$

Where A_n = plan area of pillar

R = perimeter

$$We_{ff} = 4 \frac{70,0m}{34.0m}$$

$$= 8.0$$

The strength of the pillar was therefore:

$$(P_s) = 133 \frac{W^{0.50}}{H^{0.75}}$$

$$= 133 \frac{8,2m^{0.75}}{2,5m^{0.75}}$$

$$= 133 \frac{2,9}{2,0}$$

$$= 193.0 \text{ Mpa}$$

6.2.3 **Pillar stress**

The maximum APS calculated from the numerical model was 192Mpa. This effectively reduced the FOS of the pillars to unit and in the HIB6 area some pillars in fact failed. Pillar dimensions were increased to 8,0m x 10,0m which increased the strength of the pillars to 200,0Mpa and increased the FOS to slightly in excess of unity, thereafter, no more problems were experienced.

6.3 The bowl area

Similar pillar dimensions were employed in the above-mentioned area, namely 8.0m x 10.0m with 6.0m bords. No rock related problems were experienced during the mining of this area.

CHAPTER 7

Selection of Trackless Equipment

7.1 Introduction

The selection of equipment for trackless mining operations and their application is extensively covered by Dr H Scott-Russell in his Doctoral Thesis[17]. In the case of the selection of equipment for these operations, use had obviously to be made of available equipment primarily at Cooke 1 but expanded to available equipment on Randfontein Estates. Randfontein Estates had a vast fleet of trackless mining equipment with projects coming to an end while others are being planned and brought into production. This section will describe the equipment selected and some of the reasons therefor.

7.2 95 Haulage Pillar

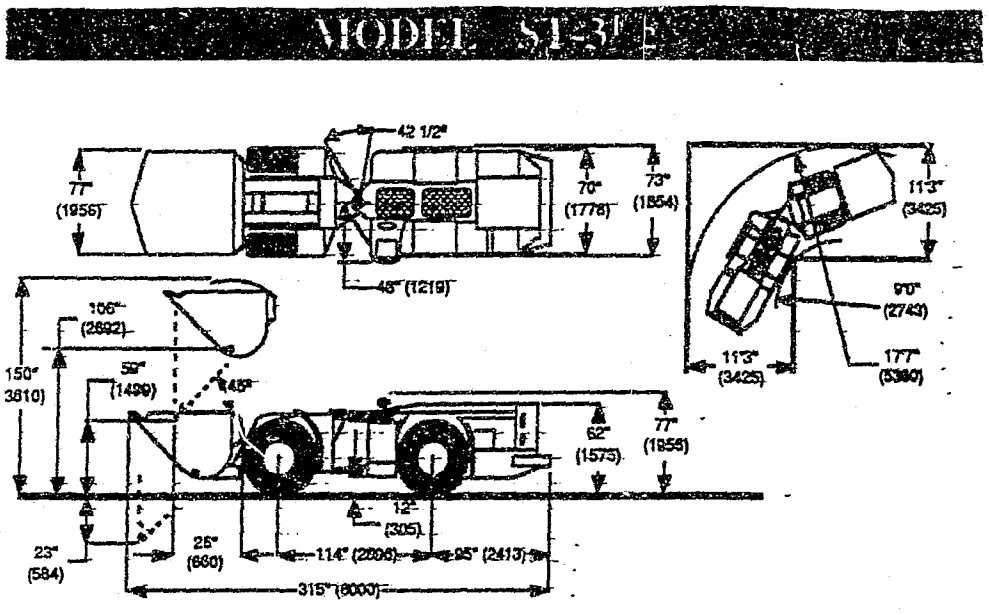
7.2.1 The LHD unit selected for this area was a 2,5m^3 (3,5 yard^3) unit. The details of this unit are shown in Figure 7.1 and from this can be seen that the height of 2,05m to the top of the cab as well as the narrow width of 1,9m made it ideally suited to this operation. In addition, the tipping height of 2,4m meant it could easily discharge its load into the 4 ton rail bound hoppers. The use of a larger unit (3,5m^3) would have resulted in the LHD being far under utilized and apart from the unfavourable dimensions would have resulted in higher unit costs per ton mined.

7.2.2 The only other piece of trackless equipment applied to this project was a scaler (Plate 7.2). This was probably the most critical piece of equipment both in opening up and subsequent mining operations. Working in fractured or potentially fractured rock requires a lot of time to make it safe but with the use of the scaler this work could be safely and very effectively done including the breaking up of large rocks prior to loading by the LHD.

7.3 101 Area

This area required a very different approach as compared to the 95 haulage pillar. Firstly, the area was far larger, the daily tonnage requirements were considerably higher, the tramming distances were far greater, the reef package was potentially wider and higher productivity was required. These considerations meant that a very different fleet was planned for the area.

Figure 7.1



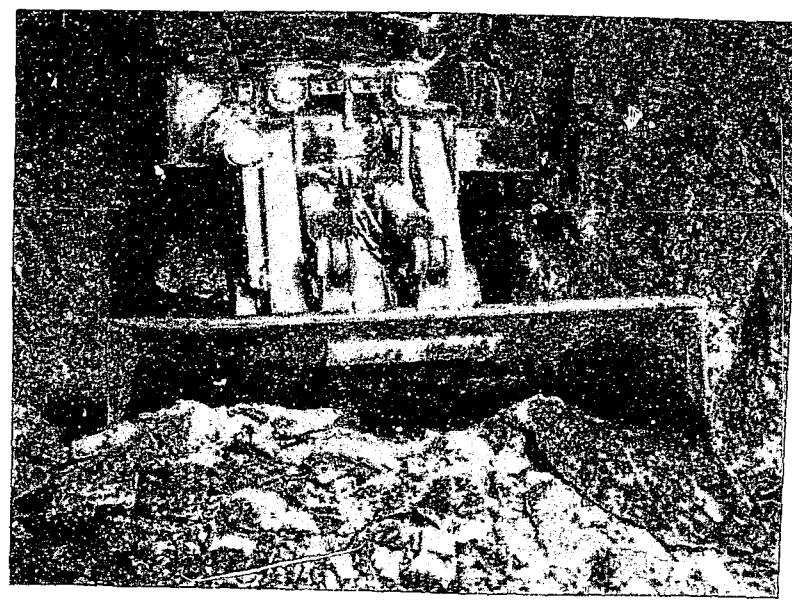
NOTES: • DIMENSIONS INSIDE PARENTHESES ARE SHOWN IN MILLIMETERS.
• HEIGHT OVER THE OPERATOR IS ADJUSTABLE DEPENDING ON APPLICATION.
• ALL HEIGHT DIMENSIONS ARE BASED ON A TIRE RADIUS OF 26" (660 MM)

Plate 7.2



Scaler Unit

Plate 7.3



3,5m^3 L.H.D.

7.3.1 Load Haul Dumpers

The LHD units selected for this operations were 3,5m^3 (5yd) units primarily because they were the largest available as well as being standard units on the mine and were available at the time. The dimension of this unit is depicted in Figure 7.2. Despite the facts expressed earlier the best all round unit for the work in this area was the 3,5m^3 unit. The costs of these units makes it virtually impossible to swop unit sizes when the conditions change but this will be covered under lessons learned.

There is the theory frequently propounded that the largest possible unit should be applied as this leads to higher productivity and lower working costs, however, machines need be matched to the conditions in which they are expected to work.

The large units are more suited to negotiating the steeper gradients as well as the extra power being useful in rapid loading cycles.

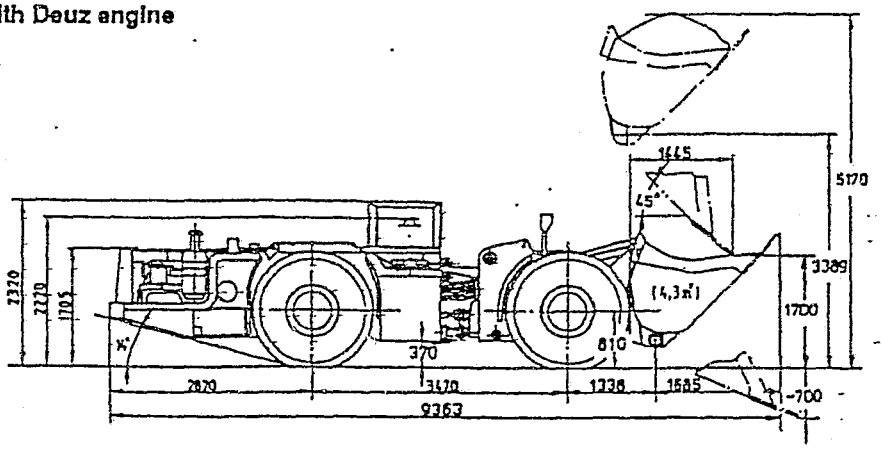
The 3,5m^3 bucket in practice has an 80% fill factor on average which means that some 2,8m^3 is loaded per trip which equates to approximately 4,8 tons and the filling of a 20 ton truck usually takes 4 passes.

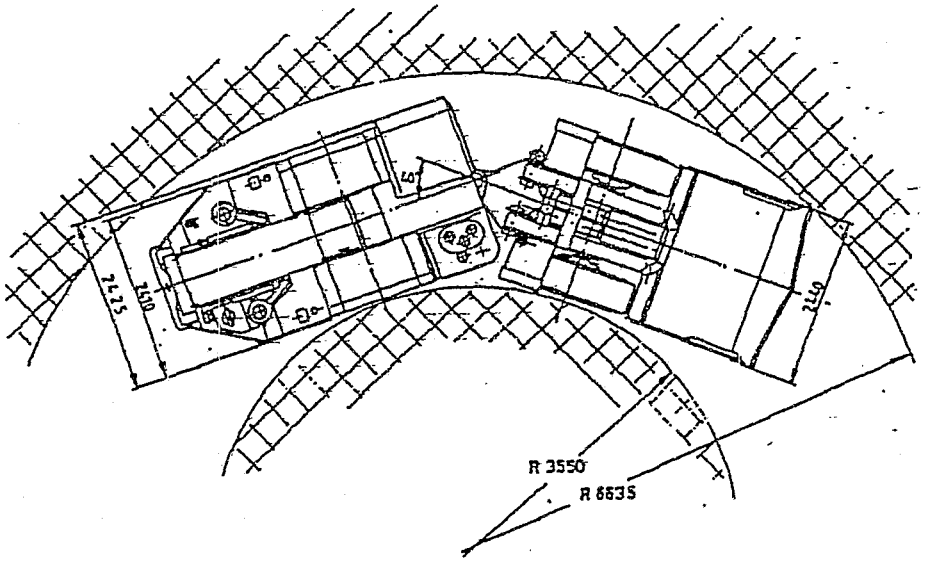
Random weighing of the trucks using transportable loadcells showed that loads varied between 18,4 tons and 24,2 tons but by far the greater amount tended to be in the region of 20 tons per load. The average over the tests indicated that slightly more than 20 tons was being loaded.

When loading directly into dump trucks single pass loading was encouraged. Operators at times believed that they had to take fully loaded buckets and this resulted in two and three passes at the rockpile, thereby increasing the loading time by 1 to 2 minutes because of the to and fro loading, dumping and re-loading. A good cycle time with single pass loading was in the order of 3,5 minutes per cycle while it was not uncommon to find, under the same conditions, the cycle times of in excess of 4 minutes were occurring and at times going out to 5,5 minutes.

MAIN DIMENSIONS

With Deuz engine





3,5 m³ LHD

A major disadvantage of trying to fill the bucket 100%, apart from wear, tear and wasted fuel is that a lot of fumes and heat is added to the air, thereby resulting in unpleasant working conditions.

The cycle time of the truck should be such that it gets back to the loading area just as the previous truck is leaving. Unfortunately, the underground layouts seldom make provision for this and it is not always as effective as what it should be because of having to move vehicles around.

With the output planned 3 x 3,5m^3 (table 7.1) units were allocated to this work which included the development, the clearing of the stope areas as well as the wide reef areas.

7.3.2 Drill Rigs

Twin boom rigs were allocated to the project, these were designed to drill 3,2m holes using 3,5 metres 28mm hex rods with R28 and R32 rope thread, and R28 45mm diameter button bits. A 75mm stab hole was drilled for the cut in each round. Initially, the drill rigs were also used to drill the roofbolt holes but because of the lengths of the boom the holes could not be drilled at 90° and this activity was replaced with hand held pneumatic machines. This not only resulted in better roofbolt installation but it also increased the availability of the drill rig for face drilling.

The overall result is that on a good day 2 full rounds were achieved per shift, where in practice between three and four rounds on a double shift was maintained.

As with all the TM^3 equipment, each unit gets a daily service inspection by an artisan as well as the pre-start checks by the operator. With the drill rigs though the artisan goes to the machine, mainly to reduce travelling time. Each drill rig, as with other units, is serviced for 1 shift per week in the underground workshop.

Other advantages of Electro Hydraulic drill rigs are:-

7.3.2.1 Faster penetration rates.

7.3.2.2 Reduction in use of compressed air.

7.3.2.3 Longer rounds can be drilled.

7.3.2.4 More accurate drilling is possible.

A typical drilling cycle for a twin boom drill rig is shown in figure 7.3.

The overall availability of the drill rigs remained acceptable for the project. The hydraulic system was responsible for the majority of the downtime which was still considerably less than that of the LHD and trucks.

7.3.3 **Dump Trucks**

Initially, two 20 ton dump trucks and one 15 ton truck were allocated to the project. Further into the project life, particularly as the distances increased, it was found the 15 ton was not a suitable choice and after a major structural failure was replaced with a third 26 ton truck. As the project wound down, the truck complement was again reduced to two. (Fig 7.4).

Roadway layouts are extremely important if dumptruck output is to be maximized. Bends, particularly 90° turns should be kept to a minimum and inclines kept at reasonable gradients, with 8° or less being desired whenever possible. Where inclines are required, they should be kept as short as possible. Roadway widths are important in order to maintain reasonable speeds and roadway underfoot conditions are critical if tramming cycles are to be kept short and operating costs kept under control.

In pillar mining ideal layouts are seldom possible and compromises need be made but not at the cost of safety.

Figure 7.3

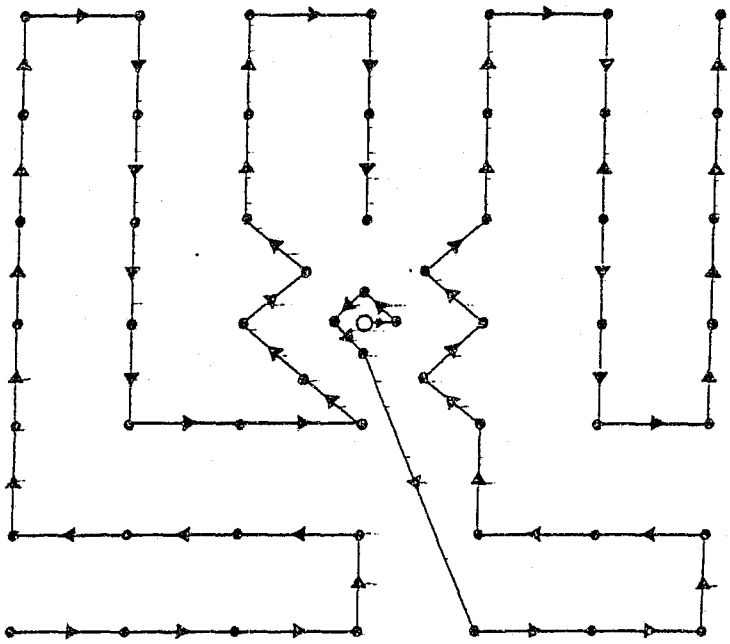
A TYPICAL DRILLING CYCLE PER DRIFTER

STAB HOLE = 76mm Ø

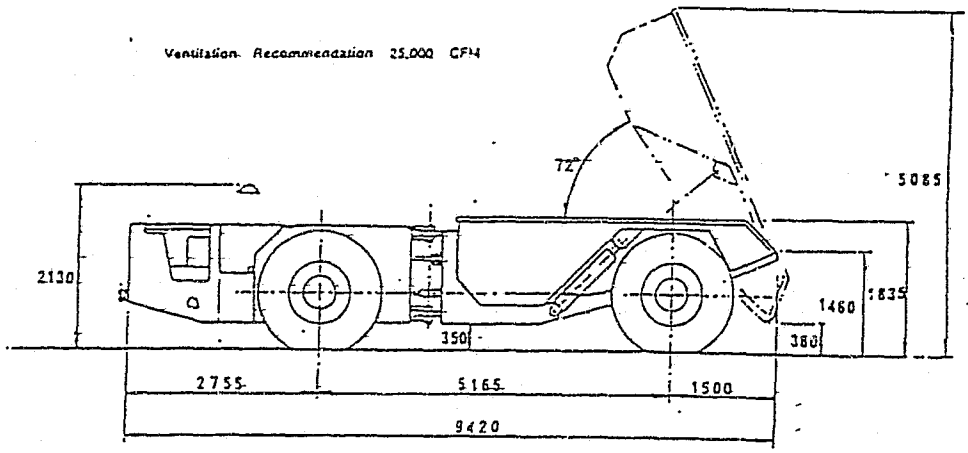
ROUND HOLES = 48mm Ø

L.H. BOOM DRILL 29 HOLES

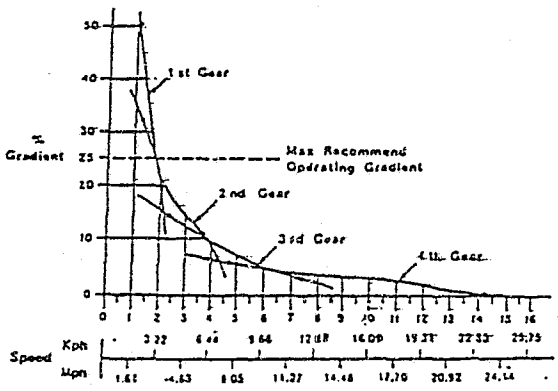
R.H. BOOM DRILL 28 HOLES

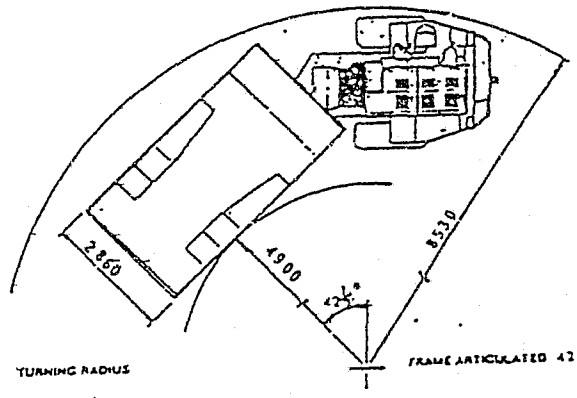


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Gradeability





VEHICLE SPEEDS LOADED – LEVEL PERFORMANC

GEAR	1	2	3	4
KM/H	3.8	7.2	13.7	24
MPH	2.4	4.5	8.6	15

Capacity & Performance

CAPACITY*

MAXIMUM ..... 22 Tonnes (24 Short Tons)

VOLUME RATING (SAE RATINGS)*

NOMINAL HEAP 13 m3

SEMI-NOMINAL HEAP ..... 11,5 m^3

STRUCK ..... 10 m^3

DISCHARGE

MAXIMUM DUMPING CLEARANCE @ 72°. 5085

DUMPING TIME ..... 15 SECONDS

Designed and Manufactured South Africa

7.3.4 Utility Vehicles

When TM^3 projects become of a significant size it is imperative to have suitable and adequate utility vehicles to service the mining, construction and maintenance operations.

The height of the roadways, which were up to 3,7 metres high, necessitated the use of scissors lifts to enable the pipe crews to suspend ventilation and air and water pipes and support crews to install roofbolts. Generally, the more you can keep the equipment out of the way the less damage occurs, particularly at bends. Where roadways can be carried very wide this is of lower importance but in the pillar area many of the roadways were carried at the narrowest that machines could be operated in without damage to equipment.

When insufficient utility vehicles are provided, as is often the case, it is soon found that very expensive production equipment is used to transport all sorts of equipment and material, from Jackhammers and hoses, through support equipment as well as sand, stone and cement.

A typical utility vehicle is shown in Plate 7.4.

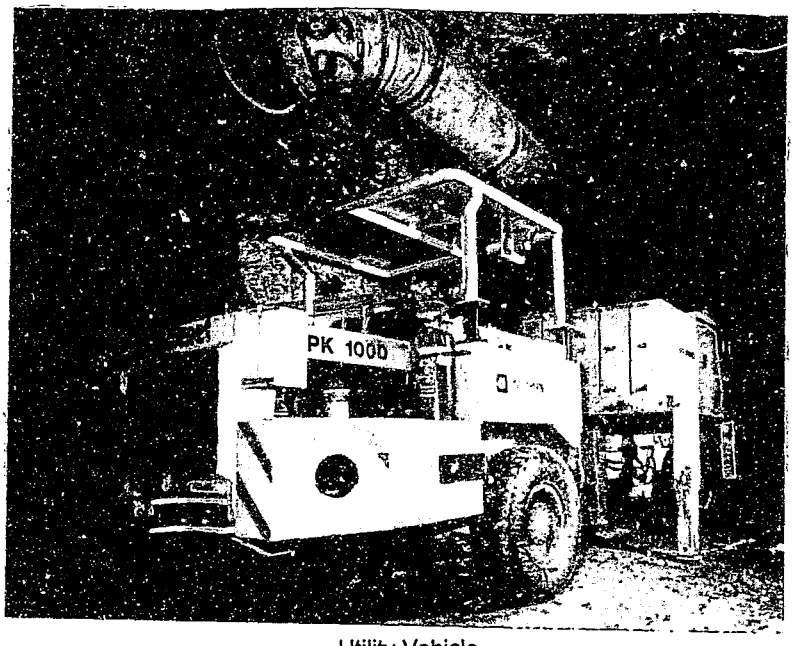
Transport of long material i.e. air and water pipes and often ventilation pipes remained a problem throughout the time of the project and were usually transported manually.

Important utility vehicles are the explosive transporters and the charging up scissors lift.

A major drawback of the utilities was their slow speed, which even on the horizontal plane has a speed of 6kms per hour and very much slower on the inclines.

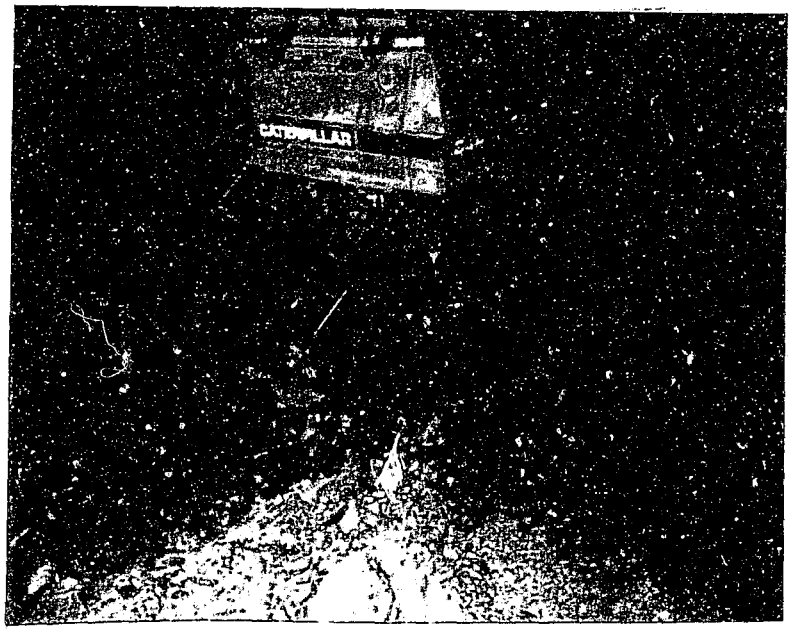
The height of the loading deck is too high for efficient work and although the units were supplied with cranes, these soon became unserviceable and the result was manual loading and unloading of equipment.

Plate 7.4



Utility Vehicle

Plate 7.5



D4 Dozer

7.3.5 Scalers

So often an after thought, a scaler is extremely important in areas where the faces are high and more so in areas of poor ground conditions where the quality of barring becomes even more critical particularly because of the size of rocks involved.

A major advantage of the scaler is that the operator is quite far from the barring face as well as having a protective screen in front of him. The enormous forces exerted by the hydraulic hammer ensures that errant rocks are put where they should be, on the ground. The scaler breaks up the larger blocks which are then loaded by the LHD.

These units often helped out on the secondary tipping points that were established whenever mining reached old ore passes. These ore passes were generally re-equipped with a box front and a grizzly installed over the opening. It was not cost effective to do the normal TM^3 tip construction because of the limited utilization of the area. This method significantly reduced tramming distances but their usage was limited to the availability of the scaler to clear the rocks.

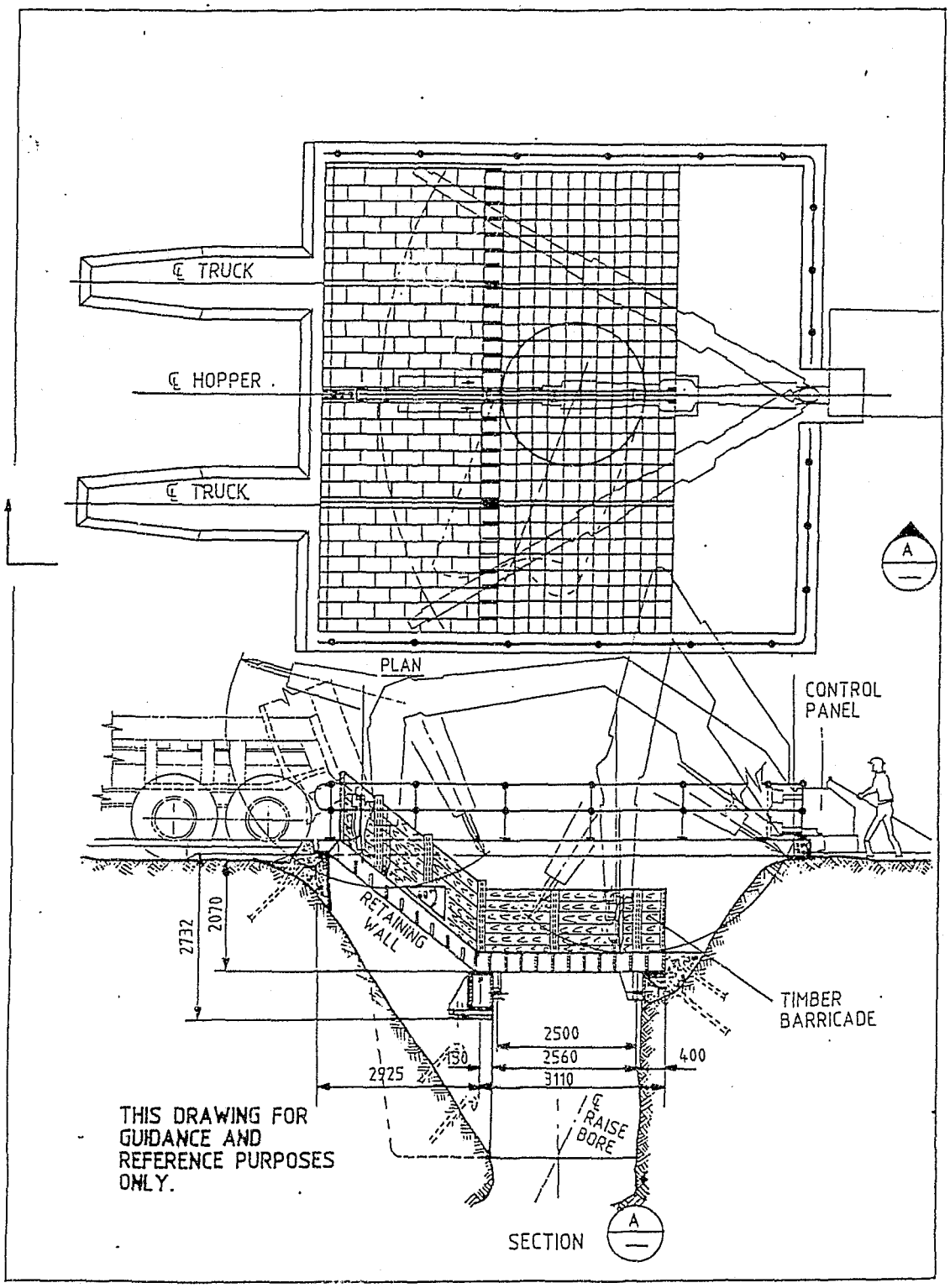
7.3.6 Impact Breaker and TM^3 Grizzley

This is a major construction done in most TM^3 operations and its distance needs be kept within the economic tramming distance for the vehicles utilized in the project. The tip is equipped with a stationary rockbreaker and a composite grizzly as depicted in Figure 7.5.

Tips are ideally situated within 150 metre tramming distance from the working faces for LHD's and 500 metres for truck operations. When tramming distances are likely to exceed these the situation must be evaluated regarding the amount of mining left and the cost of the additional tramming and reduced output and to

Figure 7.5

TM³ Grizzley Layout



evaluate these against either the construction of a new TM^3 grizzly or the use of a temporary tipping facility for LHDs. Provided of course that one can be established close to the working faces and to utilize a mobile breaker to remove the large rocks.

7.3.7 **Bulldozer** (Plate 7.5)

An extremely useful piece of equipment when having to construct roadways and to do vamping operations in trackless areas.

One of the biggest problems with doing vamping in trackless areas is the loosening of the road bed material which with time and water and the continuous running over them with heavy trackless equipment results in a well compacted roadway. Good for operating but terrible to reclaim unless a dozer is applied to rip and doze up the material for loading away.

The dozer was also used to construct the roadway required when old areas are traversed. The unit shaped the roadway and pushed excess material into heaps for loading away.

A particular advantage, found by the team, was that when waste stowing was being done, by using the dozer greater volumes of waste could be pushed into the available space.

Productivity of dozers falls off rapidly with increases in distances the material is pushed and at the same time the wear on the undercarriage increases. Careful planning and supervision is required to maximize output. Dozing distances must be kept short and the LHD needs to load away the heap at least twice per shift. A common failing is to leave the dozer to rip and doze material into a heap, when the heap gets too large the operator pushes the heap forward to continue working, this increases costs, results in double handling and reduced output with overall very much lower productivity.

7.4 Engineering Maintenance

In order to keep costs within the desired parameters maximum output of the units in a trackless section needs be consistently achieved.

One of the major factors effecting maximization of output is maximizing the availability of the machinery. In order to achieve this a good planned maintenance programme drawn up in conjunction with the manufacturers recommendations and the experience gained since the start of trackless mining within the group.

The provision of adequate work areas that can be kept clean, which in general meant concrete floors must be provided. These must as far as practical be free from dust, and by implication must be in the intake air system or adjacent to it. These facilities are of utmost importance if availability and productivity of trackless mining equipment is to be maintained at a high level.

7.4.1 95 Haulage Pillar

A small workshop, 20 metres long by 5 metres wide was established in one of the old cross cuts and relatively close to the workings. The units were assembled in the facility and thereafter all maintenance was done in this area.

The area was slipped from 3,5m wide to 5m wide and a concrete floor was put down and graded such that the water would flow out. The whole area was properly supported with roofbolts and whitewashed. The space and facilities were adequate for the particular operation.

7.4.2 101 Area

A substantial workshop facility was established in the area and fully equipped. This was an excellent facility situated on 101 level with direct access to the 101 haulage enabling the workshop to be supplied, relatively simply, with large components as the need arose.

The workshop facilities on 101 level comprised a store, working facilities complete with overhead crane, a service ramp, a boilermaking facility, a lockable

electrician facility as well as an area to repair hydraulic equipment and a combined tea room / meeting place / lecture room.

