CHAPTER 1

Twenty Five Years of Mechanisation of Tabular Orebodies in South African Gold and Platinum Mines

An Introduction

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Twenty Five Years of Mechanisation of Tabular Orebodies in South African Gold and Platinum Mines: An Introduction

1. Introduction

In this first chapter of this treatise it is the intention to provide details of the qualifications and experience, specifically related to mechanised mining, of myself, K.A.Rhodes (KAR) the candidate. In addition, a brief introduction will be given of my pioneering work related to trackless mining on South African gold mines in the 1980's and later on platinum mines in the Rustenburg area of South Africa.

1.1 Higher Education Qualifications

I attended the University of Leeds in England in the 1950's and was awarded a B.Sc. (Honours) degree in mining engineering. I hold a British First Class Managers Certificate for coal mines and South African Mine Managers Certificates for both coal mines and metalliferous mines. I have also been a Professional Engineer (Engineering Council of South Africa) for more than 40 years. Refer to **Appendix 1.1** at the end of this chapter for details of my qualifications.

1.2 Experience and Career Achievements

Since leaving university I have had six decades of experience in the mining industry. For more than forty years I have held senior management and consultancy appointments in southern Africa in diverse mining operations on coal, copper, gold and platinum mines. Refer to **Appendix 1.2** at the end of this chapter for a more detailed summary of my practical experience.

In 1953 I was awarded a National Coal Board (NCB) scholarship to attend university. After graduation and on completion of a three year directed practical training programme with the NCB there followed several years of junior official positions before my final appointment as an under(ground)manager. During this time significant mechanisation experience was gained at the coal face.

In 1964 I left England to work on the Zambian Copperbelt where I gained my first experience of mechanisation on a metalliferous mine. This was followed by appointments in South Africa on highly mechanised coal operations, and in South West Africa, now Namibia, at a newly developing copper mine where mechanised methods were being used. This experience on mechanised mining led to my pioneering trackless mining in South African gold mines in the 1980's. At Randfontein Estates Gold Mine (REGM) I successfully motivated for and then introduced a full range of trackless machines for a flat dipping tabular orebody (so called wide reef) and subsequently for narrow reef stoping, also at REGM. Based on this experience at REGM, when I moved to the new H.J.Joel Gold Mine I was responsible, as mine manager, for the design of the first totally trackless gold mine in South Africa. All these projects were planned and managed by myself from the outset and these achievements have been documented in published technical papers which are attached to this chapter as appendices and are entitled as follows.

Appendix 1.3: ***"The Use of Nonel at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited"*** K.A.Rhodes, Association of Mine Managers of South Africa, 1986.

Appendix 1.4: ***"Wide Reef Mechanised Room and Pillar Operations at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited"*** K.A.Rhodes, Association of Mine Managers of South Africa (AMMSA), 1986. This paper was awarded the gold medal by AMMSA for 1986.

Appendix 1.5: ***"Planning for a Trackless Access Stoping Operation in Narrow Reef Conditions"*** K.A.Rhodes, presented at a JCI Technical Meeting, 1986. This pioneering paper was submitted to the Association of Mine Managers of South Africa for publication in 1986 but was later withdrawn by JCI, the parent company, for confidentiality purposes.

Appendix 1.6: ***"Shaft Sinking and Mid-Shaft Loading Operations at H.J.Joel Gold Mine, Orange Free State, South Africa"*** K.A.Rhodes, The Mining Engineer, The Institution of Mining Engineers, August 1988.

Appendix 1.7: ***"The Design of a New Trackless Gold Mine"*** K.A.Rhodes, Association of Mine Managers of South Africa Trackless Mining Symposium, February 1988. This paper was awarded a special medal for the best paper presented at the symposium.

My experience on these projects underlined the necessity for the establishment of standard procedures for the operation of trackless equipment, including the need for driver discipline from the start of any such project, the commitment of mining managers to the engineering function and a 'hands-on' management style.

This treatise will further detail the work carried out by myself on the development of low profile equipment introduced to South African platinum mines and specifically my work at Amplats' new Waterval Mine where I was involved from the beginning in the design of the mine and on-going 'hands-on' consultancy work up to steady state production.

Finally, in this treatise there will be a detailed discussion of several new projects and trials of mining methods which came about from the direct involvement of myself; these included certain new projects for Amplats (Styldrift Mine and the Boschfontein Mines) and trials with long hole stoping methods and tunnel boring. I was the project manager at Bafokeng Rasimone Mine when a tunnel borer was used to develop a reef raise and the published technical paper on this project is attached as an appendix at the end of this chapter; **Appendix 1.8** "***Reef Development with a Tunnel Boring Machine on a South African Platinum Mine",*** by M.Stander, K.Rhodes, P.Horrell, D.Sammons, G.Harrison, J.Dean, presented at the 6th International Symposium on Mine Mechanisation and Automation, South African Institute of Mining and Metallurgy, Johannesburg 2001.

Since 1995 I have been an independent mining consultant and have worked with many clients, mainly in central and southern Africa. My consultancy work has focussed on the practical planning of new mechanised mines with specific emphasis on the management of trackless mining operations.

As a consultant I was also responsible for the design of a fully mechanised new underground gold mine at Shakisso in Ethiopia: the Legadembi Gold Mine. Trackless horizontal cut and fill was the selected method of mining. However, geotechnical investigations restricted the mining spans to be less than the orebody width and it was therefore necessary to provide for in stope pillars. Various in stope pillar and bay layouts (modified room and pillar layouts) were considered in order to determine the most favourable option. Refer to the technical paper "***Design of In Stope Pillars in Cut and Fill Mining for a Gold Mine in Ethiopia"*** by K.A.Rhodes and T.Rangasamy, published in the transactions of MassMin 2008 and presented at the 5th International Conference and Exhibition in Mass Mining in Lulea, Sweden in June 2008 and attached as **Appendix 1.9** of this chapter.

In the upcoming chapters in this treatise the trackless mining projects pioneered by myself, which proved to be successful on South African gold mines and platinum mines, will be described. However, as a mining consultant, other attempts to motivate for changes to new mechanised mining methods, such as long hole stoping and tunnel boring, will also be discussed.

All these projects carried out by myself, as the responsible manager or as a consultant, took place over a period of twenty five years.

CHAPTER 1

APPENDIX 1.1

Higher Qualifications: K.A.Rhodes

KENNETH ALEXANDER RHODES

Pr.Eng., B.Sc.(Hons) Mining

HIGHER EDUCATION QUALIFICATIONS

1. B.Sc. Honours in Mining Engineering awarded July 1958 by the University of Leeds, England.

2. British First Class Managers Certificate for Coal Mines: No.8638 dated February 1962.

3. South African Mine Managers Certificate for Coal Mines: No.2341 dated April 1967.

4. South African Mine Managers Certificate for Metalliferous Mines: No.2746 dated April 1972.

PROFESSIONAL MEMBERSHIP

Professional Engineer (Engineering Council of South Africa) since August 1970.
Registration number: 704177

ASSOCIATIONS AND INSTITUTIONS

1. Member of National Association of Colliery Managers, February 1963.
2. Ordinary Member (now retired), Association of Mine Managers of South Africa, 1976.
3. Member (qualified by examination) of the Institution of Mining Engineers, London, March 1965.
4. Member of the Institution of Mining and Metallurgy, London, September 1965.

CHAPTER 1

APPENDIX 1.2

Summary of Practical Experience:

K.A.Rhodes

SUMMARY OF EXPERIENCE

Consultancy Work from 1995 to Date

As an independent mining consultant KAR has worked with the following clients.

- Anglo Platinum: Design and project management of new mines, feasibility studies, audits and mechanised mining projects.
- Anglo American Corporation (Zimbabwe): Mining consultant for the new Unki platinum mine.
- Ashanti Goldfields (Zimbabwe): Consultancy for trackless mining work at the Freda Rebecca Mine.
- JCI Limited for Delta Gold: Mining consultancy work for the design of the new Hartley platinum mine in Zimbabwe.
- IMC Knight Piėsold Mining for Samancor: Chrome mine feasibility studies.
- Hernic Ferrochrome: Mine design work.
- Time Mining: Mine feasibility study for a gold mine in The Yemen.
- JCI Projects: Pre-feasibility study for an ultra-deep mine for Western Areas Gold Mine.
- Placer Dome Western Areas JV: Trackless costs benchmarking.
- Anglo Platinum/Bomar: Project Manager of TBM reef raise project at Bafokeng Rasimone Mine.
- PGM (Canada): TBM advisory consultancy for a new platinum project in South Africa.
- TWP Consultants: Mining consultant for Waterval UG2 Project.
- Knight Piėsold for Assmang Limited: Feasibility study for Dwarsrivier Chrome Mine.
- Anglo American Corporation: Mining consultant for an energy balance study for different mining methods at platinum mines.
- De Beers Canada: Consultancy work for the new Snap Lake Mine.
- BHP Biliton: Consultancy work at Samancor Western Chrome Mines.
- Meridian Securities: Consultancy work at President Steyn Gold Mine.
- MIDROC Gold Mines: Project Manager for the design of a new underground gold mine at Legadembi, Ethiopia.
- Barplats: Consultancy for the selection of a new trackless fleet of equipment at Crocodile River mine.
- Konkola Copper Mines (Vedanta): Trackless consultancy for a large copper mine in Zambia.
- Saumya Mining Joint Venture: Consultancy work for the design of a new uranium mine for UCIL in India.
- Kamoto Operating Limited: Consultancy related to a maintenance action plan for trackless equipment at a large copper mine in DRC.
- South Deep (Goldfields): Consultancy related to trackless mining costing.
- Anglo Platinum: Technical advisory work for trials of ultra low profile trackless equipment at Amandelbult Mine.
- Zimplats: Consultancy for the introduction of a new fleet of trackless equipment at Ngezi Mine, Zimbabwe.
- Anglo Gold Ashanti: Part of a consortium looking at means to mechanise operations and introduce automation at ultra deep levels.

Appointments with Johannesburg Consolidated Investment Company (JCI) between

<u>1973 and 1995</u>

<u>1994 – 1995:</u> <u>Platinum Division, JCI Head Office</u>
Permanent appointment to JCI Head Office as Consulting Mining Engineer in the Platinum Division. Primary responsibilities were the preparation of several feasibility studies for new mining projects in terms of a Strategic Planning Initiative for the Platinum Division. Following unbundling of JCI into three separate entities, namely Amplats, JCI Limited and Johnnic, status was that of Consulting Mining Engineer for Amplats (Anglo American Platinum Corporation Limited) later to become Anglo Platinum.

<u>1991 – 1994:</u> <u>Mine Manager East Mine, Rustenburg Section</u>
Mine Manager responsible for all operations. Rustenburg East Mine was a very large underground mine exploiting narrow reefs. Between 1991 and 1993 new longwalls were established and with a substantial increase in the use of hydraulic props, safety and productivity improved. In addition, new mining methods with revised development layouts were introduced by KAR including downdip mining.

<u>1989 – 1991:</u> <u>Platinum Division, Head Office</u>
Appointed to be responsible for the planning of new shaft systems for ore reserve development at Rustenburg Mines Rustenburg Section, the largest platinum mining complex in the world. During 1991 technical evaluations were carried out by KAR on certain chrome mines in the Rustenburg district and technical reports were submitted. Following the acquisition of Purity Mine, KAR was appointed as Consulting Mining Engineer for the mine responsible to the Managing Director of CMI (an associated JCI Company).

<u>1988 – 1989:</u> <u>Mine Manager, Western Areas Gold Mine</u>
Responsible for all operations at the North Division of Western Areas Gold Mine. In order to reverse the losses being experienced following the installation of trackless equipment to the low grade orebody in 1985, specific objective plans were prepared by KAR and implementation commenced in 1989.

<u>1985 – 1988:</u> <u>Project Manager/Mine Manager, H.J.Joel Gold Mine</u>
On appointment to the H.J.Joel Project (a new development in the Orange Free State) a new mining plan was initiated by KAR to introduce a mechanised option utilising trackless mechanised mining methods in narrow reef conditions. This mine was the first gold mine in South Africa to be designed from the outset as a trackless mine; KAR's responsibilities were for the design, development and commissioning of the mine. The mine was commissioned in 1988 when KAR was still the mine manager.

Two shafts were sunk to depths of 1035 metres with mid shaft loading arrangements (MSL) established on two levels whereby development of the mine was carried out simultaneously with sinking. Of major importance to the management of this mine was the necessity to avoid an inrush of water to the workings which represented a real threat, the danger of methane and the complications of changing ventilation conditions during sinking and MSL development; this demanded a total 'hands-on' style of management.

<u>1983 – 1985:</u> <u>Manager Mining, Randfontein Estates Gold Mine</u>

Manager Mining for the Cooke 2 Shaft Complex at Randfontein Estates Gold Mine. At this mine KAR pioneered the use of trackless equipment in narrow tabular reefs on gold mines and in late 1983 initiated studies for a trackless mining operation to replace existing conventional scraper cleaned wide reef stopes and in 1984 successfully motivated the introduction of a full range of trackless equipment for a wide reef room and pillar operation, which commenced in July 1984. In late 1984 further motivations by KAR commenced for a narrow reef trackless mining operation; approved in 1985 and development work began immediately.

1982 – 1983: Project Mining Engineer, JCI Coal Division
Project Mining Engineer in the Coal Division with duties to carry out the following work.

- Feasibility study for the proposed Phoenix Opencast Project for export coal.
- The expansion of the Arthur Taylor Colliery, a highly mechanised underground mine producing coal for the export market. This work necessitated an extensive study of pillar extraction operations on coal mines throughout South Africa.

1979 – 1982: Manager Mining, Randfontein Estates Gold Mine
Manager Mining at the Old Randfontein Section of Randfontein Estates Gold Mine, responsible for all underground operations. The operations were mainly centred at SD 32 Shaft where narrow reefs varied from steeply inclined (near vertical) to flat dipping with extensive faulting. Very poor gold and uranium values necessitated the most stringent management control of the operations with continuous labour reductions in order to constantly improve productivity. In 1981 KAR re-opened several old shafts after many years of being idle and these shafts were returned to profitability albeit on a small scale.

1977 – 1979: Underground Manager, Rustenburg Platinum Mines, Rustenburg Section
Underground Manager for Townlands Shaft at Rustenburg Section. Longwall mining had just been introduced and this appointment provided the opportunity for KAR to successfully develop this method of mining achieving high face advances which came about from the establishment of new layouts and specific control systems developed over a two year period by means of 'hands-on' management.

1976 – 1977: Manager Mining, Elsburg Gold Mine (Western Areas Gold Mine)
Appointment to Elsburg Gold Mine (later to be merged with Western Areas Gold Mine) as Manager Mining responsible for underground mining operations. In this period several major fires occurred at Western Areas and significant experience was gained in bringing these fires under control: this work was exacerbated by the complex mining layouts necessary to exploit the extensively faulted multi-reefs.

1976: Assistant to the Consulting Engineer (Coal), JCI Head Office
A feasibility study was carried out by KAR for the Middelburg Uitkyk Coal Prospect; this study provided for a large mechanised colliery serving the export market and was planned for both underground and surface mining methods. The study was completed in six months by KAR and initial planning experience was gained in opencast mining.

1974 – 1976: Project Manager/Underground Manager, Otijhase Copper Mine, SWA

Appointment at Otijhase Copper Mine, a developing greenfields copper mine in South West Africa (now Namibia). Over a two year period KAR was responsible for the development of the mine from outset to first production with the use of trackless mechanised equipment.

1973 – 1974: Mine Manager, Tavistock Colliery
Mine Manager at Tavistock Colliery and during this period KAR successfully introduced mechanised equipment to the bord and pillar operations for the first time at this colliery.

1973: Underground Manager/Mine Manager, Phoenix Colliery
Initial appointment in the JCI Group.

Appointments prior to employment with JCI

1972 – 1973: General Mining Corporation
Following the takeover of Coalbrook Collieries by Gencor, certain transfer appointments occurred as follows.
1972 – Relieving Mine Manager, Transvaal Navigation Collieries.
1973 – Assistant to Operations Manager (Natal).

1968 – 1972: Assistant Manager/Acting Mine Manager, Coalbrook Collieries, OFS
Assistant Manager at Coalbrook Collieries and for four years KAR was responsible for all underground operations. The mine, a major producer to Escom, employed mechanised bord and pillar methods, with conventional mechanised equipment and the only continuous miners in South Africa, working under difficult mining conditions: high inflows of water to be controlled; poor roof and sidewall conditions; seams liable to high rates of methane emissions and to spontaneous combustion; limited pitroom.

1967 – 1968: Technical Assistant, Cornelia Colliery, OFS
Appointed by the Anglo American Corporation as a Technical Assistant and assigned to Cornelia Colliery, OFS. Difficult geological conditions predominated at Cornelia and the mine was also prone to underground heatings and fires and practical experience was gained in the control and sealing of underground fires on coal mines.

1964 – 1967: Shift Boss, Mufulira Copper Mines, Zambia
Shift Boss at Mufulira Copper Mines, a very large underground mine which employed many different mining methods including open stoping (with and without sand fill), sub-level caving, block caving and the cascade system. At this time the mine was introducing trackless mechanisation in their sub-level caving and cascade methods and early experience was gained by KAR with the use of trackless equipment on a metalliferous mine.

1960 – 1964: Junior Underground Official, National Coal Board, United Kingdom
On completion of the directed practical training programme there followed a period of contract work at the coal face before a series of appointments as an underground official at various collieries. All experience gained with the National Coal Board was associated with the longwall method of mining in seams of varying thickness. Systems of mining included hand loading methods and fully mechanised operations including shearers, trepanners, ploughs and armoured face conveyors with hydraulic support. During these early years of

work this was the first opportunity for KAR to obtain experience in the transformation from manual methods of working to totally mechanised operations at the coal face; this experience was to prove invaluable.

1957 – 1960: Mining Engineering Trainee, National Coal Board

Three year training programme as a mining engineering trainee in the No.8 Castleford Area of the National Coal Board. During this extensive training period experience was gained at various collieries; this experience incorporated all aspects of underground operations and associated technical, administrative and management work.

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APPENDIX 1.3

"The Use of Nonel at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited

K.A.Rhodes

Transactions of the Association of Mine Managers of South Africa, 1986

APPENDIX 1.4

"Wide Reef Mechanised Room and Pillar Operations at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited"

K.A.Rhodes

Transactions of the Association of Mine Managers of South Africa, 1986

CHAPTER 1

APPENDIX 1.5

"Planning for a Trackless Access Stoping Operation in Narrow Reef Conditions"

K.A.Rhodes

August 1986

CHAPTER 1

APPENDIX 1.6

"Shaft Sinking and Mid-Shaft Loading Operations at H.J.Joel Gold Mine, Orange Free State, South Africa"

K.A.Rhodes

The Mining Engineer, The Institute of Mining Engineers, August 1988

CHAPTER 1

APPENDIX 1.7

"The Design of a New Trackless Gold Mine"

K.A.Rhodes

Association of Mine Managers of South Africa
Trackless Mining Symposium, February 1988

CHAPTER 1

APPENDIX 1.8

"Reef Development with a Tunnel Boring Machine on a South African Platinum Mine"

M.Stander, K.Rhodes, P.Horrell, D.Sammons, G.Harrison, J.Dean

6th International Symposium on Mine Mechanisation and Automation, Johannesburg 2001

CHAPTER 1

APPENDIX 1.9

“Design of In Stope Pillars in Cut and Fill Mining for a Gold Mine in Ethiopia

K.A.Rhodes, T.Rangasamy

MassMin 2008, 5th International Conference and Exhibition in Mass Mining, Lulea, Sweden, June 2008

CHAPTER 2

The Background to the Start of Mechanised Mining of Tabular Reefs on the Gold Mines of Johannesburg Consolidated Investment Company Limited

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The Background to the Start of Mechanised Mining of Tabular Reefs on the Gold Mines of Johannesburg Consolidated Investment Company Limited

In this chapter the candidate, K.A.Rhodes (KAR), sets out the background to his work, specifically related to the introduction of mechanised mining operations on the tabular reefs at the gold mines of Johannesburg Consolidated Investment Company Limited (JCI).

In August 1983 I was appointed as Manager Mining at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Ltd. (REGM) with responsibilities for all operations at the shaft. Immediately prior to my taking up this appointment a major fall of ground had occurred in a wide reef stope on the E8 Reef horizon and four mineworkers had been killed. The first time that KAR went underground at Cooke 2 Shaft it was to accompany the General Manager, Mr W.J. van der Meulen, on a visit to the 85N2 E8 stope where the multi fatality had occurred; the General Manager was to carry out a follow-up inspection of the stope on that day.

On completion of the inspection and whilst still underground, Mr van der Meulen had asked KAR what he thought of conditions in the stope to which KAR replied that it appeared to him that he had entered an 'underground quarry'. At the time it was what came immediately into my mind; as I remember there was an excessive amount of large broken rock throughout the stope indicating an ore clearance problem and hence my reference to a quarry. Refer to the photograph in **Figure 2.1** and other photographs in **Annexure 2.1** in **Volume 2** showing excessive rocks in strike gullies in other wide reef stopes (these photographs were not taken on the day of the visit but are typical of conventional wide reef stopes at the time). The other striking issue at the time was the obvious practical problem of drilling, barring down, making safe, charging up and the carrying out of construction work in the working height which was of the order of 3 metres; refer to the photograph in **Figure 2.2** and other photographs in **Annexure 2.2** in **Volume 2**, also taken at a later date.

FIGURE 2.1

Excessive Rocks in Strike Gulley in a Conventional Wide Reef Stope

FIGURE 2.2

The Need for a Ladder to Carry-out Construction Work in a Conventional Wide Reef Stope

Of course, what was really meant by my response to the General Manager was that under the conditions seen on the visit there would be more control over the safety of the operation if the operations were mechanised using equipment designed for the conditions. The suggested change to mechanisation was not that well received and was immediately challenged by the General Manager. I remember that he responded to my proposal by saying that in order to get large equipment through a small vertical shaft, along long rail haulages and into the stoping area would raise severe practical difficulties. I replied that I believed that it could be done. It was at that time I decided to set out a motivation and prove to the General Manager, Mr van der Meulen, and JCI Head Office that mechanisation was a viable option. There was going to be resistance, that I knew, but it is important to record that when Mr van der Meulen realised my determination to mechanise the E8 Reef at Cooke 2 Shaft (which I proved to him during the coming months) he became very supportive, notwithstanding his initial scepticism of my proposal.

In summary and with the benefit of hindsight, the timing of my appointment immediately following upon a multi fatality accident from a major collapse of hangingwall on the E8 Reef would lead to an opportunity for me to introduce mechanisation on that specific reef horizon at Cooke 2 Shaft.

2.1 <u>Early Background</u>

At Cooke 2 Shaft in late 1982 there had taken place a change in the method of mining for wider reefs. This change had enabled wider stoping widths, at lower overall grades, to be mined differently from the normal method of mining conventional stopes. The revised method for stoping widths between 2,5metres and 4,0metres would rely on pillars (a type of bord and pillar layout) for support with rock bolting between the pillars. The original conventional systems had relied on grout based packs and stick support. This change, although still employing conventional methods, had initially been introduced on the UE1A Reef and after several months of employing the method on that reef the new method had been shown to be both productive and safe. The method was then adopted for the underlying E8 Reef. Although, until July of 1983, no falls of ground or unstable hanging wall had been reported in the E8 stopes a sudden collapse, spanning several pillars, occurred in the 85N2 E8 stope. In my mind, on that very first visit underground at Cooke 2 Shaft, it was clear from the outset that the use of mechanised trackless equipment could improve both productivity and the overall safety of the operation. The conclusion reached at that time by myself was based on my previous experience with mechanisation. At this point in the chapter it will be useful to consider some of this experience in brief detail.

2.1.1. ***Experience with Mechanisation***

Following graduation in the late 1950's from the University of Leeds, England, I (KAR) completed a three year programme of training (Directed Practical Training) with the National Coal Board, United Kingdom. This was at a time when manual coal face work was being phased out and mechanisation of face operations was taking place rapidly in the nationalised industry. This was therefore the first opportunity to be part of a major transformation at the coal face from the old manual methods to mechanisation. Significant experience was gained over a period of several years as an official in charge of mechanised face operations and at Under(ground) Manager level.

On leaving the National Coal Board in the mid 1960's, experience was gained on the Copperbelt of Zambia, where at Mufulira Copper Mine, one of the largest underground mines in the world at the time, KAR had first-hand experience of mechanised trackless equipment in hard rock mining. In fact, this was the real beginning of mechanised operations in metalliferous mining world-wide. However, although the availability to the industry of a full suite of equipment for drilling, loading, transport and other ancillary operations was still a long way off this initial exposure to mechanisation on a large scale on a large metalliferous mine was invaluable.

In South Africa in the late 1960's into the early 1970's KAR managed a highly mechanised operation at Coalbrook Collieries in the Orange Free State; at this time very little mechanisation was being practised on South African coal mines. In addition to the use of conventional mechanised equipment, Coalbrook was the only coal mine in South Africa using continuous miners. The employment of such equipment instilled in a manager the necessity to develop a hands-on management style which was to prove essential for KAR in later years in the development of new mechanised projects in South African gold mines.

In 1973 KAR joined the Johannesburg Consolidated Investment Company and was appointed colliery manager at Tavistock Collieries and was immediately involved in the conversion of Tavistock from a hand-loading labour intensive operation, which was the widely accepted system prevalent at that time in the Witbank Coalfield, to a completely mechanised mine. This was when there were still very few mechanised collieries in South Africa.

In the mid 1970's KAR was transferred, within JCI, as the responsible mining manager to a new mine at the start of its development: Otjihase Copper Mine in South West Africa (now Namibia). Otjihase had changed its original mining policy from labour intensive methods, similar to gold mines in the group, to increased mechanisation utilising drill rigs and LHDs. KAR was, therefore, the manager responsible at the time mechanised mining was introduced at Otjihase.

The above brief references to previous experiences that KAR had before 1983 with mechanised mining on various mines provided the platform to enable him to focus on mechanised options at Cooke 2 Shaft when he was transferred there, within the JCI Group, in 1983.

2.2 <u>Arguments for Wide Reef Mechanisation at Cooke 2 Shaft, REGM.</u>

Following the underground visit by KAR to the 85N2 E8 stope where the multi fatality had occurred, the most striking argument from the outset was safety. It is axiomatic that in wider orebody operations where conventional labour intensive methods are employed, it becomes more difficult to maintain safe working conditions as the mining height increases. However, when mechanised mining is practised in the same circumstances, the safety of persons at the face is markedly improved when mechanised equipment designed for the conditions is used.

During 1983, the first year that wide reef conventional mining was first introduced at Cooke 2 Shaft, there had been many serious accidents from falls of ground and also persons falling off ladders and drilling platforms. In this respect refer to **Table 1**.

Table 1

Working Place	Date	Nature of Injury	Description of Accident
95 NIE8	21.1.1983	Suspected fracture right hand dorsal	Struck by rock whilst barring hanging
	25.6.1983	Laceration wound right index finger	Struck by rock from hanging whilst drilling
	14.7.1983	Loss of one upper tooth (handling of equipment)	Struck by jack whilst fastening same
	29.10.1983	Laceration left cheek and loose teeth	Struck by rock from hanging whilst cleaning holes after drilling
	25.11.1983	Contused right wrist	Struck by rock from hanging whilst drilling
85 N2 E8	10.3.1983	Laceration wound right dorsum foot	Struck by rock from hanging whilst drilling
	23.7.1983	Fatal	Major fall of ground
	23.7.1983	Fatal	Major fall of ground
	23.7.1983	Fatal	Major fall of ground
	23.7.1983	Fatal	Major fall of ground
	23.7.1983	Compound fracture right tibia Closed upper right tibia Laceration lateral and medial side right foot	Caught by hanging whilst fastening eye bolt
	28.7.1983	Cornea left eye	Struck by rock from face whilst drilling
	4.11.1983	Contused right shoulder	Struck by rock from hanging
	4.11.1983	Fractured pelvis	Struck by hanging whilst fastening prop
106 N1 UE1A	6.8.1983	Laceration wound left middle finger	Struck by rock whilst lashing
	22.8.1983	Contused back	Fell from ladder whilst installing roof bolt
	30.9.1983	Severe laceration upper arm	Struck by rock from hanging whilst drilling
	1.10.1983	Contused right foot	Struck by rock from face whilst drilling
	11.11.1983	Medial malleolus – severe laceration heel and severed Achilles	Struck by rock from hanging whilst barring
106 N1 E8	15.10.1983	Contused lumber region	Slipped and fell from platform whilst drilling
	20.9.1983	Laceration forehead and upper lip	Injured by rock from hanging whilst drilling

In addition, and notwithstanding that the change to a bord and pillar mining method from grout based packs and sticks had enabled the mine to exploit more profitably the low grade ore reserve, it was nevertheless believed in late 1983 by KAR that further significant improvements in productivity were possible by the employment of mechanised equipment in the stopes.

At the time, in the mind of KAR, there were clear arguments for a trackless mechanised option. Firstly, in conventional mining a footwall grid has to be developed before stoping on the reef horizon can commence and this is a lengthy process. Access from a footwall crosscut off the main haulage by means of a travelling way up to the reef horizon has to be established for every stoping connection; only then is a centre gulley raise developed on reef following which ledging and and construction work for winches and grizzlies takes place. In addition, orepasses have to be developed from the footwall crosscut to serve the stope. In the trackless operations envisaged by KAR, once the reef horizon has been accessed all operations would take place on reef and from the developed declines (winzes) stoping operations could commence; there is no ledging and construction phase. Some boxhole development would still be necessary for LHD's to tip; this changed with the use of trucks tramming to a single tipping point, but this came later in the project. Build-up of reef production could therefore be expected to be more rapid with the trackless mining option. In other words, when the full fleet of equipment was working on the reef horizon KAR likened the concept at the time to a mechanised 'panzer division' which could move freely on the same horizon at will.

Notwithstanding that the footwall development for a mechanised method would be minimal there would be obvious advantages on the reef horizon related specifically to drilling, support work and cleaning. Face drilling would be more easily controlled for reason that there would be only a few drill rigs in use compared to the many rock drill operators required for conventional mining. Also, more accurate drilling with drill rigs would result in less damage to the surrounding strata. Although no real change would occur in the support requirements for both the conventional and trackless options the means of installation would be far more effective using trackless equipment such as roofbolters, and a marked improvement in quality of support would occur. With regard to the cleaning phase, the use of LHD's would without doubt prove far more effective than scraper winches.

Therefore, in late 1983 KAR decided to prepare motivations for a trackless mechanised alternative in wide body conditions. These series of motivations would be based on the following key factors. Firstly, there would be an improvement in productivity due to a reduction in manpower. Secondly, profits would increase due to the reduced labour complement; a significant reduction in development costs; the partial elimination of ancillary operations. Finally mechanisation would provide for a safer operation.

2.3 Motivations for Trackless Mining

These motivations would lead to the approval of the Cooke 2 Shaft E8 Reef Mechanised Project; details of this project will be set out in Chapter 3 of this treatise.

The successful start-up with mechanisation of the wider E8 Reef at Cooke 2 Shaft encouraged KAR to consider the mechanisation of the much narrower UEIA Reef at Cooke 2 Shaft and in a following chapter, Chapter 4, the motivation for and the introduction of narrow reef mechanisation on the 95 Level at Cooke 2 Shaft will be set out.

Following the successful two years of introducing mechanisation at Cooke 2 Shaft REGM KAR was transferred to the new HJ Joel Gold Mine to be developed by JCI in the Orange Free State. This new mine had initially been designed and planned as a conventional gold mine but would, in fact, become a new trackless mine designed, planned and managed by KAR from the outset. Full details of the development of the HJ Joel Gold Mine will be addressed in Chapter 5.

CHAPTER 3

The Wide Reef Mechanised Mining Project at Cooke Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited

CHAPTER 3

The Wide Reef Mechanised Mining Project at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand, Limited

On the 4th November 1983 at REGM's offices K.A.Rhodes (KAR) made a brief presentation to the Technical Director of JCI on a proposal for a mechanised operation at Cooke 2 Shaft. In this presentation it was stated as the intention to present a report which would motivate the introduction of a trackless mining operation on the E8 Reef horizon. This presentation was to be followed up with a series of motivational reports over the next three months.

The initial report was very short and was presented later in November and it was followed by a more updated report in mid-December. The final provisional motivational report for the approval of the project was presented in January 1984.

In the course of this chapter these preliminary motivational reports will be examined. Further, the so-called Phase 1 of the project will be described leading up to the approval of the final Phase 2 of the project and finally the on-going build-up to the planned production from the project during 1984 and 1985 will be discussed.

3.1 Initial and Interim Report

Following the above meeting with the Technical Director, a short interim report on the proposal was submitted on 23rd November 1983 to REGM's Consulting Engineer. In this report it was stated that the mechanised operation would be carried out on the E8 Reef horizon between 85 Level and 95 Level in the north eastern portion of the lease area of Cooke 2 Shaft. In this area the reef varies in width between 2 metres and 5 metres and with the dip from 2° to 10° in an easterly direction. It was contended that a highly mechanised trackless operation based on a bord and pillar mining layout was feasible.

In a general overview of the proposed operation the following aspects of the project were briefly set out.

3.1.1 *Geology and Geological Reserves*

The geology of the area of E8 Reef which had been targeted was relatively undisturbed. The reserve available had been calculated as 4,219 million tons at a grade of 4,6g/ton. Assuming a geological loss (10%) and a loss for leaving regional pillars (4%) and an extraction of 92% within stopes, the actual mineable reserve was estimated at 3,118 million tons at a reef width greater than 2 metres.

3.1.2 ***Rock Mechanics Considerations***

The important rock mechanics considerations were stated as follows.

The necessity to overstope the E8 Reef horizon was important. The UEIA Reef lies 20 to 40 metres above the E8 horizon and therefore the highest possible extraction of the UEIA Reef had to be attempted. The percentage extraction of the underlying E8 Reef would only be maximised if pillars on the UEIA Reef could be minimised.

Any regional pillars on the UEIA Reef horizon would have to be superimposed on identical regional pillars on the E8 Reef horizon.

A sequence of primary and secondary extraction was seen to have advantages. Initially pillars would be developed larger than required and therefore there would be a high safety factor during primary mining. Bord spans during primary mining were recommended not to exceed 10 metres in width. However in a two stage extraction these spans could be exceeded during secondary extraction when the mined area would be abandoned and barricaded off on retreat.

When the E8 Reef was overstoped the E8 horizon would be de-stressed and consequently pillar loads would be low. Notwithstanding the necessity for computer modelling it was considered feasible for pillar widths to be designed for 1 to 1,5 times the stoping width and an extraction rate of 92% would then be possible.

3.1.3 ***Design and Planning***

It was envisaged that steady state production would be 40 000 tons per month operating a double shift. At the reserve stated, the life of the operation was estimated in excess of six years.

Access to the mining area would be from 90 Level and all reef development and stoping operations would be carried out with trackless equipment. All trackless equipment would have to be transported through the shaft and along 90 Level haulage to the mining area. The diameter of Cooke 2 Shaft and internal shaft steelwork would dictate that equipment would have to be stripped on surface and re-assembled underground at an inbye assembly bay at the end of the rail haulage in 90 Level North 11 Crosscut.

The overall mining layout on reef would provide for winzes (downdip roadways) developed on true dip at 150 metre centres following which bord and pillar mining would take place on strike. Actual dimensions of the stopes would be determined after computations by rock mechanics but bord widths would not exceed 10 metres during primary mining.

Ore tramming on the mining horizon would be to orepasses developed at 150 metre intervals down dip. Ore clearance would take place on 101 Level where the rail haulage was being upgraded; this haulage would need to be extended to establish an effective ore clearance to the shaft using trolley line locomotives and high capacity hoppers.

In terms of the required ventilation, multi blast conditions would have to be provided for: blasting twice in a 24 hour period. The total volume of air to satisfy the necessary criteria was estimated at $135m^3$/sec: the criteria to be considered were the production rate, diesel exhaust fumes dilution, heat removal and multi blast requirements with early re-entry periods.

Trackless equipment for the project was detailed: LHDs, drill rigs, roofbolters and utility vehicles. For improved effectiveness it was proposed that electro-hydraulic drill rigs be considered, notwithstanding the increased strain this would place on maintenance skills. At the time of this motivation trucks had not yet been considered by KAR.

3.1.4 ***Engineering Considerations***

At this initial phase of the motivation process it was clearly understood by KAR that there were important and critical factors related to the engineering aspects of a trackless operation. Trackless mining was going to be a new concept for all the officials on the project, both mining and engineering, and it was important that they should realise that, from the beginning, they would have a steep learning curve to climb and the control and management of this operation had to be grasped from the outset. If this was not realised and understood from the very beginning then the project would, in the opinion of KAR, fail.

Some of the important engineering factors at the time can be stressed.

Workshops

Workshop facilities would be provided for on 90 Level in close proximity to the access ramp from 90 Level to the E8 Reef horizon. The establishment of these workshops would commence as soon as the project had been approved.

Maintenance

Maintenance had to be done in accordance with strict schedules and mining personnel had to have the discipline to enforce this.

Fuel Supply

Fuel supplies had to be readily available; diesel transported by rail mounted tankers. In fact, a far more streamlined system was to be planned for later.

Stores

An underground store had to be established in the workshop area.

Service

An efficient back-up service from the original equipment manufacturers (OEMs) had to be put in place. It was critical that the availability of the equipment had to be high if the project was to succeed.

Training

Training of operators, artisans and supervisors would commence prior to commissioning of the equipment underground. OEM's would provide the training programme and supervise the training.

Although, in hindsight, these few basic considerations set out in this interim report in late 1983 are but a far cry from a maintenance action plan guide developed by KAR many years later, in 1983 they represented the early stance taken by KAR in order to ensure that the engineering function was being committed to by mining managers and supervisors. As the project grew additional factors would focus on operating standards for both mining (operators) and engineering (maintenance) and in turn would lead to the compiling of standards manuals.

This interim report then had briefly set out the principles of the project but the costs and efficiencies of the operation had still to be worked out and this would lead to a second (follow-up) report which was submitted in December 1983.

3.2 Second Report (Follow-up to the Interim Report)

This report, submitted by KAR on the 14th December 1983, was an updated follow-up report to the Interim report submitted in November 1983. Consideration was now given to a comparison of the proposed method and the current conventional layout in respect of efficiencies, working costs and safety.

3.2.1 ***Efficiencies***

In terms of efficiencies, it was estimated that the trackless project would provide for a productivity of 33 tons/non-skilled worker compared to the conventional planned efficiency of 7 tons/non-skilled worker: a reduction of 192 persons at a production rate of 40 000 tons/month.

3.2.2 ***Costs***

At this point in time an estimate was made of the working costs (stoping and development) for the trackless project and also a comparison with the actual costs of conventional mining. This cost comparison was still not definitive but did provide a comparative guide. At this stage the capital replacement costs and major overhauls had not been separated but were included in the overall estimated working cost for trackless mining. The comparative costs were estimated at R7,77/ton and R6,21/ton for conventional and trackless mining respectively. The comparative guide did therefore indicate that the working cost estimate for a trackless method would be markedly less (20%) than the actual conventional costs.

3.2.3 ***Safety***

With such a reduction in stope labour there was justification in claiming that in mechanised operations, with the lower complement of workers exposed to the conditions, there could be less accidents from falls of ground. Also, with mobile equipment designed to operate in higher workings the persons injured from falling from fixed ladders and drilling platforms could be eliminated.

In summary then it could be stated that for the trackless project the geology of the target area on the E8 Reef was favourable for trackless equipment and it was now proposed to employ a room and pillar layout. Further, it was argued that the proposed change to mechanisation would prove to be significantly more efficient in the use of labour and there would be a reduction in working costs. Finally, the method was safer and a reduction in accidents could be expected.

3.3 **Final Preliminary Motivation Report**

The final preliminary report, submitted by KAR on 31 January 1984, provided more details of the project and enforced what had previously been stated that a mechanised trackless operation would be both feasible and viable and also safer. There were some changes from the Interim Report and a summary of the main aspects of this motivation can be set out below. KAR's own copy of the original ***Final Preliminary Motivational Report*** (with some random highlighting and notes by KAR) dated 31st January 1984 can be seen in **Annexure 3.1** in **Volume 2.**

3.3.1 ***Geology***

The target area comprised the largest block of potentially payable E8 Reef at Cooke 2 Shaft. The reef in the area was given to be relatively undisturbed and the general dip of the fan shaped body was between 2° and 10°.

3.3.2 ***Reserves***

The actual mineable reserve was now estimated at 3,050 million tons at a reef width of 2 metres and greater.

3.3.3 ***Rock Mechanics Considerations***

It was re-iterated that the percentage extraction on the E8 Reef horizon would only be maximised if maximum extraction of the UEIA Reef took place. This conclusion was the result of observations in the E8 stopes at Cooke 2 where a highly loaded pillar on the UEIA horizon had necessitated that larger pillars be left on the underlying E8 Reef in order to ensure stope stability on that horizon. Also there had been a pillar failure in the 85N2E8 stope (the stope where the multi fatality had occurred previously) due to an overlying pillar on the UE1A Reef; refer to **Figure 3.1** for a photograph. Once again it was stated that all regional pillars on the UEIA horizon would be superimposed on identical regional pillars on the E8 horizon.

It was now planned that mining would take place in two stages; this policy being confirmed when the mining method changed from bord and pillar to room and pillar. A sequence of primary and secondary extraction has the advantage that the primary operation when mining on advance will provide for larger pillars with an overall high factor of safety but can allow for larger spans when on retreat during the secondary extraction phase. Pillar sizes on primary mining would be 7 metres x 10 metres and following a secondary extraction stage on retreat the final pillars would be 5 metres x 5 metres; these sizes had been agreed, for the two stages of extraction, with the Group Rock Mechanics Engineer. Overall extraction was now estimated at 90%; refer to the calculation below.

Room width	= 10 metres
Pillar size	= 7 metres (wide) x 10 metres long
Holing between pillars	= 4 metres
Final pillar size	= 5 metres x 5 metres

Therefore final extraction after secondary mining on retreat is as follows

$$\frac{\left[(7 + 10) \times (10 + 4)\right] - (5 \times 5)}{(7 + 10) \times (10 + 4)} \times 100\%$$

$$= 89,5 \text{ (say 90\%)}$$

FIGURE 3.1

Pillar Failure in 85N2E8 Stope Due to Overlying Pillar on UE1A Reef

3.3.4 ***Mining Design and Planning***

In terms of the final mining design and layout, steady state production, expected to be achieved during the second year (1985), would be unchanged at 40 000 tons/month for six years.

Development of the area would commence on 90 Level elevation. Initially contour reef drives would be developed from 90 N11 crosscut and from these drives decline winzes (access ramps) would be developed on true dip at approximately 150 metre centres. When an access ramp holed into a bottom access airway/travellingway only then would stoping commence. Dimensions of the drives and declines were designed to be 8 metres wide x 4 metres high.

The method of mining selected for this operation was the stepped room and pillar system. Rooms would be developed 5° down dip of true strike with access holings 60° down dip of strike being developed at 14 metre centres. The general development showing contour drives and access ramps or declines, the general stoping configuration and detailed stope layout are shown in **Figures 3.2, 3.3 and 3.4** .

Secondary extraction would be carried out when primary stoping in any winze connection is complete. During secondary extraction operations partial extraction of pillars would take place on retreat; pillars being reduced in size in stages to minimum dimensions. The stages in the partial extraction of pillars are shown in **Figure 3.5,** signed off by the Group Rock Mechanics Engineer. A fuller explanation of these stages is given in the Final Preliminary Report.

At this point I can refer to a personal visit KAR made to England in June 1985, when time was spent at the International Mining Exhibition in Birmingham and the application of a remote radio controlled system (ToroTel) for use with Toro LHD's was discussed with A.R.A. (Toro) engineers. The use of this system was envisaged for the project to provide for maximum extraction during secondary stage stoping when final cleaning of the blasted pillars would take place.