The workshop had its own fresh air intake and gas monitoring was regularly done by the foreman and the shift boss. Regular inspections were done by the Environmental department did regular inspections. It is essential to ensure that gas buildups do not occur, as the result of several units being run simultaneously, in the workshop area.

Although no significant problems occurred, possibly because of the attention paid to this aspect, strict control must be exercised to protect the health of the maintenance crew. Plate 7.6.

7.4.3 Planned Maintenance

A computerized planned maintenance and record-keeping system was in use on all JCI Mines.

Each piece of equipment has its own history file, which is gathered from the artisan job cards as well as from the operator's checklist. In addition to recording the down time and the causes there of, fuel consumption, oil consumption, run hours and a host of other information is input into the computer.

A monthly printout by machine for each area, machine class, and each machine was received by maintenance and production people.

Oil analysis was done with every oil change in order to monitor component wear and as an early warning of impending component failure, as well as being able to identify and eliminate the increase of undesirable materials such as silica dust, water, fuel dilution and oil viscosity.

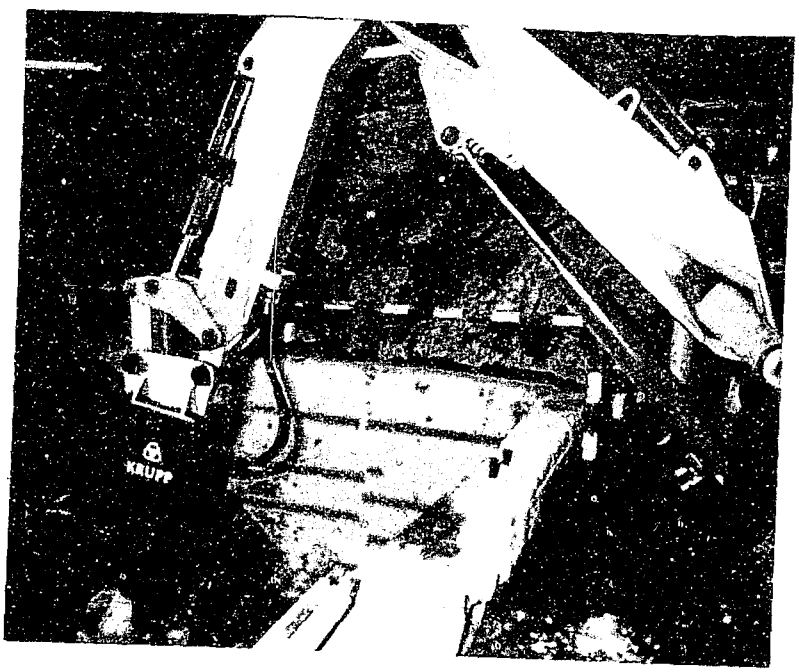
Tyre management formed an integral part of mining and engineering personnel's day to day function. In addition all records of tyres, type, life type of failure, retreaded were maintained by the planned maintenance section. This information was distributed to management monthly and analyzed and where necessary

Plate 7.6



U/G/Workshop

Plate 7.7



Stationary Rock Breaker

corrective action was taken. When one considers that there were some R1,3 Million (1993) worth of tyres in operation at any one time at Cooke 1 alone, this represents a substantial investment.

In order to get the best supervision on tyre usage as well as having an independent assessment done, the Mine had the tyre contractor do a monthly survey of every tyre as well as the working places where the machines were found to be operating.

The survey included:-

1. Tread depth.
2. Remaining operating time/hours forecast.
3. Cuts and their causes.
4. Mismatched tyres.
5. Rim conditions.
6. Wheel stud and nut conditions.
7. Valve caps.
8. Uneven wear.
9. Unacceptable wear/possible claims.
10. Tyres run too far and not able to re-tread.
11. Oil contamination's.
12. Roadway conditions
13. Loading point conditions.
14. Excessive water.
15. Tyre pressures.
16. Poor operator techniques.

As can be seen from the above list, a very comprehensive investigation was carried out and not only does it prevent unnecessary damage and minimize tyre costs it also keeps the operating staff aware of an independent roadway/loading survey which in part eliminates some of the friction between the Mining and Engineering team members.

The planned maintenance system was and must be a pro-active system. To change an engine or transmission in the workshop where good lighting, crane facilities and good floor conditions exist is many, many times easier than struggling in the working place with restricted height, no proper lifting facilities, cap lamps for lighting and totally unsuitable underfoot conditions.

The writer foresees the question being raised by many people as to why the broken down machine is not towed back to the workshop for repairs. Well, there are two very important reasons which are that without the motor running there is no hydraulic pressure to release the brakes as they are of the fail safe system and even if they were disengaged the machine has no steering as it also required the hydraulic pressure to be generated by the running motor. Klintworth and Cox set out in great detail to describe a planned maintenance system in their presentation "Maintenance of Trackless Mechanized Mining Equipment – Trackless Mining Symposium 1988.

TABLE 7.1

TRACKLESS EQUIPMENT ALLOCATED TO THE 101 PROJECT:

3.5 Cubic metres Load Haul Dumpers	-	3
Twin Boom Electro Hydraulic Drill Rigs	-	2
20 Ton Low Profile dump trucks	-	3
Utility Vehicles	-	4
Mobile Scaler	-	1
Supervision Vehicles	-	2
D4 Dozer used in vamping operations	-	1
Stationary rockbreaker	-	1

CHAPTER 8

Mining Operations

8.1 Introduction

With the thorough planning that had been done on the 95 haulage pillar during the initial phases, particularly with respect to the opening up and layouts, the mining operations, per se, went extremely well. The small multi skilled group under the direct supervision of the miner with the trackless equipment provided, certainly made the whole operation simpler and easier to control.

The 101 area commenced in 1986 with the aim of extracting as many payable pillars left between conventionally mined out areas on the UE1A reef horizon as possible. By the end of 1992 the available reef on the UE1A was nearing depletion with only some tertiary extraction remaining. This gives an excellent opportunity to describe the mining operations that had successfully been applied during the project.

At Cooke 1 shaft both conventional mining methods, as well as trackless mining methods and a number of hybrid systems utilizing the benefits of trackless mining as well as narrow reef stoping were applied depending on the conditions applying in the specific area.

The conventional mining was done on a scattered method on various reef horizons across the mine in part due to the very definitive pay shoots.

Ore removal to the shaft was done by track bound 6 ton hoppers pulled by battery powered locomotives on 90, 95 and 101 levels and by 10 ton overhead trolley line locomotives with 10 ton hoppers on the 106 haulage level.

Refuge chambers were laid out and equipped and all persons working in the area were made aware of the escape routes as well as the location of these refuge chambers. The necessary persons were put in charge and simulated training was conducted from time to time.

8.2 95 Haulage Pillar

With the 95 haulage, pillar mining operations as such could not commence immediately. Firstly, the accumulated rubbish and old material had to be loaded up and sent out of the mine. The haulage was then made safe, as described in paragraph 5.2, and supported concurrent with clearing and making safe.

8.2.1 Selection of Crew

Careful consideration was given to the selection of the crew members. People, particularly the supervisory staff were selected because of their attitude towards safety, their ability to do the job at hand, their known ability to overcome obstacles.

To mine the area at the rate planned required the following crew:
1 Mine Overseer (Part Time)
1 Shift Boss (Part Time)
1 Ganger
12 Category 'B' Personnel
Composite crew except for the LHD and Scaler operators.

The last two categories mentioned being those who would be involved in the day to day operations. All crewmembers were thoroughly briefed, before operations began, on what was required of them. No additional engineering personnel would be allocated to this work with all engineering input of cables, construction, maintenance and services would be undertaken by the nearby 90 South 3 project crew.

The 90 South 3 Mine Overseer was in charge of the operations and the most suitable Shift Boss from the area was selected and the project allocated to him as part of his section of responsibility.

8.2.2 Requirements

Two thousand four hundred tons per month or
One hundred tons per day on average.
Average grade twenty gram/ton.
Semi trackless mining with hand held jackhammers and LHD cleaning.

Once a face started mining every effort had to be made to maintain a continuous advance and overall a steady rate of extraction was, as far as practical, maintained throughout. This was necessary to prevent undue stress build ups ahead of the working face. Only 1 shift per day operation was permitted with no night shift work being allowed in the pillar area. This ensured better supervision. The same crew was responsible for tramming of the reef to the tips on the station. The crew was multi disciplined.

8.2.3 **Preparation**

The cutting of the mesh was achieved by using a specially adapted cutting torch thus enabling the person to stand at a safe distance from the actual cutting operation.

This re-opening was done in small sections of approximately two metres at a time. Once the two metre section had been bled, barred and made safe, the hanging wall was re-supported by means of 2,4 x 25mm diameter, fully grouted, re-bar that was drilled through newly suspended wire mesh. The opening up of 2 metres at a time was to be a tedious job that had to be done in short sections in order not to put the safety of people at risk.

The method of removing the mesh and bleeding out the scaled rock was a follows:

1. Temporary, mechanical, support was installed at the last point where the mesh was safe or had already been made safe.

2. Permission had been obtained to use the LHD bucket as a platform and the necessary safety chains were used to prevent inadvertent tipping of the bucket.

3. Using a torch that had been "extended" the lacing and mesh was cut away. The long torch meant that it was not necessary to pass the line of mechanical support.

4. Once the wire mesh and lacing was out of the way and all the loose rocks had fallen (beyond the support area) the scaler was brought up and the exposed area barred solid.

5. The temporary support was then extended into the newly prepared area.

6. The wire mesh was put in place and supported with mechanical jacks.

7. The roofbolts were then drilled through the mesh to prevent any injuries from falling rocks. The bolts were inserted using resin capsules. Once set, the washers were put in place and the mesh pulled up.

8. No persons were allowed into the area still to be made safe.

9. The lacing followed behind the wire meshing.

10. As the team progressed further into the haulage it was found that once the wire mesh and scaling had been removed and the area barred solid, there was no need to replace the mesh. The ground conditions are clearly visible in Plate 8.2.

A lot of effort was put into the planning of this section of the operations but none the less it was modified as the work progressed and safer or better methods were found which made the individual activities safer and more productive.

8.2.4 Mining

Once the haulage had been opened, cleared and re-supported, the next phase was to expose the reef fully. As a result of the numerous faults, the reef horizon had been displaced both up and down a number of times. Reef exposure was achieved by lifting footwall or slipping hanging whichever was appropriate. The final section along the haulage is depicted in figure 8.1 which clearly indicates the reef displacements and the amount of footwall lifting or hanging wall slipping required to fully expose the reef.

Plate 8.1



Showing wire mesh that would require bleeding

Plate 8.2



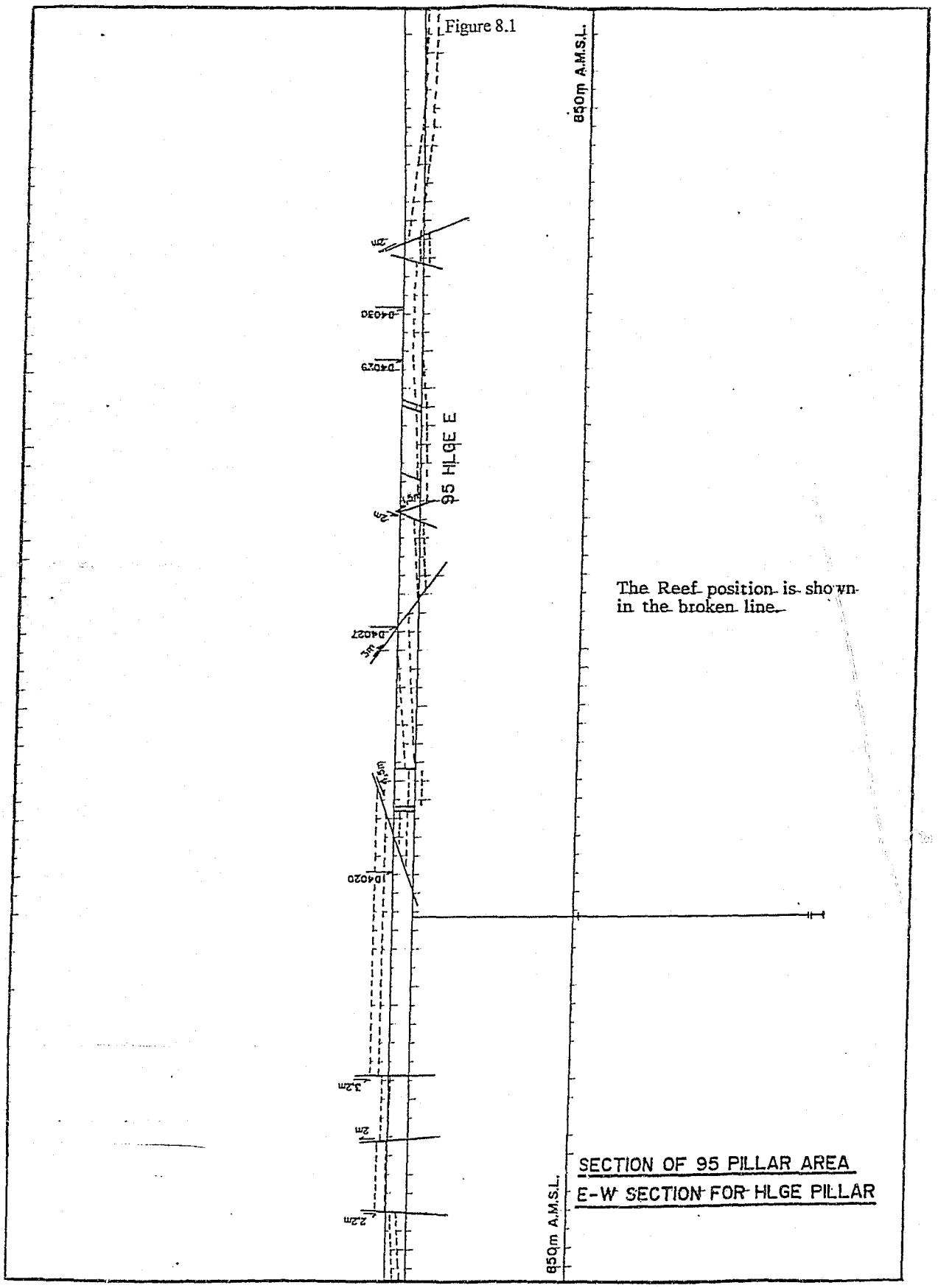
One of the old stope faces

Plate 8.3



Opening up Phase on 95 Haulage East

Figure 8.1



SECTION OF 95 PILLAR AREA

E-W SECTION FOR HLGE PILLAR

The pillar was then pre-developed by blasting 3,5m wide x 2,5 metre high (full reef width) drives at approximately 25 metre centre (figure 8.2). In order to minimize areas of potential problems a rigid pattern could not be adhered to and cognizance had to be taken of the positions of faults when laying out this development.

The planned development was modelled on the Rock Engineering computer prior to any work commencing, in order to ensure that no unacceptable stress patterns were evident. The layout resulted in seventeen panels being made available, each having a length of ± 7 meters. The panels were advanced from the breaking point created by the development, along the direction of elongation as shown in figure 8.3.

The next phase, which is where the mining proper commenced, was to expose the reef fully. As a result of the faulting the reef in hanging required stripping to fully expose the reef band and on the other sections the reef was in the footwall which then required lifting of footwall to fully expose the reef band.

As can be appreciated, this resulted in an apparently very disjointed layout with roadways having to negotiate highs and lows. The reef exposure after the footwall lifting and hanging wall slipping is shown in figure 8.4.

The full reef exposure was achieved by establishing a face the width of the haulage and then daily drilling and blasting this face, fully exposing the reef, until the next fault was intersected.

In the case of hanging wall slipping, the material that was blasted down by and large was left to form the roadway with only excess material being removed. This also meant that very little tonnage was trammed from the project during this stage.

Figure 8.2

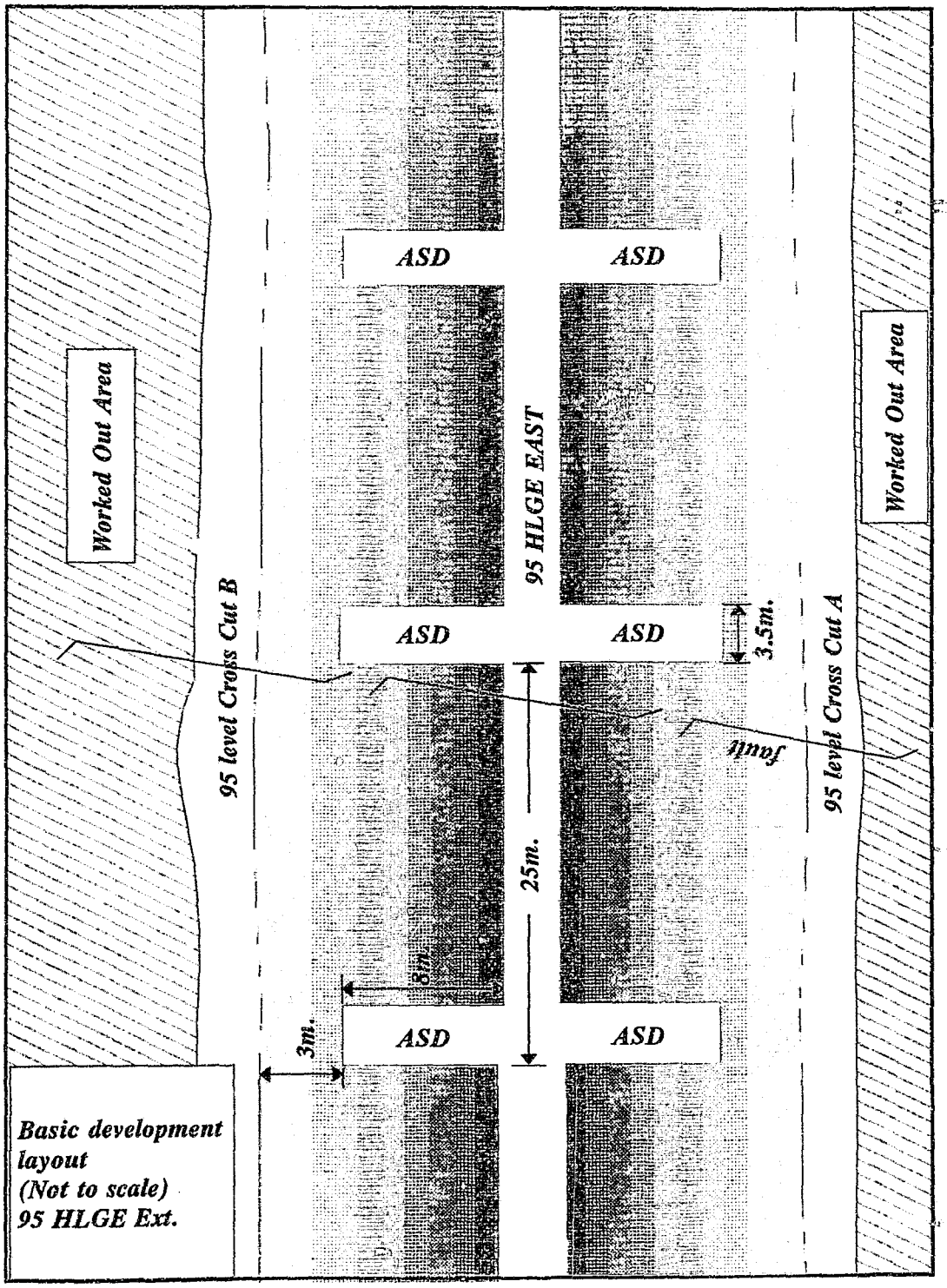
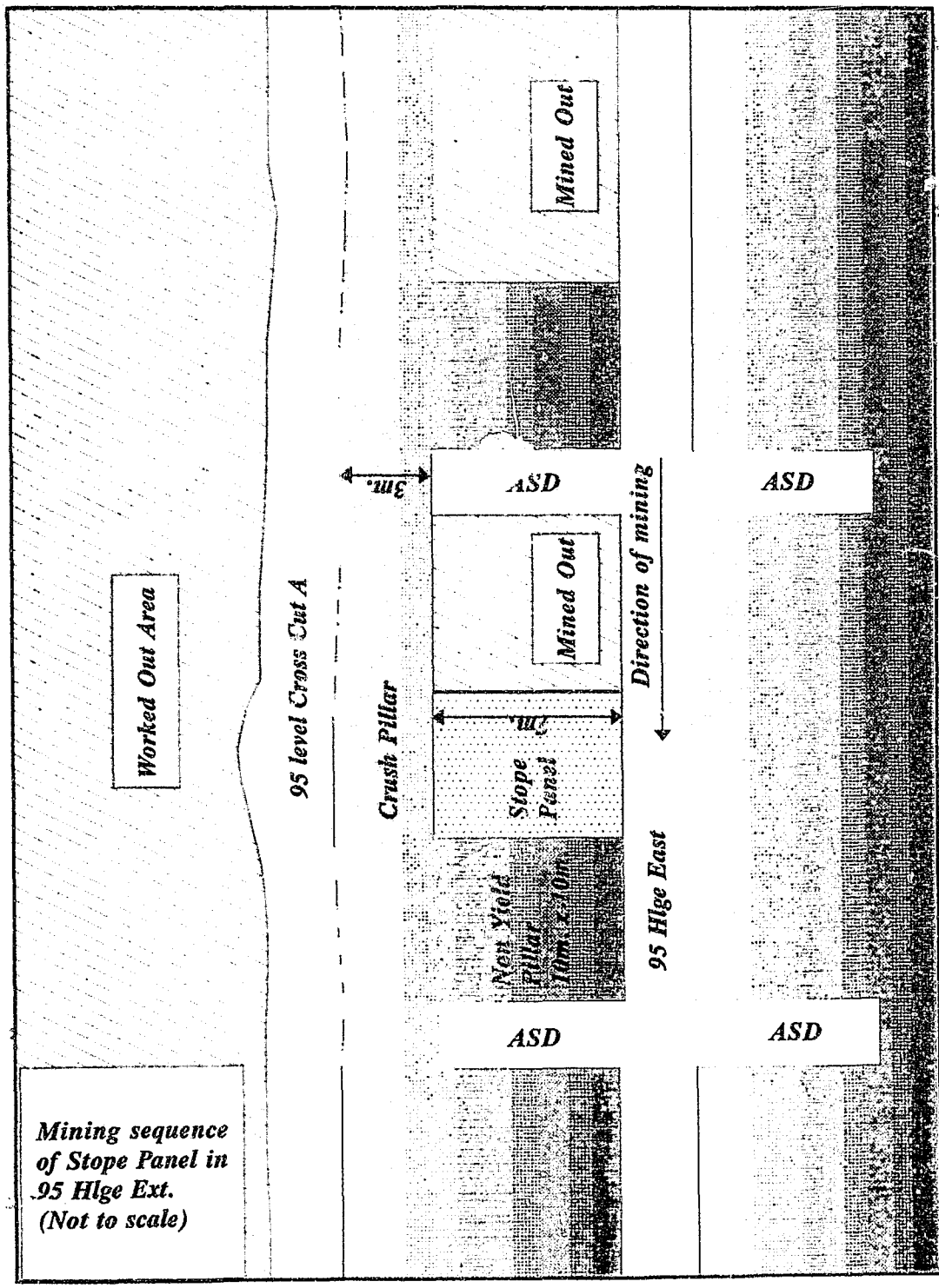

Worked Out Area
95 level Cross Cut B
ASD
ASD
95 HLGE EAST
ASD
ASD
3.5m.
25m.
6m.
3m.
ASD
ASD
95 level Cross Cut A
Worked Out Area
Basic development layout
(Not to scale)
95 HLGE Ext.

Figure 8.3


Mined Out
ASD
ASD
3m
Worked Out Area
95 level Cross Cut A
Mined Out
Direction of mining
Crush Pillar
7m
Stope Panel
Net Yield Pillar 10m x 10m
95 Hlge East
ASD
ASD
Mining sequence of Stope Panel in 95 Hlge Ext. (Not to scale)

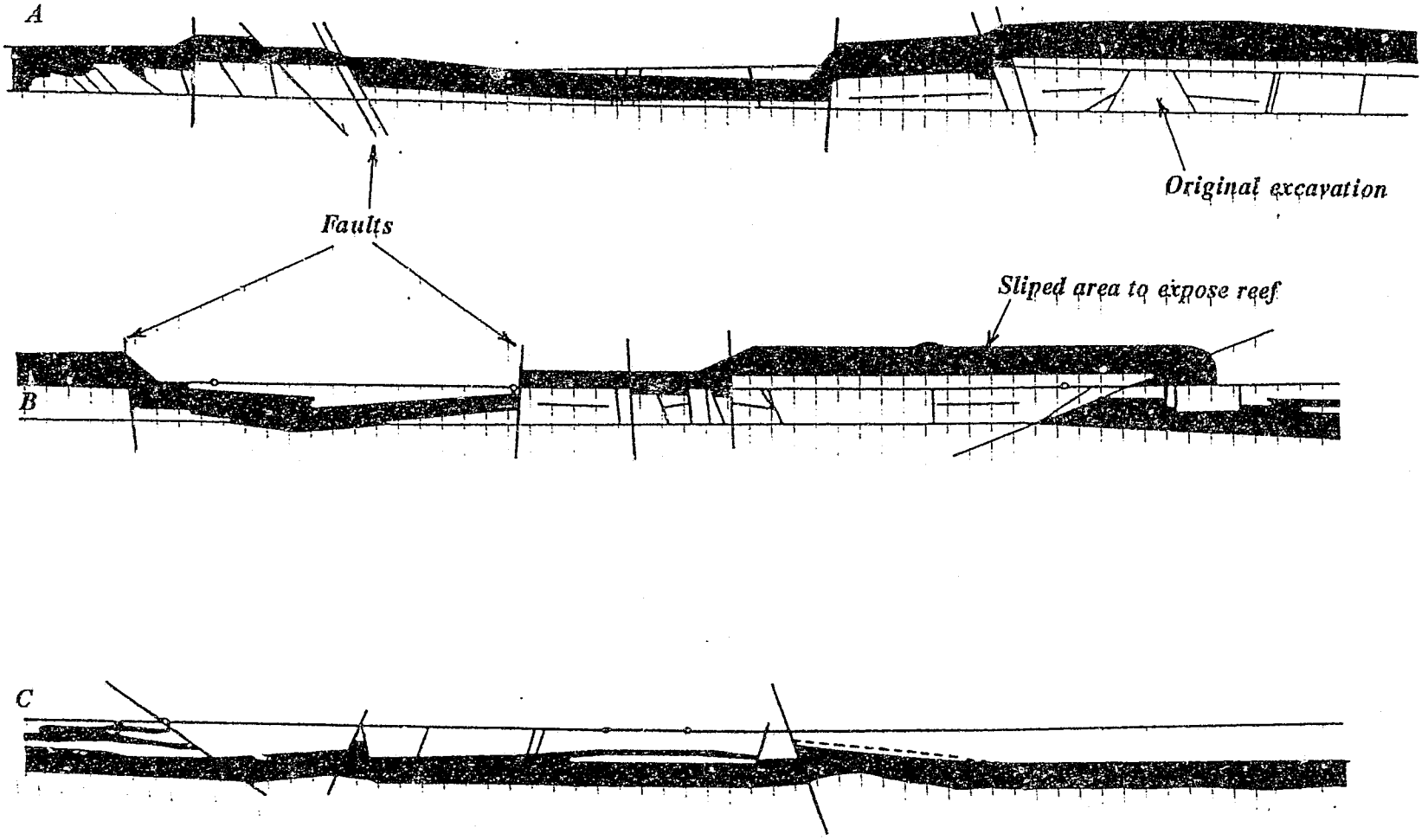


Figure 8.4

Section of 95 Pillar indicating sliping to expose the reef.
Indicating the amount of hangingwall and footwall sliping necessary to expose the reef (shown in black).

Drives of 3,5m x 2,5m high were then excavated at ±25 metres centres or as dictated by faulting. In some cases this resulted in bringing the drives less than or slightly more than 25 metres apart.

This resulted in 17 panels with a face lengths of 7 metres.

The panel was advanced from the created breaking point for a distance not exceeding 14metres. These panels were mined on retreat from the furthest point towards the shaft pillar. Full remnant precautions were applied from the outset and strictly enforced until the very end of the operation.

One of the recommendations of the rockburst communion of 1964 was that faces should be blasted regularly and in this case 1,2 metre advance was blasted in a panel daily until such time as the panel was completed. It was then cleared and swept before the next panel commenced. This very soon became a smooth operation with very little time between the last blast and the commencement of mining in the next set of panels.

A mobile scaler was allocated to the project from the outset as the high impact forces a long reach was imperative for safe efficient operations. The pressure that had occurred earlier as the result of mining around the pillar caused the rock to scale significantly and was the reason the pillar area was originally wire meshed. This rock scaling had generated a lot of large rocks in places up to 2m deep which were released when the wire mesh was cut. The shaft ore pass system was equipped with 30cm x 30cm grizzlies and therefore it was important to reduce rock sizes so that they would pass through these grizzlies. The breaking of rocks on shaft grizzlies is not only difficult and time consuming but also delays the trains and causes slower cycle times. Having the same crew do both operations resulted in far better teamwork with rocks generally being broken in the mining area.

A major unknown was how far the rock scaling had progressed from the tunnel sidewalls or hanging wall. From physical inspections it was evident that a lot of movement had taken place as can be seen on plates 8.2 and 8.3. The fault

Plate 8.2



One of the old stope faces

Plate 8.3



Opening up Phase on 95 Haulage East

zones were known to be problem areas and as slipping was necessary in order to expose the reef fully these areas would be high and the scalers reach would be essential for the extensive barring that would be required.

The working height created by the mining of the composite reefs enabled the mobile scaler to access the mining areas with ease.

8.2.5 **Drilling**

The 1.5m holes were drilled through wire mesh supported on mechanical jacks against the hanging wall and held against the working face. The practice of drilling through the wire mesh prevented any injuries caused by rocks from the often friable faces.

The 1,2 metre advance per blast was chosen for the following reasons:

8.2.5.1 Avoidance or minimizing of hanging wall damage.

8.2.5.2 The unsupported span after the blast was relatively small and was Largely covered by the last row of support.

8.2.5.3 The tonnage was such that the face could be cleaned in the allotted time.

8.2.5.4 Drilling control with short holes was better.

8.2.5.5 The length was conductive to regular face advance.

The drilling pattern was 3 rows of holes spaced 80 cm apart and drilled at 80° into the face using hand held jackhammers with standard development airlegs.

8.2.6 **Blasting**

Various types of explosives and initiating systems were considered as it was essential that minimal hanging wall damage occurred from blasting operations.

The initial feelings were that Anfex should not be used because of the possibility of the explosive gases entering the fractures already existing in the hanging wall,

thereby creating further support and safety problems. Consultations and discussions were held with Geologists, Rock Engineering and the explosives suppliers in an effort to assimilate all the information on the expected rock behaviour and the effects of the different types of explosives and initiation systems.

The decision taken was that initial trials would be conducted using the standard mine explosives, this being an Ammonium Nitrate blasting agent and the standard 8D detonator with slow stope igniter cord.

The reasons for the decision were as follows:

8.2.6.1 A distinct parting was evident on the hanging wall contact.

8.2.6.2 The hanging wall was competent.

8.2.6.3 With correct burden, spacing and timing, very little back damage occurred.

8.2.6.4 It was the standard explosives system and therefore all persons would be completely familiar with the use of the explosives and the treatment of misfires.

8.2.6.5 It was possible that a certain amount of hole distortion would occur due to the stresses already in the pillar and the changing stress pattern as the pillar was mined. This type of hole distortion would be readily overcome by using Anfex which would not be possible with any of the cartridge explosives.