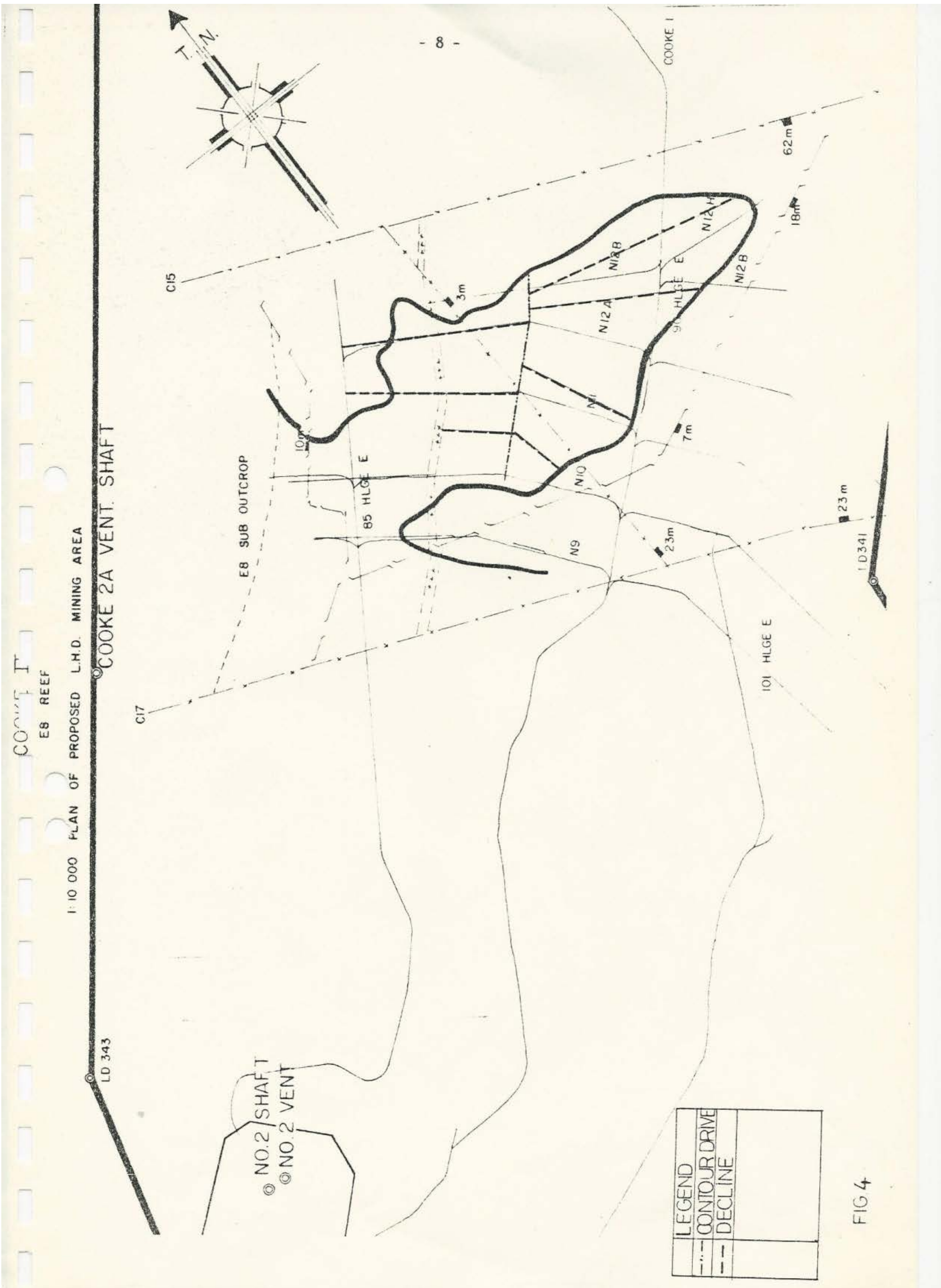

FIGURE 3.2

Contour Drives and Access Declines

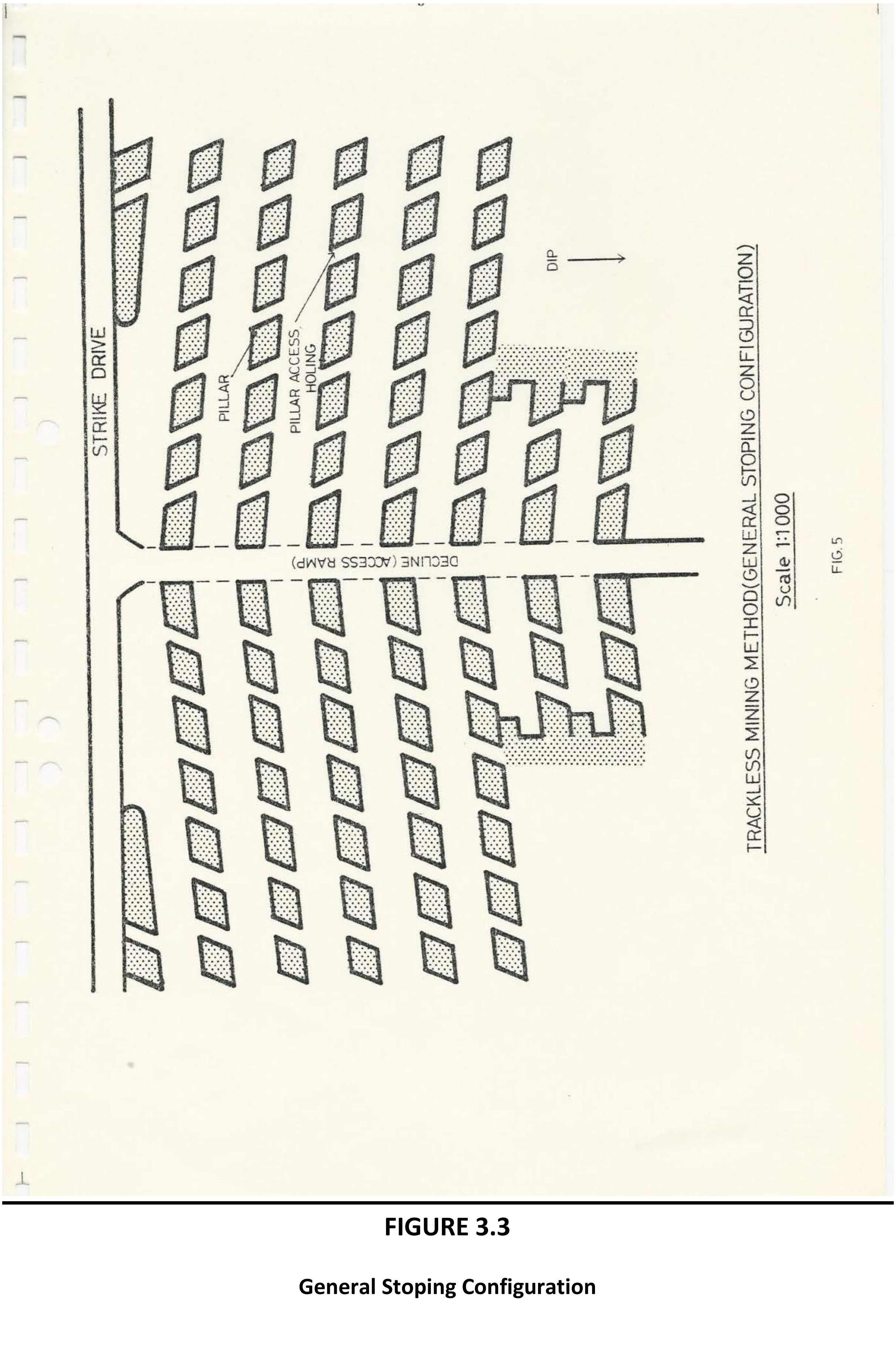

FIGURE 3.3

General Stoping Configuration

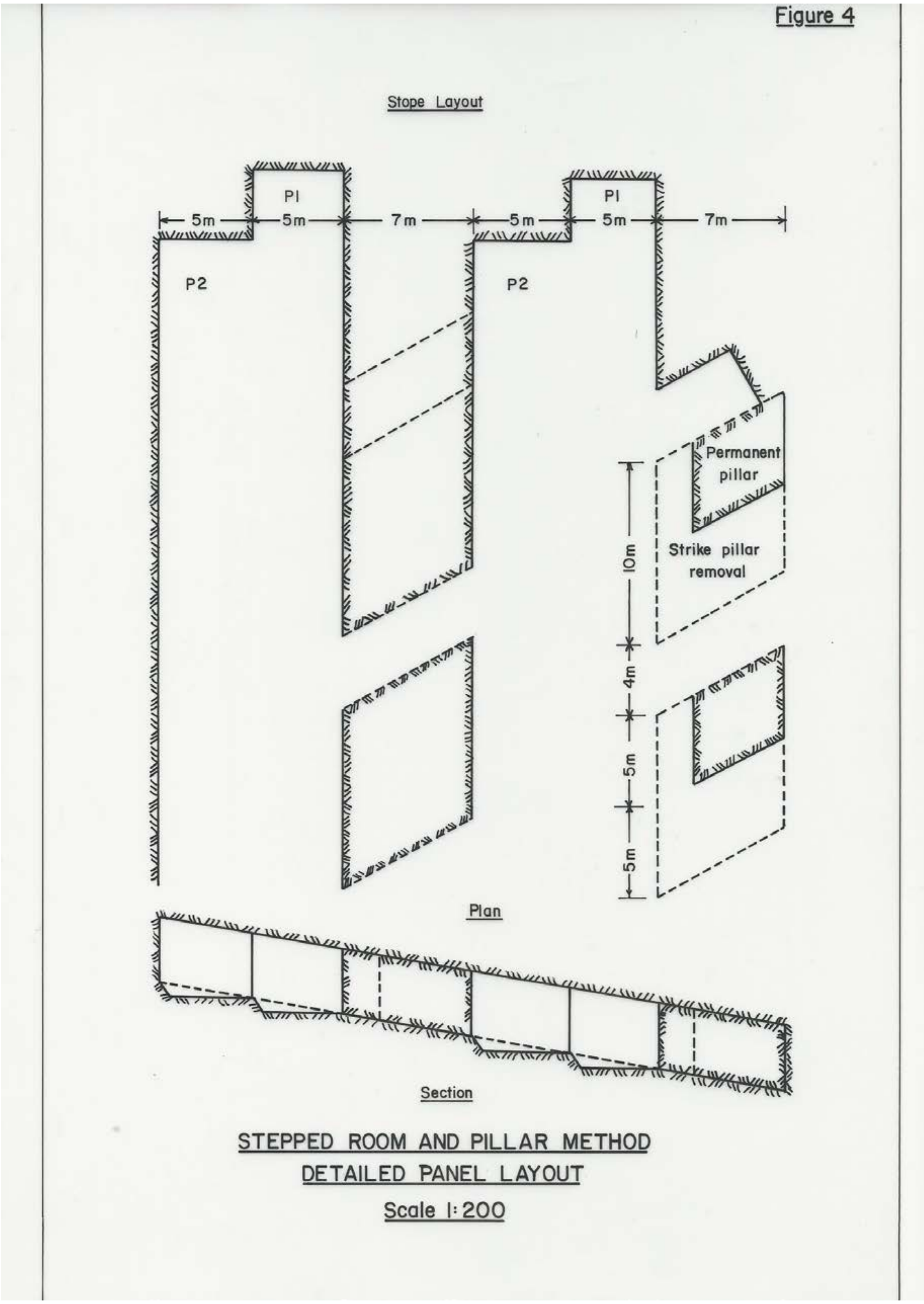

FIGURE 3.4

Detailed Stope Panel Layout

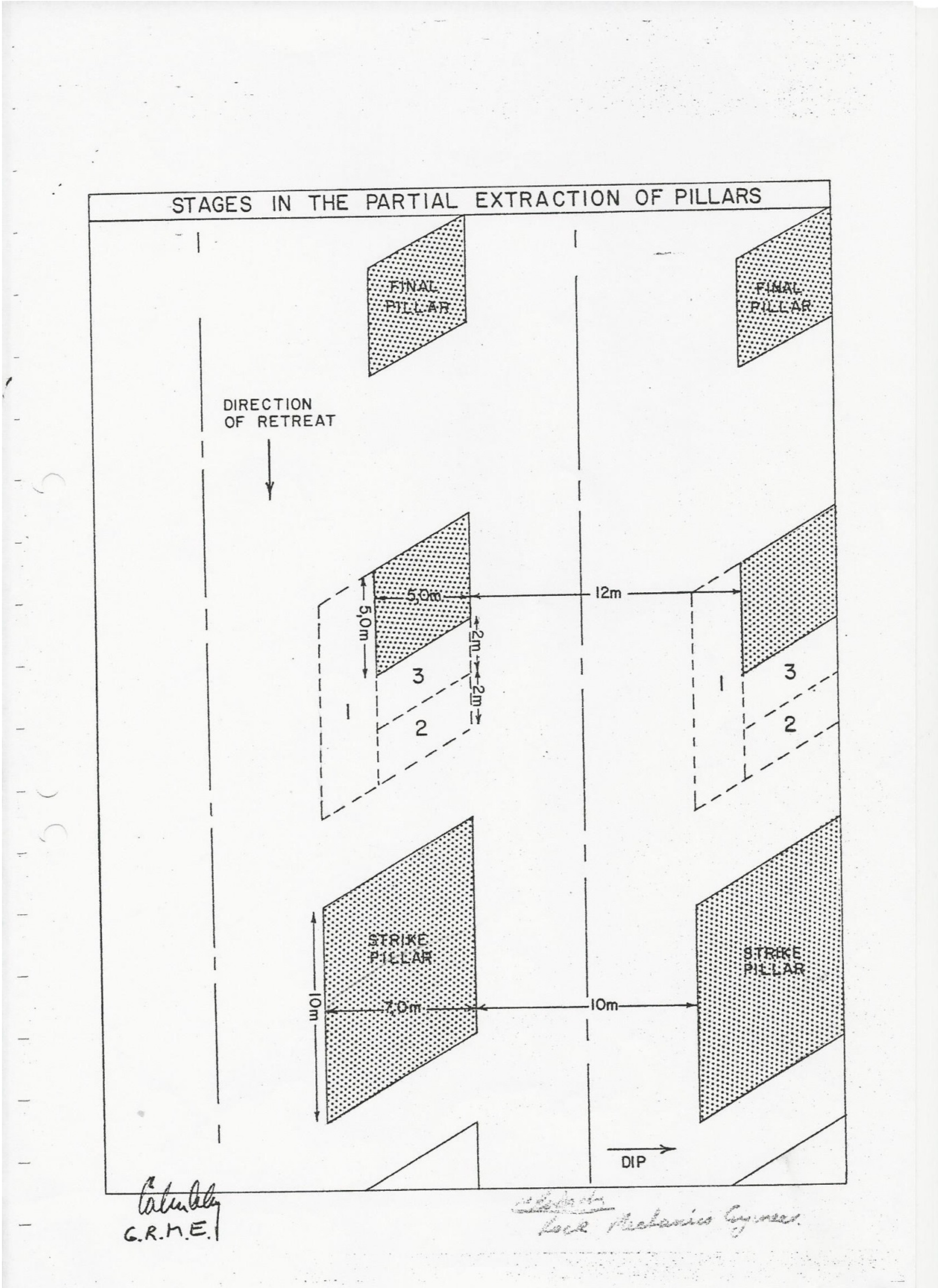

FIGURE 3.5

Stages in the Partial Extraction of Pillars

3.3.5 ***Cycle of Operations***

Further details were now set out in this final preliminary motivational report related to the cycle of operations during stoping.

Drilling and Blasting

The decision had now been made to use electro-hydraulic (two boom) drill rigs for stoping; they were also planned for the development work. A further decision was made to use Nonel short period delay detonators (SPD's) in stoping. The face shape would be staggered providing for a leading panel with a slashing panel lagging by approximately 5 metres. It was expected that by using Nonel SPD's that most of the reef blasted on the slashing (lagging) panel would be thrown into the lower leading panel. In the leading panel the use of the proven Nonel long period delay detonators (LPD's) would be in use.

The use of Nonel will be discussed in greater detail later in this chapter.

Cleaning

Trucks were now being envisaged by KAR for this project and the reasons for this decision will be seen in more detail later in this chapter.

Cleaning operations would be carried out by 2,7m^3 LHD units (consideration would be given later to larger units) into 16 ton trucks. Loading of trucks would take place in the access ramps where a height of 4 metres would be available, the LHD unit tramming to the access ramps on strike. Loading of reef by LHD would mainly take place on the lower leading panel, any reef left on the slashing panel being transferred only to the lower panel in order to prevent any machine slipping over the edge of the slashing panel.

The trucks would transport the reef to a main tip up the access ramps and along strike haul roads (reef drives). These roadbeds would be prepared using crushed stone from development operations and concreted where necessary. The haul roads, developed at 8 metres wide, would allow two vehicles to pass each other without the necessity for passing loops.

Final transfer of ore to the shaft system would take place by locomotive haulage on 90 Level and 101 Level.

Support

The recommended support for the stopes (and development) was 2,7 metre x 25mm end anchored resin rebar on a 2 metre x 2 metre pattern. During secondary extraction, being on retreat, it would not be necessary to install any support when pillars were reduced in size.

3.3.6 ***Ventilation***

Few changes to the interim report were necessary for the ventilation design. The total volume of air for the project, including double shift multi-blast stoping, would be 140m^3/sec. Stope faces would be ventilated by air jet fans (see a photograph in **Figure 3.6**). The air jet fans were planned to work in conjunction with force and exhaust fans and columns; refer to **Figure 3.7** for general stope ventilation. The Environmental Control Department at REGM had now compiled a detailed report of ventilation requirements which provided for a three hour re-entry period after the blast.

3.3.7 ***Equipment, Workshops and other Engineering Aspects***

At this time details were made available for the equipment schedule for a build-up to full production. However, planning was still in progress for the streamlined haulage on 101 Level for final clearance of ore.

Workshop facilities would be provided for in close proximity to the 90 Level elevation reef development. Development of a permanent workshop would take place immediately the first LHD unit was made available. The workshop would provide for two major bays (initially A, and when in full production A + B). Refer to **Figure 3.8** in this chapter, taken from the Final Preliminary Motivation Report. This would in fact be the No 1 Workshop in the final design.

FIGURE 3.6

Air Jet Fan in Stope

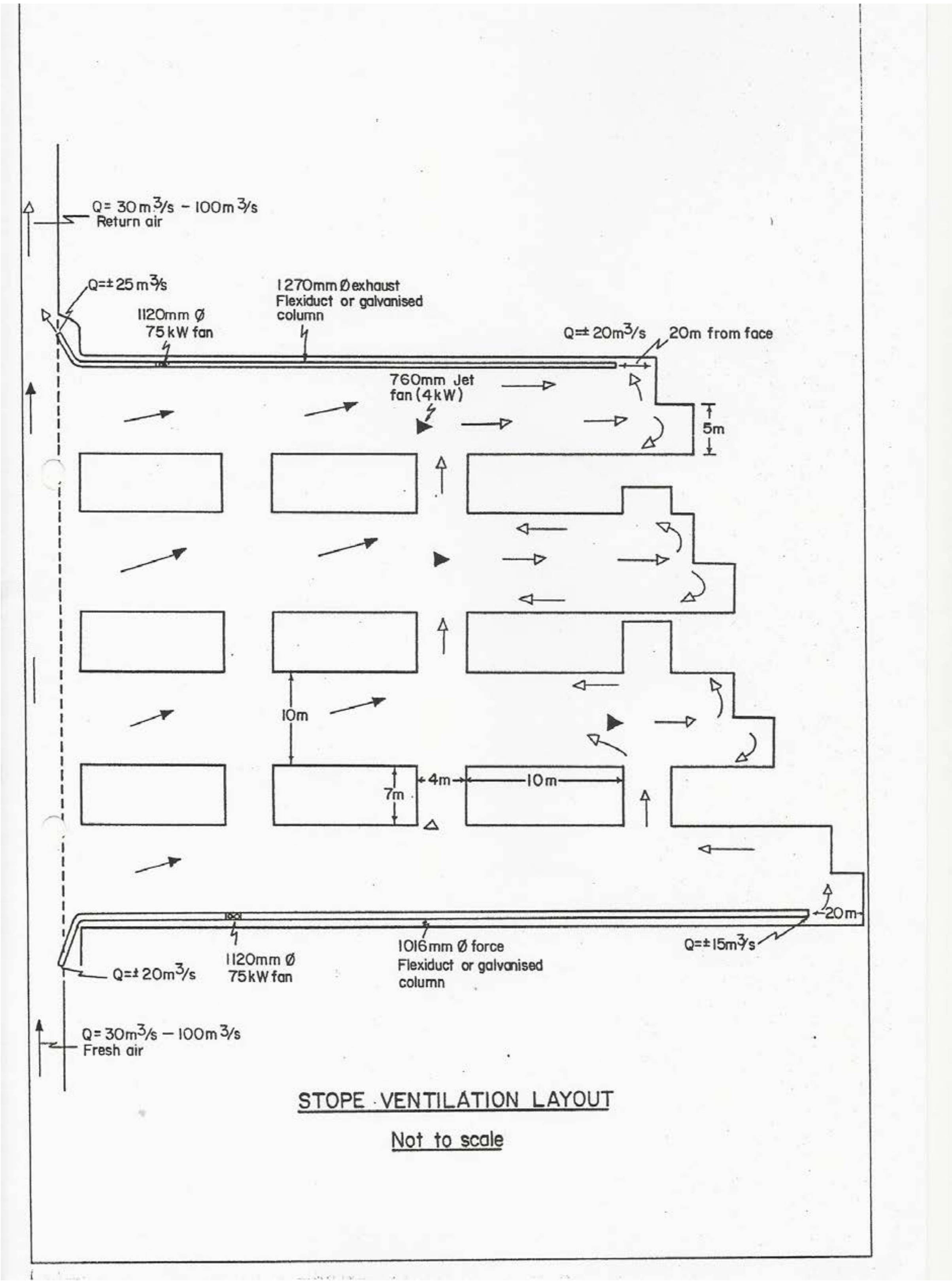

FIGURE 3.7

General Stope Ventilation Layout

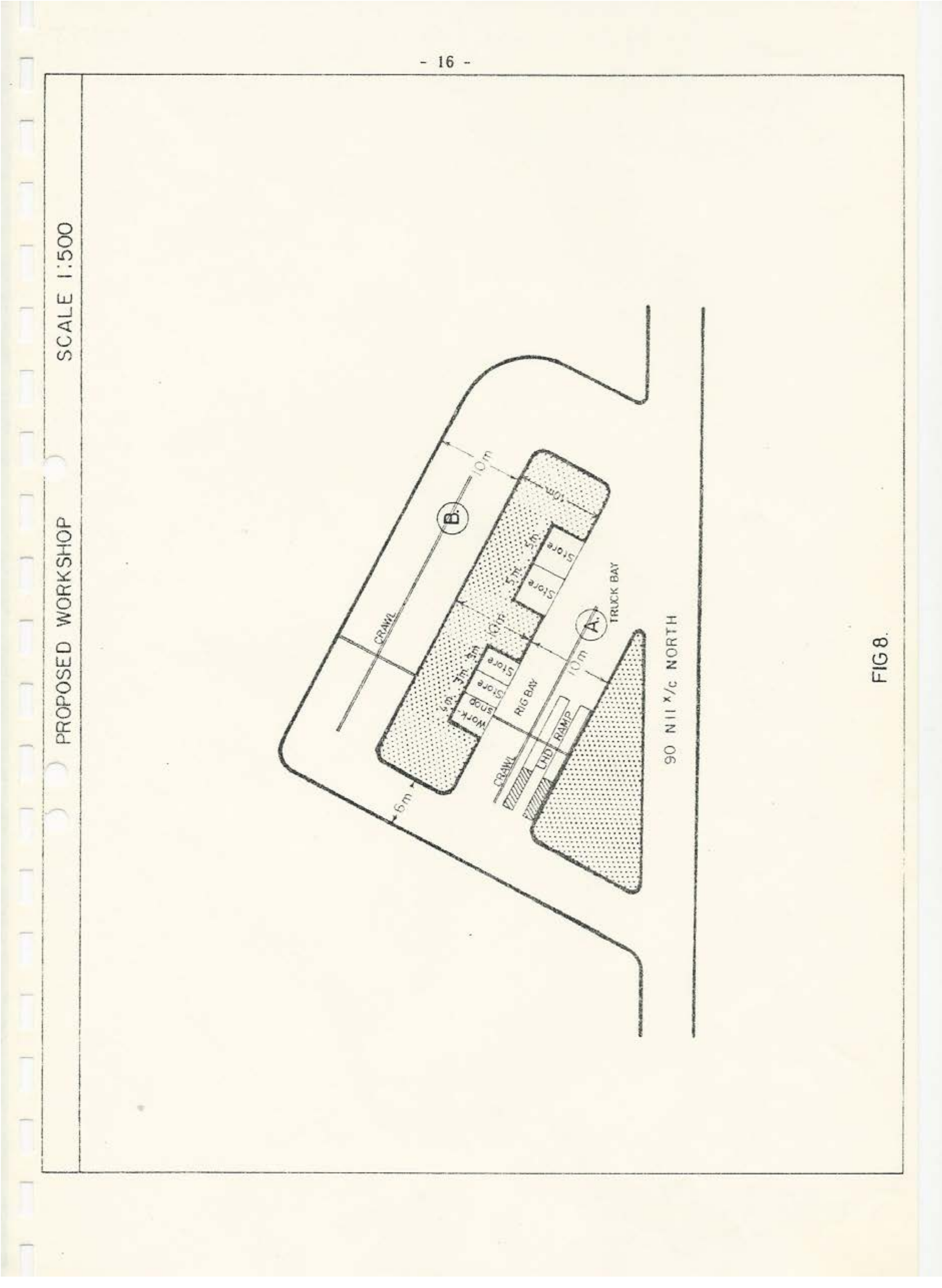

FIGURE 3.8

Proposed No 1 Workshop

The proposed workshop complex was extensive and would be developed in the footwall of the E8 Reef horizon and it was therefore axiomatic that rock mechanics aspects be considered. Full recommendations from the rock mechanics engineer for the support of the workshop and pillars to be left on the E8 stoping horizon are referred to in the Final Preliminary Motivation Report.

In the initial period whilst the permanent workshop was being constructed a temporary satellite bay would be made available on the reef horizon.

In terms of fuel supply, KAR was now giving consideration to pumping fuel to underground storage tanks by direct pipeline from surface. However, initially fuel would have to be transported underground by rail-bound tankers.

With regard to the access of capital equipment into the underground workings, all equipment had to be stripped on surface before being transported down Cooke 2 Shaft and on the 90 Level haulage to a place where re-assembly would take place . Documentation which provided details of dimensions and masses of sub-assemblies of certain favoured equipment for the project had been made available, by the responsible manufacturers, to the project. This information confirmed what KAR had believed from the beginning that the proposed equipment could be transferred underground. Full documentation generated by the original equipment manufacturer (OEM's) was collated at the request of KAR by the Trackless Mining Project Mine Overseer and submitted to KAR.

3.3.8 ***Labour***

Full details of complements were now given in this report for both C.W.S. and N.C.W. personnel. As an explanatory note, at that time JCI had changed to a new nomenclature for what was previously seen as white and black labour: C.W.S. (common wage scale or skilled) and N.C.W.S. (non-common wage scale or unskilled) respectively.

3.3.9 ***Training***

Schedules were now being planned for all persons (CWS and NCWS) to be sent for training for the first full year (1984) of the project.

3.3.10 ***Efficiencies***

The labour planning for conventional wide reef stoping was 7,5 tons/NCWS per shift and in general actual performance did not exceed this figure. Therefore, in conventional stopes, it would be planned for a complement of 222 in order to achieve 40 000 tons production. Stope preparation crews and winch movers would probably necessitate a further 20 persons, giving a grand total of 242 NCWS workers.

A labour estimate for the trackless operation was 23 NCWS per shift stoping complement, detailed as follows.

Job Category	Complement
Drill Rig Operators (3 rigs)	6
ST 3½ yd^3 LHD Drivers (5 machines)	5
Truck Drivers	4
Team Leaders	2
Rock Bolt Helpers	5
Tip Attendant	1
Total per shift	23
Total for double shift operations is therefore	46
Assuming a crew of 4 for pipe construction	4
Total stoping complement	50

Such a complement would therefore provide for an efficiency of 33 tons per stoping employee per shift compared with a planned efficiency of 6.9 tons per NCWS per shift in the conventional calculation (planned figure adjusted to account for additional crews referred to above).

The proposed trackless system would therefore provide for a reduction of stoping labour of the order of 192 persons; such a reduction being a strong motivation for the introduction of a highly mechanised operation if consideration was given to the continuous escalating cost of NCWS labour and in addition, the obvious advantages of employing a reduced labour force.

A comparison of CWS labour for trackless and conventional mining show slight reductions in favour of trackless mining (details in the Final Preliminary Motivation Report in Annexure 3.1 in Volume 2).

These calculations therefore confirmed the statements given in the follow-up notes to the Interim Report.

3.3.11 ***Costs***

At this stage of the planning further calculations had been made (which were not to be the final figures) for a comparison of working costs for the two options: conventional and mechanised. The costs included development costs which would prove to be significantly reduced for the trackless operation. In this respect, it had been recorded that the footwall development required to develop the reserve for the 90 Level E8 Project would have been 10 000 metres if stoping was done conventionally.

Comparative Estimated Costs in January 1984

Operation	Cost R/Ton	
	Conventional	Trackless
Development	0,97	0,03
Labour	3,07	1,45
Drilling	0,74	1,29
Blasting	1,12	1,12
Cleaning	1,40	1,39
Support	0,60	0,60
Ventilation	0,06	0,06
Other	0,37	0,27
	8,33	**6,21**

3.3.12 ***Safety***

Once again the same arguments for a safer operation, as stated in the previous motivation reports, were stressed. A mechanised room and pillar operation would be safer for the conditions and accidents could expect to markedly decrease.

3.4 **<u>Final Comment on the Motivation of the 90L E8 Mechanised Project</u>**

This final preliminary motivational report concluded that a trackless mining operation on the E8 Reef was believed to be technically feasible and safe.

The geology of the area targeted on the E8 Reef was favourable and the selected mining method of room and pillar was a proven method. The report showed that the proposed trackless operation would be more efficient in the use of NCWS labour and the comparison of labour efficiencies for both NCWS and NCWS were favourable to trackless mining over the conventional methods.

The costs were estimated to be R2,12/ton less for the trackless option (R6,21 versus R8,33 for conventional). The expected savings in costs at a production rate of 40 000 tons/month would be in excess of R1 million (or in today's terms of the order of R12 million) per year.

In summary, over a period of three months KAR had set out in three reports the motivation for a trackless mechanised operation on the E8 Reef horizon at Cooke 2 Shaft, REGM. Each subsequent report contained further details of the project. On the 31st January 1984, the date of the final preliminary report, it was considered that sufficient motivation had been done to warrant the approval of an initial phase for the project.

3.5 **<u>1984: Phase 1 of the Project</u>**

In early 1984 work continued on the preparation of an application for a capital vote for what may be called Phase 1 of the 90L E8 Trackless mining project.

Following the submission of the Final Preliminary Motivation Report on 31st January 1984, a draft application for the capital vote was made on 16th February 1984 and finally revised on 30th March 1984. Approval for the capital, to be known as Vote 574, was authorised by the Consulting Engineer on 3rd May 1984. The capital equipment authorised in this vote, in addition to significant underground workshop equipment, was as follows.

LHD 2,7m³	2
LHD 3,8m³	1
Twin boom electro-hydraulic drill rig	2
Roof bolter	1
18 Ton truck	2
Toyota land cruisers	3
Utility scissors vehicle	1

In terms of the above fleet it can be seen that 18 ton trucks were now being planned for whereas in the very early motivations trucks were not included at all. The 18 ton trucks (upgraded from the 16 ton size quoted in the final preliminary report) matched well with the 2,7m³ LHD unit; four passes would fill the truck.

At the time, the use of any trucks underground in tabular bodies in gold mines did not command support at senior level in the company and therefore the decision to introduce trucks required significant motivation from KAR. In considering the use of trucks it had been determined that trucks would be markedly less costly to operate than LHD's. They were able to travel in the workings at twice the speed of LHD's and, therefore, it would be possible and cost effective to reduce the tramming distance of LHD's to the minimum by the use of trucks; in this respect LHD's should tip into trucks as close as practicable to the face. Also, this decision would provide for the further advantage of allowing the trucks to travel to a single tipping point where an impact breaker could be constructed and cause all the reef to pass through a single orepass to 101 Level. The haulage on 101 Level was being upgraded to transport all the reef from the trackless operations, including the new 95 Level UEIA Project. This 'super' haulage and the new 95 Level UE1A Project will be discussed in Chapter 4.