8.2.6.6 The reef was known to be very brittle.

8.2.6.7 The 80cm burden selected in conjunction with modelling and advice from AECI for the holes to clear rapidly resulted in very little back damage into the hanging wall.

8.2.6.8 No benefit in changing the explosive type was apparent.

The first set of panels were evaluated on a daily basis by Mining, Geology, Rock Engineering, and the explosives advisor in order to determine whether or not the correct system was being applied.

An alternate system was available and should the chosen system not have proved satisfactory, then the use of a higher V.O.D. explosive with less gas generation would have been evaluated.

Plates 8.4 and 8.5 show the results of good blasting and support.

8.2.7 **Cleaning Operations**

A 2,5m^3 load-haul-dumper (LHD) loaded directly into 6 ton hoppers at the entrance to the pillar area. A single battery powered locomotive provided with six hoppers trammed directly to the main station tips.

There was sufficient time to complete the cleaning of the blasted rock as well as to transport all the equipment. Working one shift only does not make full use of the L.H.D. and were it possible the unit could have been utilised elsewhere on night shift. A one-shift operation was applied to ensure good, consistent supervision.

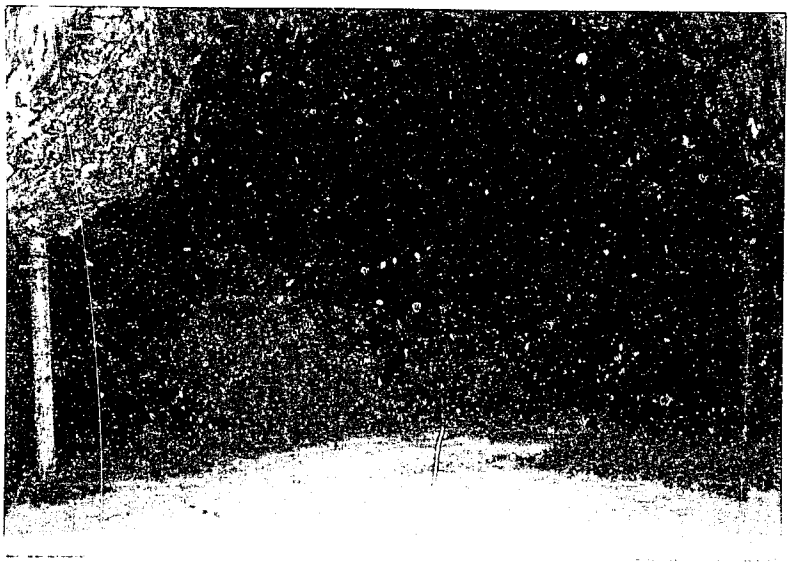
The application of the LHD enabled the faults to be negotiated and the hanging and footwall stepped up or down with the units flexibility being a distinct advantage. The fact that there would be no scraper ropes and different snatching and tipping points would result in less cluttered areas and quick safe exits from the area would be possible.

As the pillar was mined on a retreat basis with short panels the only additional escape way that was maintained at all times was along the old haulage that had not become the trackless roadway. The width of the pillar being mined and the support pillars left was such that no additional escapeway was deemed to be necessary.

8.2.8 **Sequence of Operations**

Each set of panels, on both sides of the haulage were mined out, properly swept and barricaded off before mining operations started in the next set of panels.

Plate 8.4



A Holing into the Old Reef drive

Plate 8.5



More than a year after the area was left
A little scaling is evident

In this manner it was totally unnecessary for any person to enter the worked out areas and these could be barricaded off. This eliminated the risk that is always evident in mined out areas but more so in pillar operation. Plate 8.6, 8.7, 8.8.

8.3 101 Window/Bowl Project

Due to the diversity of the reef structure encountered, a variety of layouts and methods were employed to effectively mine the area, with reef widths varying from 80cm up to nearly 4 metres and the dip not only varying from flat to as steep as 70°, it also formed a bowl shape.

8.3.1. Steep Mining Areas (dip grater than 35°)

After extensive modeling on the Rock Engineering departments, computer in order to simulate mining sequences and evaluate stress patterns, it was decided that the area would be mined in a series of breast panels.

Strike roadways were developed from an incline access way giving panel lengths of ±30 metres. Raises ±60 m apart were developed between these roadways.

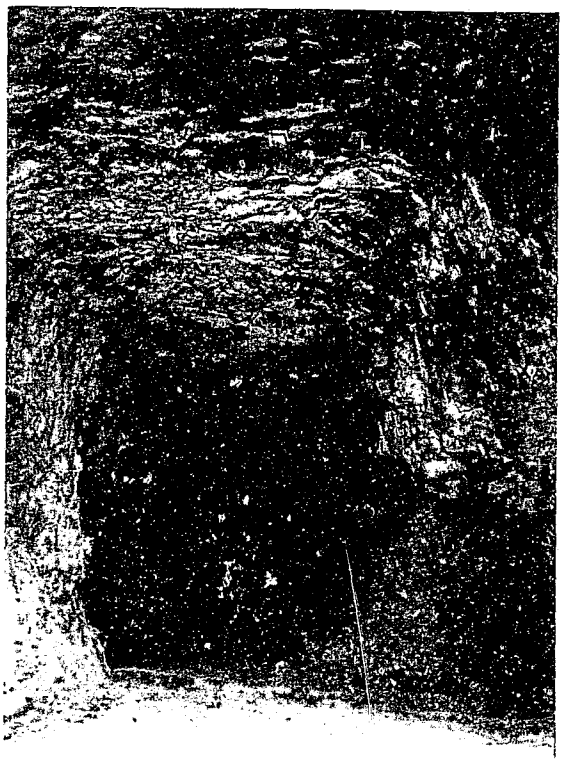
The primary reef mined in the area was the UE1A which is underlain by the E9Gd. These two bands were either contiguous or separated by a narrow waste parting. The composite reef package in places contained good values but in general only the UE1A reef carried regular values and therefore only the UE1A reef was planned to be extracted.

8.3.1.1 Selecting the Crew

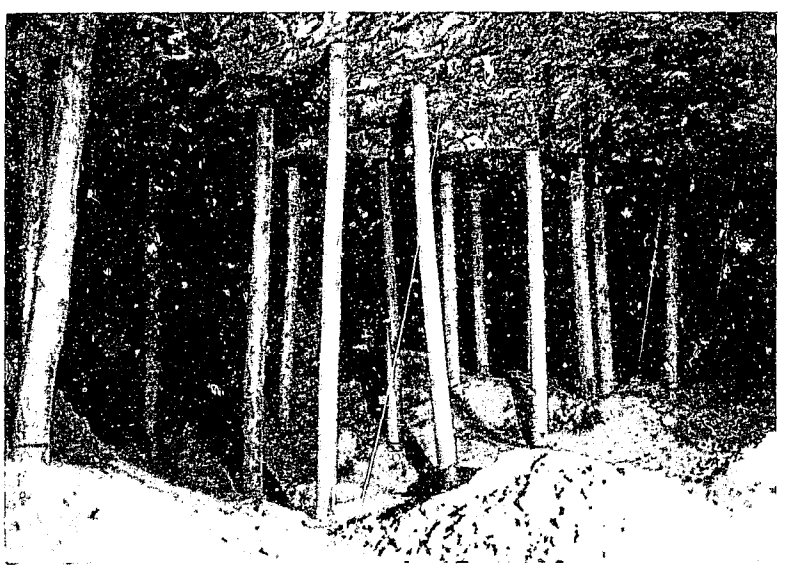
Although steep mining had previously been extensively applied in the old Randfontein Estates section it was not a method of mining generally practised in the Cooke Section.

A miner that had the necessary experience with steep mining was identified.

Plate 8.6

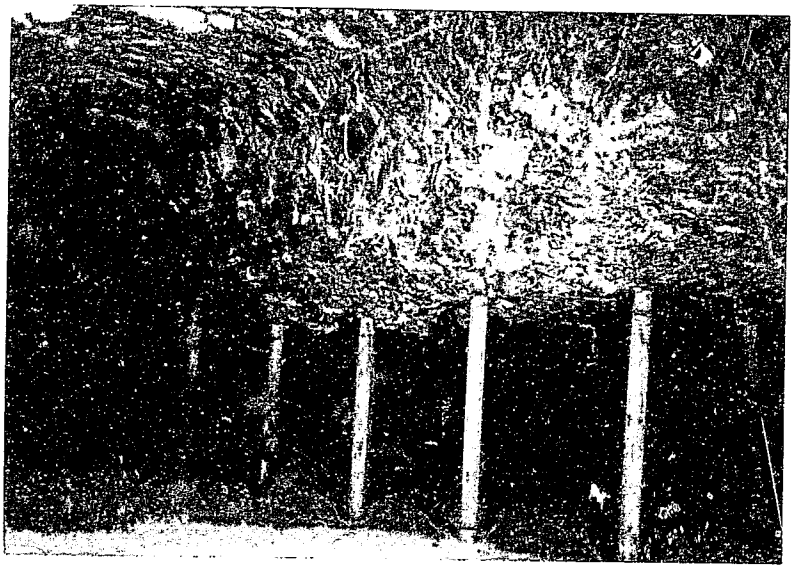


The old haulage is at the top showing support and
Footwall lifting necessary to get into reef



A stope panel on limit being swept

Plate 8.7



A stope showing the brows that were created

Plate 8.8



An indication of the height of the excavations

The machine crews were made up of persons experienced in steep mining as well as some that had worked with the particular miner previously. The main objective in the crew selection was to gather a nucleus of people experienced in steep mining activities. This selection was a crucial factor in the safety of persons who were to perform the various functions.

The process was repeated particularly with the Team Leader and the timber crews. The remainder of the persons were in the less critical functions and could be taught their job requirements.

The crew comprised:

Day shift
1 Mine Overseer (Part Time)
1 Shift Boss (Part Time)
1 Ganger
14 Category 'B' personnel as follows:
1 Team leader
6 Jack hammer handlers
Timber Crew
2 Miners Assistants
1 General worker

Night shift
1 Shift Boss (Part Time)
1 Cleaner (Part time)
2 Category 'B' personnel comprising of LHD operator and a Gang Supervisor.

8.3.1.2 **Preparation**

While the development was being done and the raises established the old standards for steep mining were reviewed and brought up to standard with the mines present high safety requirements. (The mine holds the

industry record for the most million shift awards achieved). The work in progress was already covered by existing standards.

Once the standards had been finalized the crew was thoroughly briefed and instructed in the safe mining practices. When the ledging operation started an instructor was allocated to the crew to assist the ganger and team leader in training and overseeing operations particularly from a safety viewpoint.

A matter of serious concern was protection from falling or rolling rocks. Swinging stulls were installed below each raise to protect the LHD driver and any other persons who may have been in the reef drive. These swinging stulls were moved forward as the face advanced with solid stulls installed in the area already mined.

8.3.1.3 **Mining**

The area was to be mined with breast faces from each of the raise connections with a continuous crown pillar. Figure 8.5. indicates the method of mining. Drilling and blasting took place on the dayshift. On the night shift an LHD would load the rock from the blast which had gravitated to the roadway below the face. Loading was done on the night shift to ensure that the minimum number of people would be in the area below the face when barring and drilling were taking place.

At the start of the shift the faces were barred from the top down, as is the mine standard, by the team leader and one assistant.

8.3.1.4 **Drilling**

1,2 metre long holes were drilled at 70° to the face with a spacing of 60 cms. Holes are marked in a staggered pattern with 2 rows of holes for stoping widths of less than 120cms and 3 rows where the stoping width increased to beyond 120cms. Drilling was done using hand held jackhammers and airlegs. Special chain ladders had to be used and anchored to enable the machine crews to have a safe place to stand and to facilitate drilling. Figure 8.6.

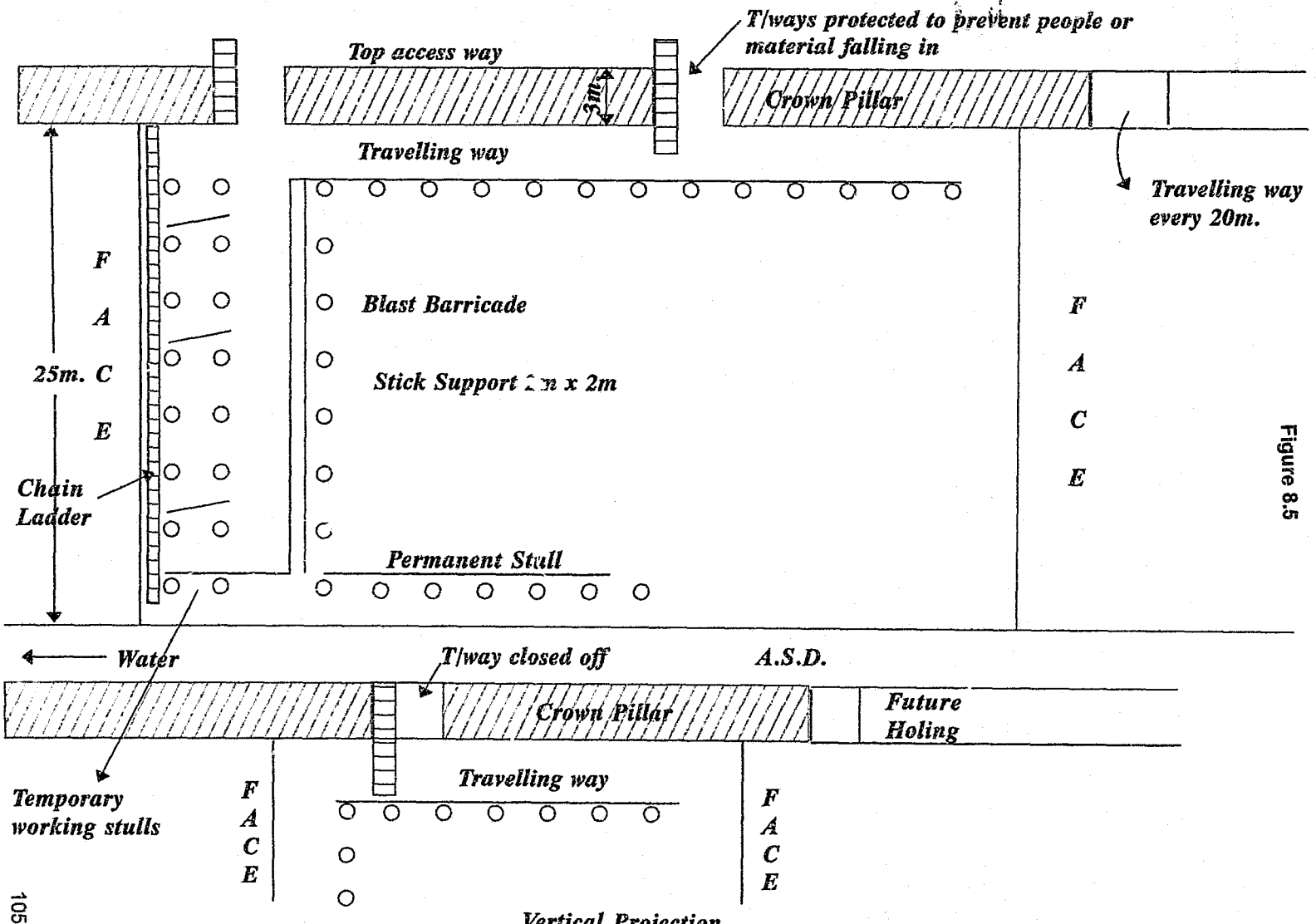
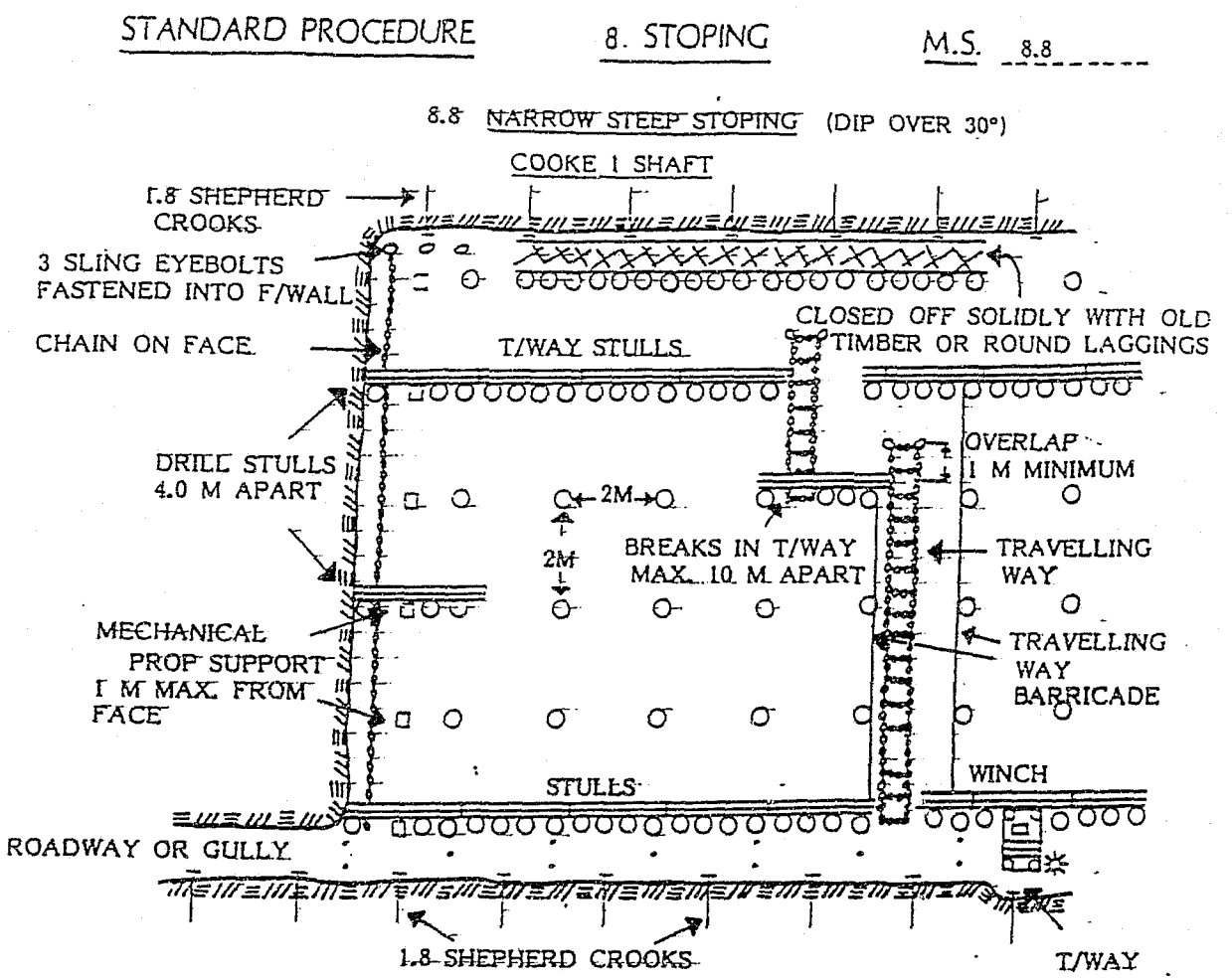


Figure 8.5

Diagrammatic layout of steep stoping area

Figure 8.6


STANDARD PROCEDURE
8. STOPING
M.S. 8.8
8.8 NARROW STEEP STOPING (DIP OVER 30°)
COOKE I SHAFT
1.8 SHEPHERD CROOKS
3 SLING EYEBOLTS FASTENED INTO F/WALL
CHAIN ON FACE
T/WAY STULLS
CLOSED OFF SOLIDLY WITH OLD TIMBER OR ROUND LAGGINGS
DRILL STULLS 4.0 M APART
2M
2M
BREAKS IN T/WAY MAX. 10. M. APART
OVERLAP 1 M MINIMUM
TRAVELLING WAY
MECHANICAL PROP SUPPORT 1 M MAX. FROM FACE
TRAVELLING WAY BARRICADE
STULLS
WINCH
ROADWAY OR GULLY
1.8 SHEPHERD CROOKS
T/WAY

8.3.1.5 **Blasting**

Pneumatically loaded Anba was the standard explosive used as was used on the rest of the mine with stope fuse equipped with 8D detonators and twin igniter cord. As with the rest of the mine 1 hour stay alights were used and blasting was on a time basis at the end of the day shift.

8.3.2 **Stack Zone**

This area was a highly complex zone resulting from numerous faults and the reef being "stacked" in a series of steps. Refer to figure 5.10. The numerous faults not only made the mining of such an area extremely difficult but also potentially hazardous.

The planning was done in great detail and simulated in steps on the Rock Engineering p.c..

The narrow reef areas where the faulting was closer than 10m and the stacking very complicated were left unmined. In the wide reef areas a safe and efficient trackless method could be laid out and mined.

8.3.2.1 **Preparation**

An incline roadway from the 101 reef drive was developed at a minor dip of -8° in order to access the area. Panels of 4 to 7 metres (maximum) wide were then laid out to run down between the faults. In areas where there was sufficient undisturbed ground two or more panels were developed parallel to one another with 8m wide pillars in between.

LHD and truck tips were established and LHD's operating distance to tips was kept to less than 150 metres and trucks to a maximum 1 way trip of 500 metres.

A fully functional workshop facility was available on 101 level.

The development of the panels was in effect the initial mining of the area as these trackless roadways were up to 3.5 metres high and between 4 metres and 7 metres wide, depending on the ground between the faults and pillar requirements.

8.3.2.2 **Selection of Crew**

As this was part of the greater trackless mining activities that were being undertaken for the bulk of the extraction of the 101 area the crew was part of the overall complement.

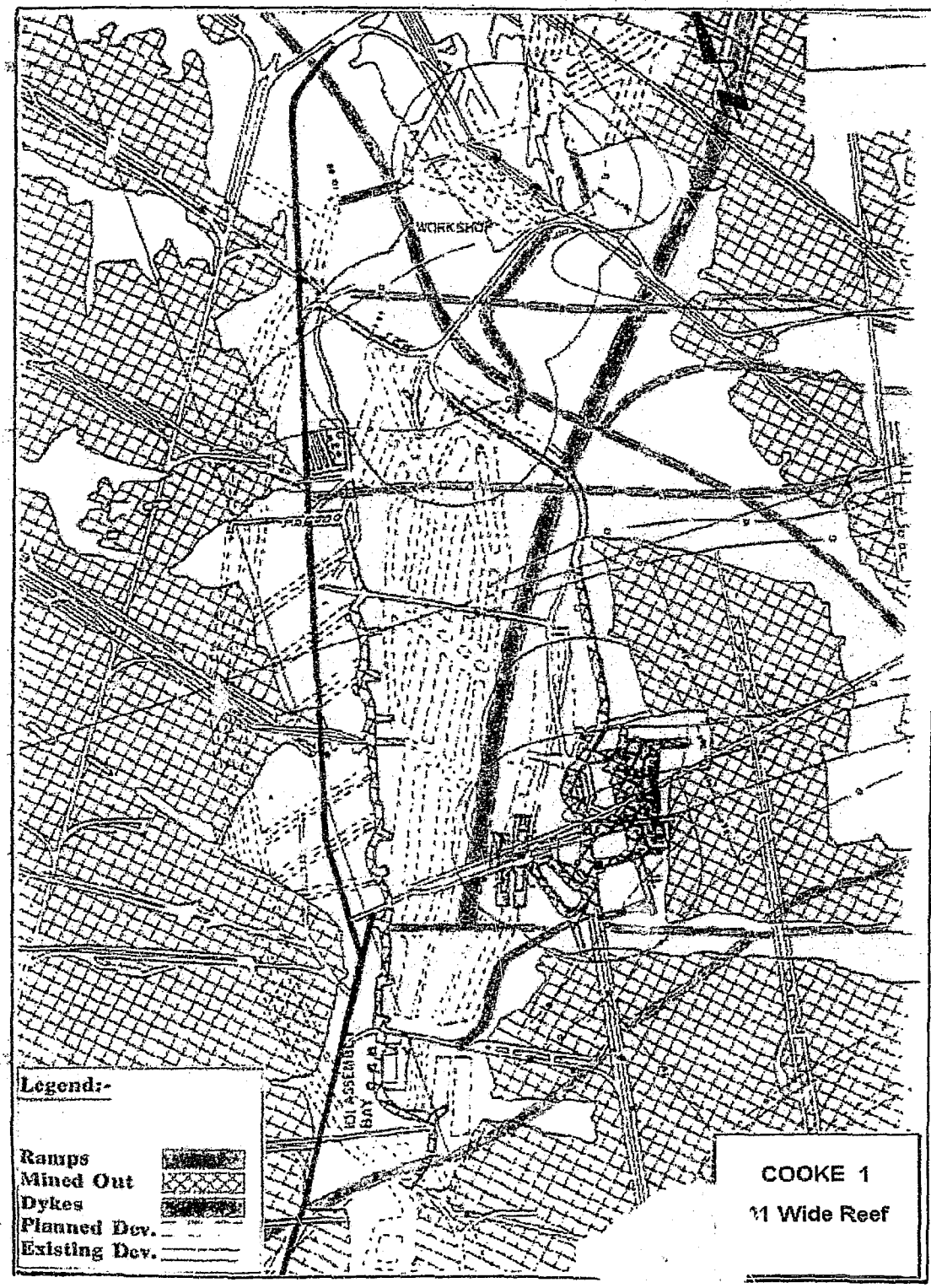
8.3.2.3 **Mining**

The mining was as per layout shown in figure 8.7 with access to the various elevations obtained from an 8° access ramp developed on reef.

8.3.3. **Wide Reef Area**

Trackless mining had proved to be cost effective at Randfontein Estates. Another major advantage was the flexibility of the method in negotiating faults, dykes and changes in the dip of the reef. A lot of footwall waste development would be excluded and the obvious choice at the time had to be trackless mining. The greatest portion of the block of ground would be mined using trackless mining equipment on a pillar and stall or a room and pillar method. As a result of the reef dip, the amount of faulting and previously mined areas, no neat symmetrical layout was possible.

Figure 8.7


WORKSHOP
Legend:-
Ramps
Mined Out
Dykes
Planned Dev.
Existing Dev.
COOKE 1
11 Wide Reef

8.3.3.1 **Preparation**

Trackless mining operations, not unlike any other mining operation, requires significant preparation work prior to full scale mining operations. One of the advantages of trackless mining is that mining can commence a lot sooner than with conventional mining layouts and considerably less development was required both on reef and off reef.

Critical to a trackless mining operation is the provision of workshop facilities. In this case a workshop was provided on 101 level Figure 8.8. This is described in greater detail under trackless equipment chapter. Access roadways to the mining areas were developed. In some cases roadways through the previously mined areas had to be established in order to gain access to an existing ore pass. A TM^3 grizzley and impact breaker were sited at each major tip.

Ventilation intakes and returns were established. This will be covered in detail under the chapter on ventilation. The workshops needed to be in fresh air at all times to not only facilitate repair work during non mining times, particularly in the event of a major breakdown but also to maintain a clean dust free environment.

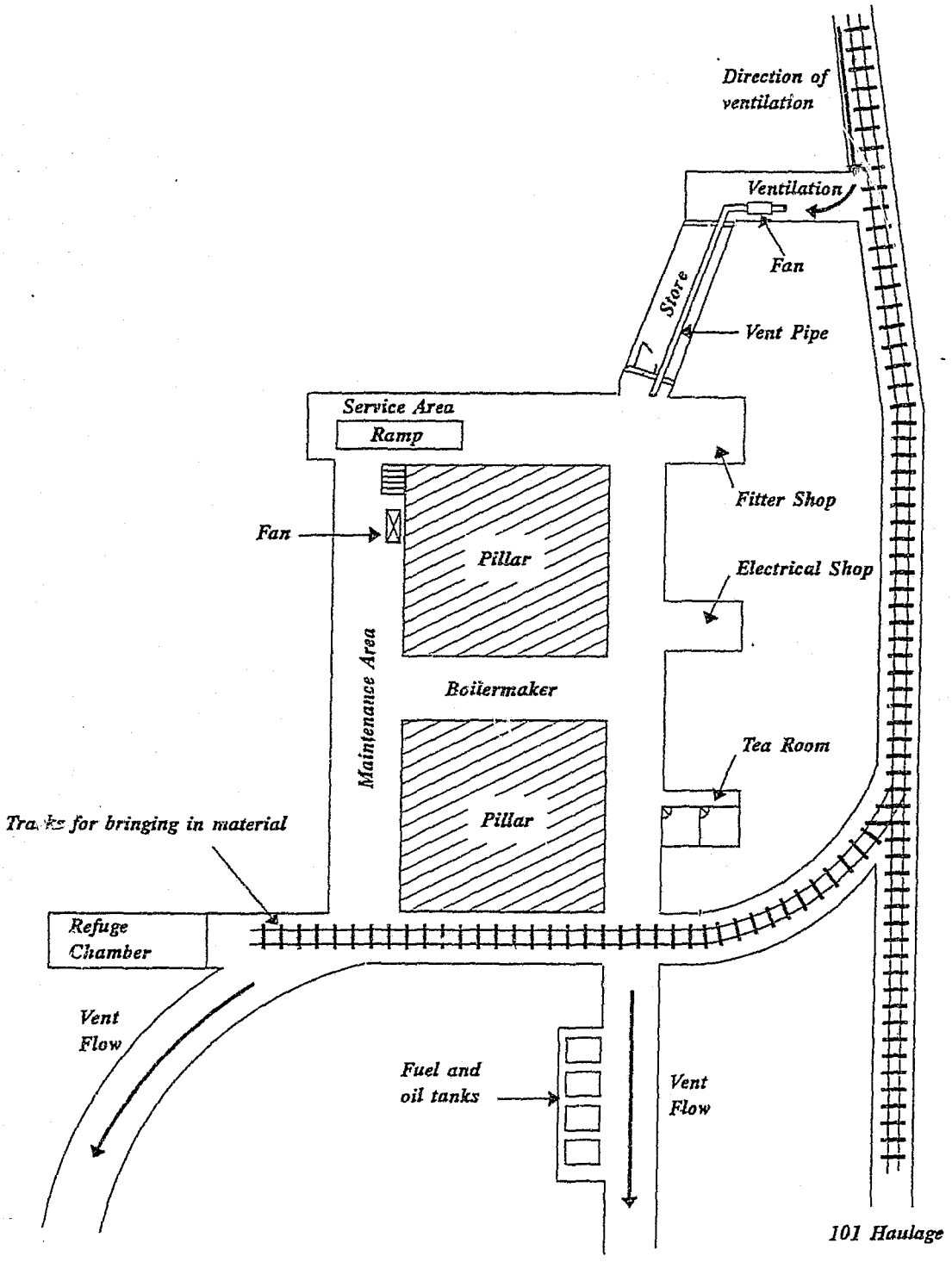
8.3.3.2 **Selection of Crew**

Trackless mining requires a different approach to normal mechanized stoping. The equipment is very expensive and the team that manages and operates the fleet requires very extensive knowledge of the type of equipment.

For the miners involved it required the making safe of headings that were 3,5 metres high, the marking off of these high excavations, the keeping up to date of ventilation and other service columns and the maintenance of roadways.

Figure 8.8

SCHEMATIC LAYOUT OF 101 WORKSHOP AREA



Equipment operators were selected and trained on the equipment and at the mines training centre. The full list of people required to do the work is shown in Table 8.

8.3.3.3 **Mining**

An example of the mining layouts is shown in figure 8.9. Roadways 7 metres wide were laid out with a 7 metre pillar. Holings were effected at 30 metres centres to facilitate tramming as well as the ventilation system.

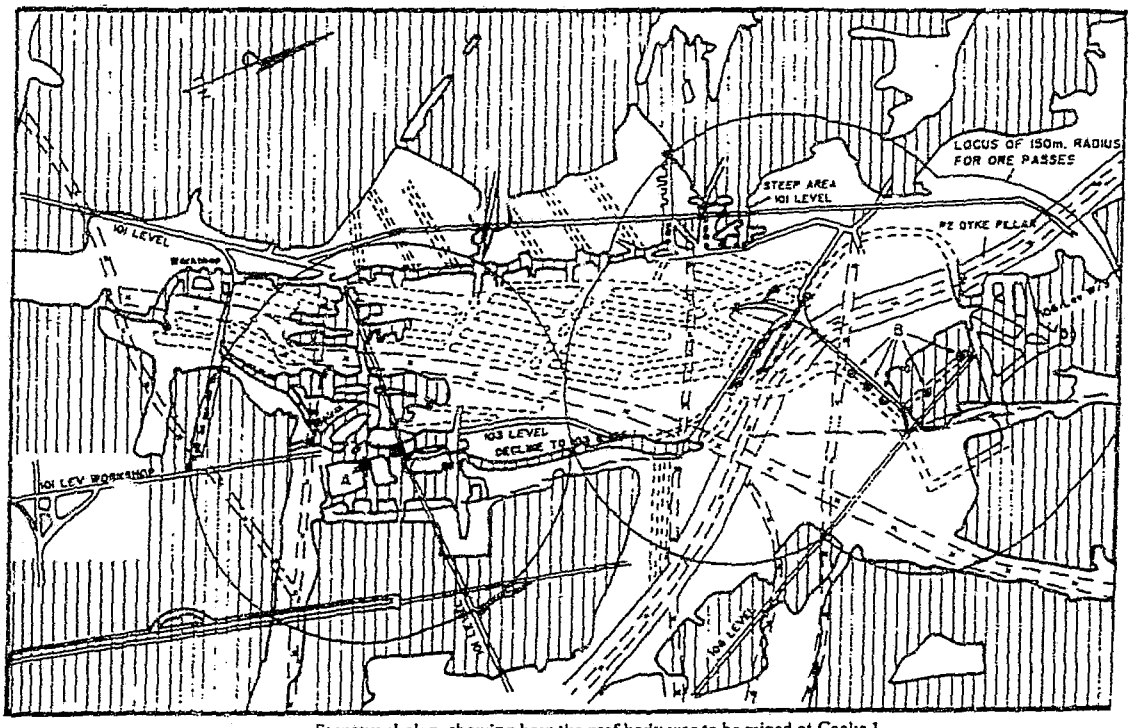
A three roadway system was used with one of the headings leading the other by ±30 metres. This roadway would intersect any faults or variations in the reef. The exact position of the reef was identified by drilling, the layouts altered to negotiate the fault before the other headings caught up.

This resulted in minimum lost time, and reduced the dilution from either having to lift footwall or drop hanging wall in the two following ends. The width of the rooms varied depending on the stress patterns as predicted by the rock Engineer. The widest rooms mined were 10 metres wide in the relatively low stressed ground to a maximum of 3 metres wide in the most highly stressed areas.

A schematic section as depicted in Figure 8.10 indicates the reef relationships and roadway layouts.

The pillar sizes ranged generally from 7 by 10 metres to 8 by 10 metres later as the stresses increased and further practical experience was gained.

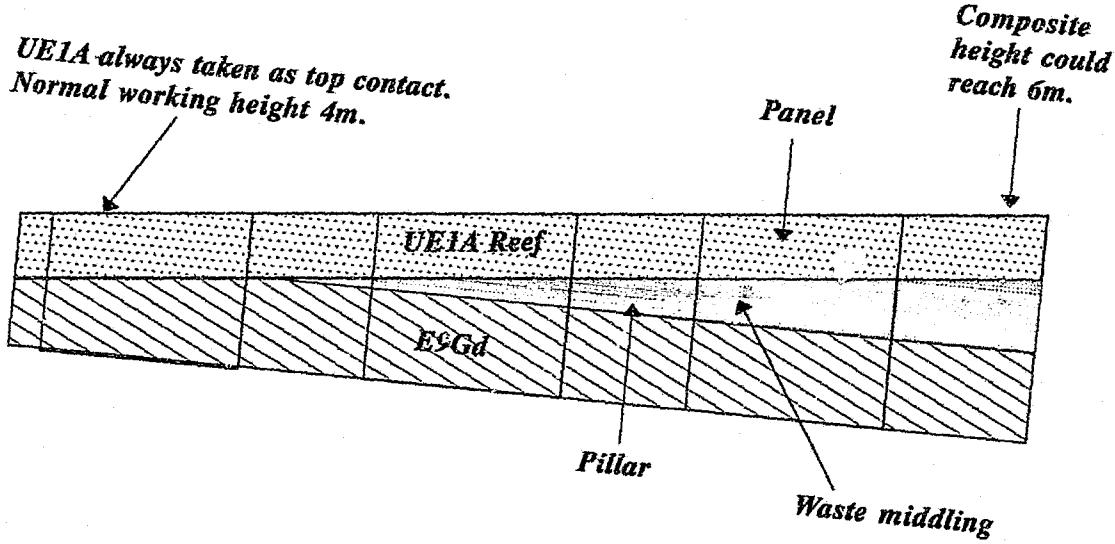
Figure 8.9



Structural plan, showing how the reef body was to be mined at Cooke 1

Mining Layout

Figure 8.10



Where E9Gd values where greater than the breakeven value they were mined in total. However, where these values did not warrant extraction, the minimum height panel as the UE1A reef was excavated.

(Paylimit at the time was 3,45 g/ton and the breakeven at the Trackless was calculated at 2,75 g/ton.)

Section showing UE1A and E9Gd Reefs with panels.

8.3.3.4 **Drilling**

An Electro Hydraulic drill rig (Fig 8.11) was used to drill the faces. The drill rig was capable of 1 to 2 headings per shift, depending on travelling time of the rigs between faces, and operated on a two shift basis. The planning was 3 rounds in two shifts which, except for when a few major breakdowns occurred was regularly achieved on a monthly basis. The hole length of 3,2 metres with 45mm diameter holes were used on a pattern as shown in figure 8.12.

Studies undertaken showed that it took from 3,5 minutes to 5,8 minutes to complete a hole including moving to the next hole. Bits had to be changed after every 5 holes to prevent over drilling and to maximize bit life. For a long period in the region of 80 meters per bit was all that could be achieved but with better control and good sharpening this increased to an average of 100 meters per bit. The 20 meters may not sound like much but a 20% decrease in bit costs is a substantial saving.

Studies found that to complete a 52 hole round, with a 75mm stab hole, took between 2,5 and 3 hours. However, the travelling to and from the workshop and to the next face consumed a large portion of the shift. The speed on the flat and the delays in negotiating bends results in an average speed of less than 100 meters per minute with the speed up or down the declines being in the order of 50 meters per minute.

Three electro hydraulic drill rigs, drilling a 3,2m hole were used and a typical unit is depicted in plate 8.9.

The major advantage of the drill rigs, particularly in the pillar areas, was that the drill operators work away from the rock face. This was found to be of particular advantage when working in fractured ground. It was generally found that after the face had been drilled, 2 or more LHD buckets of scaled rock had to be cleared away before charging up could commence.

Figure 8.11

High capacity hydraulic drill rig for medium sized tunnels and mine production

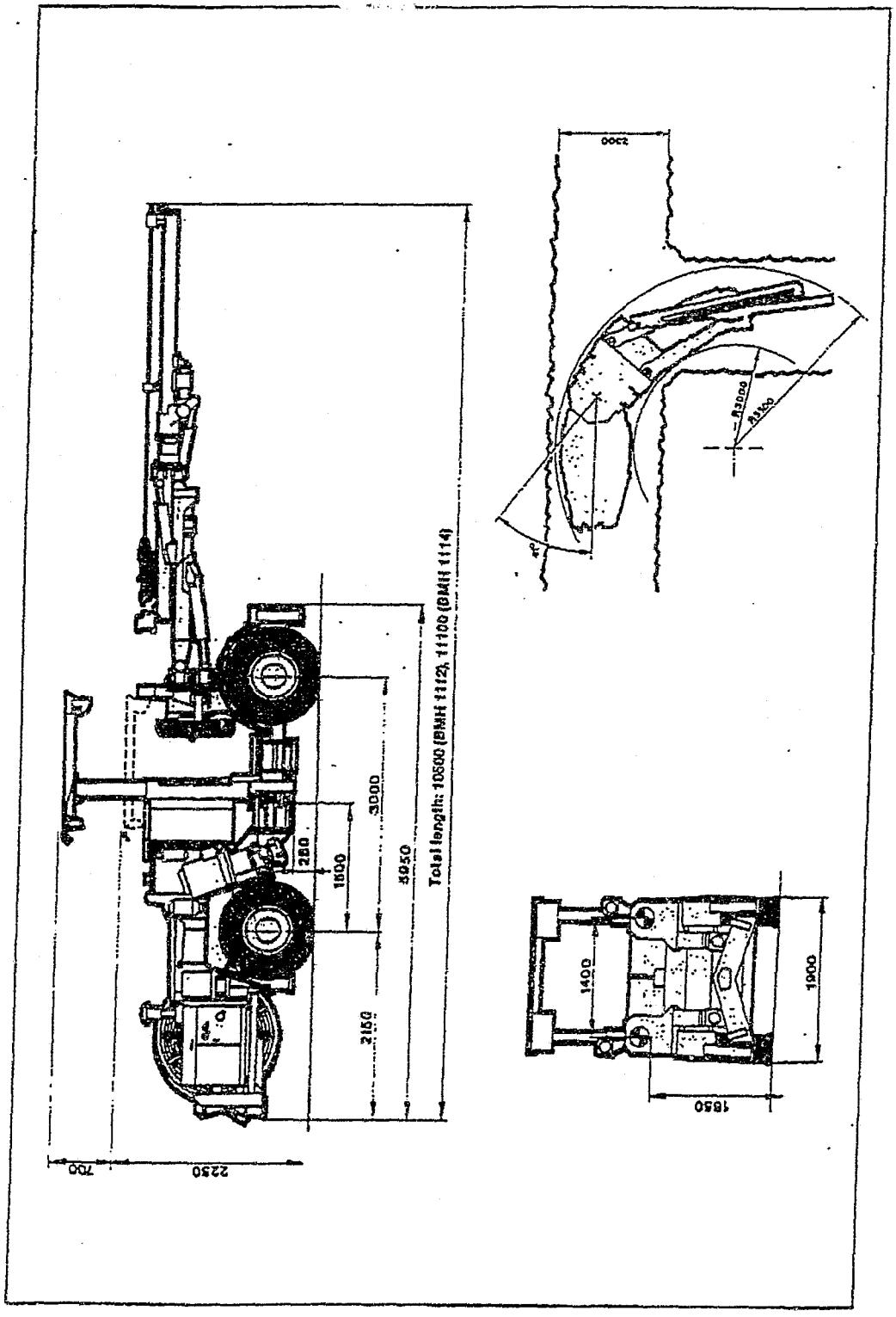
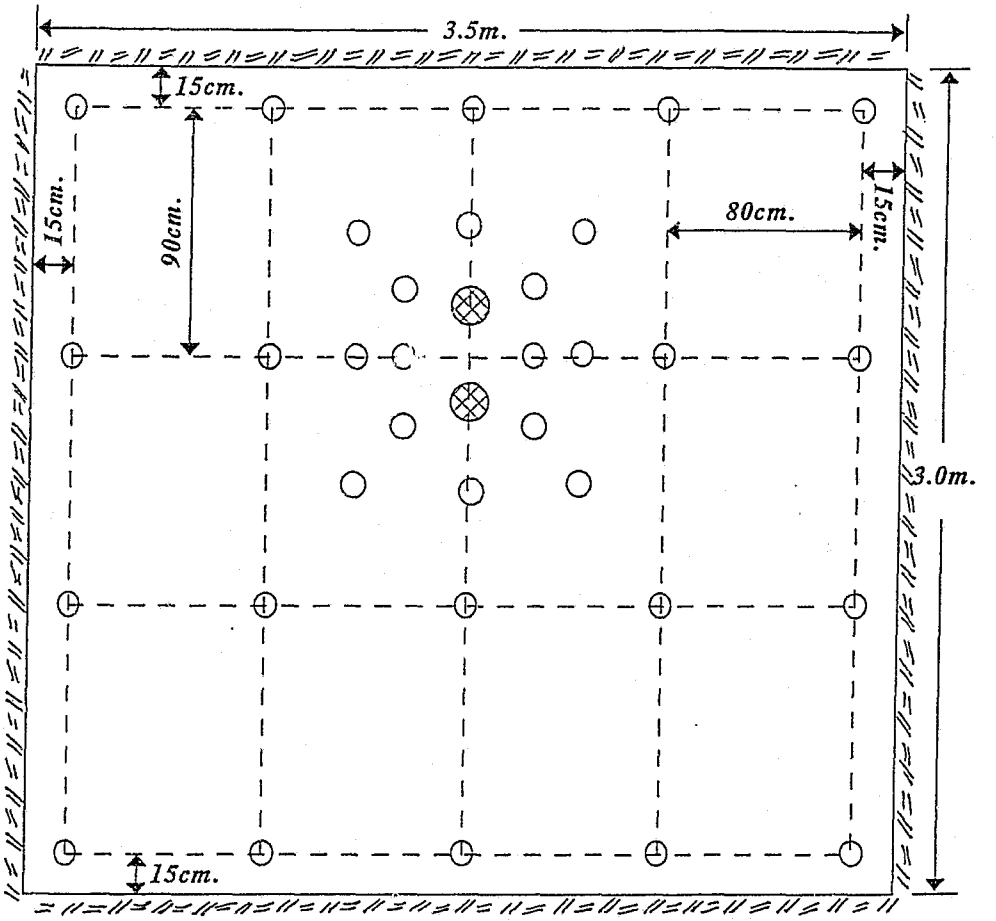


Figure 8.12

TM³ ADVANCED STRIKE DRIVE (A.S.D.)



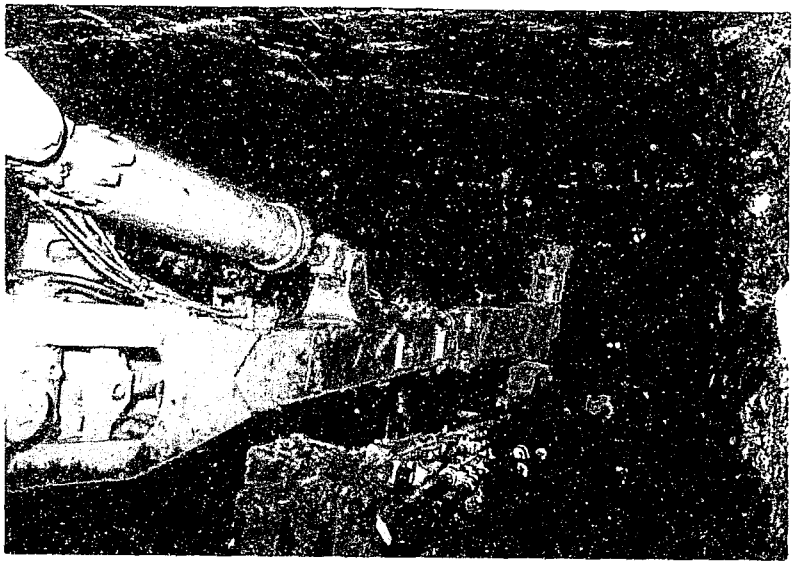
FACE HOLES = 35 *SHOT HOLES = 33*

The two stab holes ⊗ *are not charged.*

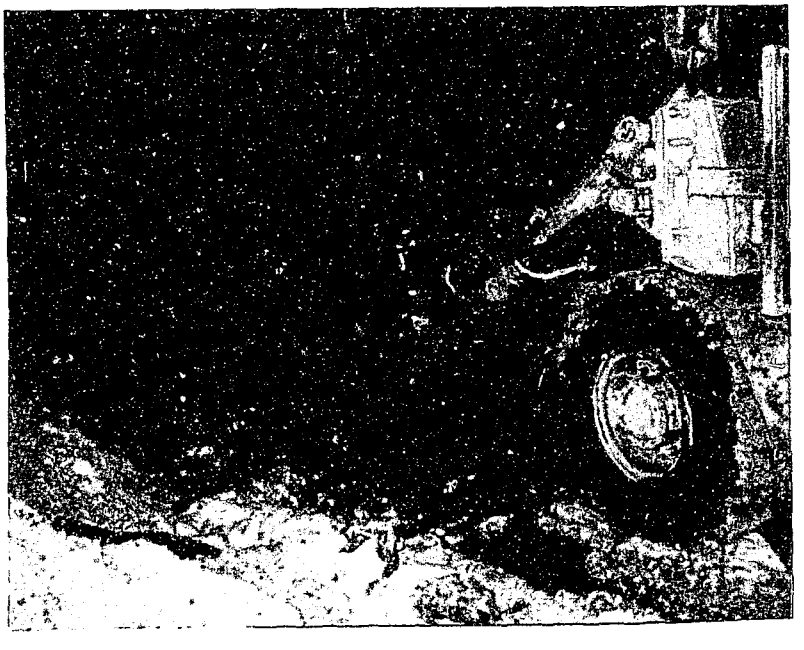
(Not to scale).

Drill Hole Pattern

Plate 8.9



Electro Hydraulic Drill Rig Drilling Face



Other advantages of electro hydraulic drill rigs are:-

1. Faster penetration rates.
2. Reduction in use of compressed air.
3. Longer rounds can be drilled.
4. More accurate drilling is possible.

When examining the causes of downtime and non drilling time of the rigs in an endeavour to improve productivity the following were the main factors affecting the output. They are not listed in any specific order:

1. Mechanical availability
2. Utilisatoin.
3. Operator ability
4. Mine policies – where machines parked at end of shift etc.
5. Travelling distances – where machine is refueled etc.
6. Maintenance schedules, hourly based or calendar based.
7. Conditions and gradients of roadways.
8. Water handling system.
9. Condition of bits.
10. Fracturing of the face.
11. Quality of face preparation.
12. Communication with workshops in event of breakdown.
13. Artisan transport facilities.
14. Unavailability of prepared and marked headings.

8.3.3.5 **Blasting**

Pneumatically loaded Ammonium Nitrate blasting agent was used in all except the lifter holes, which were charged with Dynagel. Fuses were 3,4 metres long and a 32m x 200mm stick of Dynagel was used as the primer. All holes were bottom primed. Medium speed igniter cord was used with a 1hour stay alight. Blasting was done on a time basis and only at the end of the day shift.

8.3.3.6 **Cleaning**

Cleaning was done on two shifts using 3,5m^3 Lads with or without 20-ton dump trucks. The bulk of the cleaning took place on the night shift as there was less congestion and all the machines were available while on Day shift machines were serviced and maintenance work undertaken.

The main factors effecting the output of the LHD's in this project were:-

1. Mechanical availability.
2. Effective utilization (not an expensive wheelbarrow).
3. Operator ability
4. Bucket configuration.
5. Machine configuration
6. Tramming distance to tip or truck.
7. Quality of fragmentation.
8. Gradient and underfoot conditions at loading point.
9. Amount of broken ore.
10. Condition of roadways and gradients.
11. Single pass loading.
12. Breakout force of the unit.
13. Number of trucks allocated to a loader.

It was found, not unlike surface mining operations, that the most effective work was obtained when two or more trucks were allocated to an LHD. The result was that the LHD cleared an end very quickly and the units then moved on to the next lot of ground with the LHD doing the final cleanup.

Initially two 20 ton dump trucks (Plates 8.10 and 8.11) and one 15 ton truck was allocated to the project. Further into the project, particularly as the distances increased it was found the 15 ton was not a suitable choice and after a major structural failure was replaced with a third 20 ton truck. As the project wound down the truck complement was again reduced to two.

As the tonnage tended to fluctuate quite widely a study was done to determine the factors influencing the output in this particular project. The factors were:-

1. Mechanical availability
2. Effective utilization.
3. Operators ability.
4. Layouts (some angles were too sharp).
5. Tramming distances and gradients to the tips.
6. Sufficient and efficient tipping capacity.
7. The efficiency of the LHD operator.
8. Vehicular congestion.
9. Condition of roadways.
10. LHD tipping position / site.

8.3.4 **Flat "Narrow" Reef**

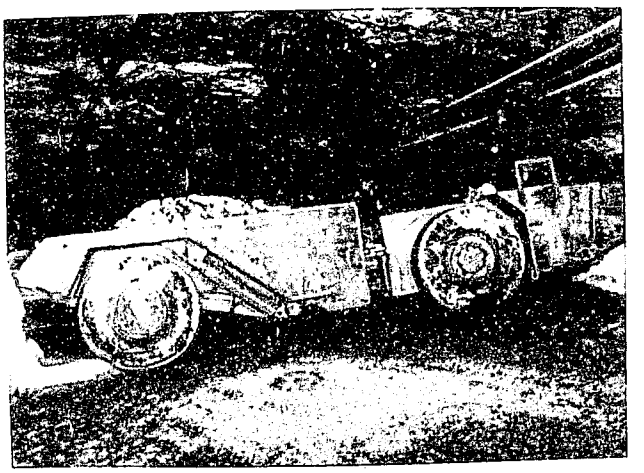
Where the reef width and grade was such that it could still be profitably mined at 2,8 metres the same basic method as applied to the wide reef areas as described above was applied. In a number of areas the double cutting technique as described in Paragraph 15.7 was very successfully applied. Plate 8.12.

Plate 8.10



LHD Loading Dump Truck

Plate 8.11



Fully Loaded Dump Truck

Plate 8.12



A doubie cut mining operation showing the waste cut
And the reef cut already drilled

TABLE 8.0

TITLE	NO. OF PEOPLE
Section Manager	0,25
Mine Overseer	0,5
Shiftboss Day Shift	1
Shiftboss Night Shift	1
Miners	2
N/S Cleaner	1
Engineer	0,5
Engineering Overseer	1
Engineering Foreman	3
Artisans	10
General Miner	1
Mechanical Equipment Overseer	0,5
Rock Engineering	0,5
Survey Staff	2
Ventilation	0,5
Geology	0,5
L.H.D. Operators	6
Dump Truck drivers	6
Drill Rig operators	4
Utility Vehicle Driver	8
Team Leaders	4
Jack Hammer Handle	8
Jack Hammer Spanner	8
Miners Assistants	8
Team Bar	2
PTV	6
P.T.V. Gang Supervisor	1
Engineering Cat B	6

CHAPTER 9

Ventilation

9.1 Introduction

The use of heavy diesel powered equipment underground creates three immediate ventilation problems. Firstly, the diesel engines generate significant amounts of heat which is passed into the air system. Secondly, is the emission of toxic gasses particularly from these engines and thirdly the machines have the potential to generate a lot of dust along roadways and when loading or tipping rock.

In the case of Cooke 1 Shaft which was still relatively shallow the ability to move large volumes of air was somewhat restricted by shaft and airway sizes, which were designed for conventional mining methods. This resulted in air velocities far exceeding normal design parameters. The problem would become more critical in deeper mines as the natural environment gets hotter due to the increasing geothermal gradient as well as the autocompression of the intake air.

Randfontein prided itself with good working conditions and these were maintained throughout.

One of the problems that was to face the team in both the operations under review was to be ventilation control particularly with the extent of mined out areas surrounding the operations and frequent holings into old raises, boxholes and mined out areas. But then on the other hand it was to be of assistance where the area could be exhausted into the old areas which eventually made their way to the upcast shafts.

9.2 95 Haulage Pillar

This area being almost totally surrounded by mined out area initially appeared to be a ventilation officer's nightmare, however, once the situation was investigated it was found that air from the 101 level stoping was upcasting through the old areas and although it contained some pollutants in the form of NO, CO and CO_2 and some fumes from trackless equipment the overall quality of the air was good. In addition air could have been drawn directly from the downcast shaft but the air was needed in other locations and it was decided to use the upcasting air.

Ventilation of the development was done using 760mm vent columns and 30kW force fans. Once the haulage had been cleared out a good quantity of air flowed through it and the development therefore only required very short columns which provided very comfortable working conditions and diesel fume emission never became a problem.

Trackless equipment used in the area only amounted to 102kW, which did not pose any environmental difficulties.

9.3 **101 Area**

The ventilation problems were to prove to be far more intense in this area with the ventilation doing unpredictable things when holings were made into some or other old workings. Ore passes connected to different levels, stopes and cross cuts were found to link into the existing system on either intake or exhaust sides and sometimes when you thought the air was going to do one thing it did the opposite or nothing at all. It is a tribute to the Cooke 1 ventilation department that they kept the conditions generally to a very high standard despite the vagrancy's of the system.

The mining and ventilation people communicated on a daily basis and it is this close co-operation, sometimes at night when the night shift crew found the system doing something different, that ensured that no serious problems were encountered.

The area is served by a trackless vehicle fleet of some 1480kW rated diesel power. This amount of power simply had to increase the generation of heat and exhaust gases into the air. Some authors (Fourie, 1988) indicated that the total heat generated by a trackless fleet would be similar to that as generated by equipment in conventional mining for the same tonnage output. This is not as straight forward as that, as the variables of tramming distances could result in more or less trucks being applied as well as the gradients will vary the heat input per ton/kilometre. In fact Middleton 1989 stated that localized buildup of heat could be significantly worse with large diesel powered units referring particularly to development but then TM[3] mining at times is as much development, all be it on reef, as it is stoping. Patterson, 1989 indicated that 0,06 to 0,08 m^3/s of air per kilowatt of rated engine capacity needs be supplied at the point of operation.

A supply of 0,08 m^3/s would suffice if international standards were applied to Nitrous Oxides.

At the time JCI planning parameters called for an air dilution factor of 0,12m^3/s of fresh air per rated kilowatt of diesel power for the dilution of Nitrous Oxides and the other noxious emissions associated with diesel combustion units. This planning rate is high as it is known that 0,1m^3/sec of fresh air keeps concentrations within acceptable limits and in fact with the start of the project 0,06m^3/s was supplied per rated kilowatt but at times gas levels neared or exceeded the critical levels and the volumes were increased to ensure no undesirable exposures.

The area was ventilated with 220m^3/s of air, of which 75% was directly from the downcast shaft via 90,101 and 106 levels. The balance of the air was re-used air which was received via the old workings from other areas. A fortunate point for the people at Randfontein Estates Cooke 1 Shaft was the relatively shallow depth and the low heat pickup thus enabling the air to be re-used.

The overall air factor for the project calculates as 0,15m^3/s/kW or 10m^3/s/kiloton/month. As a comparison where air is related in conventional mining to monthly tonnage produced the standard factor of 3,3m^3/s/kiloton/month. The high air factor supplied was partly due to the vagrancy's that result when holings into old workings occurred and in part may be described as some factor of safety.

As the areas were served by multiple roadways varying generally between 16m^2 and 20m^2 in area, the main tramming routes were defined and the air distributed such that velocities in the region of 2m/s were maintained in these access ways. This was primarily done to accommodate tramming speeds of the vehicles in order to prevent the vehicles travelling in a plug of fumes which would not only make the temperatures very uncomfortable but would likely raise gas concentrations to unacceptable levels.

Primary air distribution and control within the project was done mainly by the installation of ventilation seals both permanent and temporary. Permanent seals were predominantly constructed from reinforced concrete poured into shuttering. Temporary seals consisted mainly of gumpoles, gumplanks and polypropylene curtaining. Permanent seals have been a problem in all trackless layouts and a cost effective and quickly installed seal is still under investigation.

Concrete blocks either crush or are damaged by the blast, bagged material does not stay in place, Gunnited brattices peel off or leak very rapidly and the concrete cracks when closure occurs.

Obviously air controls are also required where machinery is required to move through. In cases such as these doorframes were constructed from which split conveyor belting was suspended in an overlapping curtain which formed a very effective flexible air control system.

Because of the vast amount of controls, both permanent and temporary seals required in a project like this a construction crew comprising a supervisor and eight assistants were continuously employed in maintaining and constructing ventilation controls.

Where old mined out areas were traversed, polypropylene vent curtaining was used to course the air through the area. These areas could cause a lot of lost ventilation if neglected.

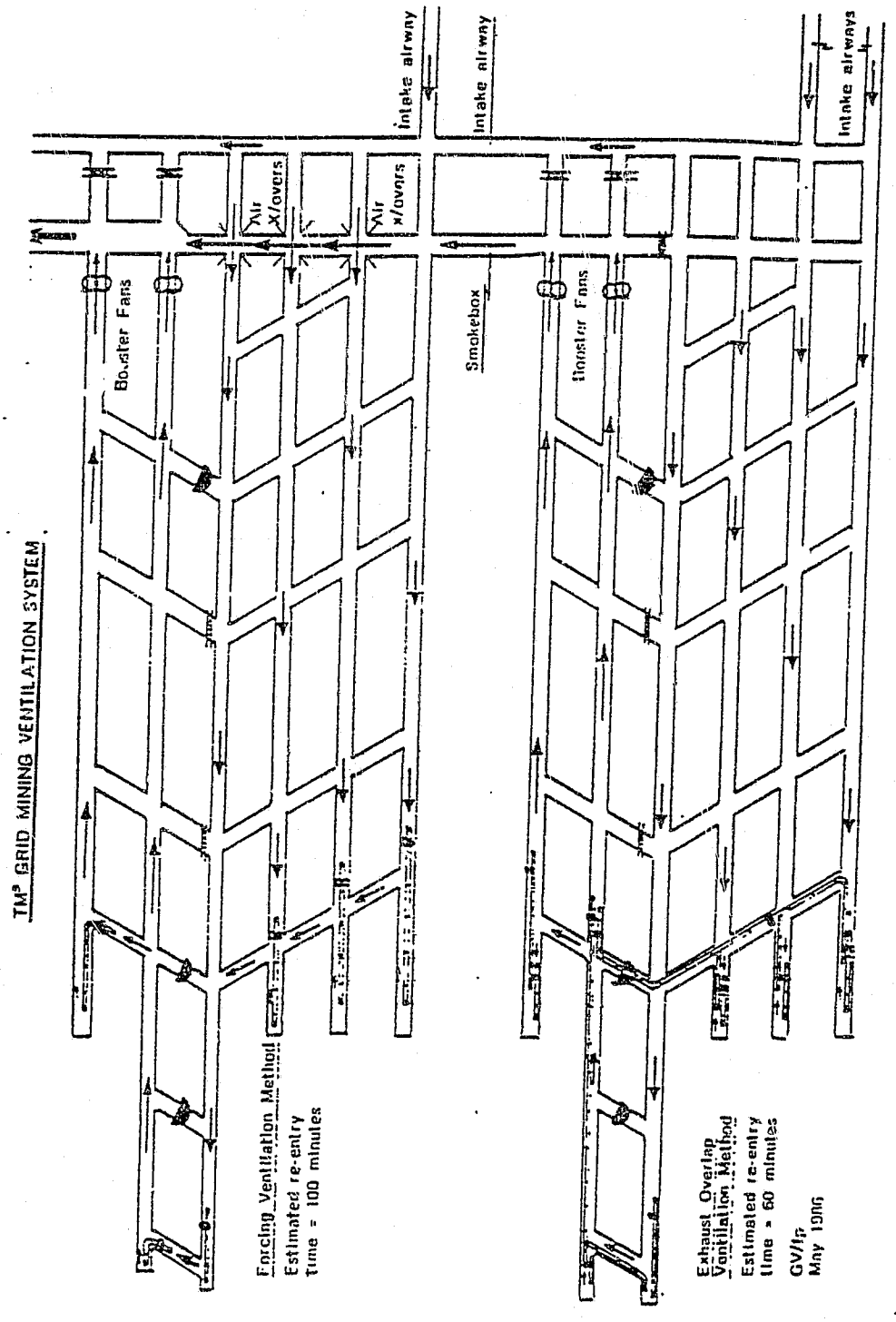
The development of the roadways and the stoping drifts were ventilated by means of 1 016mm diameter galvanised ducting and using 45kW axial flow force fans supplying between 14 and 16m^3 of air to the working face. Because of the limiting air quantity vehicular capacity in these headings are limited to 150kW which is also the rating for the LHDs currently used in the area.