However, notwithstanding all the obvious necessities for the documentation and official approval of any project through a motivational process as has been described up to now, it can be recorded that there was an increase in the confidence of the project to such an extent that verbal approval had been given as early as December 1983 (before the submission in early 1984 of the application for a capital vote) for the purchase of two LHD's and two drill rigs for the wide reef project; both subject to the need for a three tender system. This approval had been confirmed in a memorandum, dated 12

December 1983, from the General Manager REGM. It was then that I knew I had overcome the doubts of Mr van der Meulen, the General Manager and therefore it was from the end of 1983, only four months after the arrival of KAR at Cooke 2 Shaft, that it could be said that the project really started to accelerate.

In terms of progress with the project in early 1984, it would now be relevant to examine the involvement of KAR with the use of Nonel, referred to in the final preliminary motivational report, and its relevance to this project.

3.5.1 ***Nonel***

Standard blasting techniques at REGM (as on other gold mines in South Africa at that time) utilised fuses and igniter cord for the firing of charged shot holes. Although these techniques had proven effective for many years they nevertheless contained certain undesirable features. A major disadvantage of the system had always been its inability to guarantee consistent sequential firing; out of sequence shots in the range of 1% - 3% had been recorded in controlled tests at REGM. Notwithstanding the acceptance of that system over the years, any attempt to improve the system had to be considered. Therefore, in late 1983 discussions had taken place between the Gold Division of JCI and African Explosives Chemical Limited (AECI) regarding the use of Nonel assemblies. Nonel is a non-electric ignition system which can eliminate cut-offs and, further, guarantee sequential firing. The system was invented by NitroNobel (Sweden) and is based on a plastic tube internally lined with an explosive powder which on initiation carries a propagating shock wave to ignite the delay element of a Nonel detonator. It is a safe system and cannot be initiated by flame, impact or electric current. In December of that year Mr GHS Bamford, the Consulting Engineer of the Gold Division of JCI, made the decision to carry out trials with Nonel at REGM's Cooke 2 Shaft. The trials were to be conducted with Nonel Short Period Delays (SPD's) and Nonel Long Period Delays (LPD's). The Nonel SPD's which were not commercially available in South Africa, and had to be imported from Sweden, were to be used in trials in the conventional wide reef room and pillar stopes and the narrow reef crush pillar and stick stopes. The Nonel LPD's which had been available from AECI for some years were to be tested in development operations and advance strike gulleys (ASG's) in conventional stoping. All this controlled test work would take place at Cooke 2 Shaft under the direct supervision of the Industrial Engineering Department at REGM who in turn were responsible to KAR in his capacity as manager of Cooke 2 Shaft.

The major findings from this controlled experimental work, which were to relate specifically to the 90L E8 Reef Trackless Project and have a significant impact on the project, can be summarised as follows.

SPD's

In wide reef room and pillar stopes the use of Nonel SPD's was shown to be cost effective for blast holes of 2,0 metres or longer with burden spacing of 90cms to 100cms. Blasting with Nonel SPD's improved fragmentation and the throw of the blast proved very effective; more than 85% of the stoping panel was thrown into the ASG in the wide reef stopes. These positive findings in conventional wide reef stopes with 10 metre rooms would ensure their effectiveness in trackless stopes where the slashing panel was planned to be only 5 metres long. Throw and fragmentation from using Nonel in a trackless operation would provide for ideal muckpile conditions for an LHD. In this respect it has to be emphasised that the major constraint in conventional stopes had been the inability of the ASG winch to handle the large tonnage of broken ore from a rapidly advancing face. This caused an excessive build-up of rock in the stope as seen in the photographs included in Chapter 2 and Annexure 2.1 of Volume 2 of this treatise. Therefore, the results dictated that Nonel SPD's must be used in the future trackless stopes; refer to P1/P2 layout in **Figure 3.9**. Also refer to photographs, taken at a later date, in trackless operations showing a Nonel face lit up before blasting and the effect of the throw blast from P2 into P1; **Figures 3.10** and **3.11** respectively.

In footwall lifting (benching) and pillar extraction operations Nonel SPD's would be ideal and it was planned that they would be used in all such future operations in trackless mining.

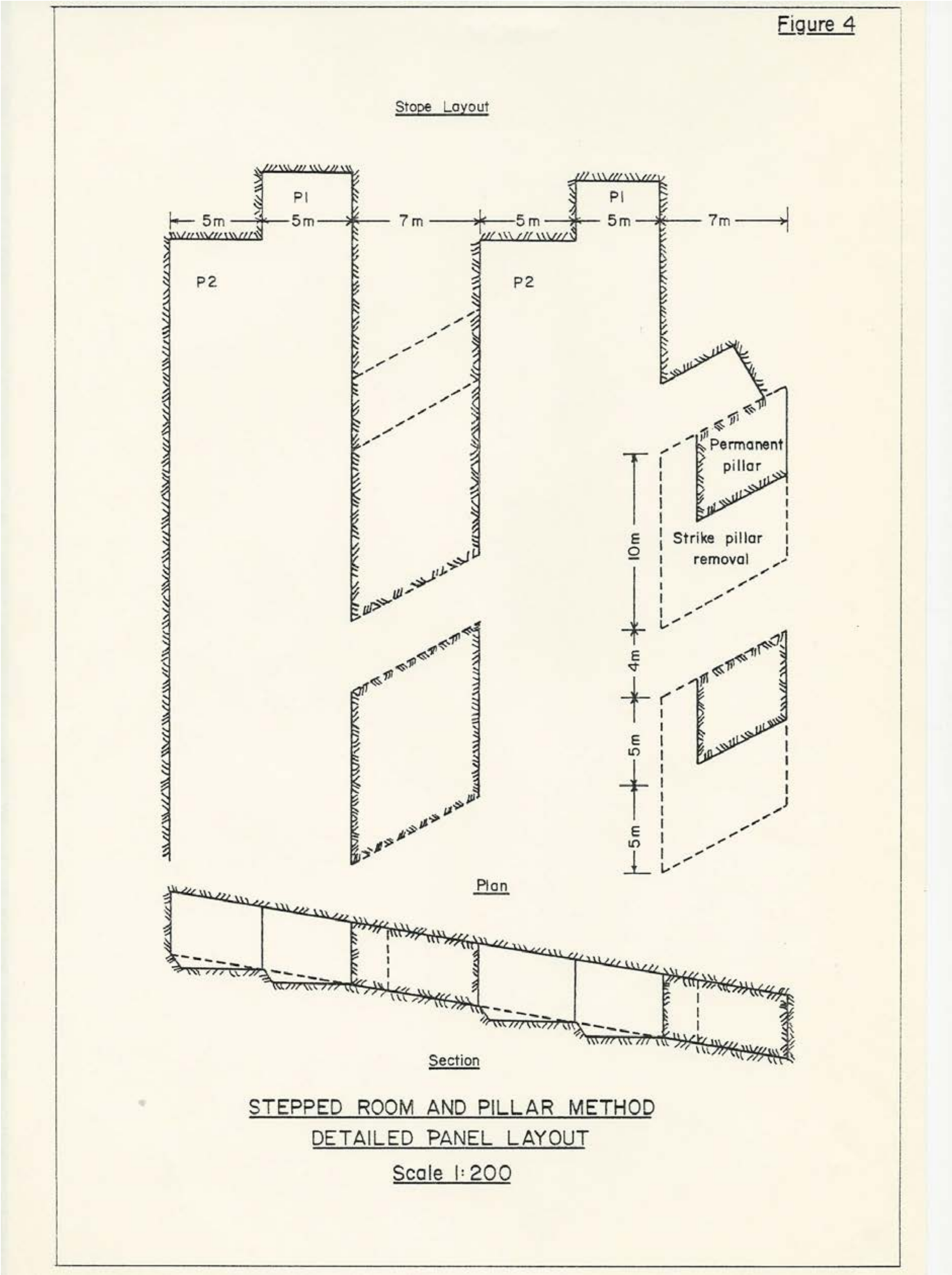

FIGURE 3.9

P1/P2 Layout

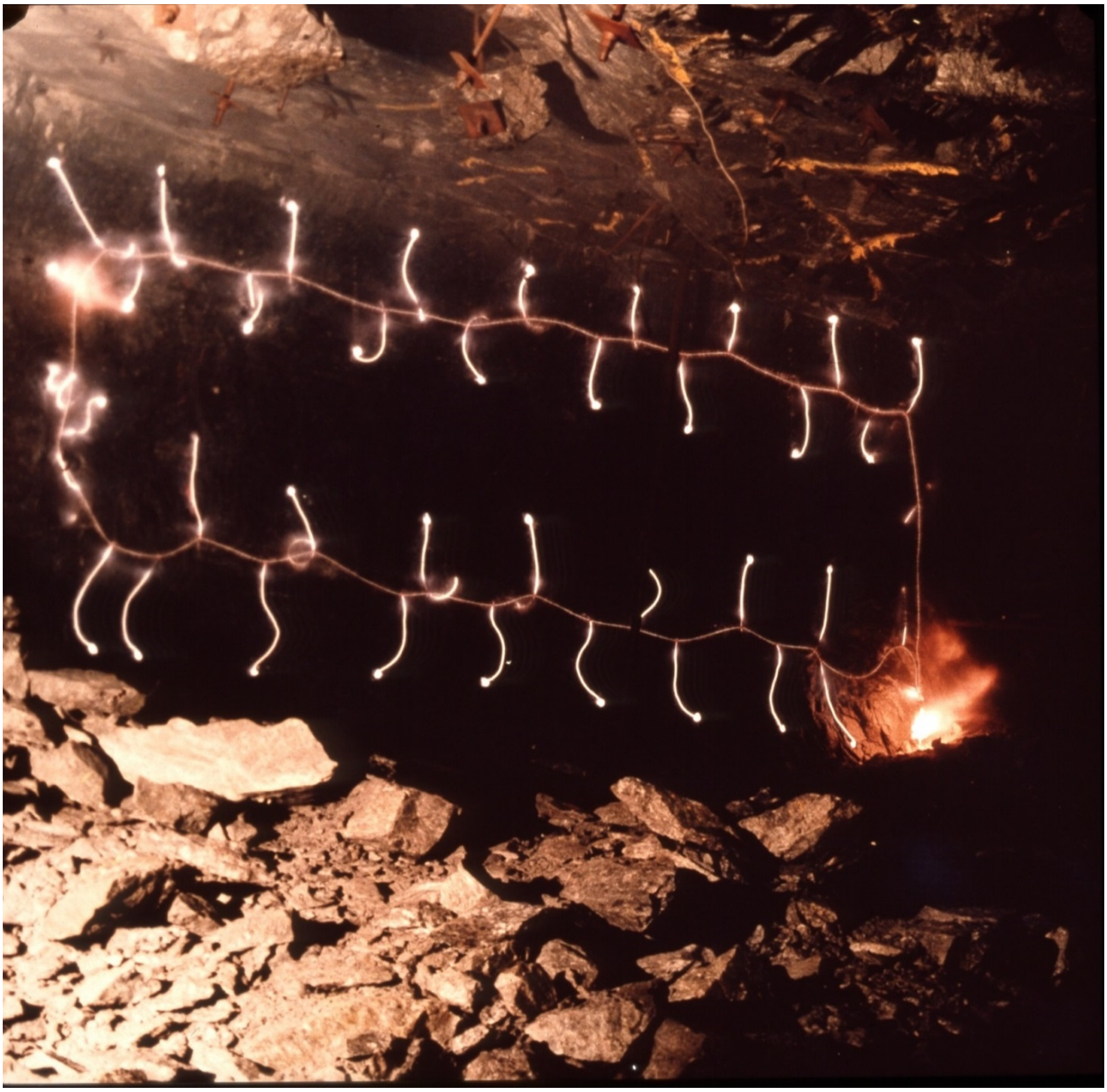

FIGURE 3.10

A Nonel Face Lit Up Before The Blast

FIGURE 3.11

The Result of the Throw Blast from P2 into P1

LPD's

It was concluded that the use of Nonel LPD's in rounds of 2 metres or less could not be justified. Nevertheless, in trackless stopes the leading panel (P1) in the stope was to be drilled by electro-hydraulic drill rig (3,8 metre blastholes); therefore Nonel LPD's would be viable and necessary in order to maximise advance and justify the cost of drilling long rounds. Further, the use of LPD's in trackless development (similarly long rounds with electro-hydraulic drill rig) would also be cost effective. It should also be stressed that with LPD's an increased advance would be achieved due to a significant reduction in socket length resulting from sequential firing as opposed to fuse blasting with igniter cord.

The major disadvantage of Nonel was always going to be the cost; Nonel assemblies were more than double the cost of fuses. Nevertheless, the conclusions from the controlled trials with Nonel had shown that with SPD's if blast holes in stoping were drilled 2,0 metres or greater and if in development LPD's were used to drill 2,0 metre rounds or more, in both cases the use of Nonel was viable.

These controlled trials at Cooke 2 Shaft had shown positive findings for the 90L E8 Trackless Project which was, at that time, building up in the Phase 1 period. They had shown unequivocally that both Nonel SPD's and LPD's held significant benefit for the new trackless project and therefore the findings from these trials were quickly integrated into the trackless operation in 1985 with significant advantages to the project.

The technical paper "***The Use of Nonel at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited"*** written and presented to the Association of Mine Managers of South Africa by K.A.Rhodes is attached as **Annexure 3.2** in **Volume 2**. This paper was published in the transactions of the Association of Mine Managers of South Africa in 1986 and was based on the detailed documentation emanating from the controlled experimentation on Nonel under the direction of the KAR at Cooke 2 Shaft, REGM.

The pace of the project was rapidly accelerating in early 1984 and it was now necessary to develop specific strategies for mechanisation of the project. It was believed by KAR that the important factors for the introduction of any mechanisation programme were engineering maintenance, training and management controls.

3.5.2 ***Engineering Maintenance Philosophy***

It was realised that a most important issue in a trackless operation was the need for machines to be available when they were required to do work; a high availability of the equipment had to be achieved from the outset and throughout the life of the machine. In the early days of the project it was necessary for all responsible officials to understand that mining and engineering personnel must work together and that senior managers (from myself down) were committed to this key objective. Trackless mechanised mining was going to be very different from the standard way of managing a gold mine and any management philosophy which took the attitude of 'us and them', which was all too prevalent between mining and engineering functions on gold mines, would cause any trackless mechanised operation to fail, and fail quickly. The most important part of this policy was that there had to be a commitment by the mining managers to the engineering function. It was therefore critically important, for the success of this first mechanised project, that this support by mining managers and supervisors for engineering maintenance programmes was entrenched from the outset; this was my responsibility to ensure this happened. In this respect the specifics of this philosophy can now be discussed.

The two major factors for an effective maintenance programme were swiftly realised in early 1984: the provision of adequate underground workshops in order to carry out maintenance and a strictly enforced maintenance schedule carried out by trained artisans.

Underground Workshops

Underground workshops had to be made available from the very beginning. Although it would not be practical to develop and construct a workshop immediately for the full fleet of equipment which would be required at steady state production, it was necessary to establish workshop facilities in stages.

The first underground workshop bay required was to become known as the assembly bay; this bay would be developed in advance of any equipment being sent underground. At Cooke 2 Shaft access to the future trackless area was more than three kilometres distant, along a rail haulage, from the vertical shaft. All machines had to be stripped on surface and transported to the end of the track system where the assembly bay had been constructed; the bay was developed by conventional drilling and cleaned by rail-mounted loaders loading into hoppers. This assembly bay was then to serve as a maintenance bay for the first equipment delivered underground whilst this equipment was being used to develop the first workshop for the project. Refer to photograph in **Figure 3.12**. Therefore, in early 1984, it was necessary to focus attention on the assembly bay.

An important factor however in the development of workshops was seen to be the necessary support of the strata of these key excavations and the necessity for pillars in the workshop area and also the proximity of the workshops to the mining horizon. All these factors were incorporated in plans provided by the rock mechanics engineer and approved by KAR as the responsible manager before any development took place.

Also in 1984 initial planning was being undertaken for the supply of diesel fuel direct from surface by fuel pipeline to underground tanks in close proximity to the future workshop complex. Details of this system will be given later in this chapter

Maintenance

At the start of this project there were certain issues relating to maintenance which were considered key to the success of the operation. There had to be strong discipline exercised over the operators of the equipment to ensure that machines would be available at the right time for their scheduled services; it was the responsibility of the mining supervisors to enforce this directive from the very beginning.

The availability of spares was identified as a major factor. Machines at the start of the project had to have high mechanical

FIGURE 3.12

Assembly Bay Being Developed Underground by Conventional Methods

availabilities (over 85%) and if spares could not be guaranteed when required, due to a breakdown on any one of the machines (and there were very few machines at the start of the operation), then the whole cycle of operations (drilling, cleaning, support) would be dislocated and development progress seriously affected. In due course major changes were to take place in the overall stores infrastructure at REGM but in 1984 it was the responsibility of the responsible engineer, in close contact with the respective OEM's, to ensure that spares were available when required. The machines would be new, and without abuse (which will be discussed later), availabilities would be expected to be high with only minimum downtime. However, as the project grew and machines aged, downtime would inevitably increase. It was therefore important at the beginning to establish an effective engineering maintenance philosophy.

Probably from an engineer's point of view the most important issue of any maintenance plan has to be the quality of skills of his artisans and therefore, early concentration was given to artisan skills and artisan training. The specialised skills required for the maintenance of the electro-hydraulic drilling rig for instance, introduced to the project instead of compressed air operated rigs, was highlighted very early.

3.5.3 ***Training***

Reference has been made to artisan skills training but a pre-requisite for success has to be the level of competence of operators or driver skills. The machines being introduced were few and they were expensive and the machines could not be allowed to be damaged or abused by any driver's irresponsibility; such a trend would represent a major risk to the project. In recognition of this risk, before any machines had been delivered or went underground, an early appointment was made by KAR of a Mechanical Equipment Supervisor (MES). The person appointed had previous mechanised experience and had completed certain training courses which specifically included hydraulic courses prepared by the OEM's. The MES had the legal appointment in terms of the Mines and Works Regulation 18.2.2(c) in that he was delegated by the responsible engineer to test the competency of a driver before issuing a licence; see attached in **Figure 3.13** the first legal appointment of an MES made by KAR and in **Figure 3.13A** the duties of the MES as set out by KAR.

The success of this appointment led to it becoming a standard appointment in any trackless project within JCI; refer to the attached JCI Policy and Procedure Manual, Trackless Mechanised Mining Methods, Ref.01-03-66 dated 1987-03-31 in **Figure 3.14,** highlighting the position of the MES, then called an overseer.

For KAR this was always going to be a key appointment and as can be seen from the duties and responsibilities described above the MES had reporting authority directly to the senior manager, which in itself emphasised his importance to the project. The appointment of an MES has proved effective in the management career of KAR, and in later years, has always been recommended for any new trackless project wherever KAR has been the appointed consultant to any mine/client.

Notwithstanding the importance of operator skills, it was also determined that training was necessary for drivers' immediate supervisors. It was not expected that any supervisor needed to be able to operate a machine to the degree of competence of a driver but they had to know how to operate the machine and its basic functions and be aware of the potential for abuse of the machine. Finally, it was required for management to be subjected to a machine appreciation programme designed to give them the necessary technical knowledge to manage trackless operations effectively.

3.5.4 ***Trackless Management Responsibility***

From the very start of this first mechanised project I had to ensure that mining managers and supervisors subordinate to myself gained the necessary knowledge to enable them to manage the project effectively. In addition to the programmes outlined above I arranged, whenever and wherever possible, to expose officials (and myself) to other mechanised operations and therefore several visits were made to mines in southern Africa during 1984/1985.These extended visits had the purpose of gaining practical knowledge of trackless mining on fully mechanised mines

R.E.G.M (COOKE 2 SHAFT) TMC2/1

DRIVERS OF TRACKLESS EQUIPMENT: MINES AND WORKS REGULATIONS 18.2.1, 18.2.2, 18.2.3, 18.2.4.

Drivers of self-propelled mobile machines.

[1]**18.2.1** *Except as provided for in regulations 18.1.1, 18.1.2 and 18.1.3, no person shall drive or be caused or permitted to drive any self-propelled mobile machine which is under the control of the manager, on private property, in or at a mine or works, unless he is authorised thereto by the manager, mine overseer, shiftboss or person appointed in terms of regulation 2.13.1, 2.13.2 or 2.13.3.*

18.2.2 *The manager, mine overseer, shiftboss or person appointed in terms of regulation 2.13.1, 2.13.2 or 2.13.3 shall not authorise any person to drive any self-propelled mobile machine unless he is satisfied that the person—*

(a) has attained the age of 18 years;

(b) does not suffer from defective sight or hearing or any other infirmity, mental or physical, likely to interfere with the efficient discharge of his duties; and

(c) has been found competent to do so, by actual test, by the person appointed in terms of regulaton 2.13.1, 2.13.2 or 2.13.3, or by some other competent scheduled person to whom this duty was delegated by such person, except that—

(i) the provisions of regulation 18.2.2 (c) shall not apply to a person so authorised if he is in possession of a driver's licence issued by a recognised provincial authority and valid for the class of vehicle he is authorised to drive, and

(ii) a learner driver may be permitted to drive any such machine under the immediate supervision and control of the authorised driver.

An authorisation issued in terms of this regulation shall be valid for the mine or works in respect of which it is issued.

18.2.3 *No person shall drive or be permitted to drive any self-propelled mobile machine unless he is properly seated in a seat provided for the driver, except where the machine is so designed as to be driven with the driver standing or walking.*

18.2.4 *If, in the opinion of the Inspector of Mines or the Inspector of Machinery, any person authorised in terms of regulation 18.1.3 or 18.2.1 has been negligent in the execution of his duties or is for any reason unable to discharge his duties safely and efficiently, the manager shall, upon notice from the Inspector, immediately withdraw the authorisation to such person.*

IN TERMS OF THE ABOVE REGULATION 18.2.2.(c) A.G.ROBINSON (MECHANICAL EQUIPMENT SUPERVISOR AT COOKE 2 SHAFT) IS DELEGATED BY THE APPOINTED SUBORDINATE ENGINEER P.J.M V.WYK AT COOKE 2 SHAFT, SUCH PERSON BEING APPOINTED IN TERMS OF REGULATION 2.13.3, TO TEST THE COMPETENCE OF DRIVERS OF TRACKLESS EQUIPMENT

P.J.M. ~~VAN~~ WYK.

K. A. RHODES
MANAGER MINING COOKE 2
Sign.:
Date: 5/12/84

FIGURE 3.13

First Legal Appointment of Mechanical Equipment Supervisor

Duties of the Mechanical Equipment Supervisor
as first set out by K.A.Rhodes, Manager Mining, Cooke 2 Shaft, REGM in 1984

1. Select all new candidates for a job as an operator and ensure any new operator is trained according to best practice.

2. Review all operators' licences, interrogate all operators training background and re-appoint or re-train all operators: this to be done in terms of the proposed programme.

3. Train and appoint instructors to assist with establishing best practice.

4. Maintain a total on-going re-training programme for all current operators.

5. Assist and co-ordinate supervisory training. Initially set up appreciation training for supervisors and management for good and bad practices.

6. Exercise driver discipline over the entire complement of operators on a daily basis. Remove incompetent or ill-disciplined operators for re-training; this to be done by the MES with the full authority of the manager in charge of the shaft.

7. Investigate damage and abuse of equipment by operators and, further, liaise with the responsible engineer in terms of damage report investigation.

8. Enforce TM3 standards.

9. Follow-up on the use of the operators' check list.

10. Enforce correct drill string procedures (collaring, bit removal etc.).

11. Work in close co-operation with OEM's on their proposed training programmes and their audits of both machines and operator practices.

12. Submit a daily report to the senior manager in charge of the shaft.

Figure 3.13A

Johannesburg Consolidated Investment Company, Limited

GOLD AND URANIUM DIVISION

POLICY AND PROCEDURE MANUAL
TRACKLESS MECHANISED MINING METHODS

ORGANISATIONAL STRUCTURE - MINE

01 : MINING		
Ref No. 01-03-66		
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MINE ORGANISATIONAL STRUCTURE

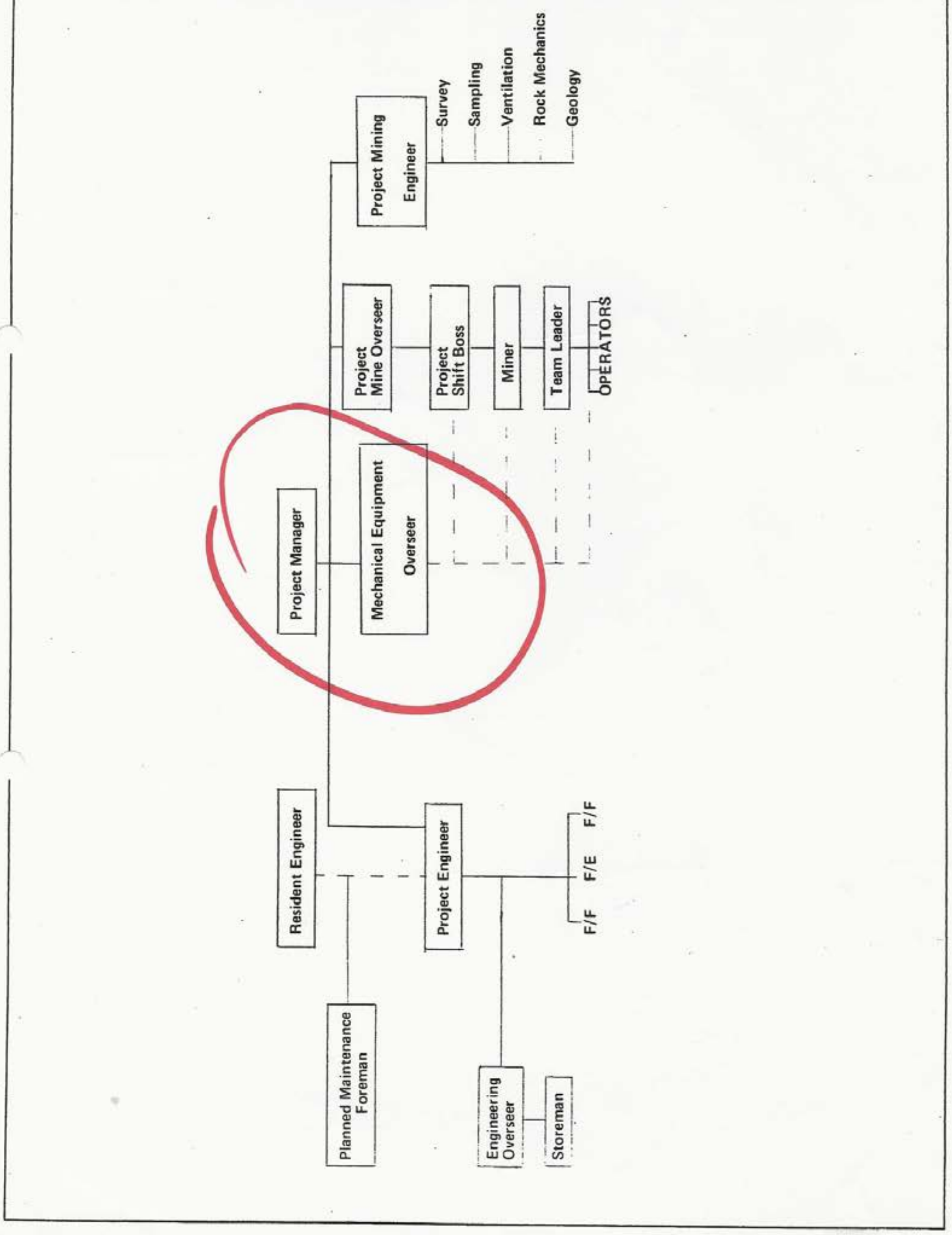


FIGURE 3.14

JCI Trackless Organisational Structure

such as the Otjihase Copper Mine where I had been manager in the mid-1970's. Over a period of a few days on such a visit it was possible for KAR to develop a team spirit and a commitment of the mining and engineering disciplines to the project. Also, these visits helped to develop a 'hands-on' management style for the daily control of all aspects of the operation, which was necessary for success.

3.6 Final Report for Motivation of the Project (Phase 2)

By July 1984 trackless development work had commenced following the approval of a Capital Vote No 574 by the Consulting Engineer Gold with a value of R2,404 million (in March 1984 terms). The approval of Phase 1 of the project, a concept new to the industry, would represent a milestone in trackless mechanisation in a gold mine.

Development of the project continued in 1984 and the final motivational report for Phase 2 of the 90L E8 Project was completed on 12 December and an application for a capital vote was submitted by KAR on 31 December 1984. This application and ***Final Motivational Report*** dated 12 December 1984 by KAR, is reproduced in full in **Annexure 3.3** in **Volume 2**. This report examined the general progress with the project in 1984 and gave a financial justification for the project; summarised briefly as follows.

3.6.1 Production towards the end of 1984 was approximately 10 000 tons/month (reef and waste); actual reef production in December 1984 was 11724 tons against a plan of 6336 tons. The report projected reef tonnages to be 18000 tons by July 1985 utilising the equipment approved in terms of the Phase 1 programme only. Full production was projected to be achieved in January 1986 at 40000 tons/month reef following the introduction of equipment approved in the Phase 2 application.

3.6.2 With regard to the all-important issue of underground workshops, the No 1 Workshop (previously discussed under the Final Preliminary Report) was available and capable of servicing all the equipment in operation in December 1984. The second facility, No.2 Workshop, was under development and situated inbye of the established No 1 Workshop in the 90 Level N11 crosscut. When completed, this workshop would provide for full workshop and maintenance facilities for both Phase 1 and Phase 2 of the project. It was confirmed that all rock mechanics considerations had been taken into account in the layout and development of these workshops, with cognizance being taken of the proposed stoping layout on the E8 Reef horizon.

3.6.3 Maintenance of the equipment was being carried out in strict accordance with the planned schedules.

3.6.4 Although diesel fuel was still being transported into the mine by rail fuel tankers to 90L N11 crosscut and then pumped into temporary storage tanks, planning was advanced for the installation of an automatic bulk diesel fuel transfer system where diesel would be pumped direct from surface storage tanks (already installed) by pipeline to storage tanks to be situated between the No 1 and No 2 Workshops.

3.6.5 It was recorded in the report that all trackless equipment for Phase 1 (stripped on surface and re-assembled underground) had been transferred through the Cooke 2 Shaft to 90L N11 crosscut along the 90L rail haulage without any problems. Refer to the photographs in **Figures 3.15** and **3.15A** and other photographs in **Annexure 3.4** in **Volume 2**.

3.6.6 Training of operators and artisans had commenced with OEM's providing training courses. Additional training programmes for all responsible officials had been prepared and implemented by year end. Refer to photographs of drill rig training in **Figures 3.16** and **3.16A**.

3.6.7 The operating costs for the trackless operation, although still not yet definitive, was estimated at R3,84/ton less than the cost of a conventional operation; actual estimates R8,60 and R12,44 respectively.

3.6.8 It was also detailed in the report that waste development for the project reserve would have been of the order of 165000 tons (165 694 tons) if carried out by conventional mining; however, when the waste development required for the trackless operation was deducted the difference would be approximately 140000tons

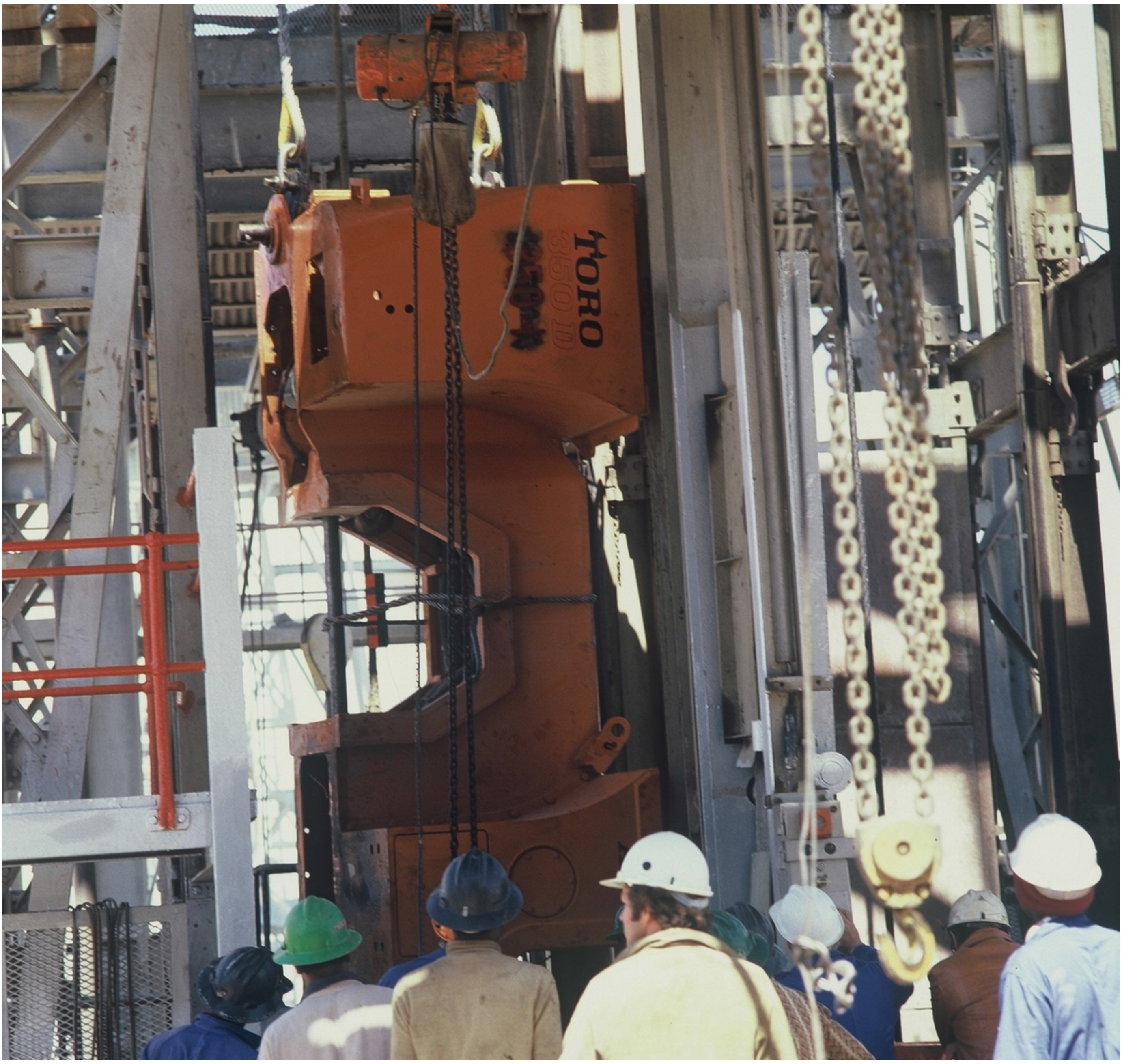


FIGURE 3.15

Slinging Underground a Component of a Stripped Machine

FIGURE 3.15A

Re-Assembly of a Machine Underground

FIGURE 3.16

Set-Up of Drill Rig for Training on Surface

FIGURE 3.16A

Underground Training of Drill Rig Operators with the Drill Master

(165000 less 25000 for waste from workshops, raiseborer orepass from 101 Level, 101 streamlined haulage). If it can be assumed that this footwall waste would be generated during the life of the trackless project, this would have released between 20000 and 30000 tons more reef hoisting capacity at the shaft per year for say five years; the shaft was then running at full capacity. If this waste could be replaced by reef then it represented a major bonus to the company. The relevant revenue for this additional reef would be R76/ton, with costs assumed to be R20/ton (incremental) profit would therefore be R56/ton or more than R7 million over the period of say 5 years.

3.6.9 In order for conventional mining to take place it would have been necessary to plan for more than 10000 metres (10039 metres) of footwall development for the same reserve at an estimated cost of development of R3,908 million.

3.6.10 Finally, the DCF (at 15% discount rate) was calculated at +R6,04 million for the project. Discount payback was 29 months with an IRR of 59%.

3.7 <u>1985: Phase 2 of the Project</u>

The application for a capital vote for the Phase 2 (and final phase) of the 90L E8 Reef Project, motivated at the end of 1984, was approved by the Consulting Engineer (Gold) and forwarded to the Board of Directors on 08 March 1985. The Control Budget Estimate for the final phase was R5,051 million escalated to forecast completion date (R4,443 million in Base Date terms).

Over 80% of the value of this estimate dealt with equipment; below are the details for Phase 2, in addition to the machines already in use up to that date and approved for Phase 1.

<u>Phase 2</u>

Twin Boom Drill Rig	2
Roof Bolter	1
3,8m^3 LHD's	2
2,7m^3 LHD's	0
18 ton Trucks	4
Utility Vehicles	2

Grader		1
Transport Vehicles		4
Impact Breaker		1
	Total	17

The approval for the Phase 2 equipment would be sufficient to meet the requirements of the planned production of 40000 tons of reef per month.

It would now be the appropriate time to comment on the equipment selected for this project in the final phase and the status of workshops in 1985.

3.7.1 ***Equipment***

KAR believed that it was correct to have selected the largest size machine for the project because the fundamental reasoning was that the largest units would achieve a reduction in the working costs due to a smaller fleet; operating costs being reduced as the cost of operating different sized machines did not vary that markedly and the number of artisans required to service the fleet does not vary with the size of the units. Nevertheless, having accepted the above argument cognizance had to be taken of the optimum size of the fleet (there had to be flexibility) and also the dimensions of the planned roadways.

When the type and size of the equipment had been decided it was still necessary to go through a tender process based on detailed specification documentation, and when the tenders had been received a decision was only then made after an adjudication process involving the responsible managers and engineers. A typical example of this process for a purchase of a utility vehicle can be seen in the attached documentation in **Annexure 3.5** in **Volume 2**. In this example a decision had been made, for technical reasons, to purchase the unit which was not the lowest tender; in this respect a variation in scope for the CBE had to be approved.

Some relevant discussion of the main groups of machines now follows.

Drill Rigs

At an early stage in the project it was decided to opt for the electro-hydraulic drill rig over the pneumatic rig. There were major advantages to be gained from this decision: faster penetration rates would maximise advance; pneumatic rigs require a high compressed air pressure possibly necessitating booster compressors; any reduction in compressed air requirements would reduce costs generally at the shaft.

Roof Bolters

Recommended support requirements on the E8 Reef horizon required that a hole of 2,8 metres in length had to be drilled (2,7 metre grouted rebar). In terms of the necessity to mine a stoping width of 3 metres it was not possible to utilise a standard drill rig for reason that a working height of 4,75 metres would be required which was unacceptable. There were two further options open; use of a telescopic chain feed or roofbolting with a single boom roofbolter for drilling the hole combined with a separate boom with a basket to enable a man to install the roofbolt. In the case of the telescopic rig a stoping width of 3,6 metres would have been required and even then the hole could only be drilled 1,8 metres long; this option was also clearly unacceptable. In effect the only acceptable choice at that time was a separate unit with a drilling boom and a boom with a basket; refer to photograph in **Figure 3.17** of an operator working from the basket. In this case the hole would be drilled in two passes in a stoping width of 3,1 metres or three passes down to a stoping width of 2,5 metres. This then was the selected option.

It was important to realise that in a cyclic operation, where multiple faces are available at any one time, it is the right choice to have a machine for each part of the cycle; if the face drill rig is used for roofbolting as well as drilling the face then that machine is carrying out two functions in the cycle. A dual purpose machine should only be planned for when equipment is captive, as in a single development end or for tunnelling work.

LHD's

It has been seen in Phase 1 that 2,7m³ capacity LHD's were purchased but in terms of the argument put forward for selecting

FIGURE 3.17

Roofbolter with Boom and Basket

the largest practical size unit, 3,8m³ capacity LHD's were selected for Phase 2 .

A further development in Phase 2 was the introduction of the ejector bucket (EOD bucket) to the 3,8m³ units. It was believed by KAR that ejector buckets were necessary to facilitate the loading of trucks where there was a height restriction, this being due to the need to control dilution, whether from waste or (unpay) low grade reef. Further, to obviate any damage to a truck body and the truck wheels and tyres, an LHD with an EOD bucket would be able to stand off from the truck and push the load across the truck bowl; when conventional loading takes place the LHD can get too close to the truck making contact then unavoidable. These advantages were considered to outweigh the disadvantage of having to maintain additional cylinders and hoses required for the operation of EOD buckets. Refer to photograph of bucket and photograph of an LHD (with EOD bucket) loading into a truck; both seen in **Annexure 3.6** in **Volume 2**. Also refer to a sketch in **Figure 3.18** showing advantages of the EOD bucket.

Trucks

The importance of matching LHD's and trucks was quickly realised. Initially the 18 ton truck was planned to be filled by four passes of the 2,7m³ LHD or three passes of the 3,8m³ LHD. When the 32 ton truck was introduced later in Phase 2 (which necessitated a variation in scope) the 3,8m³ LHD would fill the truck in five passes. The final fleet of equipment would include two 18 ton trucks and two 32 ton trucks. See **Annexure 3.7** in **Volume 2** for photographs of both trucks.

In introducing trucks to the project it was possible to plan to tram by truck to a single main tip equipped with an impact breaker; the reef being transferred down a long raisebored orepass to 101 Level where the high capacity rail haulage was under construction. This main tip provided for two trucks to tip simultaneously, with the impact breaker able to reach any section of the grizzley from a central position. Although this tip would only be available in January 1986, the design of the tip would ensure that there would never be any blockage on the grizzley at the tipping point which could cause a bottleneck to the entire ore clearance system.

SKETCH SHOWING ADVANTAGES OF E.O.D. BUCKET

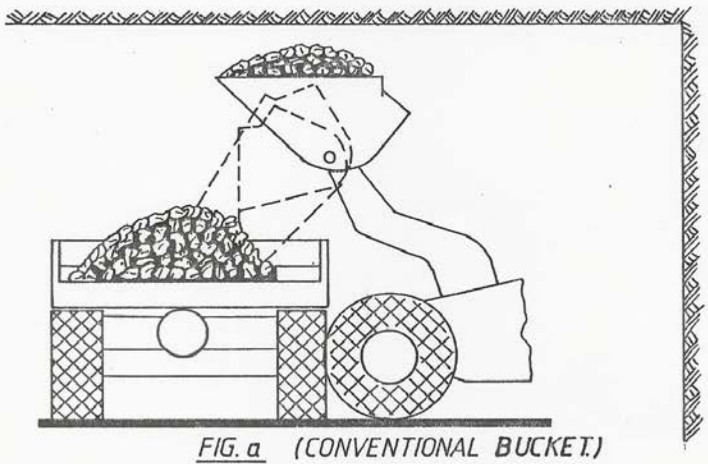


FIG. a (CONVENTIONAL BUCKET.)

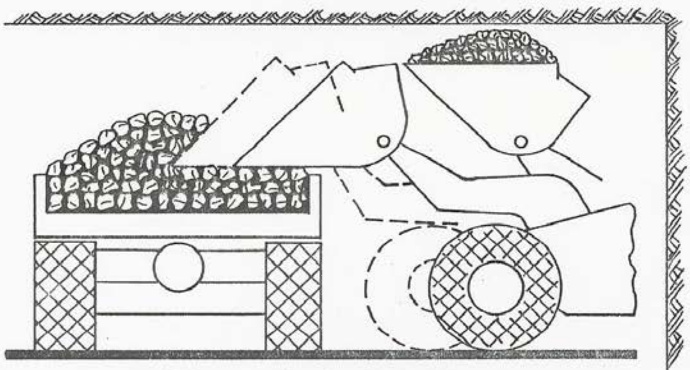


FIG. b (E.O.D. BUCKET.)

N.B. 1. Roadway height in Fig. (b) is less than in Fig. (a).

2. Reach of L.H.D (with E.O.D. bucket) in Fig. (b) is greater than for L.H.D in Fig. (a).

Figure 3.18

Advantages of EOD Bucket

There had been significant input into the design of this tip and grizzley steelwork and as such it would become a JCI standard for trackless mining. Details of the tip layout and tip grizzley, designed by KAR, can be seen in **Figures 3.19** and **3.19A** respectively.

Utility Vehicles

In this project utility vehicles (UV's) had to be available for carrying out the ancillary work both efficiently and most importantly safely. It has been said earlier that a significant number of accidents had occurred in the conventionally mined wide reef stopes due to people falling from platforms and ladders when carrying out work such as making safe by barring down, drilling and charging up holes, pipe construction and cable hanging. The photograph in **Figure 3.20** clearly shows the advantages of working from a UV; in this case charging up the face. In addition to the important safety arguments, in any cyclic operation it is vital that the main production machines carry out the work they are bought for and are not allowed to be used for other work, for example, an LHD being used as a lifting machine; a production machine like an LHD is considerably more expensive to operate than any utility vehicle. In addition, production can be obviously affected when rigs, LHD's and trucks are employed for other work; therefore the use of utility vehicles is necessary to carry out the ancillary work in a cyclic operation.

One specific machine that would become available following Phase 2 approval was a grader to carry out maintenance work on roadbeds. Constant attention had to be given to the condition of roadbeds if the cost of operating LHD's and trucks specifically was not to spiral out of control. Potholes were being filled in with broken rock but more importantly raise borer cuttings were being imported to provide for a final surface; this would prove to be ideal when the grader was brought into use.

Personnel Vehicles

Another important support vehicle necessary for the effective supervision of the operation and also for engineering and other

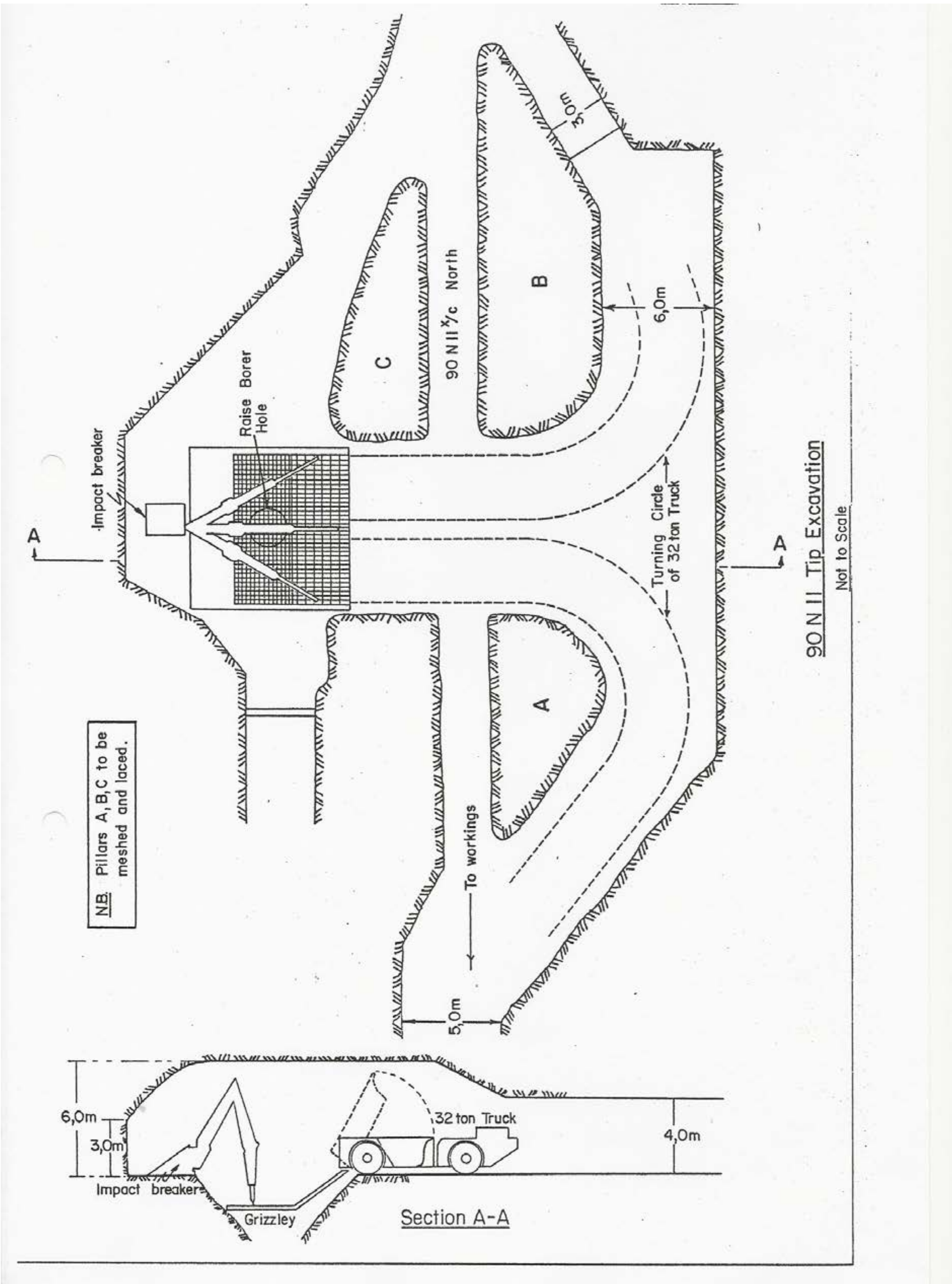
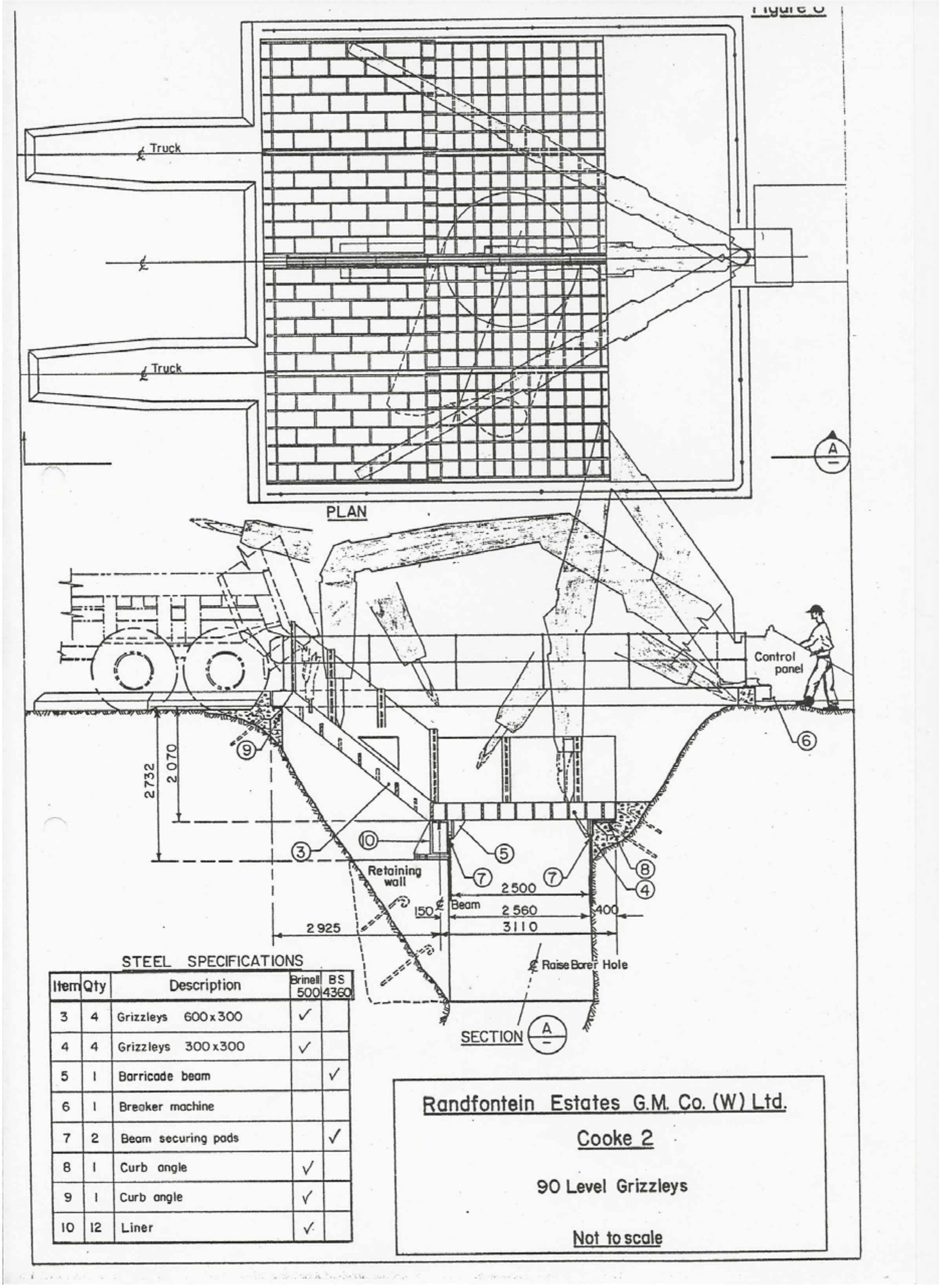