Dump trucks were loaded at the nearest point of through ventilation. Utility vehicles were only allowed to operate in the headings when there was no LHD in the area being supplied by the same ventilation column.

Holings into the old workings were effected from time to time and used as return airways either using the natural draw of the main fans or where necessary one or more 760mm diameter fans were utilized.

With the system operating as described wet bulb temperatures of 25,5°C were maintained.

Figure 9.1



Trackless Mining Ventilation Layout

CHAPTER 10

Cycle of Operations

10.1 Introduction

The mine, in general, operated on a two shift basis with the drilling and blasting taking place on the day shift. All the activities that go with stoping are also done on the Day shift. The cleaning or removal of ore took place on the night shift. The stoping operations operated on a two day or three day cycle depending on face availability and conditions in the areas. The trackless mining operations operated on a similar pattern with the blasting taking place at the end of day shift with drilling and cleaning taking place on both shifts.

10.2 95 Haulage Pillar

This area was planned from the outset to operate on a single shift basis. The reasons for this were:

10.2.1 A high standard of supervision was required in order to ensure safe working conditions.

10.2.2 It showed the best productivity would be achieved by this method.

10.2.3 The rockburst commission recommended that remnants be extracted on day shift only.

The faces were cleaned at the start of the day shift and prepared for drilling. Machine crews entered in the mornings, re-dressed the faces and commenced drilling.

10.3 101 Area

10.3.1 Trackless Mining and Wide Reef

Areas were mined as per the other TM^3 operations on the Mine with drilling and loading operations taking place on two shifts, equipping and maintenance taking place on the day shift and blasting once a day at the end of day shift with a four hour re-entry period. Faces, in general were worked on a two day cycle and no serious stress problems were encountered by applying this method.

10.3.2 **Drilling**

The use of electro hydraulic drill rigs enables fast accurate drilling to be done as well as drilling a larger diameter hole. Larger holes result in fewer holes required per face drilled.

The roadways and panels varied in height dependent on reef width. Panels were between 2.5 and 4 metres high with panels where drill rigs used having a minimum of 3 meters. Roadways were between 4,5 and 7 meters wide.

10.3.2 **Blasting**

For blasting Anfex and Dynagel were used in conjunction with standard fuse and slow igniter cord. The blast was generally contained on the face with good fragmentation. Out of sequence shots or misfires caused large rocks which in general were not a problem, and were broken on the tips by hydraulic impact breakers.

10.3.4 **Loading / Cleaning**

Loading operations take place on both the day and night shifts using 3,5m^3 LHD units. The ore was either trammed directly to a tipping point provided it was within a 150m travelling distance from the loading face, or loaded into dump trucks of 20 ton capacity. The trucks then transport the ore to the main tips, which will generally be situated within a 500 metre travelling distance.

10.3.5 **The Narrow Areas**

Were also handled as per the rest of the mine with drilling and blasting with its associated work taking place on the day shift and stope cleaning being done on the night shift. No serious problems were encountered during the period.

The narrow steep areas were essentially worked on a single shift basis with faces prepared, drilled and blasted on the day shift and the LHD loading ore in the advance strike drives on the night shift only. No other persons worked in the area or panels on the night shift.

10.3.6 Drilling

In all the narrow areas at Cooke 1 conventional face drilling using pneumatic, hand held jackhammers with suitable airlegs were generally used. Initially integral steel was supplied but later in the project detachable bits were used. Plate 10.1. This was partially a cost saving exercise and possibly to a greater degree one of ergonomics. The crews are reluctant to carry large numbers of drill steels. Plate 10.2 while with detachable bits they are only required to carry two drill steels and two drill bits. Plate 10.3.

10.3.7 Blasting

All blasting in the narrow stopes used Anfex and 8D detonators, stope fuse and stope cord. The whole operation was changed, timed and connected to 60 minute stay-a-lites.

10.3.8 Cleaning

Cleaning using 50kW and 75kW winches was a long established process at Randfontein Estates. The broken ore was scraped from the face into an advanced strike gully, then scraped back to a tip, or loading point for LHD's. The bulk of the cleaning took place on the night shift.

Plate 10.1



Detachable Bits

Plate 10.2



Operator carrying Drill Steel

Plate 10.3



Operator with 2 Bits



Comparison of what operators are required to carry
In the two cases referred to

CHAPTER 11

Support

11.1 Introduction

The primary function of support in underground excavations is to control the movement of the rock mass immediately surrounding these excavations. In shallow excavations the zone of rock is subjected to tensile stresses while in the case of the pillar extraction the stress patterns could be very different particularly with reference to the pillars in the bord and pillar or room and pillar layouts.

In designing the support systems to reduce, and as far as possible eliminate, the incidence of rockfall as well as the possibility of pillar bursts, a number of factors needed to be taken into consideration, namely, the degree of fracturing already present, the geological disturbances, the integrity of the hanging wall strata, the size and shape of the excavations, the depth below surface, the purpose for which the specific excavations would be required and the probability of rockbursts.

This chapter examines the design and implementation of mining support systems applicable to the area under study.

11.2 Factors in the Design of Stope Support Systems

South African Gold Mining is carried out in hard brittle quarzitic rock with depths varying from outcrop stopes to great depth. Much of the rock mass behaves elastically but the mining results in destabilizing the zone thereby resulting in fracturing of the rock surrounding the areas.

The design of the necessary support system therefore depends on the knowledge of three fundamental aspects:

11.2.1 The nature and extent of the rock to be supported.
11.2.2 The deformation to which the support system would be subjected.
11.2.3 The characteristics of the possible support system to be applied.

> "Not all of these aspects can be assessed accurately enough to allow a design based on strict engineering calculations to be applied to each and every situation. Heavy reliance must be placed on a qualitative understanding of rock and support

behaviour in the light of past experience" extract from "methods of ameliorating the hazards of Rockfall and Rockbursts 1988."

11.2.4 The effect of blast induced fracturing

The stope support requirements were initially grouped into 3 classifications.

1. Temporary support – applicable to narrow, narrow steep and wide ore bodies as well as for localized conditions.
2. Narrow reef permanent support.
3. Wide reef permanent support.

The latter two were separated as by the very nature of the excavations and their obviously widely differing needs and characteristics.

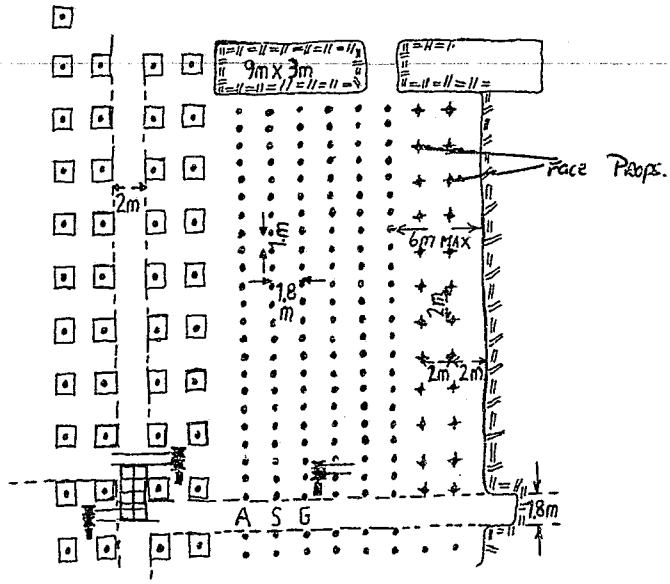
11.3 Temporary Support

11.3.1 Narrow and Narrow Steep

The mine made extensive use of mechanical stope support. Figure 11.1 indicates the normal mine standard for narrow stoping. This standard was reviewed and it was felt that although the requirements were somewhat different from normal narrow stoping, the standards already required such a high standard of support that initially the mine standard would continue to be applied but would be inspected regularly by the Rock Engineering department and should any undesirable tendencies start to manifest themselves, the standard would then be revised.

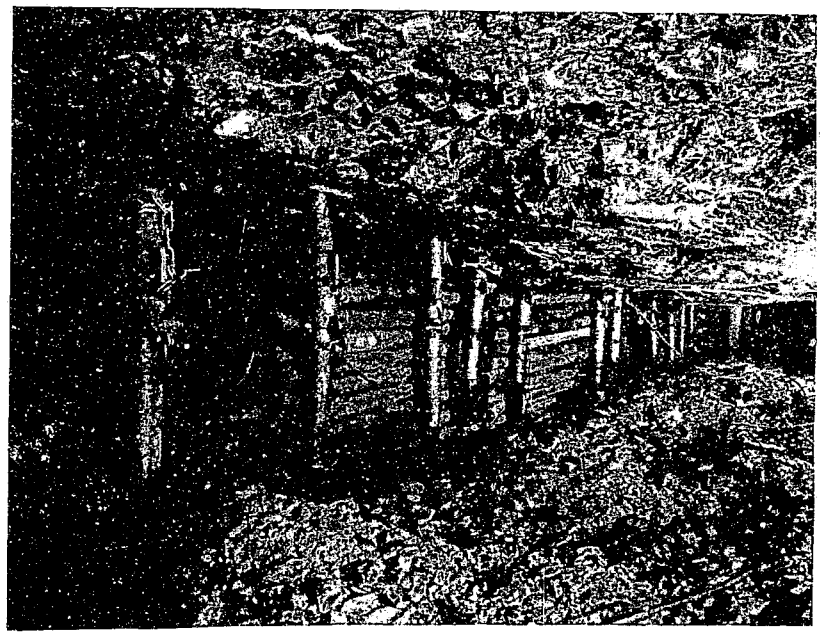
No problems were encountered. However, stricter control was exercised in the removal of these mechanical props before the blast. Plate 11.0, indicates the Hydraulic props with the blasting barricade suspended against them. When the permanent timber support was installed these were moved forward and kept within 6 metres of the face.

Figure 11.1



Mine Std Narrow Support

Plate 11.0



Hydraulic Props on Face

11.3.2 Wide

The mines standard is that permanent support is carried not more than 1,2 meters from the face before blasting. Figure 11.2.

Furthermore, that support is installed before drilling of the face commences. Prior to and during the installation of the permanent support, extensive use is made of mechanical support. Should any area be suspect temp support is installed until the area can be made safe. That obviously excludes mechanized cleaning.

Temporary support comprised of extendable mechanical jacks and in general between 3 and 6 were used at a time. Any areas that became suspect were barricaded off until inspected and the necessary action taken by the miner in charge.

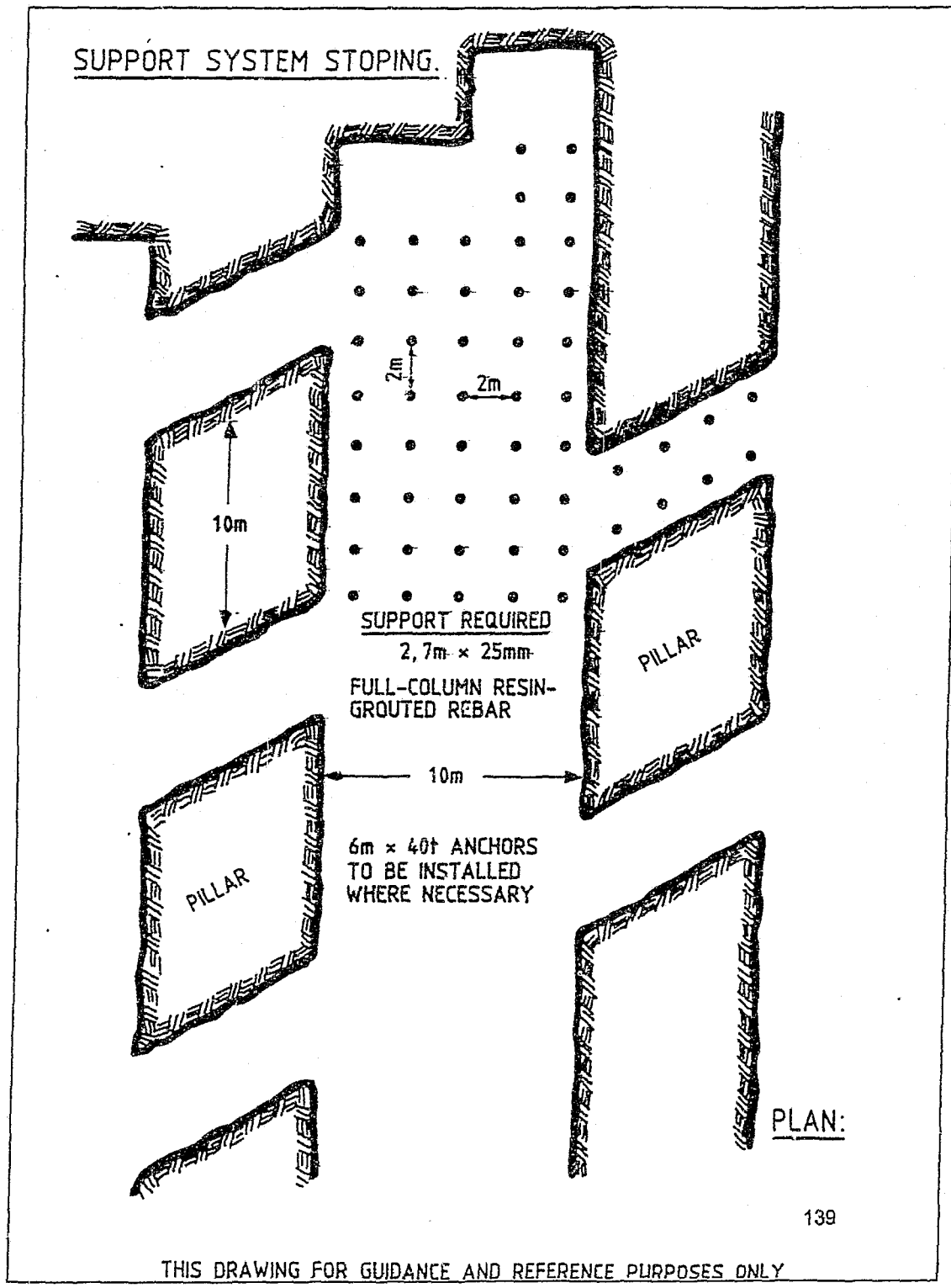
11.4 Permanent Support – Wide Areas

11.4.1 Primary Support

The primary support of the area was, by the nature of the mining layout, pillars. The criteria used during the planning of the pillar support are shown below.

Elastic Constants	Rock Mass Properties		
	Young's Modulus		70 Gpa
	Poisson's Ratio		0,2
Seam Properties	Reef Horizon		
	Young's Modulus		64 000 MPA
	Shear Modulus		29 000 MPA
Initial Stresses	Vertical at – 1000 m	Z Axis	27,0 MPA
	Horizontal	X Axis	13,5 MPA
	Horizontal	Y Axis	13,5 MPA

By analyzing other pillars on Randfontein Estates the Rock Engineering Department reached the conclusion that in order to mine the area safely and without pillar failure a maximum vertical pillar stress of 120Mpa would be used and any areas indicating stresses in excess of 120Mpa would be regarded as difficult to mine and special presentations would be applied.


SUPPORT SYSTEM STOPING.
2m
2m
10m
SUPPORT REQUIRED
2,7m × 25mm
FULL-COLUMN RESIN-
GROUTED REBAR
PILLAR
10m
PILLAR
6m × 40t ANCHORS
TO BE INSTALLED
WHERE NECESSARY
PLAN:
139
THIS DRAWING FOR GUIDANCE AND REFERENCE PURPOSES ONLY

11.4.2 **Secondary Permanent Support**

All TM[3] roadways and panels operate using 2,7m long 25mm diameter re-bar with 200mm x 200mm square washers, fully grouted using cementitious capsules were pneumatically loaded into the holes with the aid of a loading gun.

Re-bar was installed on a 2m x 2m square support pattern, as shown in Figure 11.3, giving 4 square meters per bolt. The sidewalls were supported on a similar pattern.

In areas where ground conditions were poor or suspect, the hanging wall support was altered to a 1,5m x 1,5m diamond pattern more than doubling the support rate to 1 bolt for every 1,8 square metres.

All faults and dykes have, in addition to the normal support pattern, extra grouted rebars installed either side of the fault or dyke, 1 metre away from the discontinuity and not more than 1,5 metres apart.

11.4.3 **Traversing Old Areas**

The support in these areas varied from sticks to grout packs as shown on plates 11.1 and 11.2. As can be seen from these plates the support had taken little pressure since it was mined up to 15 years earlier. The new support on immediate ledges comprised stick packs spaced at 2 metre centres along the shoulders. A stick pack is a pencil stick or profiled stick surrounded by a wedged skeleton pack. Figure 11.4. Other sticks or support which came out during blasting or were damaged or suspect were replaced with pencil sticks on a 2m x 2m pattern.

RE-BAR INSTALLATION. 2,7m × 2,5mm Ø

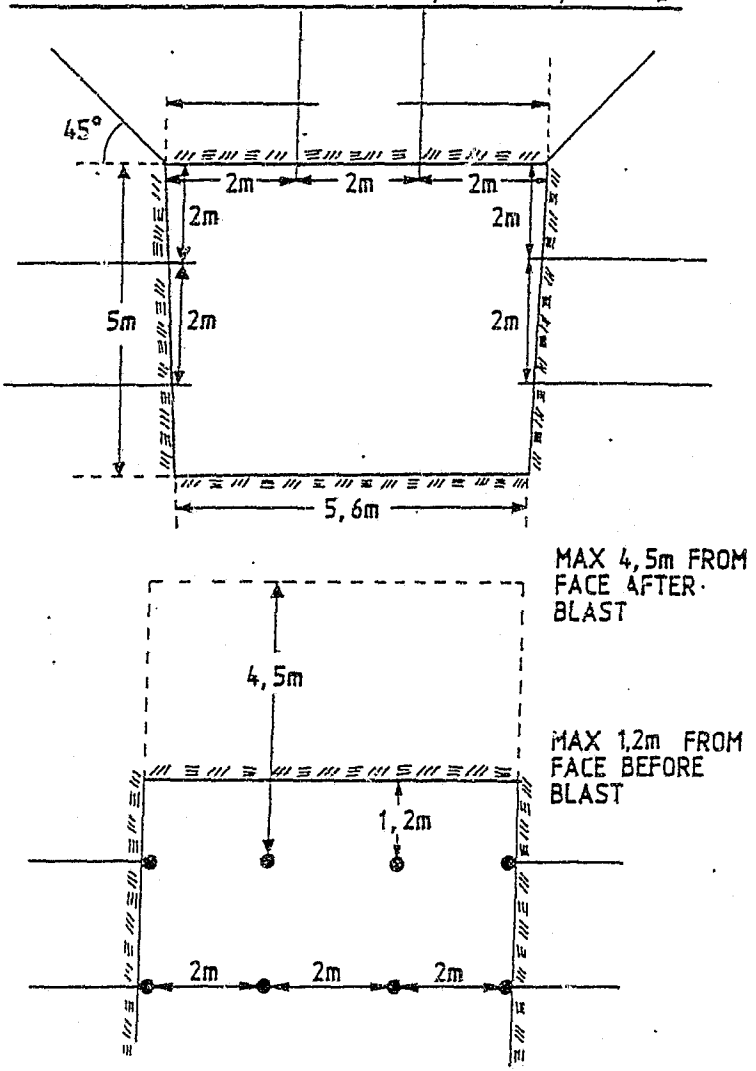


Plate 11.1



Traversing old areas

Plate 11.2

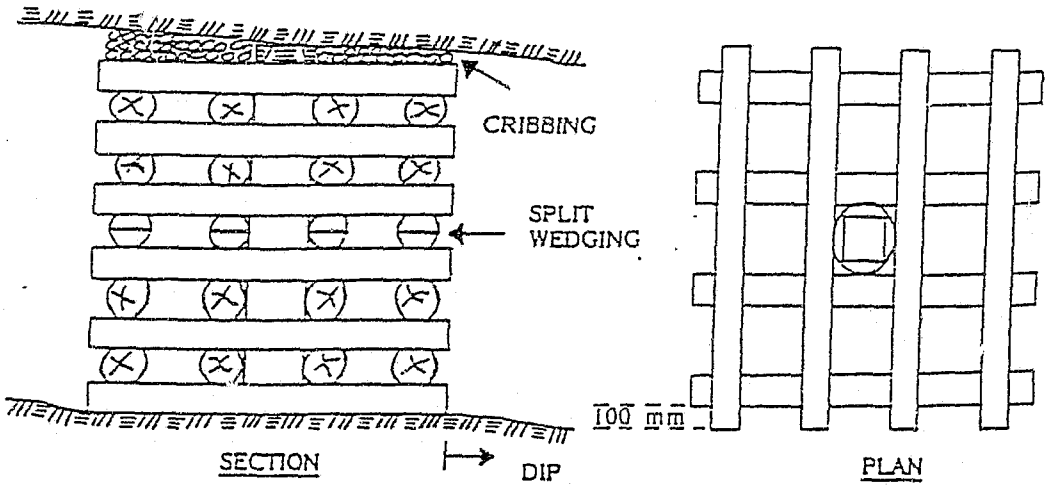


Traversing old areas

Figure 11.4

8.5 SUPPORT

8.5.6 16 POINTER E.P.S. PACKS



1. INSTALLATION

1.1 PACK POSITIONS ARE TO BE MARKED OFF BEFORE INSTALLATION BEGINS.

1.2 FOOTWALL IS TO BE PROPERLY CLEANED AND BARRED SOLID.

1.3 BOTTOM LAYER OF 1,1 M SLABS ARE TO BE INSTALLED ON DIP.

1.4 THE INNER 1,1 M SLABS ARE TO BE INSTALLED AGAINST THE YIELDING PROP.

1.5 THE OUTER 1,1 M SLABS ARE TO BE INSTALLED 100 mm FROM THE END OF THE UNITS.

1.6 CRIBBING MUST BE TIGHTLY INSTALLED BETWEEN THE TOP LAYER OF 1,1 M SLABS AND THE HANGING WALL.

1.7 SPLIT WEDGING MUST BE DONE ON THE FACE SIDE OF THE PACK.

11.5 **Permanent Support – Narrow Areas**

In the narrow steep areas gully support was by 1,6m x 20mm diameter grouted re-bar. The stoping areas were supported on pencil sticks placed on a 2m x 2m pattern and placed according to Randfontein standards not more than 6m from the face. When traversing faults or dykes additional support is placed either side of the discontinuity.

11.6 **Wire Mesh and Lacing**

Areas identified as requiring wire meshing and lacing such as workshops, tipping points or refuge bays were systematically supported as they were excavated.

Pillars that were found to be cut too small or were showing signs of failing prematurely were wire meshed and strapped or laced using 16mm dia by 2,4m long sheppard crooks full column grouted on a 2m x 2m diamond pattern hanging wall and sidewall in addition to previously installed full column grouted re-bar.

Although wire meshing was not initially planned other than in the major excavations, by the time the project was nearing completion an enormous amount of wire meshing with lacing had been installed.

11.7 **Steel Rope Anchors**

Support at major breakaways, tips, loading points and other large excavations was reviewed and invariably the installation of additional support in the form of 6m x 20mm diameter steel rope anchors were installed on a 4m x 4m pattern.

Where faulting intersected any wide area or appeared to create a situation that could be considered unpredictable in the long term, additional anchors were installed. The rope anchors are fully grouted using vertical spindle pumps and tensioned to 25 ton load capacity.

11.8 **Loading Points**

Once a loading point had been blasted, usually every second intersection, the height of the excavation was in the order of 6 metres above the footwall. This excavation required a high standard of support with particular attention paid to any brows created and the joint planes exposed.

CHAPTER 12

Vamping and Sweepings

12.1 Introduction

A comment made at a mining colloquium on massive and mechanized mining was that it was reassuring to know that some subjects such as sweeping and vamping still existed in this high technology world of trackless mining. Having paid the money to break the gold reef, all the gold needs be recovered, however, a word of caution needs to be put in here. The recovery of the last gram of gold must not exceed the worth of the product. So often the standard is set so high for the sweeping that the recovery of the last of the gold and fines far exceeds the return. This is without doubt poor financial management.

12.2 Narrow and Steep

The standards at REGM on sweepings is graphically depicted in plate 12.1. All stopes that were mined as narrow areas were kept up to standard at all times. These standards are strictly enforced by line management and over inspected by the grade department.

In order to prevent repeated sweeping of the same area effective blast barricades must be installed, moved regularly and kept to the required standard.

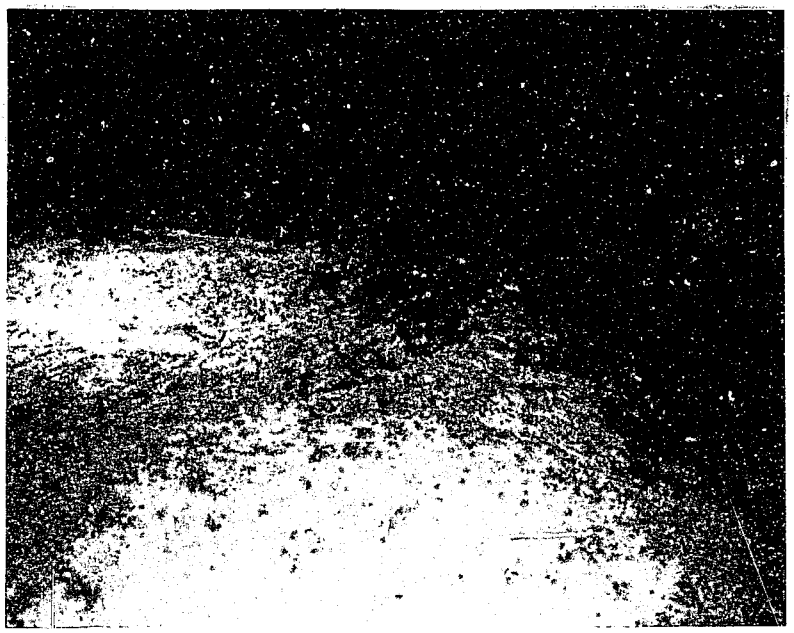
The steep areas have the benefit of gravity to assist with the cleaning of sweepings and in this case the back areas are washed down using a "Bazooka" or air and water cleaning tool. The standard of barricades is just as important in these areas as in the flatter stopes.

12.3 TM^3 Areas

The majority of the area was laid out as bord and pillar mining or adaptations of the method to suit local conditions. Therefore the vamping and sweeping of large mechanized areas had to be addressed.

At inception the decision was taken that as far as practical all bords with the exception of access roadways will be vamped and swept soon after completing the area or kept a maximum of 2 bords behind the working faces. One of the primary concerns was that

Plate 12.1



Standard of stope Sweeping

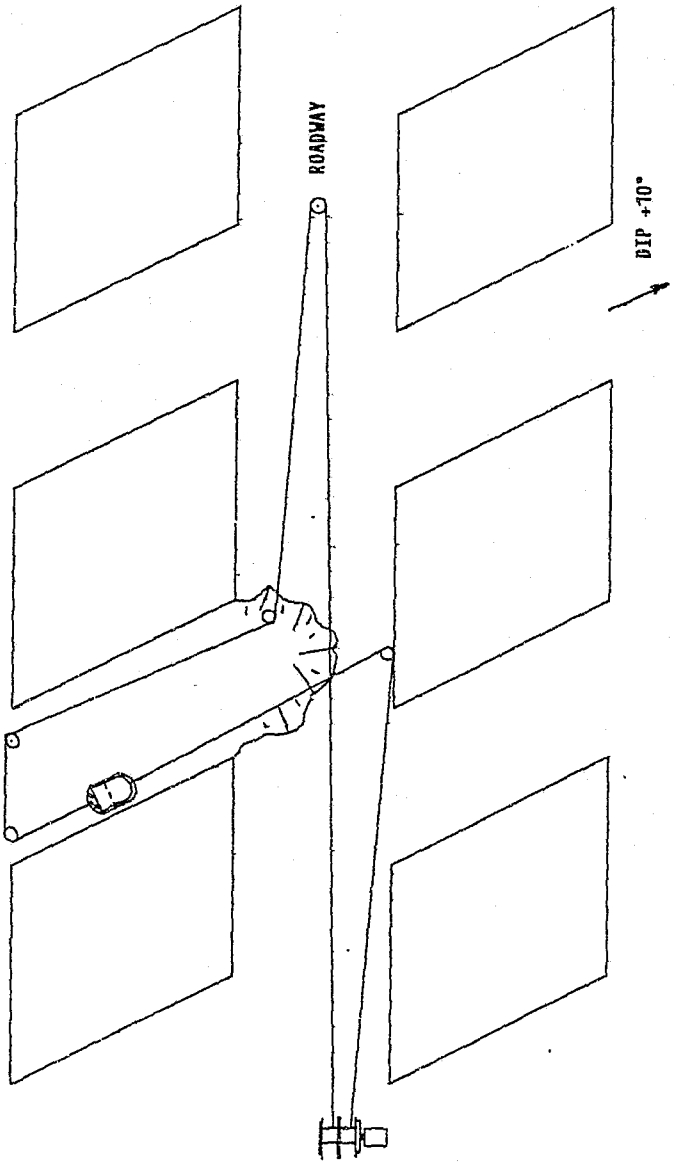
should the area become unstable during the mining, the ore used as ballast for road making material would be lost. Primary vamping was effected by using a D4 size dozer to rip the roadways, plate 12.2, and then to doze the material into heaps, plate 12.3, so that the LHD could load away the material.

In an attempt to speed up the cleaning of ore of the wide areas which had a true dip of around 10°+ it was decided to install a 75kW winch and then to scrape out the material in the bords, figure 12.1. Under normal conditions where rubber tyred vehicles had run over an area effectively compacting the ballast, it would have been a futile operation to try and clear this material, however, in this case because the dip of the stulls was such that it was decided to drill and blast the pillar holings with two blasts up dip and one blast from the roadway above down dip. The sequence is depicted in figure 12.2. The material from the 3rd blast was initially left until the roadway faces were some distance ahead. The material was then scrapped down and loaded out on the lower roadway. This was found to have some advantages. As the dozer was hard pressed in another area where the pillars were showing signs of taking a lot of pressure and it was thought that the area might be lost. The roadways not required for tramming in the area where the scraper was installed were ripped by the dozer which then went back to the higher priority area and the scraper was used to remove the vampings. As hand cleaning
(Plate 12.4) was the next phase, the scraper greatly assisted in moving the material gathered by the sweeping crew into a heap for loading by the L.H.D..

An attempt at high pressure water jetting using a diesel powered portable high pressure pumps was tried in order to improve productivity. The unit was marginally successful and was later converted to an electric unit, utilizing a trailing cable, which improved the output slightly and eliminated the noise and pollution control. The main shortcoming was that the unit had insufficient capacity both volume and pressure wise to be really effective.