FIGURE 3.19

Truck Tip Layout



Item	Qty	Description	Brinell 500	BS 4360
3	4	Grizzleys 600 x 300	✓	
4	4	Grizzleys 300 x 300	✓	
5	1	Barricade beam		✓
6	1	Breaker machine		
7	2	Beam securing pads		✓
8	1	Curb angle	✓	
9	1	Curb angle	✓	
10	12	Liner	✓	

FIGURE 3.19A

Truck Tip Grizzley Design

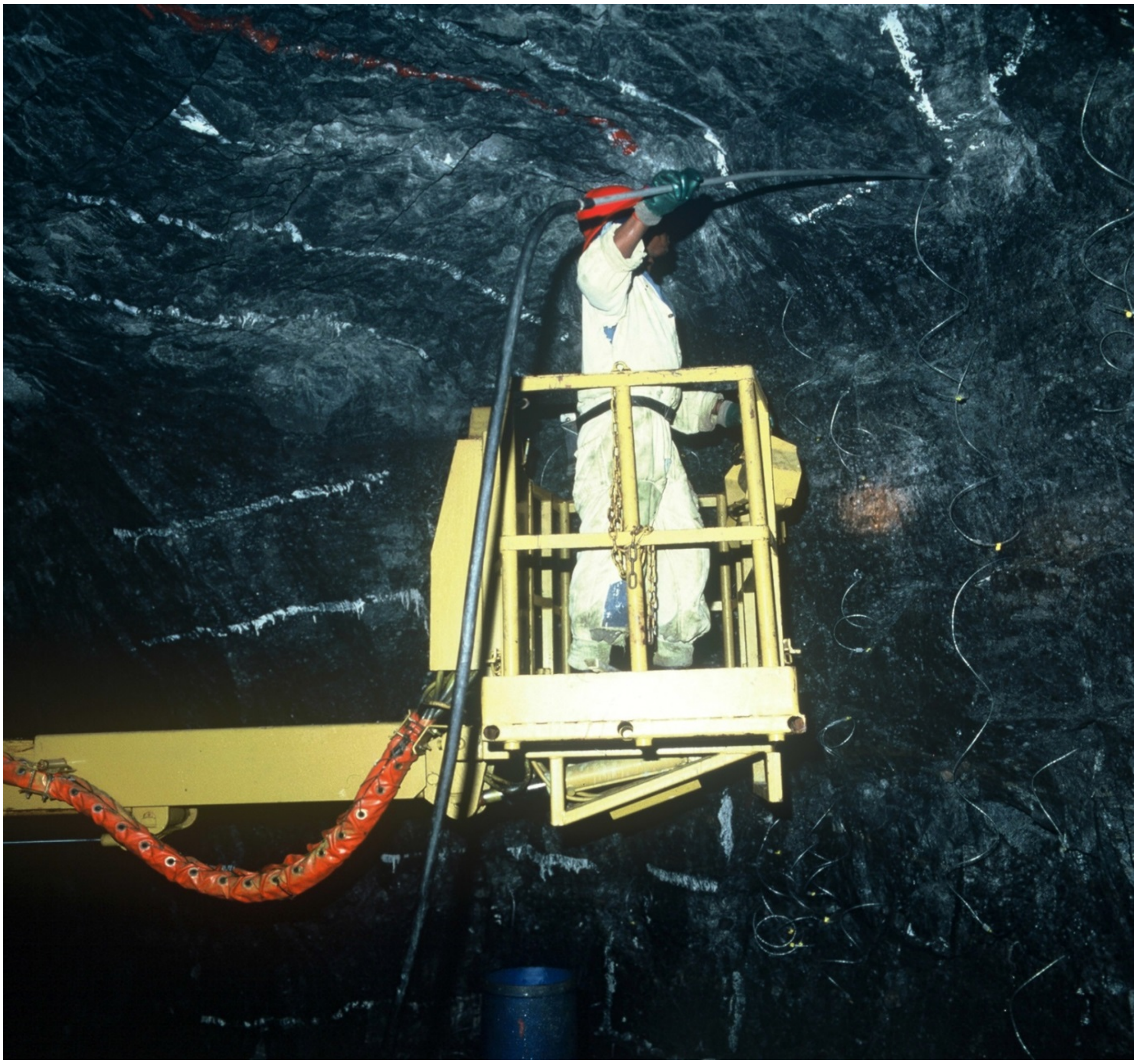


FIGURE 3.20

Charging up from a Utility Vehicle

services was the personnel vehicle. The favoured choice by KAR at that time was the Land Cruiser provided for in the CBE's. This vehicle proved to be a versatile machine which could transport people, explosives, spares for machines, roofbolts, in fact anything that would be required to service the project. For example, in the case of an artisan working out of an underground workshop, many hours of machine downtime could be prevented by such a vehicle being available for fast access into the workings and, where applicable, for the transport of spares direct to where the breakdown had occurred. See photograph of a Land Cruiser in the underground workshop in **Annexure 3.8** in **Volume 2.**

Mechanised Vamping Operations with LHD

During mid-1985 a small (1,5m³) LHD had been hired on a trial basis for the mechanised vamping of the old disused footwall crosscuts at Cooke 2 Shaft. The trial had proved successful in that the productivity target of 50 tons per shift, calculated by REGM's Industrial Engineering Department, was achieved. In a cost exercise it was therefore claimed that mechanised vamping by LHD was viable and a motivation was made by KAR for the purchase of the machine. In **Annexure 3.9** in **Volume 2** is a justification report by KAR and the application for the purchase of the LHD.

3.7.2 ***Equipment Performance***

The equipment requirements for the project were based on specific planning parameters (which assumed a double shift operation). These parameters could be considered to have been crude at that time as little or no experience in such conditions was available on a South African gold mine. Some degree of conservatism was therefore important; if one has to consider the assumptions, of that time, juxtaposed with later more reliable data used in this exposition, this will be seen to have been the case.

Drill Rigs

At the time it was assumed that an electro-hydraulic drill rig would be able to drill 50 metres from each boom in an hour (2 x 50 metres for a two boom rig). In a P1/P2 room for 80 holes at 3,2 metres long this would represent 256 metres. The drilling time would therefore be 256 ÷ (2 x 50) or 2,56 hours.

However, the tramming and set up times in one P1/P2 room could be assumed to be (say) 0,5 hours as the rig would have to move to P1 or P2 in a stepped room and pillar layout and also a similar time to tram to the next room. Thus a theoretical estimate would be 3,56 hours or (say) 4 hours for a P1/P2 round, equivalent to 1,5 (rooms) for a shift.
Tons/rig/month are therefore calculated to be as follows:
1,5 (rooms) x 47 (shifts/month) x tons/blast
Where tons/blast is width (5+5) x 3,0 (advance) x 2,5 (height) x 2,85 SG or 215 tons.
Therefore tons/rig/month (1,5 x 47 x 215) are theoretically estimated at 15000 tons; this compared to a more conservative 12000 tons/month/rig assumed at the time.

Roofbolters
In terms of the recommended support pattern of 2 metre x 2 metre grid, for a 40000 tons monthly production, 1400 roofbolts would be necessary, but assuming an additional 10% for extra supports or even cables this would necessitate (say) 1550 roofbolts/month. Following a blast in a single room, it would theoretically be necessary to install 7,5 roof bolts (average). If it is assumed that it would take one hour to install these supports and an additional hour to tram between P1 and P2 in the same room and to the next room, then the roofbolting performance could be three rooms bolted in a shift; equivalent to say 22 roofbolts in a shift or 22 x 47 (shifts/month) or 1034 roofbolts/month. Number of roofbolters required can therefore assumed to be 1550 ÷ 1034 = 1,5 (say) 2.
The project was planned for two roofbolters initially.

LHD's
If one had to apply, with practical realistic assumptions, formulae used later in this exposition for LHD performance (and in the same manner for trucks) the production performance would be estimated as follows.

Production Performance (P) is $51 \times L \div T + \left[\frac{2D}{S \times 16,67}\right]$ where

51 = 51 is assumed as the utilised minutes in an hour (85% utilisation)
L = LHD carrying capacity (4.2tons or 6tons)
T = time to load, manoeuvre and tip (3 minutes)
S = average speed of unit (6 kilometres/hour)
D = one way worse tramming distance (150 metres)

Therefore for a 2,7m³ unit

$$P = 51 \times 4,2 \div 3 + \left[\frac{2 \times 150}{6 \times 16,67}\right]$$

$$= 36 \text{ tons/hour}$$

For a 3,8m³ unit

$$P = 51 \times 6 \div 3 + \left[\frac{2 \times 150}{6 \times 16,67}\right]$$

$$= 51 \text{ tons/hour}$$

In terms of the above and assuming 280 working hours/month the tons per month performance is calculated at 10000 tons for the 2,7m³ LHD and 14000 tons for the 3,8m³ LHD; these compare to 8000 tons/month and 11000 tons/month respectively, assumed at the time for this project. These later calculations therefore confirm some conservatism in the original estimates.

Trucks

Utilising the same formulae as for LHD's, the production performance for the 18 ton and 32 ton trucks could be calculated based on the following assumptions.

L = 16tons for the 18ton truck and 29 ton for the 32ton truck (assuming a fillability factor of 90%)
T = 12 minutes
S = Average speed (full and empty) of 12kms/hour
D = One way tramming distance of 1000 metres

For the 18 ton truck

$$P = 51 \times 16 \div 12 + \left[\frac{2 \times 1000}{10 \times 16{,}67}\right]$$

$$= 34 \text{ tons/hour}$$

For the 32 ton truck

$$P = 51 \times 29 \div 12 + \left[\frac{2 \times 1000}{10 \times 16{,}67}\right]$$

$$= 61 \text{ tons/hour}$$

Therefore, the monthly performance for the 18 ton truck is 9500 tons and for the 32 ton truck is 17000 tons; this compares to 8000 tons and 12000 tons respectively in the original calculations, once again confirming conservatism in those assumptions at the time.

3.7.3 ***Workshop Status***

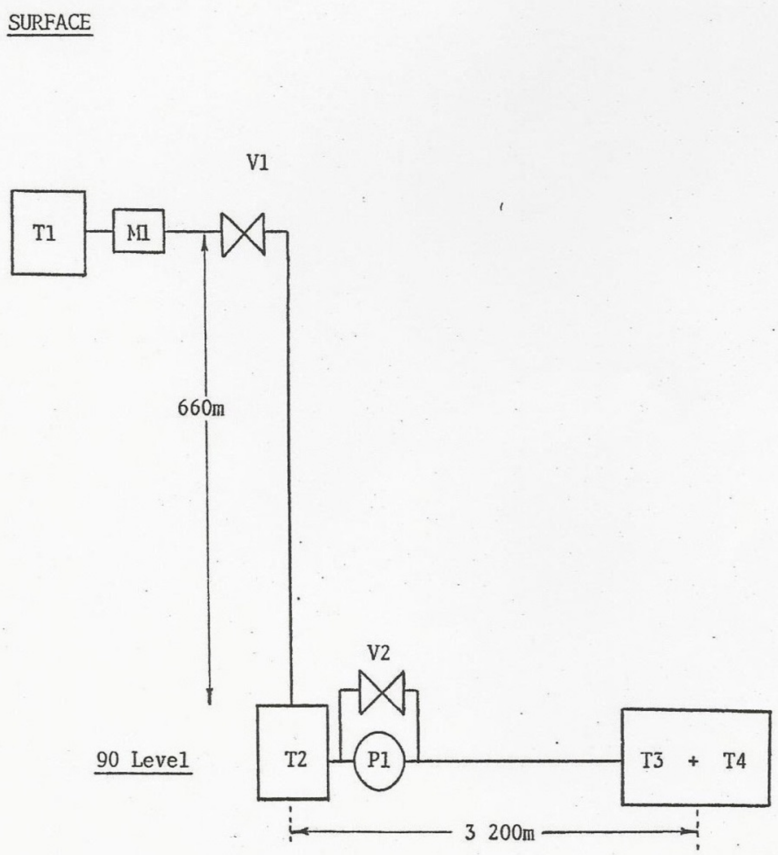
No 1 Workshop, planned to service the equipment delivered in Phase 1, had been completed in early 1985 and the development of No 2 Workshop was underway; together they would provide full workshop facilities for the total fleet of equipment when stoping operations commenced. Provision for replacement of major sub-assemblies and even major overhauls would have to be carried out in these workshops as the full fleet of machines was now captive underground.

By mid-1985 the diesel fuel pipeline system was complete. Two $9m^3$ service fuel tanks were situated in immediate proximity to the two workshops. Diesel was pumped direct from a surface storage tank ($23m^3$) down the shaft to 90 Level into an intermediate batching storage tank ($9m^3$) and then along the haulage (3200 metres) to the service tanks. The batching tank was necessary to ensure that the pipe in the shaft was completely drained; for this to be effective fuel was transported down the shaft in batches. The required quantity was pre-determined and the system was automatic. Refer to **Figure 3.21** for schematic arrangements of the system.

3.7.*4* ***Engineering Planning***

In early 1985 a start was made on the compilation of engineering costs and other related statistics; for example, the life of major

FIGURE 15



Notes

T1	$23m^3$ diesel tank
T2 + T3 + T4	$9m^3$ diesel tanks
M1	Liquid control meter
V1, V2	Solenoid valves
P1	Pump

FIGURE 3.21

Schematic Arrangements for Fuel Transfer from Surface to Underground

sub-assemblies. It was important to capture this information for future cost control purposes. LHD buckets had each been given an individual reference number. All tyres had been branded from the outset with their own number and the history of each individual tyre was kept from new until it was scrapped, with the reasons given on each tyre record card. At that stage, all this information was being captured manually as computerised planning systems were still in the future. This was the time when a monthly report, which provided for availabilities and utilisation of equipment, costs/hour of machines and equipment history, first started to be recorded. Some of the parameters being used were as follows.

Drill Rig

For every drill rig including roofbolters, the cost of spares/drilled metre and the cost of hydraulic fluid and lubricants (excluding fuel)/drilled metre.

Drifter

Cost of drifter spares/drifter percussion hour.

LHD

For each LHD, cost of spares/engine hour and fuel/engine hour.

Truck

For each truck, cost of spares/engine hour and fuel/engine hour.

Tyres

For LHD's and trucks, cost/hour, cost/ton and history and life of every tyre.

3.7.5 ***Equipment Costs***

At the end of 1985 the progressive costs of all equipment showed an actual cost of R4,12/ton compared to the motivation report estimate of R3,23/ton. There were however some reasons at that stage of the project for the higher than forecasted costs and cognizance had to be taken of the following issues.

Spares

Since the Final Motivational Report had been submitted spares costs for the equipment had increased markedly, primarily due to the deterioration in the exchange rate of the Rand.

Production Rate

The project had still not reached steady state production of 40000 tons per month and current production was only 22000 tons per month mainly for reasons of ore clearance constraints; construction of the main tip not yet being complete.

Trucks

Trucks were only just being introduced to the project and when they became fully operational, costs would fall as LHD tramming distances would then be reduced.

Tyres

Tyre costs were considered excessive due mainly to the abnormal ramp development. These costs would be reduced in 1986 due to to improved roadbeds when the grader came available; more use of trucks; the establishment of steady state tonnage following the commencement of stoping operations.

Drill String

Nevertheless, it was re-assuring at that time to be able to record that drill string costs were below the forecast estimated costs. One reason for this, which again underlined the need for a 'hands-on' style of management, was the introduction of bit sharpening to the project. In experimental trials at the end of 1984 it had been shown that the cost per metre drilled for button bits in use could be reduced by half: R0,65/metre drilled when bits were sharpened as against R1,38/metre drilled when drilling to destruction.

There was at this early stage of the project a need to take advantage of any specialist knowledge available from OEM sources. As one example of this KAR remembers discussions that were held with Paavo Horkko of Tamrock Drills in August 1985. Notes were made at the time regarding his advice related to the practical aspects of drilling with rigs. He explained how to remove a bit by positioning the bit square to the face and then removing by percussion. It was said that positioning the bit obliquely would result in bit damage; and also that 80% of all bit damage was caused by oblique collaring of the hole. Also 'free percussing' at the face would inevitably result in significant damage to the drifter, a significant contributor to overall rig costs and collaring would best be done on half percussion feed. Another issue was the importance of positioning the rig as close as practicable to the face when preparing to drill; in other words, do not drill when the machine is on the extreme limits of its boom extension. This type of discussion would prove invaluable in the control of drill rig costs in the future.

Taking cognizance of these positive aspects it could be expected that at steady state production equipment costs would be less; however the cost of equipment spares would increase if the value of the Rand continued to fall as all the equipment (and spares) were imported.

3.7.6 ***Overall Costs***

At the end of 1985 a comparison of conventional and expected trackless working costs continued to show a R4/ton difference in favour of trackless mining (R12,60 versus R8,57) and the operation was considered viable; in today's money terms this difference would be of the order of nearly R50/ton. As referred to in the Final Motivation Report, the Cooke 2 Shaft complex was working at full hoisting capacity and the significant reduction in footwall waste development rock for the trackless operation had allowed for an increase in reef hoisting capacity thereby providing for additional revenue and therefore profit.

3.7.7 ***Safety and Standards***

The serious accidents that occurred following the introduction of conventional wide reef mining which had begun in the 1983/84 period have been referred to in the motivational reports. From April 1984 when the trackless operation commenced until November 1985 there were thirty three lost time accidents recorded in the conventional wide reef stopes and four accidents in the trackless operation, albeit for a markedly less tonnage from the trackless section. The trackless operations were therefore proving to be safer with a clear improvement in the accident rate.

Nevertheless, the use of large mobile mechanised machines coupled with poor driver discipline, when operators do not follow standards and procedures, would always represent a risk of serious accidents and possible fatalities. Therefore it was important and necessary to begin to build up standard working instructions for trackless mining at the very early stage. In due course JCI manuals would be put together but this was later and therefore in late 1984 KAR issued the first of a series of managerial instructions for trackless mining at Cooke 2 Shaft, some of which are included in **Annexure 3.10** in **Volume 2** as examples. Many of these instructions were directed at safe driving techniques.

3.8 POSTSCRIPT TO CHAPTER 3

In September 1985, in an interview with the Technical Director of JCI, I was told that I was to be transferred from REGM to JCI's new mine in the Orange Free State (OFS), the H.J.Joel Gold Mine. Although I believed I had not completed all of what I had set out to do in the two years I had been at Cooke 2 Shaft, the opportunity to go to a new mine where shaft sinking had not yet started would become the greatest challenge of my career. However, in the two years at Cooke 2 Shaft, REGM, I had achieved a great deal. The 90L E8 Reef Project was well on its way to reaching steady state production; this was expected to be achieved in a few months time, January 1986. This project had proved that trackless mechanised mining methods (TM3) were indeed both technically and economically viable and importantly were proving to be safer. In fact, the Cooke 2 Shaft E8 TM3 Project was to become the standard for future TM3 projects in JCI.

In addition to the 90 Level E8 Wide Reef Project, the 95 Level Narrow Reef UEIA Project had commenced at Cooke 2 Shaft in 1985 under the direction and control of KAR, a project which was to form the basis for planning a new trackless gold mine later that year; the H.J.Joel Gold Mine in the OFS.

However, some concluding remarks are relevant before closure of this chapter. It has to be emphasised that the planning for the TM3 operations at Cooke 2 Shaft, REGM , described in this chapter and in the next chapter (Chapter 4) , had been initiated and managed by KAR only. It can also be said that, in addition to the planning for and management of these operations, they were part of the total production at Cooke 2 Shaft which KAR was responsible for and it can be recorded that the shaft was operating at a production rate of more than 200000tons/month reef and waste with all production targets in excess of plan at the time KAR was there.

In 1986 KAR submitted and presented his paper "**Wide Reef Mechanised Room and Pillar Operations at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited**" to the Association of Mine Managers of South Africa (AMMSA). It can also be recorded that this paper was adjudged by AMMSA as the best technical paper for 1986 and the gold medal for that year was awarded to KAR. A full copy of the paper can be seen in **Annexure 3.11** in **Volume 2.**

I would like to make a final comment to end this chapter: following my award of the gold medal in 1986, when I was the mine manager at H.J.Joel Gold Mine, I received a letter from Mr.W.J. van der Meulen, still the General Manager of REGM, congratulating me on the achievement. At the time (and even now, I still have the letter, shown overleaf) it seemed to me, and I am sure to Mr van der Meulen himself, to be a satisfactory conclusion to those first discussions we had had in late 1983 when I arrived at Cooke 2 Shaft, REGM.

The Randfontein Estates Gold Mining Company,
Witwatersrand, Limited

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INTERNATIONAL TELEX: 4-28416SA

COMPANY REGISTRATION NUMBER
01/00251/06

ALL COMMUNICATIONS MUST BE
ADDRESSED TO
THE GENERAL MANAGER

RECEIVED
01 APR 1987

Mine Office,
Randfontein, Transvaal.
1760

27 March 1987

Mr. K.A. Rhodes
Mine Manager
H.J. Joel Gold Mine
Mail Room
P1
J.C.I. HOUSE

Dear Ken

On behalf of R.E.G.M. and myself I would like to take this opportunity to congratulate you on winning a Gold Medal for your presentation at the Association of Mine Managers.

This is indeed a fantastic achievement and well deserved.

Yours sincerely

W.J. VAN DER MEULEN

WJVDM/an

PRODUCTION MANAGER		
RESIDENT ENGINEER		
MANAGER F AND A		
PLANNING MANAGER		
CHIEF GEOLOGIST		
CHIEF SURVEYOR		
MINE PERSONNEL MANAGER		
CHIEF SECURITY OFFICER		
EVALUATION MANAGER		
ENVIRONMENTAL SUPT.		

CHAPTER 4

Narrow Reef Mechanisation at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand Limited

CHAPTER 4

Narrow Reef Mechanisation at Cooke 2 Shaft, Randfontein Estates Gold Mining Company, Witwatersrand, Limited

Following the successful motivation of the mechanised wide reef room and pillar project on the E8 Reef horizon at Cooke 2 Shaft, Randfontein Estates Gold Mine (REGM) and its commencement in mid-1984, I submitted proposals for the introduction of trackless equipment to the UE1A narrow reef. Approval for this narrow reef project was given in early 1985 and the initial development commenced immediately.

4.1 Arguments for Narrow Reef Mechanised Operations

The introduction of mechanised equipment to the E8 Reef at Cooke 2 Shaft, REGM previously described in Chapter 3, utilised a proven method of mining: the stepped room and pillar method. However, it was its introduction to a South African gold mine employing conventional mining methods from a vertical shaft system that was to prove to be significantly different and new to the industry. Now the proposed use of trackless equipment in narrow reef conditions would represent a major advance in the mechanisation of conventional gold mines.

The background in the South African gold mining industry to narrow reef mechanised mining, up to that point, can be stated simply. In 1983 small size LHD's had been introduced in narrow reef stopes at Anglo American Corporation's (AAC) Western Deep Levels Mine. These LHD's had been used to replace the winch in strike gulleys (ASG's). However the experiments were very limited and were restricted to a small area of the mine. Also, in 1984 AAC had indicated that trackless mining techniques were being considered, specifically at Vaal Reefs Mine, but there were no significant operations taking place there.

This proposal by KAR for a mechanised trackless operation on a narrow reef would therefore prove to be the first large scale operation of its kind on a South African gold mine.

When giving consideration to the proposals for trackless mechanised mining methods (TM3) in narrow reef conditions at Cooke 2 Shaft, KAR argued that the geology of the reef in the target area and the position of the established footwall haulages in relation to the reef horizon in that area had to favour the introduction of TM3. If **Figure 4.1** (transverse section of 95N12 line) is referred to it will be seen that the UE1A Reef in the area is very flat (0° - 5° dip) and it was also lying at only 5 metres above 95 Level elevation in a basin (or syncline in section). It was therefore considered as impractical to mine from 95 Level conventionally as the area was three kilometres from the main shaft system (see **Figure 4.2**) and there would be no capacity in any system of orepasses on 95 Level. The original planned conventional development programme provided for all footwall development to take place on 101 Level with orepasses in excess of 60 metres and some footwall development on 95 Level for top access to the stoping horizon; the alternative to this layout was the establishment of an interlevel which would reduce orepass development but would introduce an extra level which would be costly. In both these conventional layouts waste development would prove to be excessive. The development layouts for both these layouts are seen in **Figure 4.3** (101 Level and 95 Level) and **Figure 4.4** (101 Level with interlevel and 95 Level).

The total metres required to be developed for the two alternatives were 17979 metres for alternative one with no interlevel and 13866 metres for alternative two with an interlevel. Full details of these calculations will be seen in the motivation report. At the time the costs of these conventional options were estimated at R7,0 million and R5,6 million respectively. In today’s money terms this would be equivalent to R80 million and R60 million respectively.

In addition to the above argument, the up-graded haulage being established on 101 Level would be used for ore clearance from the UE1A project, utilising a single tipping point on the UE1A horizon and down a raise-bored orepass to the 101 Level haulage.

In terms of these arguments a trackless operation on the reef horizon which obviated the necessity for any excessive footwall development had to be considered as a viable alternative.

In summary, the proposed system envisaged a short ramp from 95 Level onto the UE1A reef horizon which would provide for the trackless access to the project and an extension of the 101 Level streamlined haulage to a point where a raise-bored orepass could be developed to the UE1A

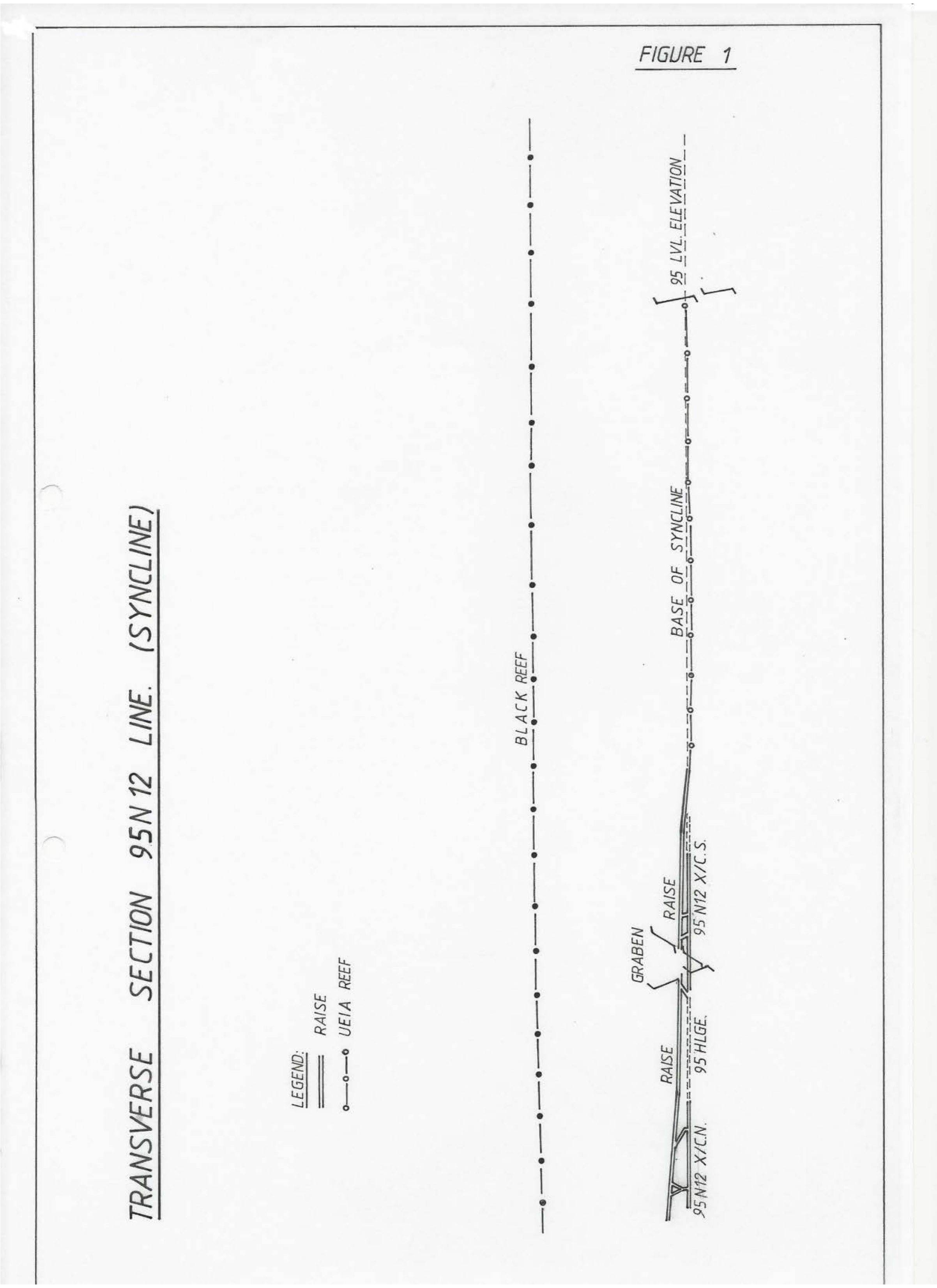

FIGURE 4.1

Transverse Section of 95N12 Line: refer to Annexure 4.2 Volume 2

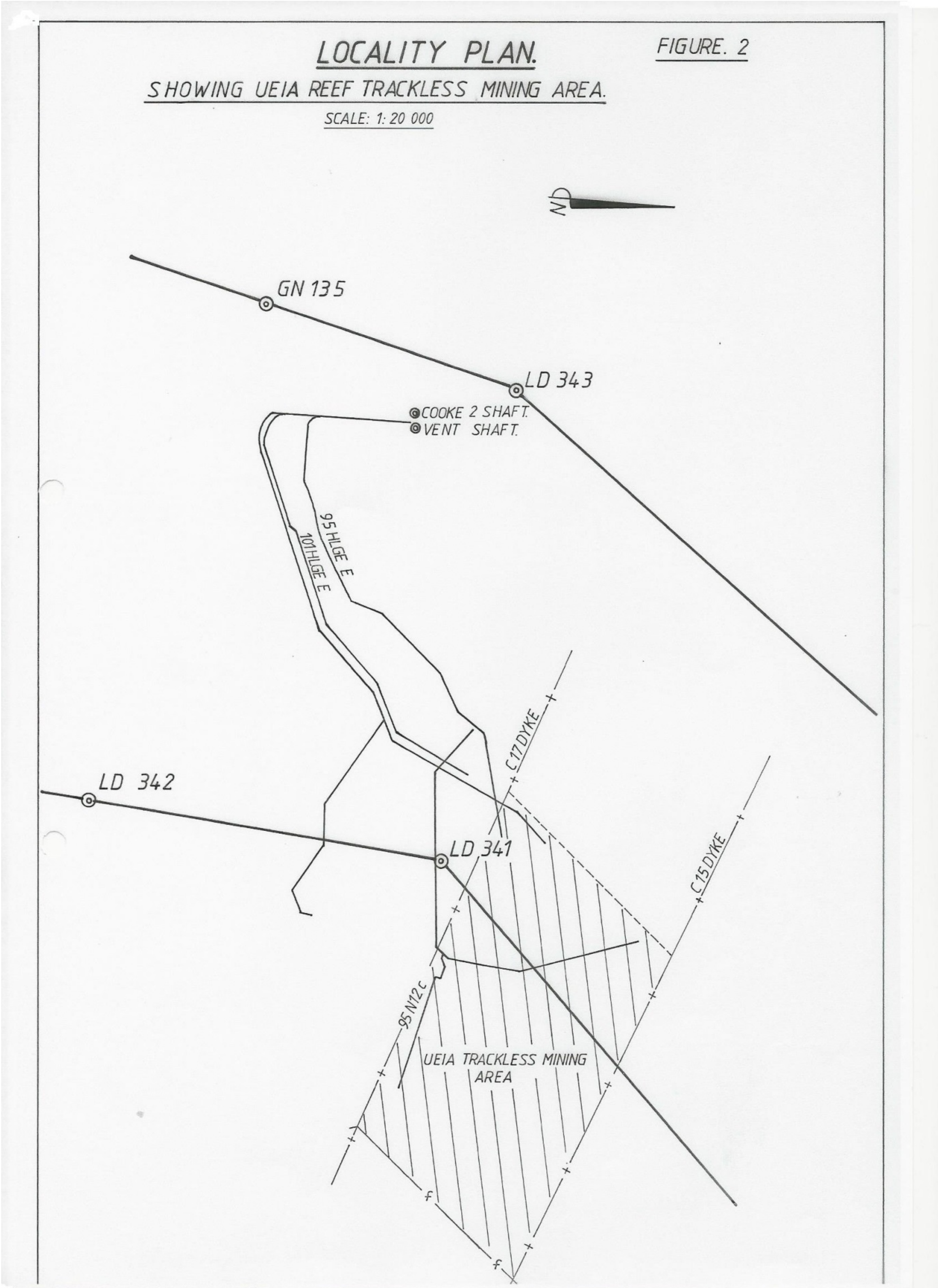

FIGURE 4.2

Proposed UE1A Trackless Mining Area: refer to Annexure 4.2 Volume 2

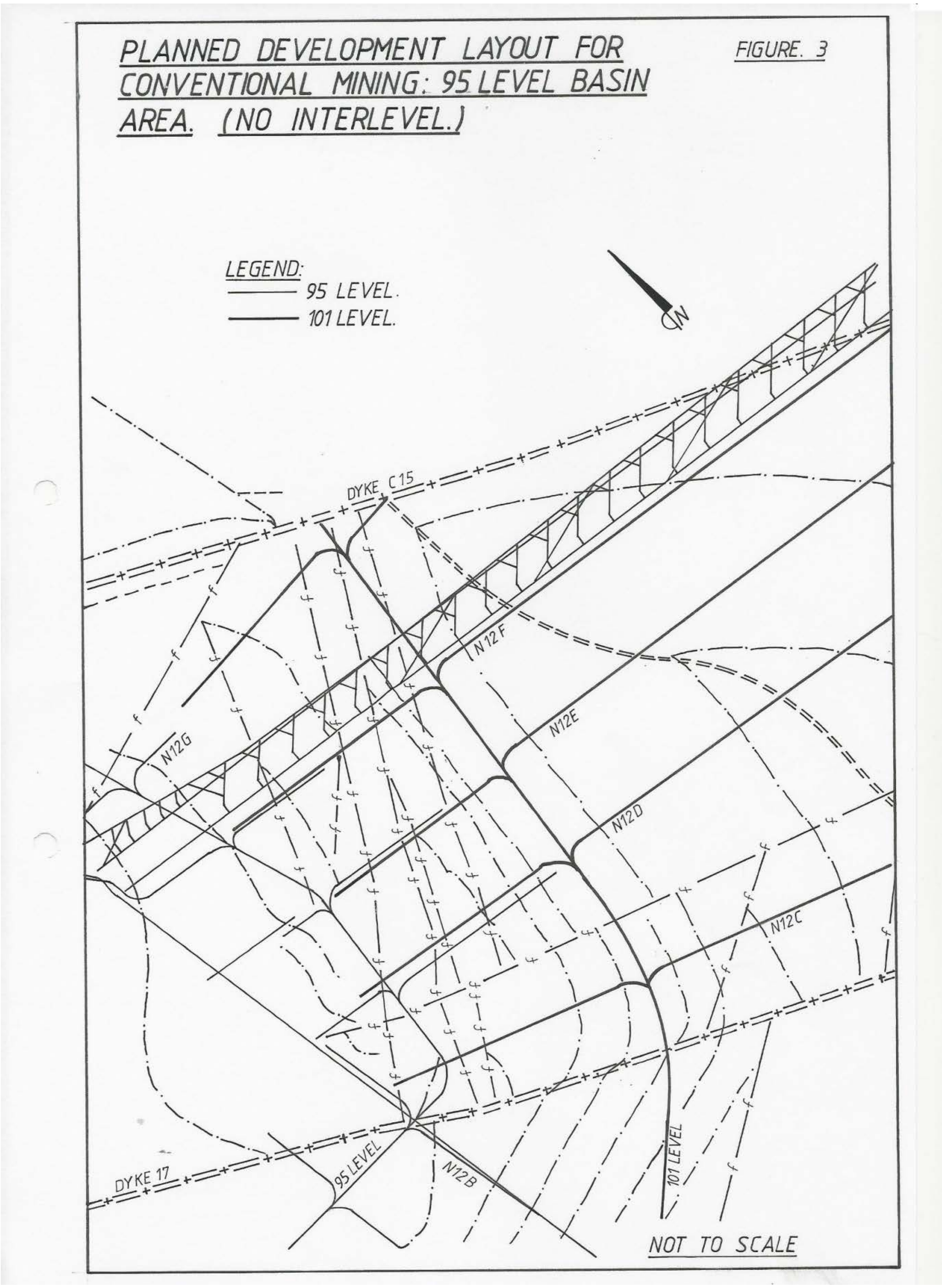

FIGURE 4.3

Conventional Development without Interlevel: refer to Annexure 4.2 Volume 2

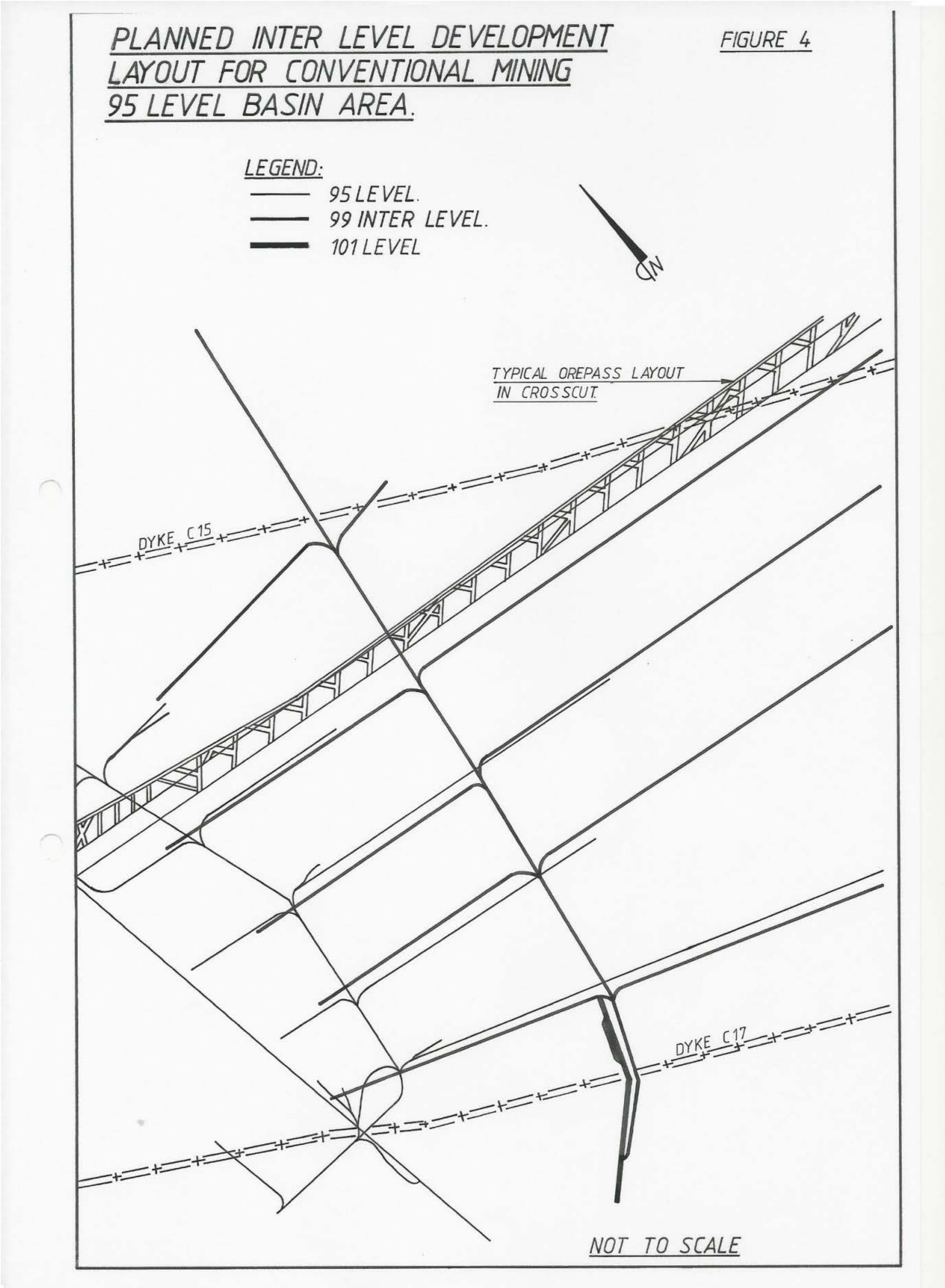

FIGURE 4.4

Conventional Development with Interlevel: refer to Annexure 4.2 Volume 2

reef horizon where a single main tip would then be established. This development, except for the workshop complex excavations on 95 Level, would represent the only waste development for the project. Thus the saving in footwall development was seen as a major argument for a trackless operation.

These arguments were the crux of the proposal made to the General Manager of REGM Mr W.J. van der Meulen and to the Consulting Engineer in late 1984. As referred to in previous chapters, Mr van der Meulen had gained confidence in KAR's proposals and motivations for the 90E8 wide reef project and in the manner in which that project was being managed, and for these reasons the motivation for the 95UE1A narrow reef project had his early approval. The motivational technical report was submitted on 01 February 1985 to the Consulting Engineer with a request for approval of the project.

4.2 Motivational Technical Report for 95L UE1A Reef at Cooke 2 Shaft

The **Motivational Report for the 95 Level UE1A Reef Project** by K.A.Rhodes can be seen in **Annexure 4.1** in **Volume 2**; technical details of the report are now discussed.

4.2.1 *Geology and Reserves*

The UE1A Reef in the area dips at between 0° - 5° and is in the form of a low profile basin (previously seen in Figure 4.1). The E8 Reef in the same area had low grades and was not as wide as in the 90LE8 project and was not therefore considered to be economic. The proven reserves of the UE1A Reef in the target area at that time (September 1984) were more than 750 000 tons over a channel width of 106 cms at a grade of 15,6 g/ton Au and 0,19 kg/ton $U_3 08$. In addition to these reserves there were further estimated reserves of 3,1 million tons at 5,3 g/ton Au and 0,18 kg/ton $U_3 08$ over a channel of 105 cms. Therefore total estimated reserves for this project were of the order of 3,85 million tons at a grade of 7,3 g/ton Au and 0,18 kg/ton $U_3 08$.

4.2.2 *Proposed Mining Method and Mine Design*

Due to the proximity of the UE1A Reef to 95 Level an access ramp was to be developed onto the UE1A Reef and once this had been done all operations would then be carried out on the reef horizon. Final ore clearance was planned to take place on 101 Level from a single orepass as stated.

4.2.3 ***Production Rate***

The maximum planned production rate from the operation was 50000 tons/month operating on a double shift basis. The mineable estimated reserve of 3,54 million tons (estimated reserve less 8% for crush pillars in stope and regional stability pillars) would provide for a life of project of about six years.

4.2.4 ***General Mining Layout***

Once established on reef, development would consist of access ramps or roadways (ARD's) across the basin and from these roadways access strike drives (ASD's) would be broken off at 40 metre intervals; these ASD's would be developed down dip of strike (-5°) which would ensure control of water from the drilling operations but necessitate pumping at the face when drilling was taking place.

The dimensions of the roadways were planned at 4,5 metres wide and 3,0 metres high to provide for truck tramming and the ASD's at 3,5 metres wide and 3,0 metres high. The general layout of the area is seen in **Figure 4.5** and the detailed layout of panels is shown in **Figure 4.6**.

4.2.5 ***Cycle of Operations***

Development

All development operations on reef (access roadways and ASD's) were to be carried out by drilling with electro-hydraulic drill rigs and cleaned by 3,8m^3 LHD's into 24 ton trucks with loading taking place in the access roadways.

Stope Drilling and Blasting

As stated in the motivational report it was always envisaged that drilling operations on the stope face would be carried out by hydraulic drill rig although for the purposes of the report, conventional face drilling had been assumed. Nevertheless, in early 1985 trials had begun with a bar type drill rig with a power pack provided by Delfos & Atlas Copco (Pty) Limited and in

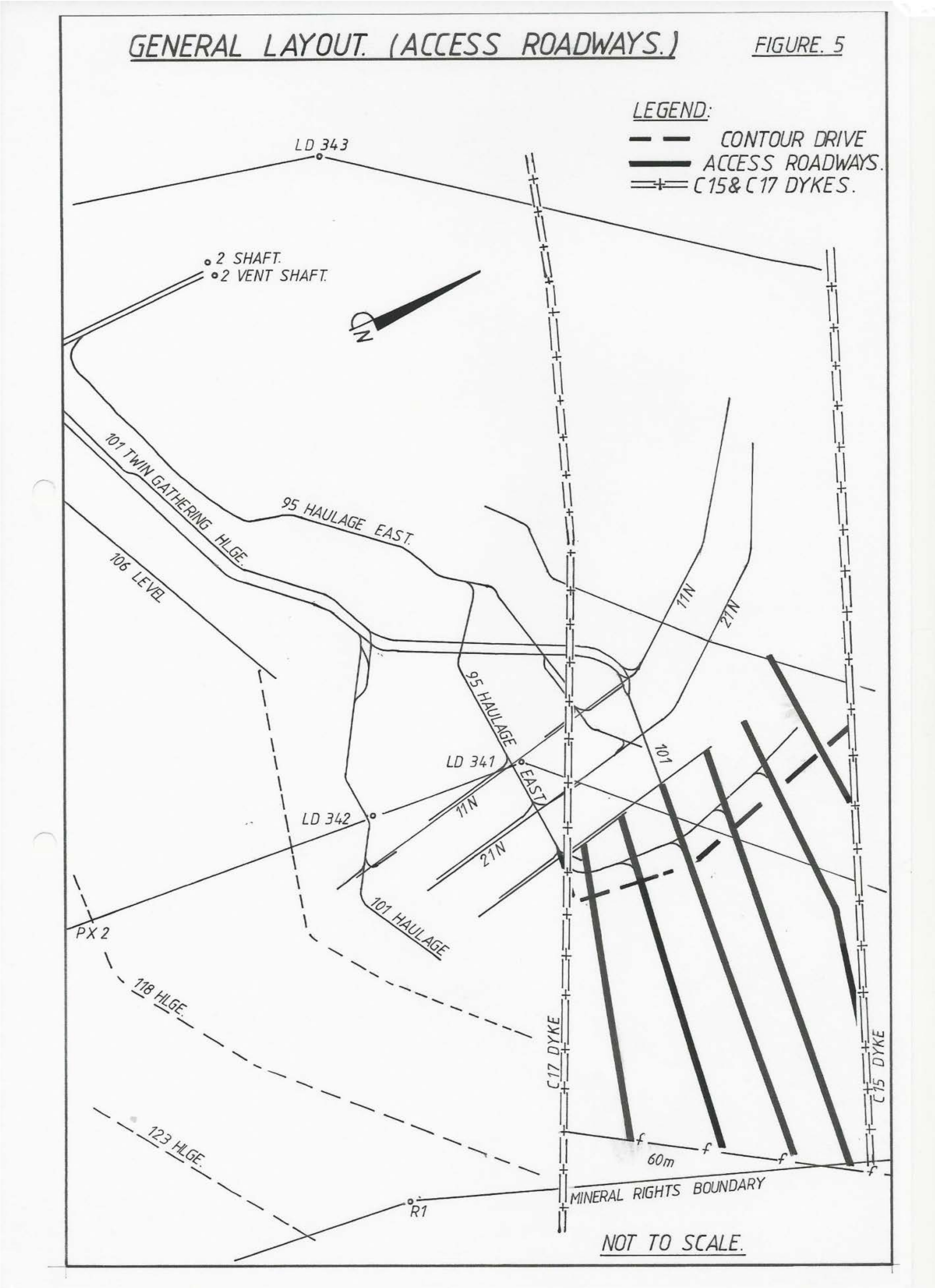

FIGURE 4.5

General Mining Layout with Access Roadways: refer to Annexure 4.2 Volume 2

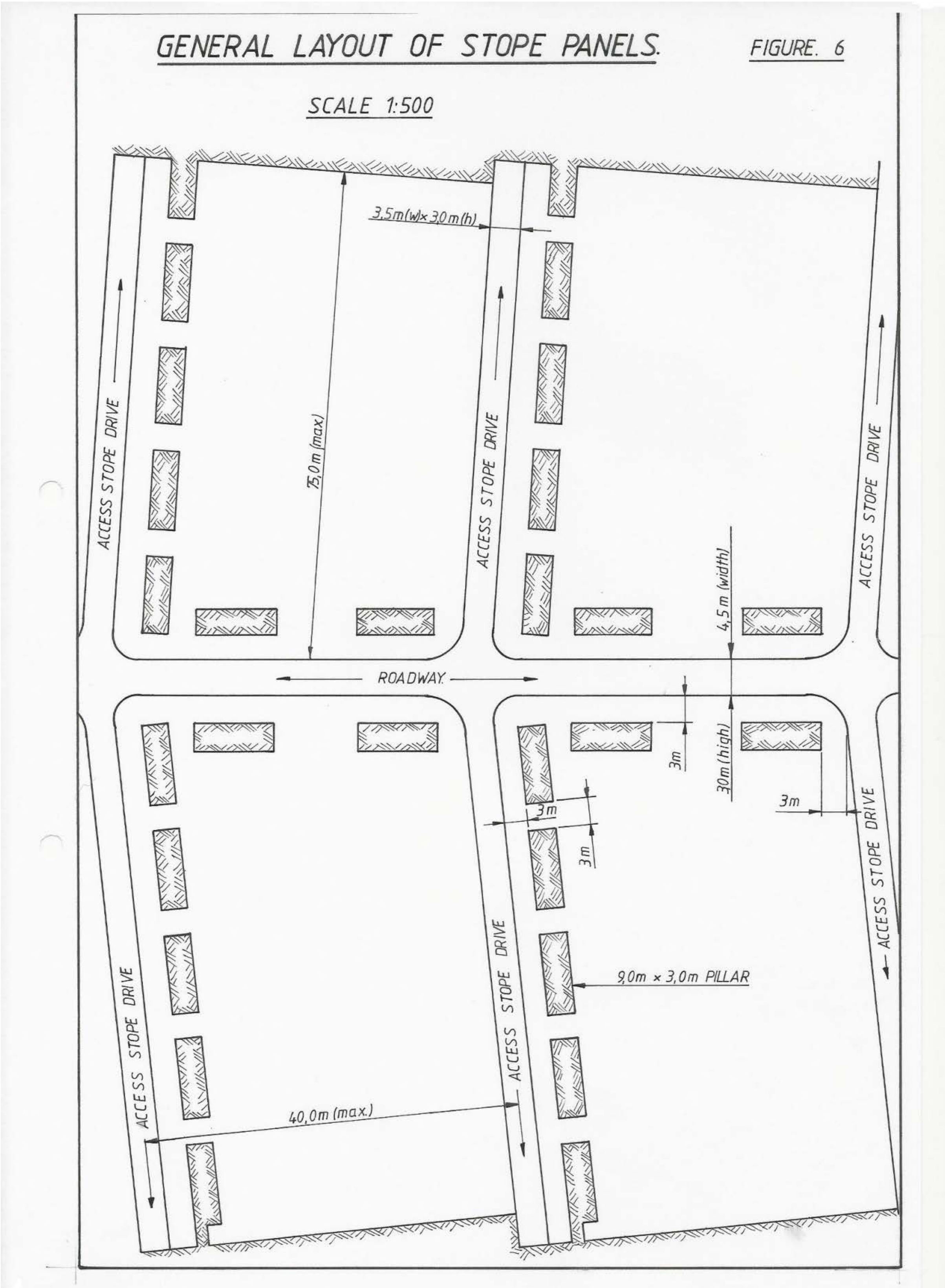

FIGURE 4.6

Detailed Layout of Stope Panels: refer to Annexure 4.2 Volume 2

April 1985 Atlas Copco had submitted initial proposals for a stope rig and were requesting REGM to share the capital cost of the project; see copy of letter overleaf sent by Atlas Copco for the attention of KAR. This then was the beginning of the development of the Stomec Drill Rig by Atlas Copco; however the prototype rig would only commence its trials after I had left REGM for H.J.Joel Gold Mine. The first studies on the progress with the prototype rig at Cooke 2 Shaft were reported on in March 1987 by the Technical Services Division of JCI. All indications at that time were that a production machine could be manufactured as the results from the trials were very positive. Refer to **Figure 4.7** for a photograph of the prototype Stomec drill rig seen operating underground at Cooke 2 Shaft at a later date.