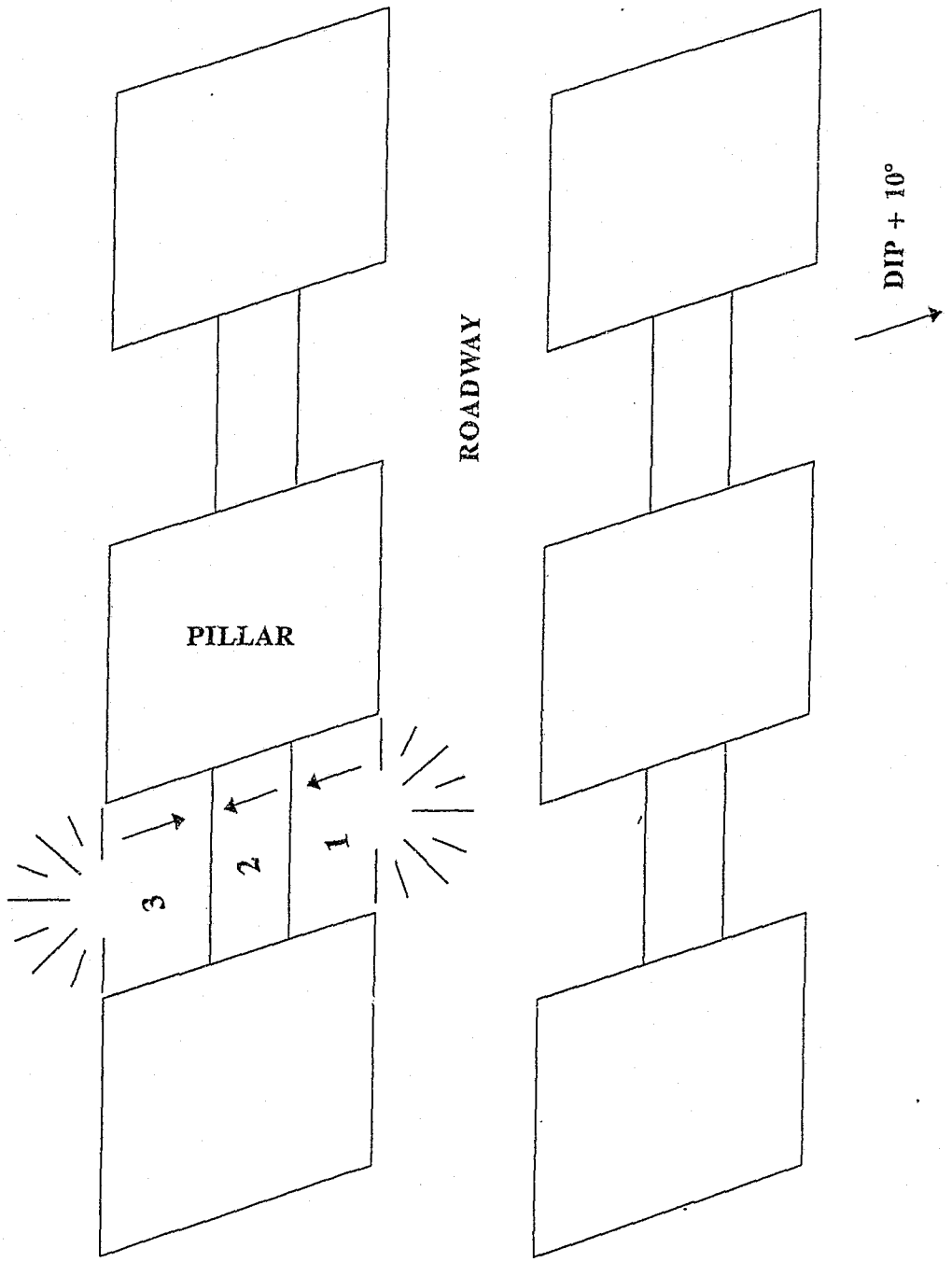
After the bulk of the material used for the roadways (varying from 100mm up to 500mm deep) had been removed by the dozer a considerable tonnage still remained which had to be removed. This was initially cleared away using shovels and wheelbarrows but the productivity was low and an alternative had to be found.

Figure 12.1



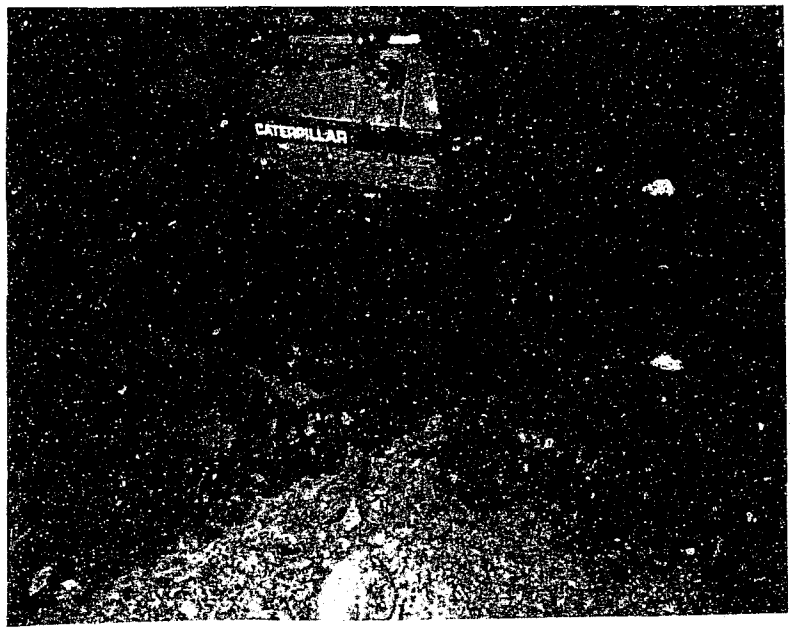
CLEANING USING SCRAPER WINCH IN TM3
AREAS WHERE DIP EXCEEDED 10°

Figure 12.2



SEQUENCE SHOWING BLASTING OF PILLAR HOLINGS WHERE THE MINOR DIPS EXCEEDED 10°

Plate 12.2



Dozer at Work

Plate 12.3

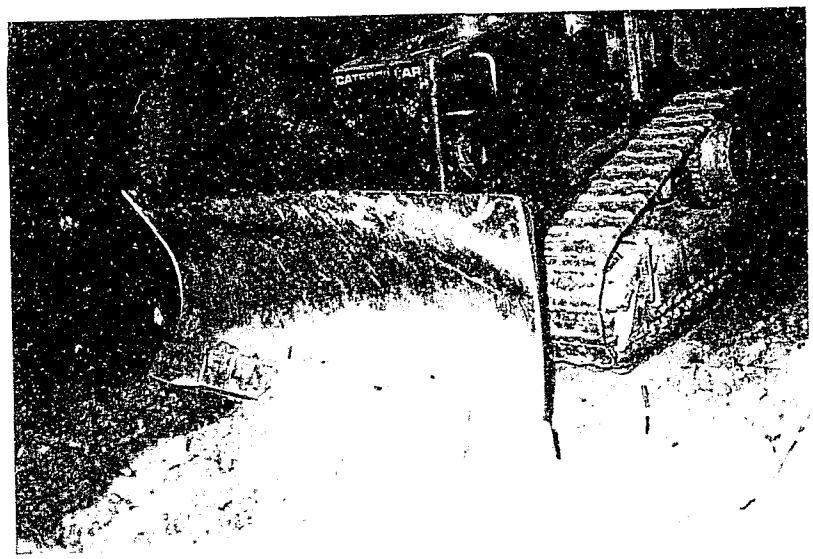


Plate 12.4



Hand cleaning

Plate 12.5



Transvac

Vacuum suction and transport was then put on trial. Although a unit had been tried some years earlier at Cooke 3 and was found to be inadequate, this unit was significantly different to warrant the investigation.

In discussion with suppliers it was ascertained that considerable progress had been made in the field and it was agreed to put a unit (Plate 12.5) on trial in the "spial" area. The results were satisfactory and the unit was purchased. In time a further two units were purchased for the shaft as a whole.

The unit operates with a small crew (Plate 12.6) who loosens any compacted material and vacuums up the fines and stones up to 100mm in diameter. Larger rocks block the nozzle and have to be removed. Reef too big for the unit is either broken up to a suitable size or loaded into wheelbarrows and taken to the loading area. Waste rock is discarded into the swept area and left.

The material is transported via a plastic column to an automatic drop box sited over a tip (Plate 12.7) or to a position where a pile is built up and loaded away by an LHD when sufficient material has been accumulated. A typical layout is depicted in figure 12.3.

Although the results obtained under observed conditions was quite high sustained output was not achieved as the hose operation caused rapid fatigue and persons had to be rotated regularly, however the results were excellent recovery of all the material wet or dry.

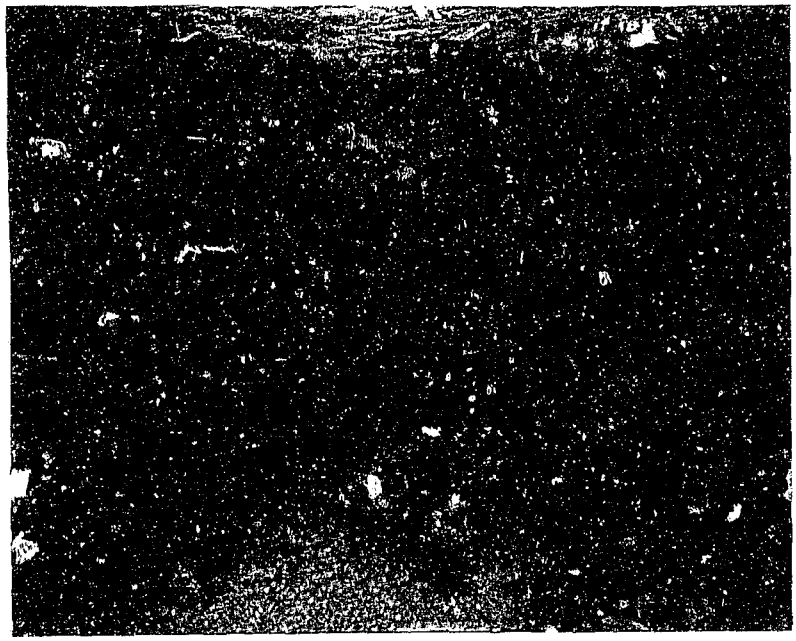
Initially it was believed that secondary enrichment of the fines would occur as the unit should theoretically suck up all the "free"gold which was likely to have collected in cracks and crevices. This was not the case and only the average grade was recovered in general.

Plate 12.6



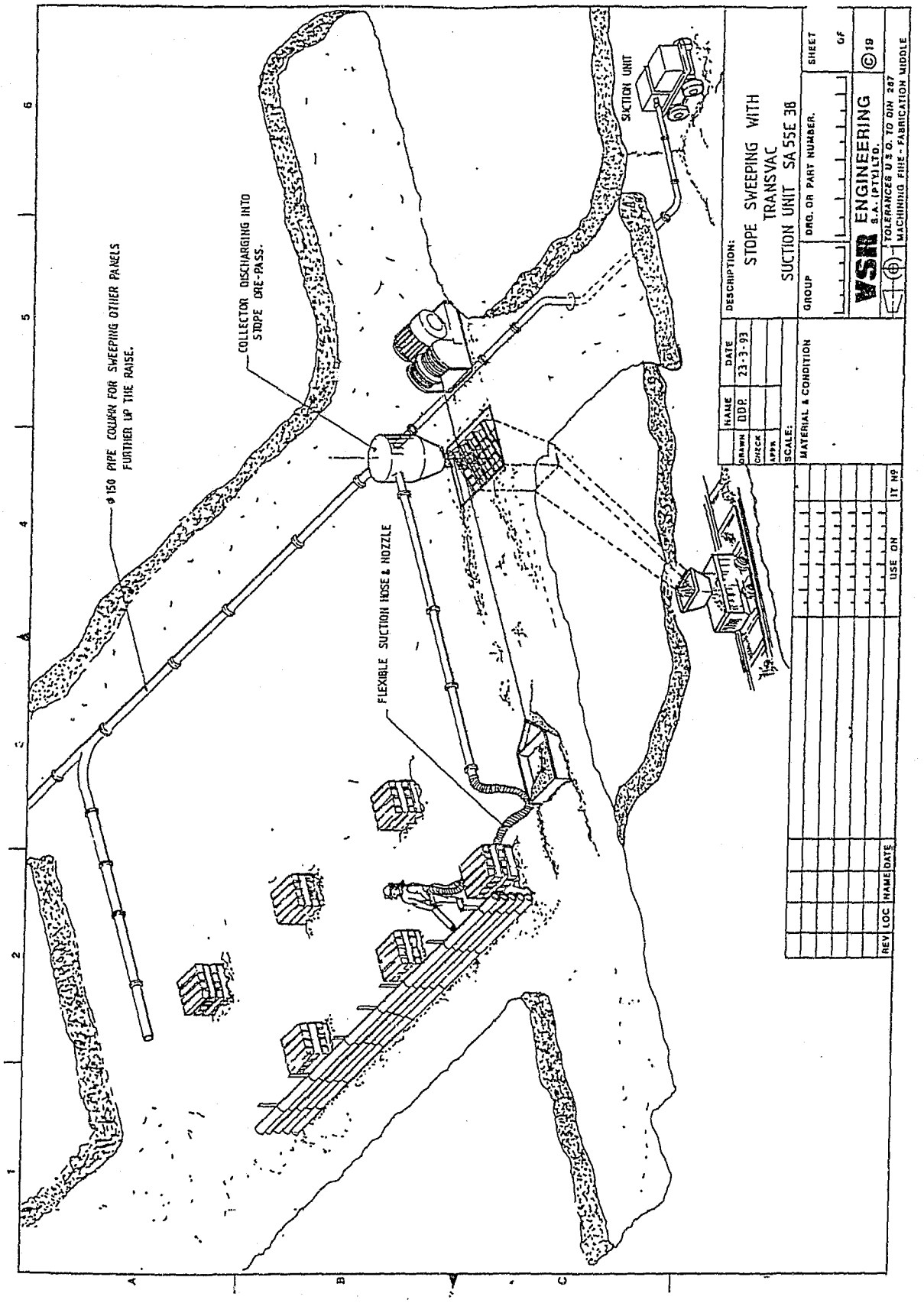
Transvac Crew working

Plate 12.7



Vacuum Drop Box

Figure 12.3


Ø150 PIPE COLUMN FOR SWEEPING OTHER PANELS FURTHER UP THE RAISE.
COLLECTOR DISCHARGING INTO STOPE ORE-PASS.
SUCTION UNIT
FLEXIBLE SUCTION HOSE & NOZZLE
DESCRIPTION:
STOPE SWEEPING WITH TRANSVAC SUCTION UNIT SA 55E 38
DRAWN DDP 23-3-93
CHECK
APPR.
SCALE:
MATERIAL & CONDITION
GROUP
DRG. OR PART NUMBER.
SHEET
VSR ENGINEERING S.A. (PTY) LTD.
TOLERANCES U.S.O. TO DIN 287
MACHINING FINE - FABRICATION MIDDLE
REV LOC NAME DATE
USE ON
IT NO

CHAPTER 13

Results Obtained

13.1 Introduction

On the completion of a project the results obtained need always be evaluated in order to determine the effectiveness of such plans in order that all persons can benefit from the experience. In both operations under review the results were very satisfactory and will be reviewed in some detail.

13.2. 95 Pillar

13.2.1 Safety

An excellent safety record was achieved during the period without a single lost time injury being sustained. This is most likely attributed to the constant attention paid to every aspect of safety particularly by the Supervisors and the extremely positive attitude of the entire crew. Considering the conditions at the start of the operation as shown on Plate 13.1 and 13.2, and that it was a pillar with likely difficult mining conditions the detailed planning, attention to detail and positive attitude all contributed to the excellent safety performance over the 50 months.

13.2.2 Tonnage Output

This operation was planned as a low tonnage operation but still planned to have a high productivity rate which was achieved. Figure 13.0.

13.2.3 Kilograms Produced

The operation, although not a high kg contributor to the overall shaft output, still was extremely profitable. The one point that needs always to be kept in mind is that this was gold locked up in a pillar and was part of the company's assets that needed to be realized.

13.2.4 Profitability

The final measure of success of any project must be its contribution to the profitability of the company. Despite the small scale of the operations, with proper planning and cost control, operations like this can be very profitable and in this case was profitable.

Plate 13.1

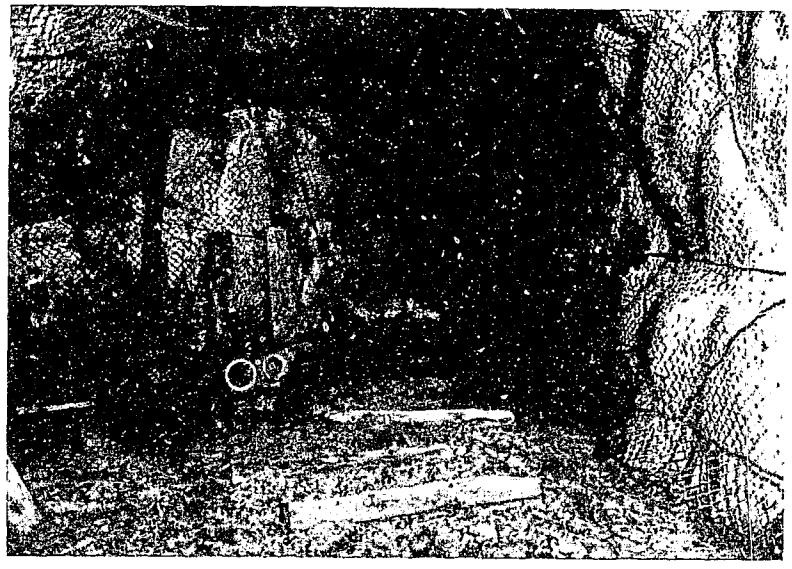
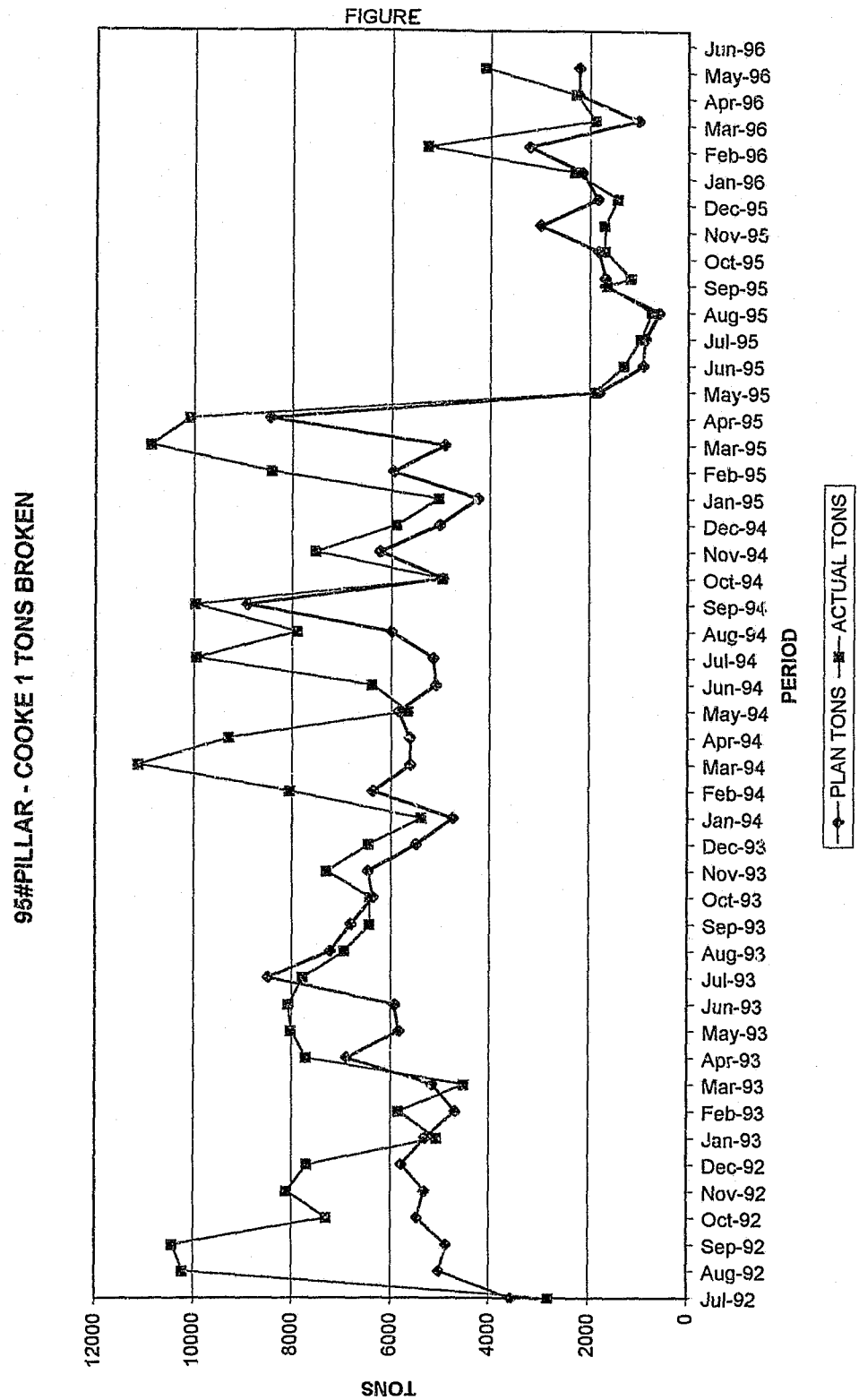


Plate 13.2



Conditions at the start of the operations

Figure 13.0


FIGURE
95#PILLAR - COOKE 1 TONS BROKEN
TONS
12000
10000
8000
6000
4000
2000
0
PERIOD
Jun-96
May-96
Apr-96
Mar-96
Feb-96
Jan-96
Dec-95
Nov-95
Oct-95
Sep-95
Aug-95
Jul-95
Jun-95
May-95
Apr-95
Mar-95
Feb-95
Jan-95
Dec-94
Nov-94
Oct-94
Sep-94
Aug-94
Jul-94
Jun-94
May-94
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Mar-94
Feb-94
Jan-94
Dec-93
Nov-93
Oct-93
Sep-93
Aug-93
Jul-93
Jun-93
May-93
Apr-93
Mar-93
Feb-93
Jan-93
Dec-92
Nov-92
Oct-92
Sep-92
Aug-92
Jul-92
PLAN TONS
ACTUAL TONS

13.3 101 Area

13.3.1 Safety

The safety achievements in this area can only be described as outstanding. When one considers the amount of old areas that were traversed, the frequency of dyke and fault intersections, and not the least that it was a pillar area with expected high stress areas that not a single serious fall of ground injury occurred from the start of the project up to the end of 1992. An excellent safety record. One of the more outstanding safety achievements is that the project has remained reportable free for 6 years.

13.3.2 Tonnage Output

Graph 13.1 and 13.2 indicates the tonnage from the third quarter 1986 though to middle 1993. The tonnage's fluctuations over the period were due to many and varied reasons. In some months lower tonnage were planned due to the anticipated negotiating of old areas, or faults and dykes or at times because tramming distances were getting to be long just before the next tip came into operation.

13.3.3 Profitability

Although the mining conditions could be described as difficult as the result of extensive faulting and the numerous other problems that affected the operations primarily as a result of the faulting, the project returned a significant profit (Graph 13.3, 13.3 and 13.5) and as a result company assets that had been temporarily unavailable were released.

Figure 13.1

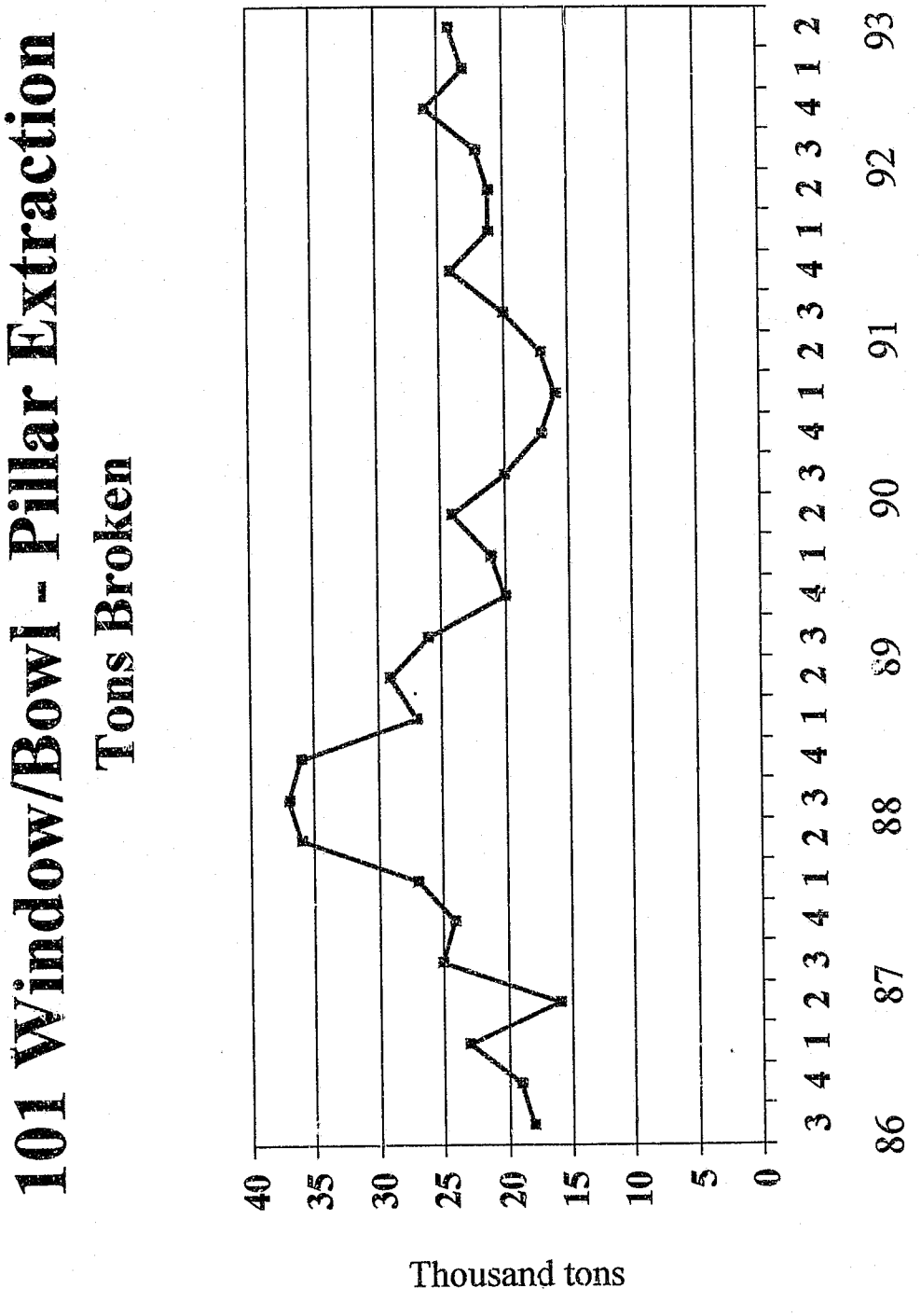
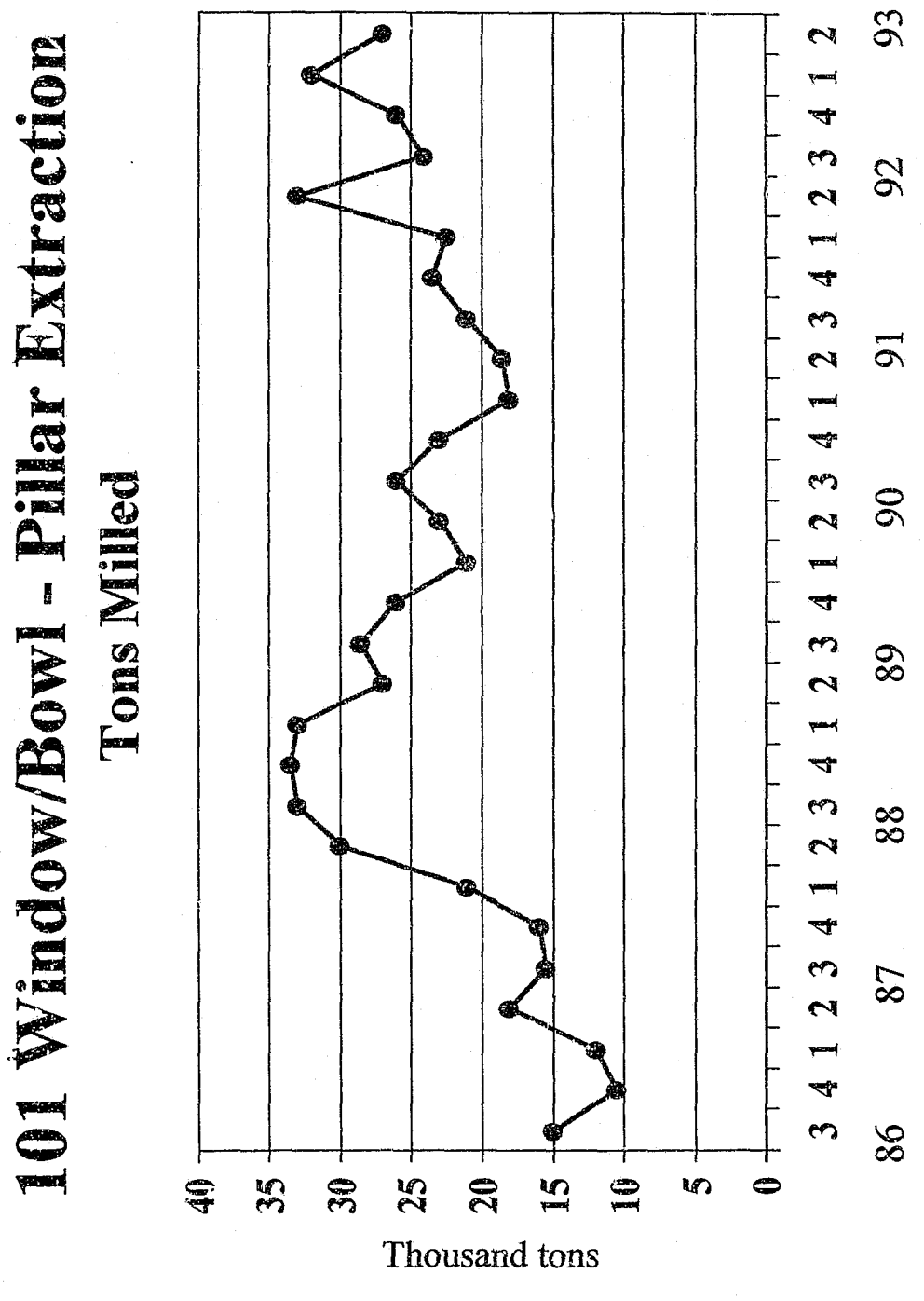

101 Window/Bowl - Pillar Extraction
Tons Broken
Thousand tons
40
35
30
25
20
15
10
5
0
3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2
86
87
88
89
90
91
92
93

Figure 13.2


101 Window/Bowl - Pillar Extraction
Tons Milled
Thousand tons
40
35
30
25
20
15
10
5
0
3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2
86 87 88 89 90 91 92 93

101 Window/Bowl - Pillar Extraction

Profit Contribution

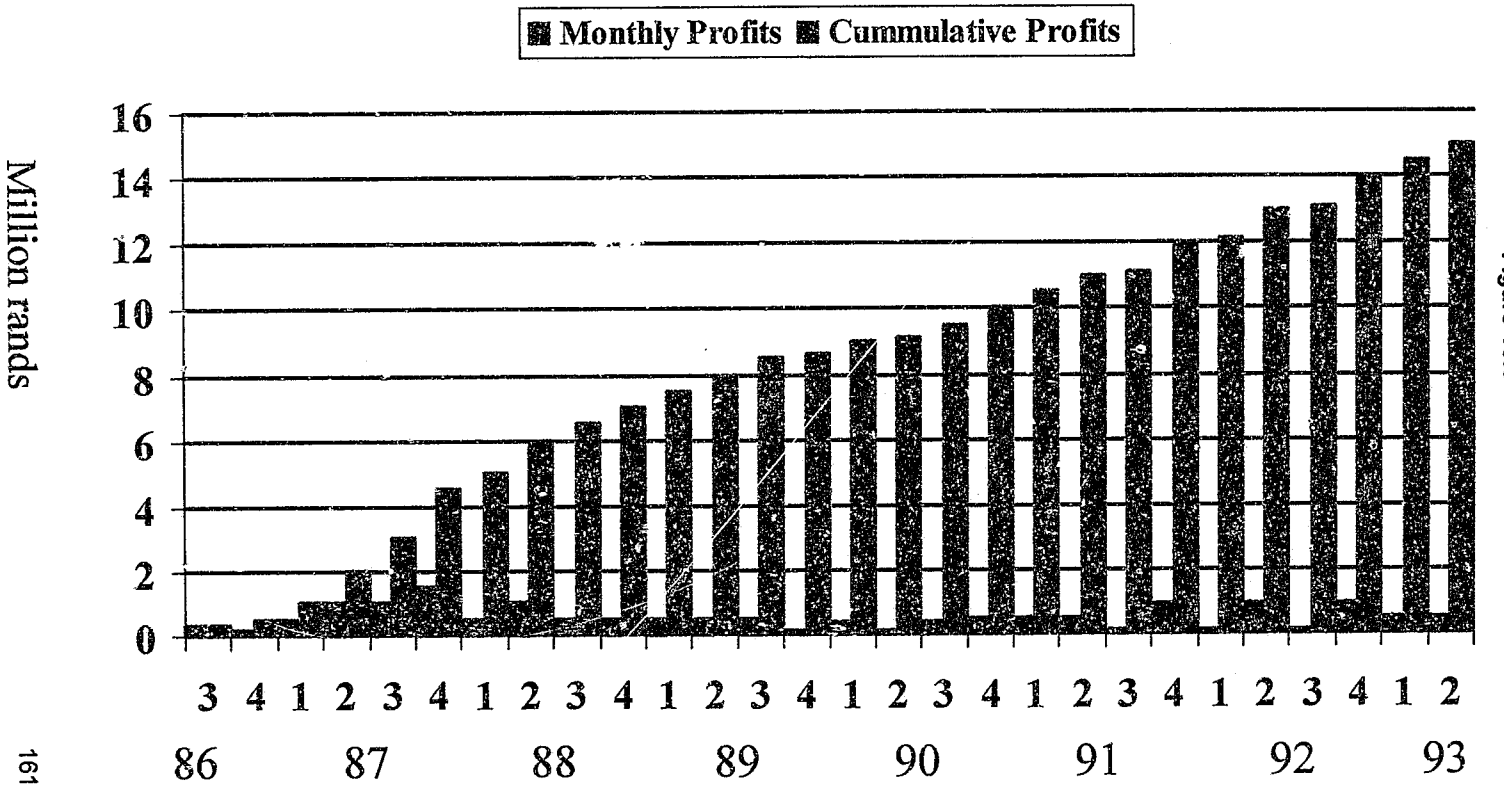


Figure 13.3

Figure 13.4

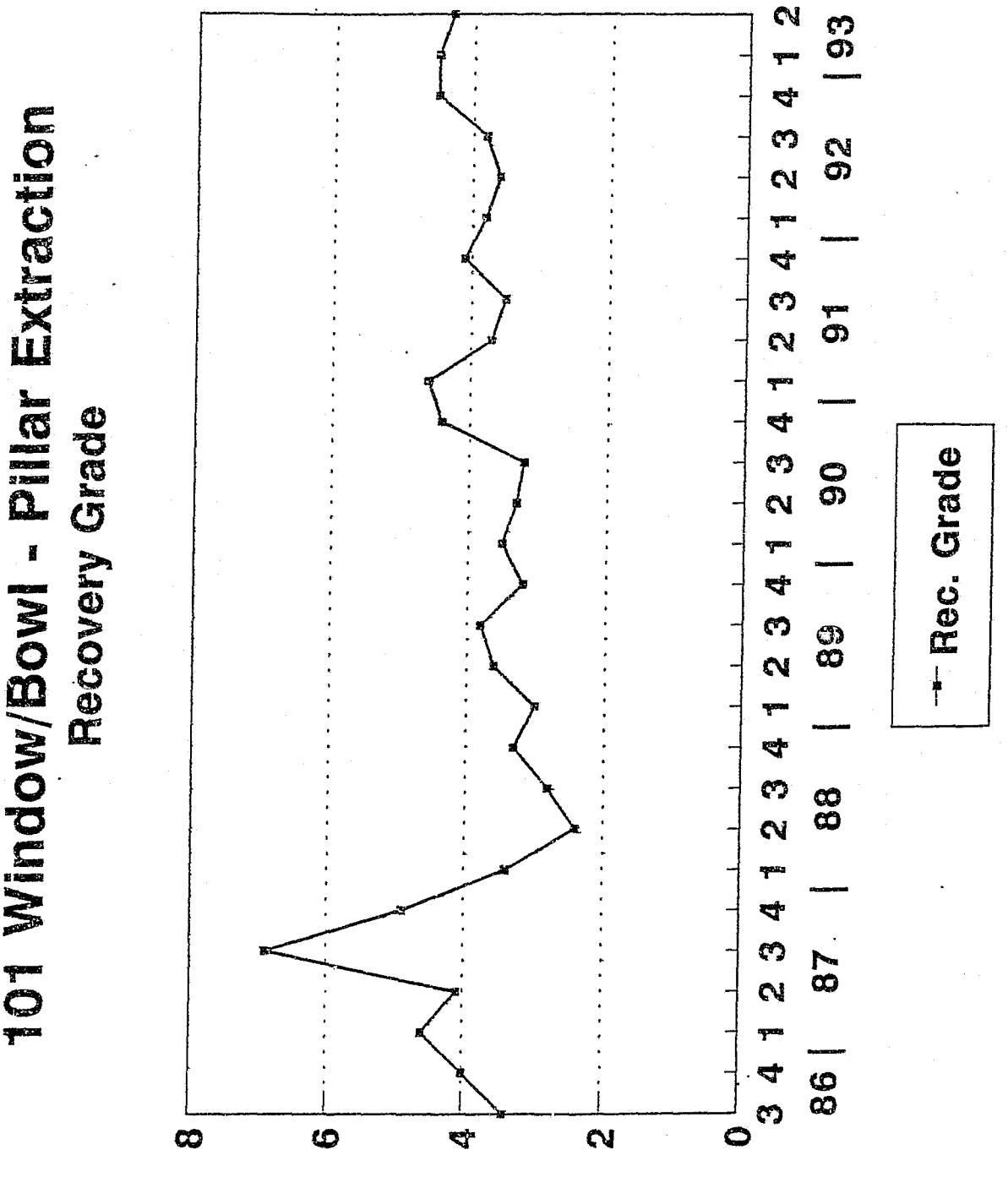
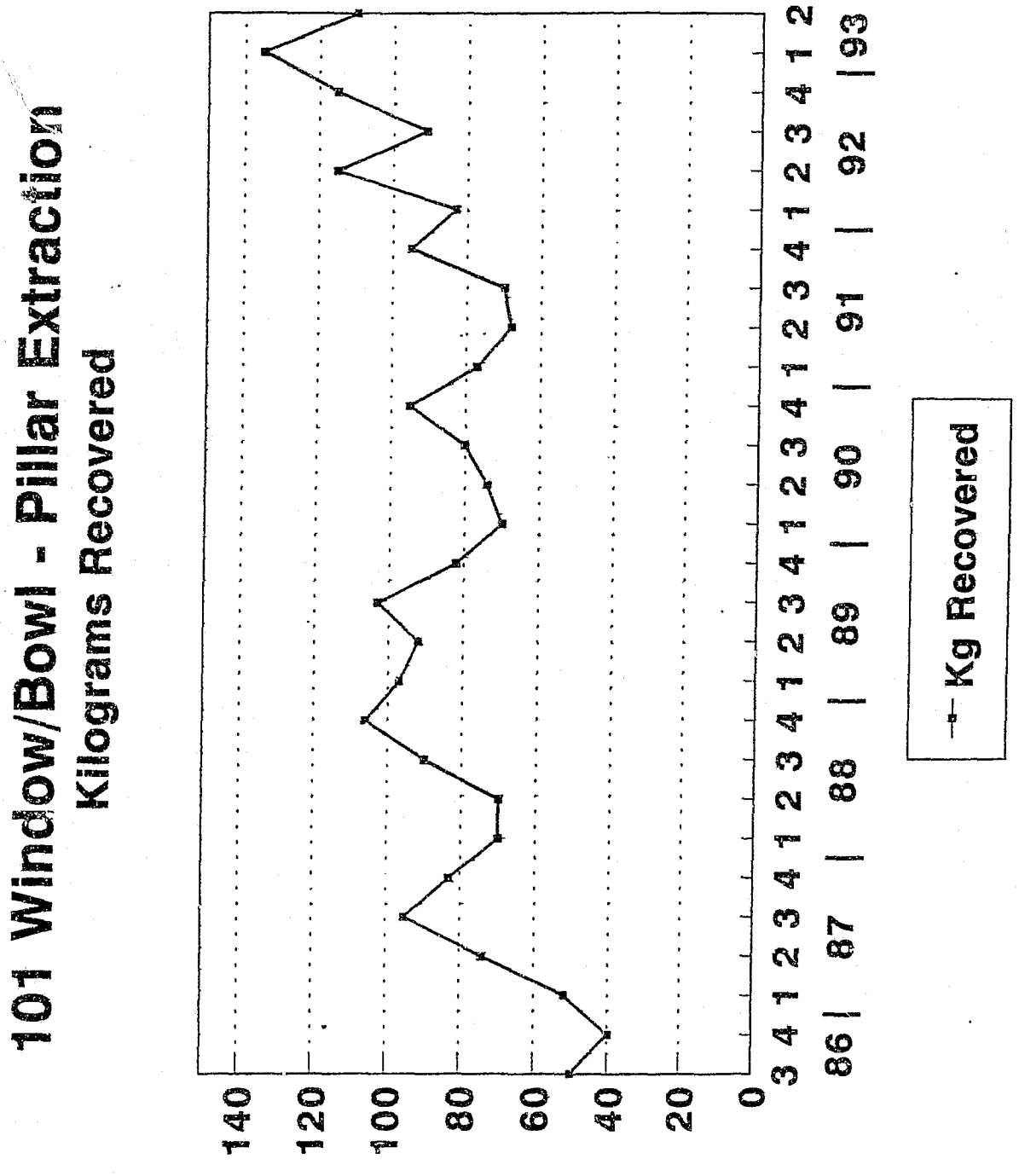

101 Window/Bowl - Pillar Extraction
Recovery Grade
8
6
4
2
0
3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2
86 | 87 | 88 | 89 | 90 | 91 | 92 | 93
Rec. Grade

Figure 13.5


101 Window/Bowl - Pillar Extraction
Kilograms Recovered
140
120
100
80
60
40
20
0
3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2 3 4 1 2
86 | 87 | 88 | 89 | 90 | 91 | 92 | 93
Kg Recovered

CHAPTER 14

Problems Encountered

14.1 Introduction

Even though extensive planning was done, alternatives considered, computer simulations done and a lot of mining experience applied to the problems at hand, some unforeseen problems always appear. Good quality planning using quality information will without doubt reduce the frequency as well as the magnitude of the problems, but problems there will be and it is up to the project team to overcome these with the minimum of disruption and in the most cost effective manner available.

14.2 95 Haulage Pillar

Possibly because of the small size of this pillar and the extensive information on it little or no notable problems occurred that had not been foreseen or forecast. The project was handled excellently and is a credit to the mining team.

14.3 101 Area

14.3.1 Pillar Sizes

The original pillar design proved to be too small, particularly when other features such as minor faults were in the area, and the pillars failed. This resulted in extensive wire-meshing and lacing being resorted to in order to stabilize the pillars. Those, in the one particular area, that were not wiremeshed and laced, failed completely, thereby adding extra strain to the surrounding area and pillars.

14.3.2 Breaking into and negotiating old mining areas although anticipated, was more time consuming and costly than planned. On a good many occasions when the area was broken into, large scale caving was encountered resulting in alternate routes having to be sought. This resulted in delays, re-planning exercises and re-routing main roadways and services.

Supporting these old areas, many of which had previously been inspected and were believed to be in good condition, proved to be far more costly and a much greater task than originally planned. Additional resources had to be applied to handle these operations.

The problem was later believed to have been caused, in part at least, by the changing stress patterns from the ongoing mining operations and in part from the effect of blasting when the holings were effected.

14.3.3 **Unexpected Faulting**

Although it was originally believed that the position of the majority of faults were known and could be projected from the surrounding area as well as haulage and x-cut development, a number of surprises occurred. Small faults splitting off from main faults caused numerous secondary faulting which then also deviated considerably from the anticipated or projected fault positions. Initially there was no easy way of projecting or predicting these faults but after the leading roadways had intersected the faults the appropriate action could be taken for the following roadways.

14.3.4 Variance in fault displacements occurred frequently. Predictions of fault displacements were often found to be incorrect, partly because of faults joining together or splitting away and because of the scissors nature of a large number of the faults. The project geologist was constantly involved with the mining team, plotting and predicting the fault displacement and positions.

14.3.5 Computer analysis predictions were not always correct and from time to time the rock engineering department had to re-assess the recommendations given for a particular situation. As the criteria were altered and the applications became more stringent, the operations probably erred more on the conservative side. This is born out that overall little or no rock engineering related problems occurred.

14.3.6 Orepass

In years gone by orepasses had been developed from the haulages and cross cuts and aimed at points above or on the present mining horizons. Although positions were given as to where the orepasses should have been, on several occasions considerable effort had to be applied in order to locate and slipe these in preparation for LHD tips.

On a number of occasions these orepasses were in worked out areas and accessing these as well as supporting the area proved to be more difficult than originally anticipated. Stripping hanging wall for a truck tip also proved to be more difficult than planned and in one case the whole operation of establishing the tip was abandoned when it became dangerous and costly. An alternative had in the interim been located which turned out to be more suitable.

14.3.7 Ventilation

Although more than sufficient total volume of air was supplied to the project certain problems arose, some of which were anticipated while the effects were somewhat different than forecast.

When breaking into old areas at times the ventilation disappeared which then resulted in ventilation problems along the roadways, particularly with tramming using trackless equipment. The reaction of the ventilation varied and as such the environmental department had to give constant assistance on a re-active basis.

Passing through previously mined out areas and having to create ventilation controls and maintaining these controls required more work than anticipated. These ventilation curtains had to be kept up to a high standard or else the air was lost to the mined out areas.

14.3.8 Roadway Gradients

The objective was to maintain roadway gradients at 8° or less. The frequent faulting resulted in the planned layouts for roadways having to be changed which usually resulted in much longer tramming distances than planned and the regular re-evaluation and re-planning of tramming routes.

With the roadway laid out at -8° the intersection of a downthrow fault of as little as 2 metres results in significant changes. Had it not been for the flexibility and adaptability of the trackless mining equipment these disturbances would not have been as easily negotiated and the layouts adapted.

14.3.9 **Broken ore left in old areas**

As stated earlier, some of the areas had been previously abandoned due to strata conditions where collapses had occurred in the ASG or the gully or even possibly major problems in the cross cuts. This meant that ore had been left on the faces, in the scraper paths and invariably in scatter piles and sweepings. The standard of the mines sweepings had also become more stringent over the ensuing years.

In the planning stages no provision had been made to reclaim this ore and in many areas it would have been uneconomical. Each area had to be made safe, sampled and evaluated and where the economics indicated that it was viable, layouts were done and methods applied to recover the ore. Amongst those methods applied included:

14.3.9.1 Installing a scraper winch and scraping the ore to a point where it could be loaded with the LHD.

14.3.9.2 Putting in a sweeping crew to hand sweep the areas.

14.3.9.3 Using a high pressure water jetting operation.

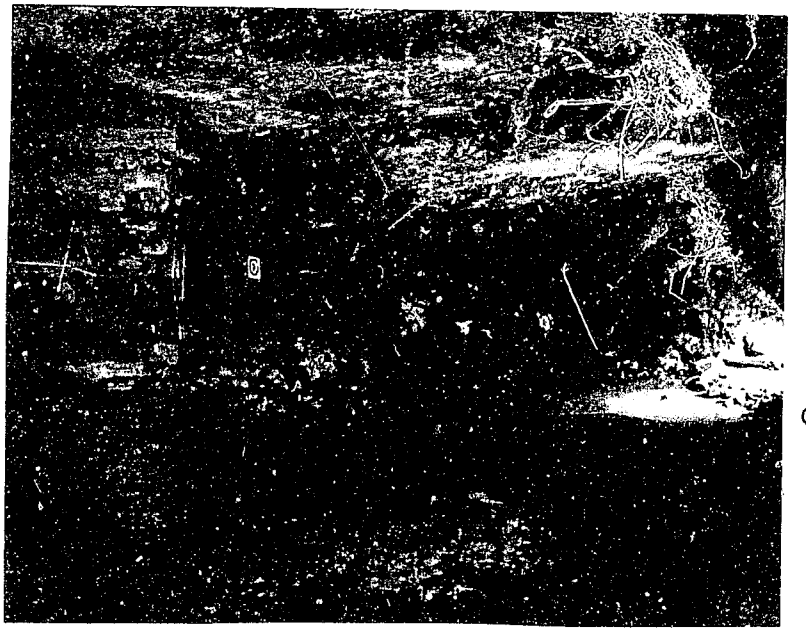
14.3.9.4 Applying the vacuum cleaners that were being used by the reclamation section.

In the final analysis no problem was too big to handle, although they resulted in changed layouts, numerous hours of re-planning and altering tramming routes. In fact the mining team became so adept at rectifying the problems as they arose that the operation ran smoothly continuously.

Plate 14.1



Wiremeshed pilllar



Ore in old areas

Plate 14.3



Sweeping Crew's in Old Areas

CHAPTER 15

Lessons Learned

15.1. Introduction

Possibly the greatest lessons that was reinforced throughout this whole project was that by careful planning and dedicated adherence to the programme is essential . Safety standards must be maintained. However, the ability of the team to take corrective action when the unexpected occurred these difficult and potentially hazardous operations were carried out with an outstanding safety record.

15.2 95 Haulage Pillar

The extent of regional and local pillars left as well as the standard of support was such that 3 years after having completed the area the conditions remain excellent. Two of the theories put forward as to why the deterioration was not more rapid were:

1. The pillar over the previous 10 years or more had become "distressed" and hence the amount of scaling initially encountered.

2. That too much was left as pillars to support the area.

At this point it would not be economical to mine more of the pillars but then the question arises as to how much more could be taken without the total area collapsing.

None the less, it is believed that in the interests of safety the correct evaluation and mining had been applied. The implementation of trackless mining to this project proved to be the safest and most economical method of extracting pillars. Provided reef width allows the application of the equipment other pillars can be addressed in a similar method.

15.3 Pillar Sizes

Although the pillar sizes were carefully planned and checked against the computer simulation, an optimum, tending towards the conservative, needs be established from the start. The dilemma exists of not wanting to leave too much in pillars as

not only does it tie up ore it results in longer sections between stalls which in general would mean lower face availability and lower extraction rates.

As with any remnant to be extracted a person can never be sure how much will be extracted before it either becomes too dangerous or the cost becomes excessive due to dilution, opening up falls of ground or support costs.

It can always be argued that by leaving larger pillars, these could be reduced or extracted on a retreat basis. It must be remembered that access to some areas are inevitably lost due to ground conditions and therefore those pillars would become lost as well.

In the case of the 101 project the initial pillar sizes were planned to be 7m x 10m which initially seemed to be correct. However, as can be seen from Plate 15.1 some of these pillars failed. Although some of the pillars failed and some closure was evident the hangingwall remained sound as can be in Plate 15.1.

On analysis it was found that the most probable cause of the pillar failure had been a minor fault with no displacement that had run roughly parallel to pillars.

Subsequent to this the pillar dimensions were increased to 8m x 10m and no further problems were encountered. This, although it does not sound significant it increased the pillar sizes by some 15%.

The area 'A' had an average dip of 14° and as such the pillars were designed in such a manner that the gradient for the trackless mining machines was kept at 8° or less. The original pillar size in this area was laid out at 7m x 10m with 7 metre rooms. Some of the pillars very soon showed signs of stress and previously unknown minor faulting was intersected. The dimensions were altered to 8m x 10m pillars with 6 metre rooms. With the change introduced the ground conditions became more stable and the work environment improved appreciably at the expense of a much lower extraction rate.

15.4 Pillar Stabilizing

As a routine mine standard, sidewalls and thereby pillars included, were supported or rather strengthened by the installation of 2,4m long 25mm diameter re-bar with fully grouted cementitious capsules. As the areas were mined some of these pillars in the back areas started self mining as can be seen in Plate 15.2 and Plate 15.3.

It became obvious, primarily in the interests of safety and to prevent as far as possible total collapse of these pillars that they would have to be stabilized.

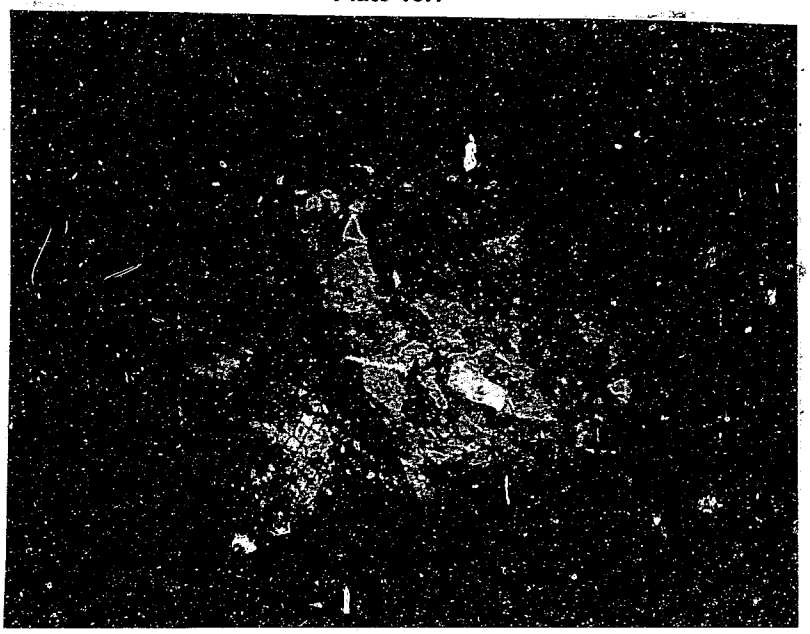
The pillars showing stress were then wire meshed and laced as is shown in Plate 15.4 using 16mm diameter, 2,4 metre long shepherds crooks and diamond mesh or weld mesh. The bolts were drilled on as near as possible a 2m x 2m diamond support pattern.

As the sidewalls were now fractured and less than stable the worst of the loose material was removed and meshing put against the face and supported on jacks and sticks against the hanging wall.

In the case of some of the pillars strapping was pulled around the mesh and tensioned. Wherever possible the pillars were drilled through the wire mesh which served as a protection for the machine crews, the Shepherds Crooks were then installed with cementitious capsules. After the cement had set sufficiently, steel rope lacing was spanned through the Shepherds Crooks and tensioned. These pillars then stabilized and continued to provide support.

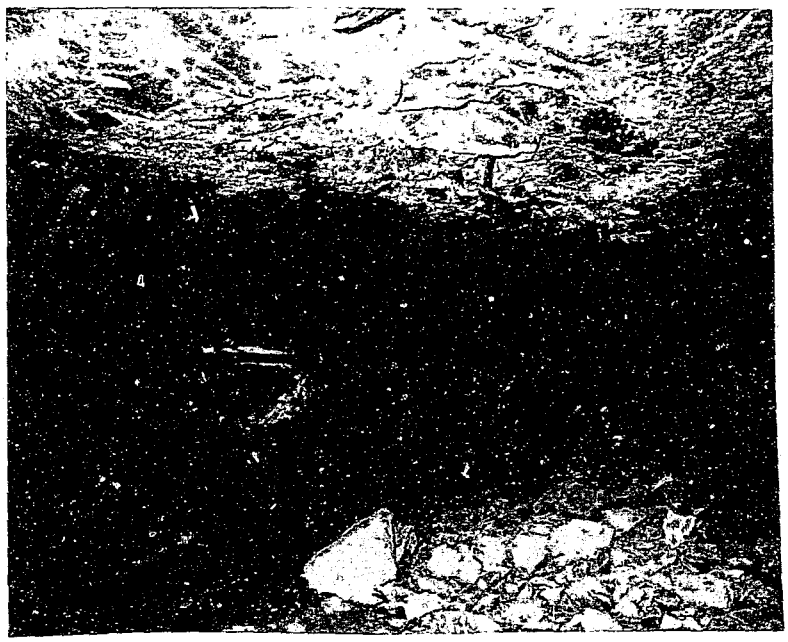
The support of partially fractured pillars is not only a hazardous occupation requiring special precautions and attention, it is also extremely difficult to drill into the fractured rock. To add further to the potential problems the pillar was then smaller than planned and did not have the same strength. The recommendation is that the main access roadways pillars be wire meshed and laced soon after development and before fracturing takes place.

Plate 15.1



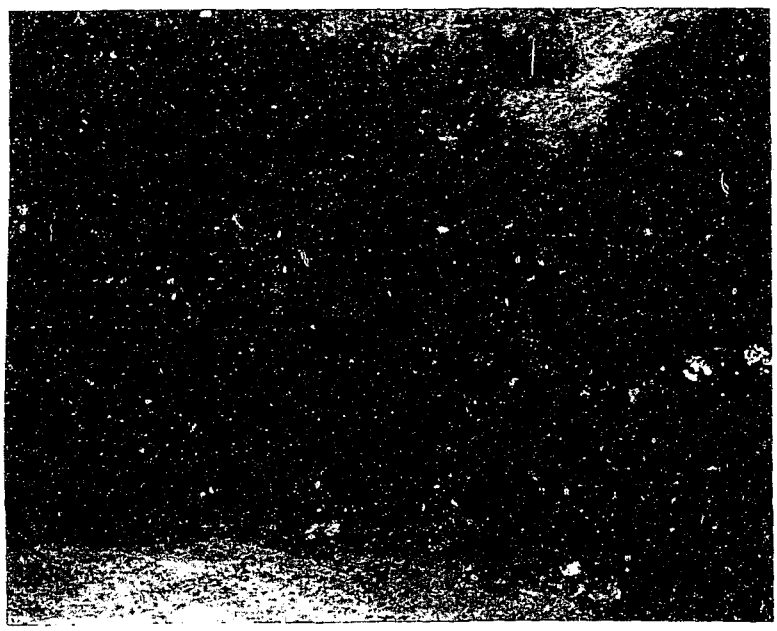
Failed Pillars

Plate 15.2



Pillars Self Mining

Plate 15.3



Pillars Self Mining

The proposal is not to wire mesh and lace all pillars. During the computer simulation pillars that are likely to give problems should be identified and secured early on. The requirements will vary from location to location, with the simulated stress profiles for that area indicating the need to taken early action.

15.5 **Mining Standards**

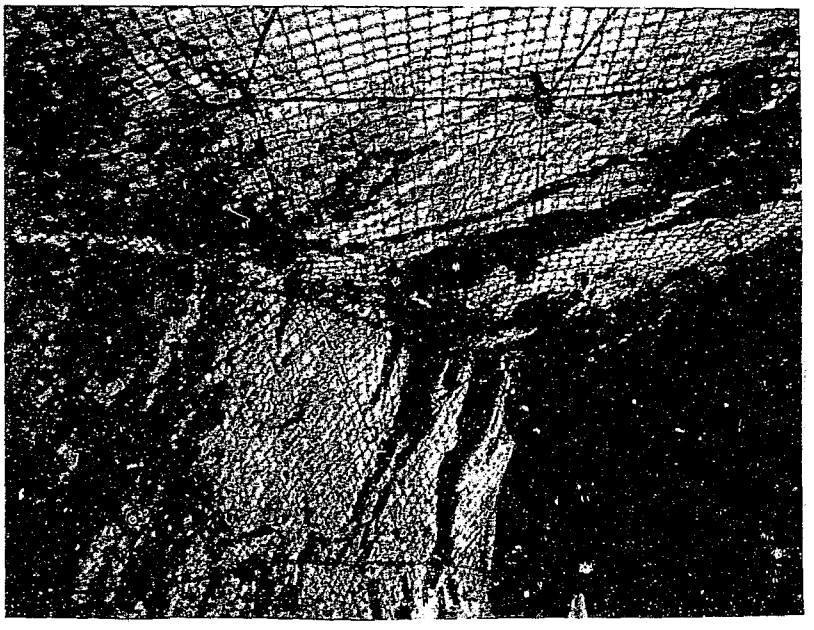
Quality drilling is important as mining men well know and when a drill rig, which drills a 3,2m long hole is used, the standard of drilling becomes more critical. The drill rigs all have very technologically advanced parallel holding facilities. Obviously some manual override is required in order to perform the required movements and to drill the look out angles on the perimeters. The drill rig operators use the manual override far too frequently because of perceived or actual non-parallel function of the booms. Unless strict drilling standards are maintained, off line holes into the hanging wall or the pillar results in damage to both which in time is exacerbated as the areas take more stress.

The cutting of the pillars particularly when turning off or having to cut at an angle because of the dip or other reasons for odd shaped pillars, was found to be extremely important. Some of the earlier pillars were found to have been cut too small and in some cases because of some sloppy mining, the pillars become useless as support.

This was soon rectified by strict enforcement of the mining plan regarding pillar sizes and the team became very good at maintaining the standards. Had this been understood and enforced right from the beginning, the problems that had to be subsequently addressed in the areas could have been avoided.

15.6 **Traversing Old Areas**

The breaking into and traversing old mining areas was costly and time consuming and at times the area selected to traverse to get to an old orepass or a remnant was found to have had a major collapse resulting in alternate routes having to be sought.



Pillar Wire Meshed and Laced

Whenever possible the old areas were pre-inspected before layouts were made for traversing the area but at times due to accesses either not being available or considered unsafe other plans had to be made. In general with the mines strict support and mining standards, areas were still in a very good condition.

Supporting these areas was a much greater task than initially envisaged, requiring the deployment of additional resources. These areas were all well supported as the roadways went through and no major problems were encountered subsequently.

The disposal of the waste from these roadways also proved to be more of a problem than initially envisage. The LHD unit can only dump a certain amount in an area and although attempts were made to push as much into the old areas space for waste dumping created major headaches at times and resulted in higher costs than what was originally planned. Plate 15.5.

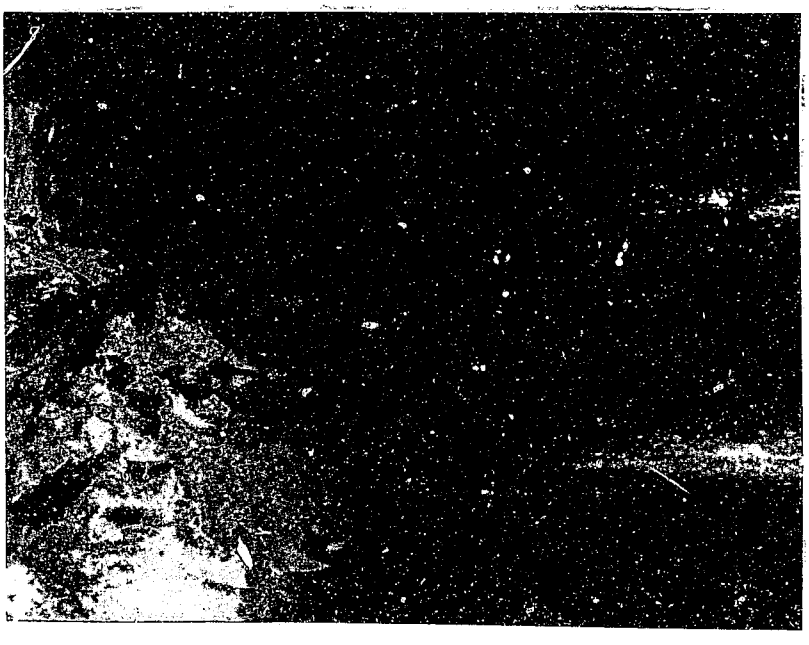
15.7 Double Cutting

The height of the roadways was to a large extent dictated by the size of the equipment that was available for the work.

Cooke 1 shaft was at this stage in its life mainly a low grade operation and every effort had to be made to enhance the value of the ore sent to the mill. As the reef was narrower in many places than the size of the tunnel required for the equipment, double cutting was resorted to.

Double cutting is an operation where the reef and waste are taken out in two separate operations. Figures 15.1 and 15.2 indicates the extraction of the primary cut and the removal of the "waste" or low grade ore which is then dumped in the mined out areas or in the case as shown in figure 15.2 the reef cut which is then trammed to the reef ore pass. On the next blasting shift the second cut is dropped and handled appropriately depending on whether it is reef or waste.

Plate 15.5



Traversing old areas

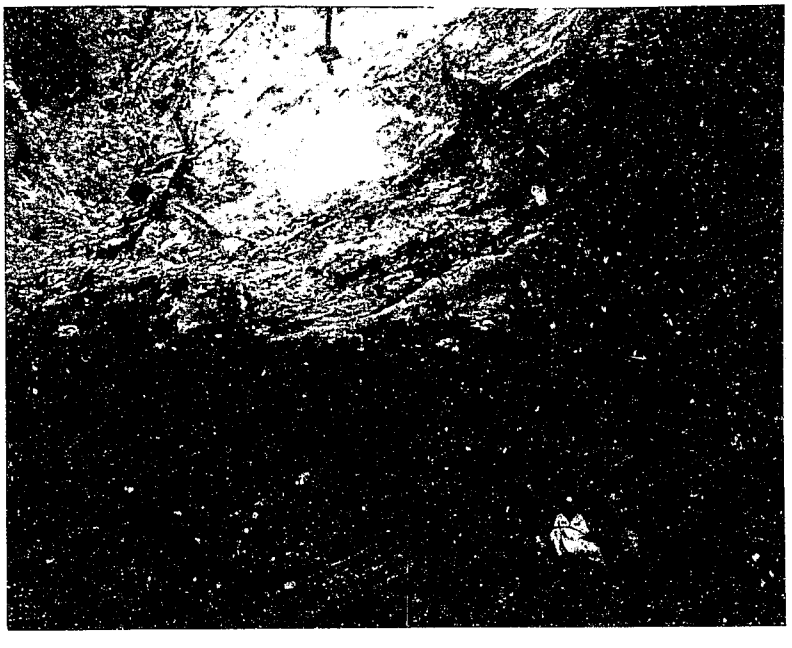
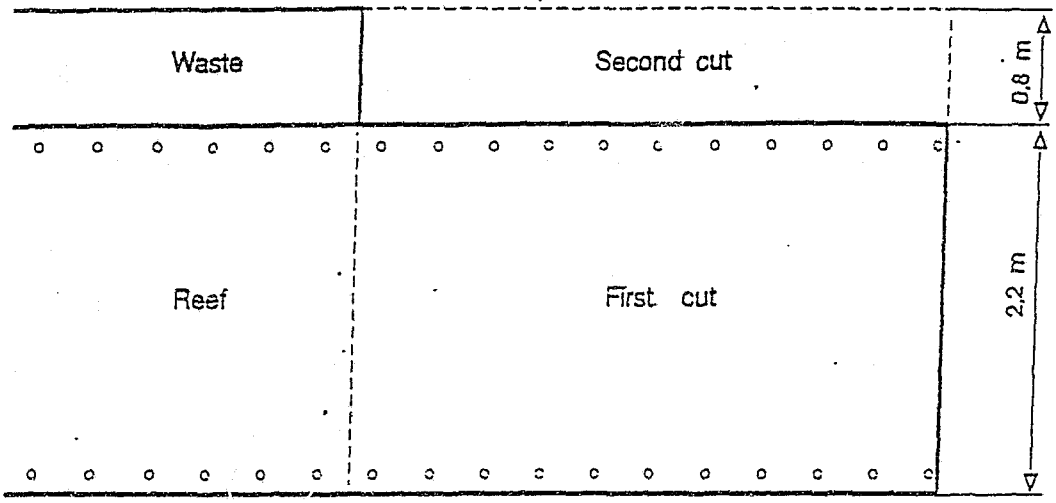


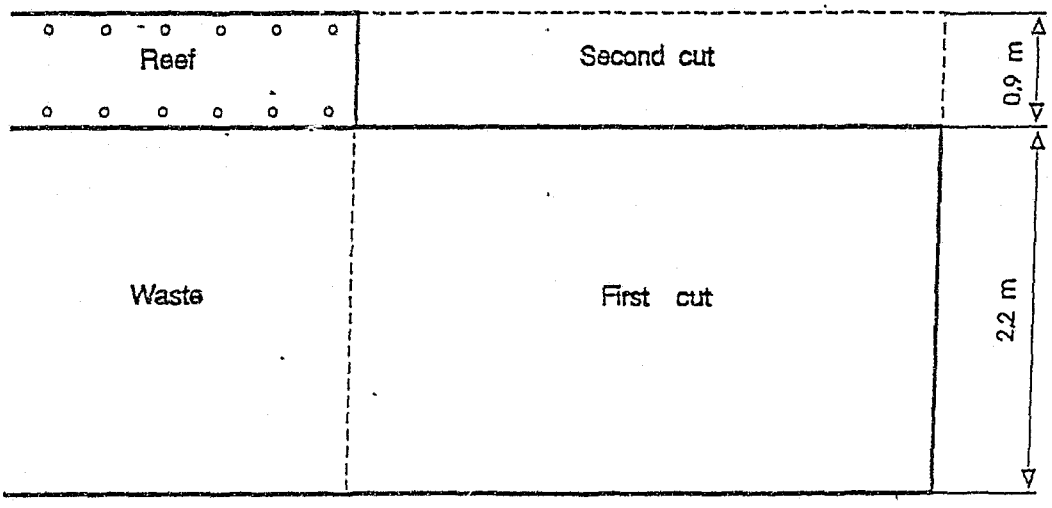
Figure 15.1



Section

Double cut with a channel of 2,2 m.

Figure 15.2



Double cut with a channel of 0,9 m.

The controversy over the benefits of double cutting has raged for a number of years with some showing it to be of great benefit while others show little or not benefit. At Cooke 1 the K9 Kimberly reef area had been successfully mined using this method and it was subsequently applied in the 101 area. An unpublished paper by Strauss, H on double cutting has at last put this into perspective and indicates with the aid of graphics in what ranges double cutting is an advantage. Figures 15.3 and 15.4.

Some of the problems encountered were:

15.7.1 Contamination of the waste by reef either scaling off, or picked up from the footwall or from blasting. Figures 15.5 and 15.6.

15.7.2 Sufficient clearance to load first cut, at times, resulted in the excavations being bigger than desired with extra material being broken. Figure 15.7.

15.7.3 In areas immediately underlying the UE1A reef horizon are the GD reef bands. These carry some values and in areas are payable. The main problem was that it was extremely difficult to differentiate between the broken ore and low grade reef.

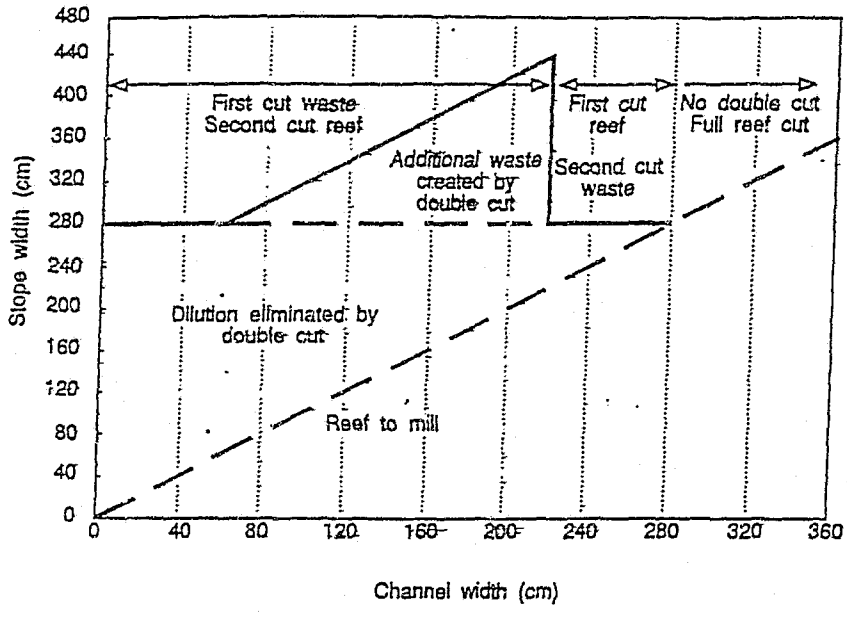
15.7.4 When sampled, the so called waste was found to contain sufficient values to pay for tramming hoisting and treatment with the cost of breaking it already provided for.

15.7.5 To find sufficient dumping place for the waste or low grade ore was a constant problem.

15.7.6 Twice as many working faces were required because of the two stage operations. In a full mining area this can be accommodated while in a remnant area this becomes progressively more difficult.

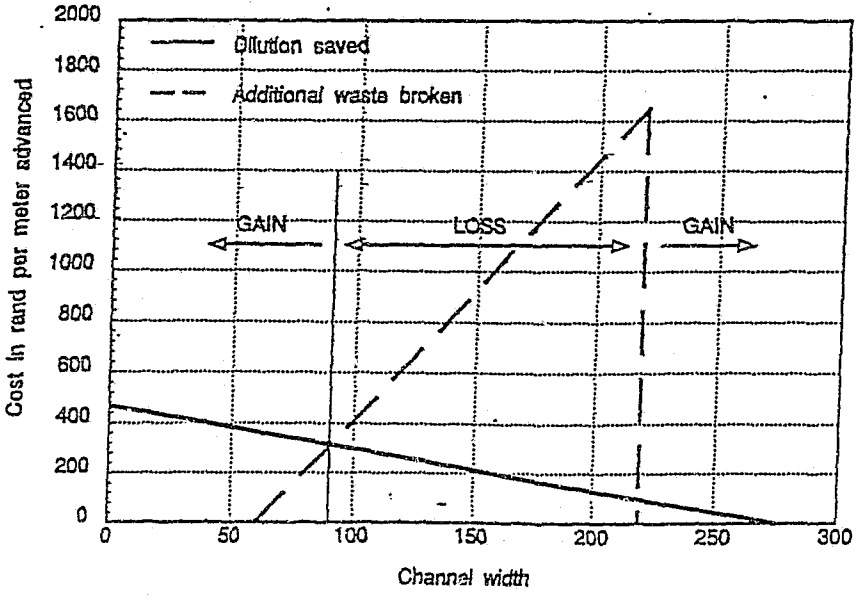
15.7.7 It was a more costly operation to run and operate.

Figure 15.3



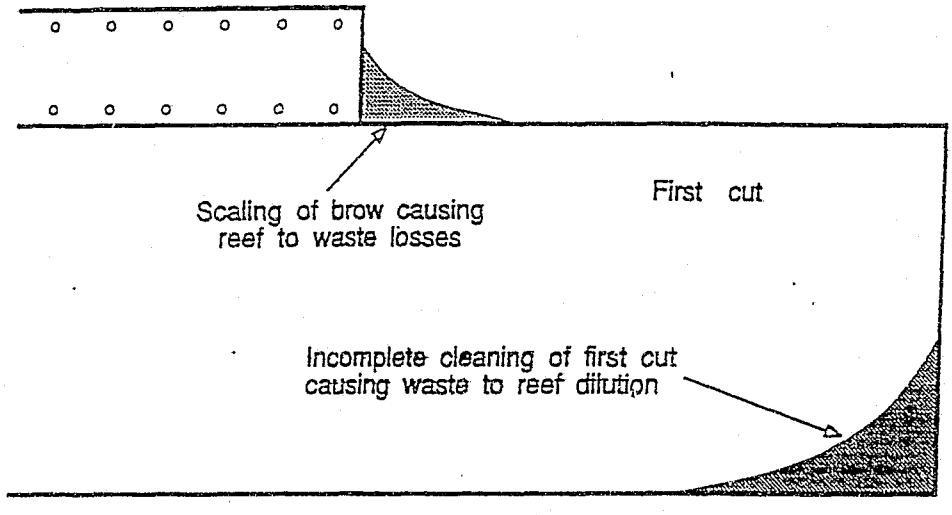
Comparison of channel width to stoping width with double cut

Figure 15.4



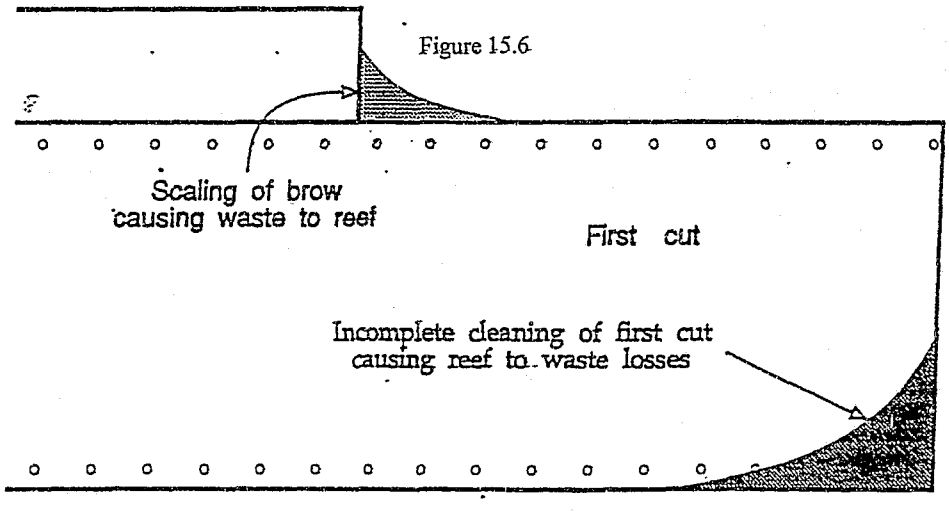
Cost comparison between dilution saved and additional waste broken.

Figure 15.5



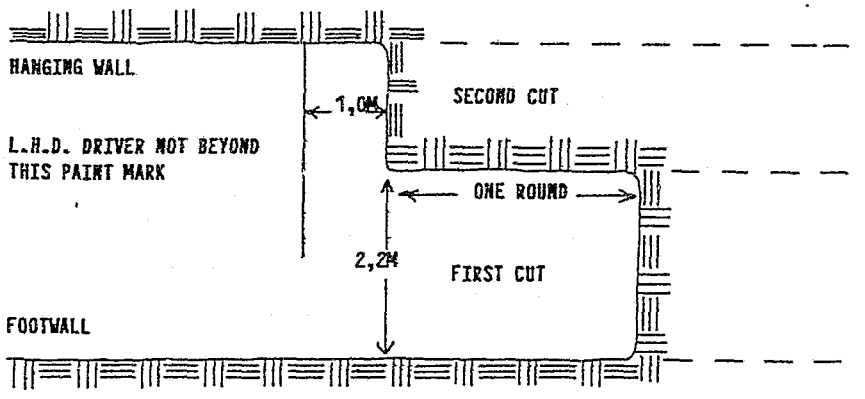
Reef losses and dilution on a narrow reef.

Figure 15.6



Reef losses and dilution on a wide reef

Figure 15.7



DOUBLE CUT CLEANING

1. OBJECTIVE

To clean double cut ends safely, using L.H.D.'s in double cut TM3 development.

2. PROCEDURE

2.1 Permanent support must be installed to within 1,0 m from the face before the round is drilled.

2.2 The waste cut is to be blasted separately.

2.3 The first cut is to be blasted a minimum height of 2,2 m to allow for L.H.D. cleaning.

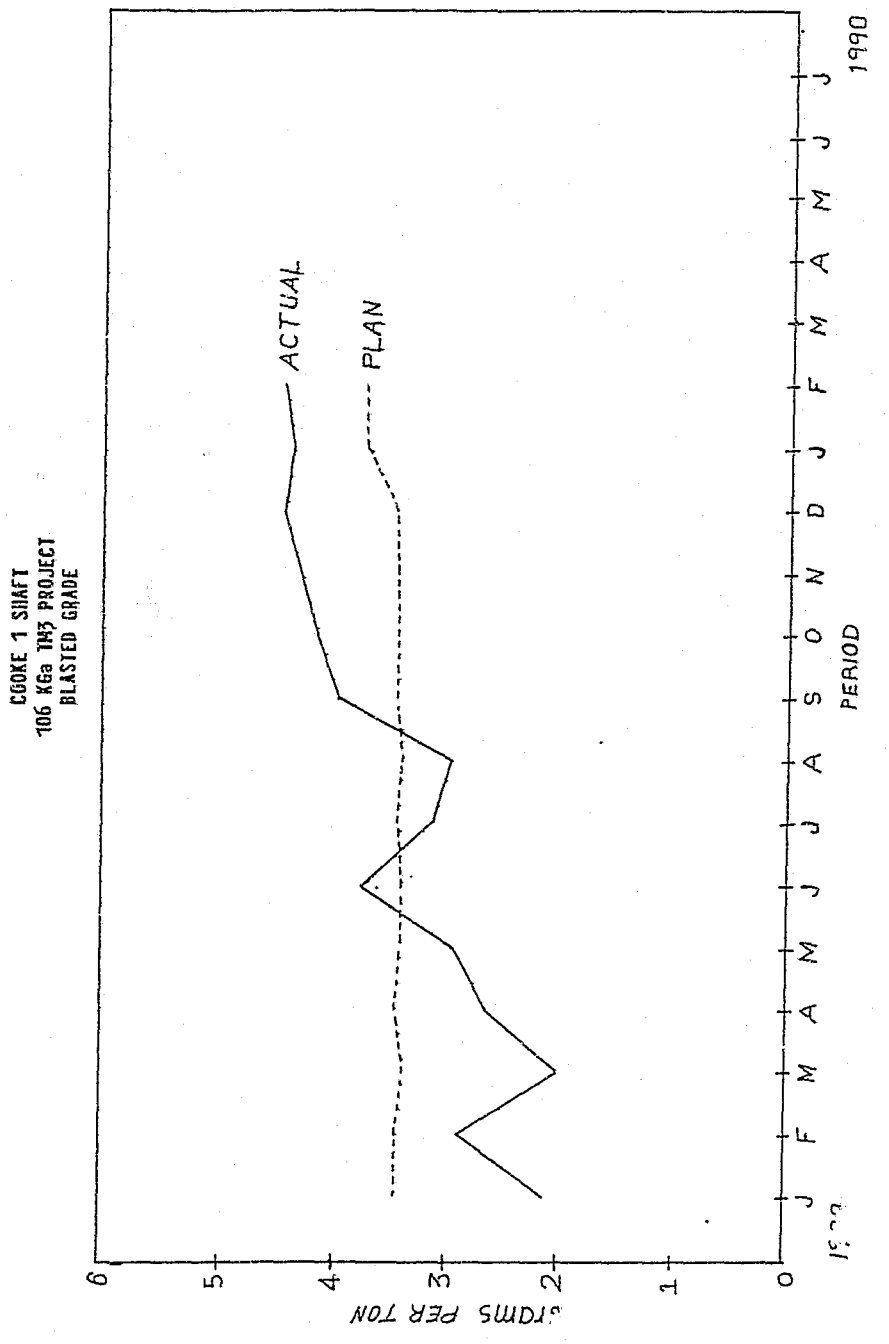
2.4 After blasting the first cut, the area shall be made safe by the appointed ganger, before cleaning operations begin.

2.5 A paint line is to be drawn on both sidewalls, 1,0 m back from the brow as a cleaning limit for the L.H.D. driver. The L.H.D. driver is not to pass beyond this mark with the front of his canopy, when cleaning the first cut.

2.6 The L.H.D. driver, who when operating his L.H.D. will not be closer than 1 metre to the brow.

pg/stddouble.dok

Figure 15.8


COOKE 1 SHAFT
106 KGa TM3 PROJECT
BLASTED GRADE
ACTUAL
PLAN
GRAMS PER TON
0
1
2
3
4
5
6
J F M A M J J A S O N D J F M A M J J
1990
PERIOD
INCREASE IN BLASTED GRADE WITH
INTRODUCTION OF DOUBLE CUTTING

The greatest advantage of double cutting is to improve the grade of ore to the mill and within the parameters as set out in the paper by Strauss H., it is viable.

15.8 **Faults**

People were well aware of the extremely faulted nature of the area being mined and many more faults and dykes or off shoots of dykes were encountered than expected. It was not uncommon to suddenly find that after the blast that the reef is down or partly left in the hanging.

In part the problem was that some of the faults were scissor faults so that having traversed the fault some distance away where it appeared as a disconformity with no throw suddenly there is a displacement. Frequently the first disconformity is only located by the Geologist when tracing the fault.

Because faults split or joined together, the original extrapolation from the previously mined areas was not totally accurate and played havoc with the mining layouts. At times the miners were expecting to lift or drop 2 to 3 metres to pick up the reef, from information at hand, only to find that the reef was little displaced or that the throw on the fault was 2 or 3 times as much.

Some of the displacements shown on plans was later found to have been estimations at the time of intersecting these in the previous stoping areas and was very inaccurate.

It was a credit to the Geologists and Survey department that so little disruption took place.

As the result of the amount of unexpected encounters the Geologist spent a lot of time in the area locating and mapping the faults and giving advice to the mining crew.

In hindsight and when the next remnant is planned the existing faults will be re-examined by the Geologists to establish the displacement accurately as well as the influence of dykes and faults. Even the minor faults are to be identified as in an

area where scissor faulting takes place, the displacement can then be predicted more accurately.

Even with additional input surprises can be expected and the team must react rapidly to minimize the disruptions.

CHAPTER 16

Conclusion

16.1 Introduction

At Randfontein Estates the statement of "Safety before production" applied throughout and to a large extent was the reason for REGM's excellent safety performance.

It can be stated unequivocally that the extraction of the pillars in the two areas under review have been done extremely successfully. The methods applied are somewhat different to past pillar extraction. The methods applied varied considerably and were selected because of peculiarities applicable to that operation.

The thorough planning done during the initial phase and subsequently to each phase of the operation contributed in no small amount to the operations success.

Pillar extraction is usually a tedious and hazardous occupation. Certainly with advancing technology in the likes of computers with specific software that is applicable to mining, the ability to do computer simulations and predict problem area and take the necessary steps, means safer mining. The continuous attention of the Rock Engineering department both as to physical inspections and predictions was of enormous benefit.

16.2 95 Haulage Pillar

The dedication of the team resulted in a very successful completion of the project. The training of the people and keeping the attention on safety aspects resulted in no serious injuries during the total project.

With the amount of faulting, the existence of fractured ground and the striving for higher productivity really means that there was not safer or better method than using Trackless Mining equipment.

16.3 101 Area

With the complexity of faulting and the problems that had been encountered earlier a lot of detailed planning was necessary to execute the project. The Rock Engineering and Geological departments were vital members of the team and guided the miners through the many problems and potential problem areas.

The Engineering department maintained high standards throughout. The project team is of the opinion that had trackless mining not been applied to this area, it could not have been extracted as successfully. Certainly, the cost of development would have been far higher and some of the areas that were extracted, by the method applied, would not have been economical to mine by conventional methods. The ability of the trackless mining system to mine below the lowest level was an asset.

16.4 **General**

The realizing of the companies assets forms part of every Managers operation and therefore every endeavour must be made to recover the gold locked up in pillars as economically as possible.

Every mining area must be reviewed on its merits and no fixed rules can be laid down for the extraction of pillars. The guidelines as referred to earlier are of significant benefit and provide valuable information which needs be considered in the planning of the operation.

APPENDIX 2

Extract from the Recommendations of the Rockburst Committee of 1964

1.Where un pay areas are known to exist mining should be towards these.

2. Stoping layouts to be planned such that stress concentrations are minimized. This refers in particular to shape and sequence of Mining.

3. Ensure that main accesses and other major / important excavation are, wherever possible, overstoped or are located in zones of minimum stress. This refers to pump stations and settlers, sub stations, etc.

4. Appoint a Remnant Officer who will report at least monthly on such matters as:-

   4.1 The approach of working places to likely areas of high stress, including dykes and faults.

   4.2 Areas of high stress likely to develop as a result of mining operations, particularly with reference to the possible effects on main access ways.

   4.3 The formation of remnants as well as losses of ground due to faulting.

5. In areas that may be prone to rockbursts, longwall stoping where practical, should be applied.

6. The formation of pillars or remnant abutments should be avoided as far as possible. In the interest of safe working conditions, some dykes or un pay ground may have to be stoped.

7. Dykes left as strip abutments between stoped out areas are liable to burst and to disturb or affect the stability of the surrounding areas in the same way as remnants. Wide dykes may be left to serve as natural support, provided travelling ways through them are adequately supported.

8. Support should be of a nature and density that will withstand sudden shock. Support should be maintained as close to the face as practical.

9. Where practical, a strike face advancing up-dip should be used for remnant extraction as this provides better and quicker access and egress. This configuration allows for greater non-violent dissipation of energy than tip faces advancing on strike.

10. The narrower the stoping width the better and reduction in roof sag can be achieved by waste filling.

11. Face advance should be regular and continuous.

12. Remnants should be extracted with all practical speed by continuous work.

13. Labour should be kept to a minimum.

14. Adequate escape ways must be provided.

15. The hanging wall portion, of the face, should be kept in advance of the footwall portion – do not allow overhanging faces to develop.

16. The manager must issue special written instructions with regard to safety precautions, working methods, support and other pertinent activities.

17. Incline shafts and service inclines / access ways should be situated in the footwall and be overstoped even if this has to be done in country rock.

18. Development of footwall drives and haulages should be in overstoped ground, if this is not possible, they should be so positioned in relation to unstoped areas that they are situated where the stress levels are not excessive i.e. preferably remote *from remnants or future remnants.*

19. No pillars or remnants should be left or be allowed to remain above, below or adjacent to crosscuts, drives or haulages.

20. Where tunnels are likely to be subjected to high stress an active support should be installed in these tunnels during excavations.

21 In the case of very high stresses, reef development should be kept to a minimum as it creates zones of intensified stresses.

Comparison of development requirements for a conventional mining layout and a trackless mining layout for the 101 area

95 Haulage Pillar

This area being almost totally surrounded by mined out area initially appeared to be a ventilation, offers nightmares, however, once the situation was investigated it was found that air from the 101 level stoping was upcast through the old areas and although it contained some pollutants in the form of carbon dioxide and some fumes from trackless equipment, the overall quality of the air was very good. In addition air could have been drawn directly from the downcast shaft but the air was needed in other locations and it was decided to use the upcasting air.

Ventilation of the development was done using 760mm vent columns. Once the haulage had been cleared out, a good quantity of air was got to flow through it and the development therefore only required very short columns which provided very comfortable working conditions and diesel fume emission never became a problem.

101 Area

The ventilation problems were to prove to be far more intense in this area with the ventilation doing unpredictable things when holings were made into same or other old working. Ore passes were connected to different levels, stopes and cross cuts were found to link into the existing system on either intake or exhaust sides and sometimes when you thought the air was going to do one thing it did the opposite or nothing at all. it is a tribute to the Cooke 1 ventilation department that they kept the conditions generally to a very high standard despite the vagrancies of the system.

The mining and ventilation people communicated on a daily basis and it is this close cooperation, sometimes at night when the night shift crew found the system doing something different, that ensured that no serious problems were encountered.

The area is served by a trackless vehicle suite of some 1480kw rated diesel power. This amount of power simply had to increase the generation of heat and exhaust gases into the air. Some authors (Fourie, 1988) indicated that the total heat generated by a trackless fleet would be similar to that as generated by equipment in conventional mining for the same tonnage output.

This is not as straight forward as that as the variables of tramming distances could result in more or less trucks being applied as well as the gradients will vary the heat input per ton/kilometre. In fact Middleton 1989 stated that localized buildup of heat could be significantly worse with large diesel powered units referring particularly to development but TM[3] mining at times is as much development, all be it on reef, as it is stoping. Patterson, 1989 indicated that 0,06 to 0,08 m^3/s of air per kilowatt of rated engineer capacity needs be supplied at the pointof operation.

Current JCI planing parameters call for an air dilution factor of $0,12m^3/s$ of fresh air per rated Kilowatt of diesel power for the dilution of Nitrosis Oxide and the other noxious emissions associated with diesel combustion units. This planning rate is high as it is known that $0,1m^3/s$ of fresh air keeps concentration within acceptable limits and in fact with the start of the project $0,06m^3/s$ was supplied per rated kilowatt but at times gas levels neared the critical levels and the volumes were increased to ensure no undesirable exposures.

The area is ventilated with $220m^3/s$ of air, of which 75% is directly from the downcast shaft via 90, 101 and 106 levels. The balance of the air is re-used air which is received via the old workings from other areas. A fortunate point for the people at Randfontein Estates Cooke 1 Shaft is the relatively shallow depth and the low heat pickup thus enabling the air to be re-used.

The overall air factor for the project calculates as $0,15m^3/s/kW$ or $10m^3/s/kiloton/month$. As a comparison where air is related in conventional mining to monthly tonnage produced the standard factor of $3,3m^3/kiloton/month$. The high air factor supplied is partly due to the vagrancies that result when holings into old workings occurs and in part may be described as some factor of safety.

As the areas are served by multiple roadways varying generally between $16m^2$ and $20m^2$ in area, the main tramming routes are defined and the air distributed such that velocities in the region of 2m/sec are maintained in these access ways. This is primarily done to accommodate tramming speeds of the vehicles in order to prevent the vehicles travelling in a plug of fumes which would not only make the temperatures very uncomfortable but would likely raise gas concentrations to unacceptable levels.

Primary air distribution and control within the project is done mainly by the installation of ventilation seals both permanent and temporary. Permanent seals are predominantly constructed from reinforced concrete poured into shuttering. Temporary seals consist mainly of gumpoles, gumplanks and polypropylene curtaining. Permanent seals has been a problem in all trackless layouts and a cost effective and quickly installed seal is still under investigation. Concrete blocks either crush or are damaged by the blast, bagged material does not
stay in place, Gunnited brattices peals off or leaks very rapidly and the concrete cracks when closure occurs.

Obviously air controls are also required where machinery is required to move through. In cases such as these door frames are constructed from which split conveyor belting is suspended in an overlapping curtain which forms a very effective flexible air control system. Because of the vast amount of controls, both permanent and temporary required in a project like this a construction crew comprising a supervisor and eight assistants are continuously employed in maintaining and constructing ventilation controls.

Where old mined out areas are traversed, polypropylene vent curtaining is sued to course the air through the area. These areas can cause a lot of lost ventilation if neglected.

The development of the roadways and the stoping drifts are ventilated by means of 762mm diameter galvanised ducting and using 45kW axial flow free fans supplying between 14 and 20m^3 of air to the working face. Because of the limiting air quantity vehicular capacity in these headings are limited to 150kW which is also the rating for the LHD currently used in the area.

Dump trucks are loaded at the nearest point of through ventilation. Utility vehicles are only allowed to operate in the headings when there is no LHD in the area being supplied by the same ventilation column.

Holings into the old workings are effected from time to time and used as return airways either using the natural draw of the main fans or where necessary one or more 760mm diameter fans are utilised. With the system operating as described wet bulb temperatures of ± 27.5C maintained.

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