Stope Cleaning

It was planned for the stope face to be cleaned by a face winch in the conventional manner into the ASD where LHD's would load and tram back to the access roadways and transfer into a dump truck; the truck would then travel to the main tip for transfer to the 101 Level (streamlined) haulage.

Stope Support

The method of support for stoping was to be the established conventional crush pillar and stick system; full details of the support system for stoping and for the development of roadways and drives had been given final approval by the Group Rock Mechanics Engineer.

4.2.6 ***Ventilation***

The total volume of air required to ventilate the area had been calculated at 160m^3/sec; such a quantity would support an output in excess of 50000 tons/month reef produced on a double shift basis with a possible re-entry period of two hours.

4.2.7 ***Equipment, Workshops and Engineering Considerations***

The primary equipment at steady state production had been defined: 4 x 3,8m^3 LHD's; 4 x electro-hydraulic drill rigs; 6 x 24 ton trucks; 2 x UV's; 6 x transport vehicles.

Atlas Copco

Delfos & Atlas Copco (Pty) Ltd
A Company in the Atlas Copco Group
Head Office

Mr Philips
Mine Manager

Your reference

Our reference ME/MS/85-109
1985.04.17

The General Manager,
Randfontein Estates Gold Mining Co. Ltd.,
P.O. Box 2,
RANDFONTEIN
1760

For the attention of Mr. K. Rhodes

Dear Sir,

Further to our discussion on Thursday 11th April 1985 regarding the Atlas Copco stope drilling rig, we wish to confirm the following :-

1. FEED BEAM : Atlas Copco are prepared to build a feed beam suitable for a hole depth of 2 m.

2. POWER PACK : The power pack used during the test period was on loan to us and should the test continue we would have to build a suitable power pack.

3. RESULTS : The results to date are as follows :-

3a. Average hole depth drilled 1,0 m.

3b. Net penetration 0,9 metres per min.

3c. Average holes drilled with a drill steel 5.

3d. Average number of sharpenings per steel 14.

3e. Average life of drill steel 75 m.

3f. Maximum number of holes drilled to the hour is 30.

3g. Number of holes drilled per set up 21.

3h. Time to set up 15 mins. time to take down 15 mins. time to move bar 10 mins.

3i. Maximum number of holes per shift is about 120. Due to face length we were not able to test this figure but it is what we could expect if the face was longer.

/2..

Directors Dr M.D. Marais - Chairman, M. Pellegrino - Managing (Italian), K.A. Belfrage (Swedish), H-B. Eriksson (Swedish), E. Liwendahl (Swedish), O. Sjöström (Swedish), Dr. A.K. Steyn, P. Wejke (Swedish)

Postal Address P.O.Box 504 Benoni 1500 South Africa | Street Address Lincoln Road Industrial Sites Benoni 1500 Transvaal | Telephone (011) 54 4411 | Telegraphic Address 'Atlascopco' Johannesburg | Telex 4-20850 SA

Atlas Copco

4. NEXT PHASE OF TEST : In order to continue with the test Atlas Copco must build a feed beam and a power pack, the supports, controls and rockdrill remain the same. Longer drill steels will also have to be used.

5. COSTS INVOLVED : The cost of a feed beam and a power pack would be R24000,00 and we would like Randfontein Estates to share these costs with us by investing R12000,00 in the project.

6. CONSUMABLES : Once again we would like the mine to supply consumables such as drill steels, hydraulic oil and electric supply to the power pack.

Drill Steels The drill steels required would be integral steels 25 mm hex with 158 x 25 shank and a forged collar. The overall length of the steel would be 2,4 m and head size 38 mm.

Part number 71712438-32 price R71,17.

7. PARTICIPATION : Once again we would like to stress our keen interest in developing a suitable stope rig for Randfontein Estates and we assure you of our full co-operation.

Yours faithfully,
DELFOS & ATLAS COPCO (PTY.) LTD.

[signature]
IAN MAC CONACHIE
UNDERGROUND EQUIPMENT

It is recommended that we proceed with this project and ~~therefore~~ we share the costs as stated above

KAN Production Manager
26.4.85.

FIGURE 4.7

The STOMEC Drill Rig in Operation Underground at Cooke 2 Shaft REGM

Notwithstanding the production parameters to be 50000 tons reef per month, in the original calculations for equipment KAR had assumed 60000 tons per month production; thus some conservatism was built into the estimate from the beginning.

LHD's

The original estimates by KAR for the number of LHD's required was four 3,8m³ units; this being based on the following simplistic calculations.

It had been assumed that there would be 47 shifts worked/month; this being the standard eleven shift fortnight worked at the time. A shift was assumed to be 7,3 hours working time. Further, assuming a machine availability of 85% and a utilisation of 80% then the production rate required would be

$\frac{60000}{47} \div (7,3 \times 0,85 \times 0,80) \quad = \quad 257$ tons/hour

It had been further estimated, by KAR, that the loading rate of the 6 ton LHD would be 100 tons/hour, therefore the number of LHD's required would be 257 ÷ 100 = 2,57 or say 3

One extra LHD would be required for waste packing.

Therefore the estimated number of 3,8m³ LHD's required would be 4.

In hindsight, the estimation of 100 tons/hour was optimistic. If the formulae later discussed in chapter 5 is used the loading rate of the 6 ton LHD, in circumstances of the 95 Level Project, would only be 68 tons/hour. However, with the machine hours being estimated in later projects at 280 hours in a month, the re-estimated fleet of LHD's would be 3,2 plus one for waste packing proving that the original estimate was marginally close to being correct.

Trucks

In terms of truck requirements, it was estimated that the tramming capacity of the 24 ton truck would be 50 tons/hour; this assuming a cycle time of 29 minutes, made of the following assumptions;

LHD loading time	=	14minutes
Tipping	=	1,5minutes

Turning round at tip = 1,5minutes

Tramming time (assuming 1000m one way run at 12kph) = 10minutes

Truck passing time (where one truck stands in lay-by) = 2minutes

Total cycle time for truck = 29minutes

Therefore tons/hour = $\frac{24}{29} \times 60$

= 50 tons/hour

Number of 24 ton trucks required = 257 tons/shift ÷ 50

= 5,2 or say 6

Therefore the estimated number of 24 ton trucks required would be 6.

Drill Rigs

In ASD's, assumptions were stoping width at 1,10 metres, panel length 40 metres with an SG of 2,75 which equals 121 tons/panel/metre advance.

At 20 metres face advance/month

Tons/month = 2420

Therefore required number of working panels for the planned tonnage = $\frac{60000}{2420}$

= 25 panels

Therefore metres advance/month for ASD's

25 panels x 20 metres/month = 500 metres

Assume 3,0 metres advance/round and 3 rounds/day/rig

Number of rigs required for ASD's = $\frac{500}{23{,}5 \times (3 \times 3)}$

= 2,4 (say) 3

Theoretically for access roadways (ARD's), if all development is pay and with ARD's at 150 metres spacing, ore reserve tons per metre ARD advance = 150x1,10x2,75

= 453

or = $\frac{60000 \text{ tons}}{453}$

= 132 m/month

It can be further assumed that only two rounds of 3 metres advance/round (6,0 metres/day) will be achieved in development due to longer tramming distances between development ends.

Therefore number of rigs for ARD's $= \frac{132}{23,5 \times 6}$

$= 0,9$ (say) 1

Therefore calculated number of rigs is 3 for ASD's and 1 for ARD's or 4 in total

In summary then, the primary equipment was planned for 4 x LHD's (3,8m³), 6 x 24 ton trucks and 4 x twin boom electric-hydraulic drill rigs.

It was intended to use the 95N12C crosscut (already developed) with additional excavations to provide full workshop facilities. The fuel supply to the project would be the same as the 90LE8 Project: initial transport by rail tankers followed up by a fuel pipeline system.

All the machines would be stripped on surface and transported through the shaft and along the 95 Level rail haulage, with re-assembly taking place in the workshop crosscut; this being the same procedure as for the 90LE8 mechanised project.

4.2.8 ***Labour and Efficiencies***

Labour complements had been set for the trackless operation and a comparison made with what would be required for conventional mining. This was as follows at the production rate of 50000 tons/month.

	Trackless	Conventional
NCWS complement	385	646
Tons/NCWS/shift	5,4	3,2
CWS complement	22	25
Tons/CWS/shift	94	83

4.2.9 ***Costs***

Comparative cost estimates had been made in the report for both the trackless and conventional options which clearly indicated that working costs for mechanised mining would be less than conventional; R11,50/ton against R15,06/ton respectively.

4.2.10 ***Dilution***

In this early stage of motivation of the narrow reef project it was obvious to KAR that the employment of large machines on a reef horizon with a channel width of 1,05 metres would be critically addressed by opponents of change. In fact the calculation in the motivational report showed that waste dilution could be less for the trackless option than for conventional mining provided that separation of the waste content, of the ASD's specifically, was effectively carried out. It was envisaged that the footwall waste in an ASD would be blasted first and trammed as waste by LHD to abandoned areas and dumped. The reef would then be stripped down from the hanging wall of the ASD in a separate blast. Even so, it was clear in the mind of KAR that this subject would continue to be a major issue in any narrow reef project and would have to be addressed again and again.

4.2.11 ***Additional Reef Hoisting***

In this motivational report KAR showed that because the waste development would be minimised it would therefore be possible to generate additional revenue from an increase in reef hoisting; at that time shaft hoisting capacity at Cooke 2 Shaft had been reached and replacement of waste hoisting capacity by additional reef could only increase profits. The calculated waste tons from conventional development had been estimated at about 265000 tons and trackless waste mining for workshops, streamlined haulage extensions, ramping and the raise-bored orepass from 101 Level was estimated at 45000 tons. The difference of 220000 tons over (say) a period of five years could realise an additional profit of R3,3 million/year based on assumed revenue and costs at that time.

4.2.12 ***Safety***

It was expected that accidents would reduce with the introduction of mechanised cleaning and tramming, when compared with the use of numerous small capacity trains on locomotive haulages. In addition, any reduction in the number of workers in the area would support the argument that accidents would be less due to fewer workers being exposed to falls of ground. Accidents due to locomotive haulage tramming and falls of ground were the major sources of serious accidents at REGM; therefore any reduction in these categories would result in a safer mine.

In terms of this motivational report it was argued that there was justification for a narrow reef trackless project on 95 Level at Cooke 2, Shaft primarily for reasons that footwall waste would be markedly reduced causing working costs to be less and enabling additional reef (to replace waste) to be hoisted and thereby increasing profits. Further, and not least, the mechanised operations would improve safety.

However the primary concern of waste dilution would remain, but KAR was confident that the practical measures outlined for its control would prove effective.

4.3 <u>The Start of Narrow Reef Mechanisation at Cooke 2 Shaft, REGM</u>

The formal approval of the 95 Level UEIA Narrow Reef Trackless Mining Project was given by letter on 12 April 1985: Vote 576. At that time it was not known by KAR that he had little more than four months left at Cooke 2, REGM before his transfer to the new gold mine JCI planned to develop in the Orange Free State.

In mid-1985 progress with the project moved quickly with both workshop development and development of an access ramp onto the reef horizon from 95 Level being carried out. Nevertheless, there were important issues to focus upon, specifically dilution control and also a follow-up on workshop strategy based on the experience gained from the ongoing wide reef project at Cooke 2 Shaft under the control of KAR.

4.3.1 ***<u>Dilution Control</u>***

This matter has already been highlighted but more discussion is believed important and necessary. It can be further stated that when big end development is carried out by mechanised equipment, questions are going to be asked about waste dilution and how will it be controlled in practice. KAR had given considerable attention to this matter and it is necessary to explain clearly how waste would be separated from reef in the development phase. In main access roadways (ARD's) and access stope drives (ASD's) it was planned to blast waste in a separate cycle and tram the waste initially to a waste tip, and later, following the normal geographic expansion of the workings, to a worked-out ASD to be packed. It was realised early that an LHD provided for selective loading and dumping which was not possible with a scraper winch working in a strike gulley in conventional mining. In addition, when waste is trammed separately by an LHD, control can be exercised over this cycle in that waste packed in a worked-out area of the mine can be seen and measured during the cycle. Nevertheless, in order to maximise the packing of waste in old workings it would be necessary to use a bulldozer to work alongside the LHD and ram the waste up to the hanging wall level; the LHD alone even with an ejector bucket would only be able to pack the waste to within 1,5 metres of the hanging wall.

Following clearance of the bottom cut (waste) the top section (reef) would be blasted down and trammed as reef. Although it could be theoretically possible to clean out all the waste in the bottom cut, in practical mining terms this would not happen, and for planning purposes it was assumed that only 60% of the waste cut would be trammed as waste and the remaining 40% waste would be cleaned out with the top reef blast; this would represent the dilution. Waste dilution calculations based on the above and the geometry of the layout showed that the dilution could be 7,0% in total for on reef development work. Originally it was thought that all the waste in the ARD's could be allowed to be trammed with the reef; however, later calculations showed that the overall dilution would then be 9,7%. It was then decided by KAR that ARD's would be treated the same as for ASD's (40% only of waste to be trammed with the reef) in order to reduce the overall dilution to 7,0%.

Referring to **Figure 4.8** it can be seen how the waste is blasted separately to the reef and **Figure 4.9** shows how the LHD is able to physically clean in an ASD where the stoping width is typically 1,20 metres. Dilution calculations are as follows.

<u>ARD Development</u>

Distance between access roadways	=	150 metres
ARD width	=	4,5 metres

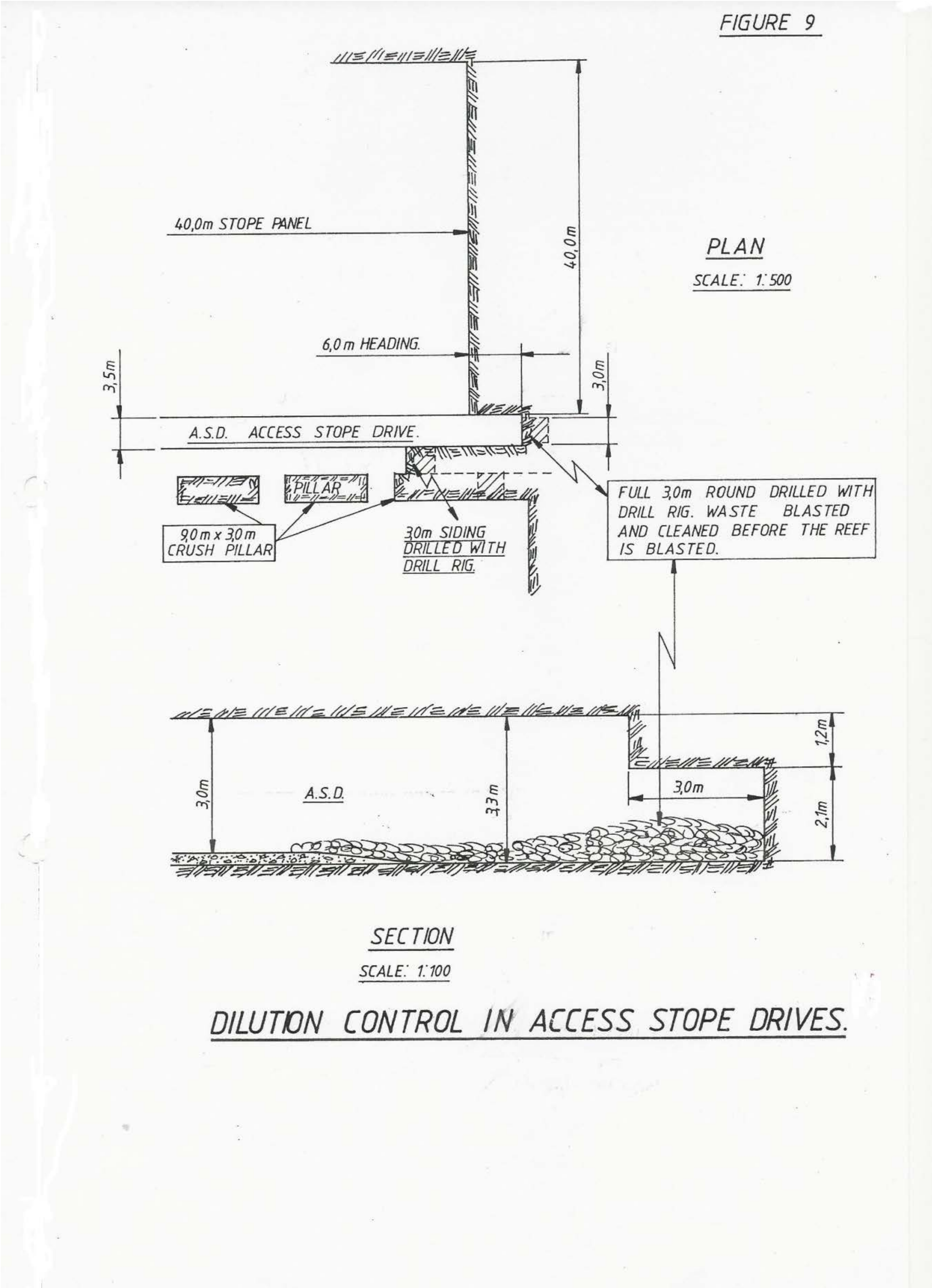

FIGURE 4.8

Dilution Control in Access Stope Drives: refer to Annexure 4.2 Volume 2

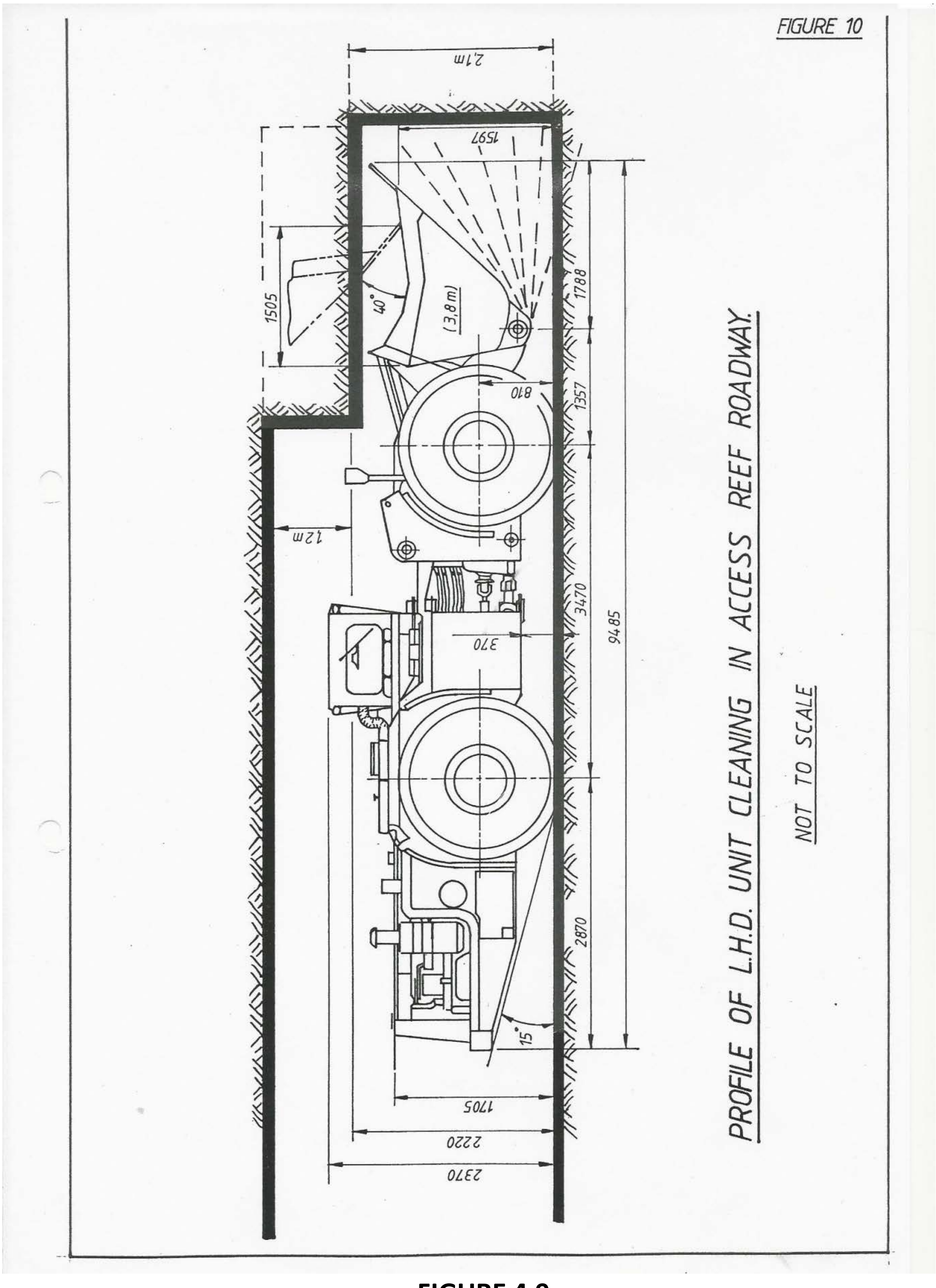

FIGURE 4.9

Profile of LHD Unit Cleaning in Access Stope Drive: refer to Annexure 4.1 Volume 2

ARD height	=	3,0 metres
Stoping width (assumed)	=	1,2 metres
Therefore:		
Ore reserve tons generated for 1 metre ARD advance		
	=	(150 x 1,2 x 2,75SG)
	=	495 tons
Development waste portion of ARD per metre advance		
	=	(4,5 x 1,8 x 2,75SG)
	=	22,3 tons
Waste trammed as reef	=	40%
Therefore:		
Dilution is calculated to be	=	$\frac{22,3 \times 40\%}{495}$
	=	1,80%

ASD Development

Panel length (centre to centre between ASD's)		
	=	40 metres
ASD width	=	3,5 metres
ASD height (excluding roadbed)	=	3,0 metres
Stoping width	=	1,2 metres
Therefore:		
Face tons blasted per metre advance by ASD		
	=	(40 x 1,2 x 2,75SG)
	=	132 tons
Waste portion of ASD advance	=	(3.5 x 1,8 x 2,75SG)
	=	17,3 tons
Waste trammed as reef	=	40%
Therefore:		
Dilution is	=	$\frac{17,3 \times 40\%}{132}$
	=	5,2%
Total dilution is therefore	=	1,8% + 5,2%
	=	7,0%

However, there would be additional waste generated from the necessity to develop turning circles for machines and also for tipping points. This waste, if allowed to be trammed as reef, would account for an additional 1% dilution. Refer to **Figure 4.10** and accompanying dilution calculations in **Figure 4.11**.

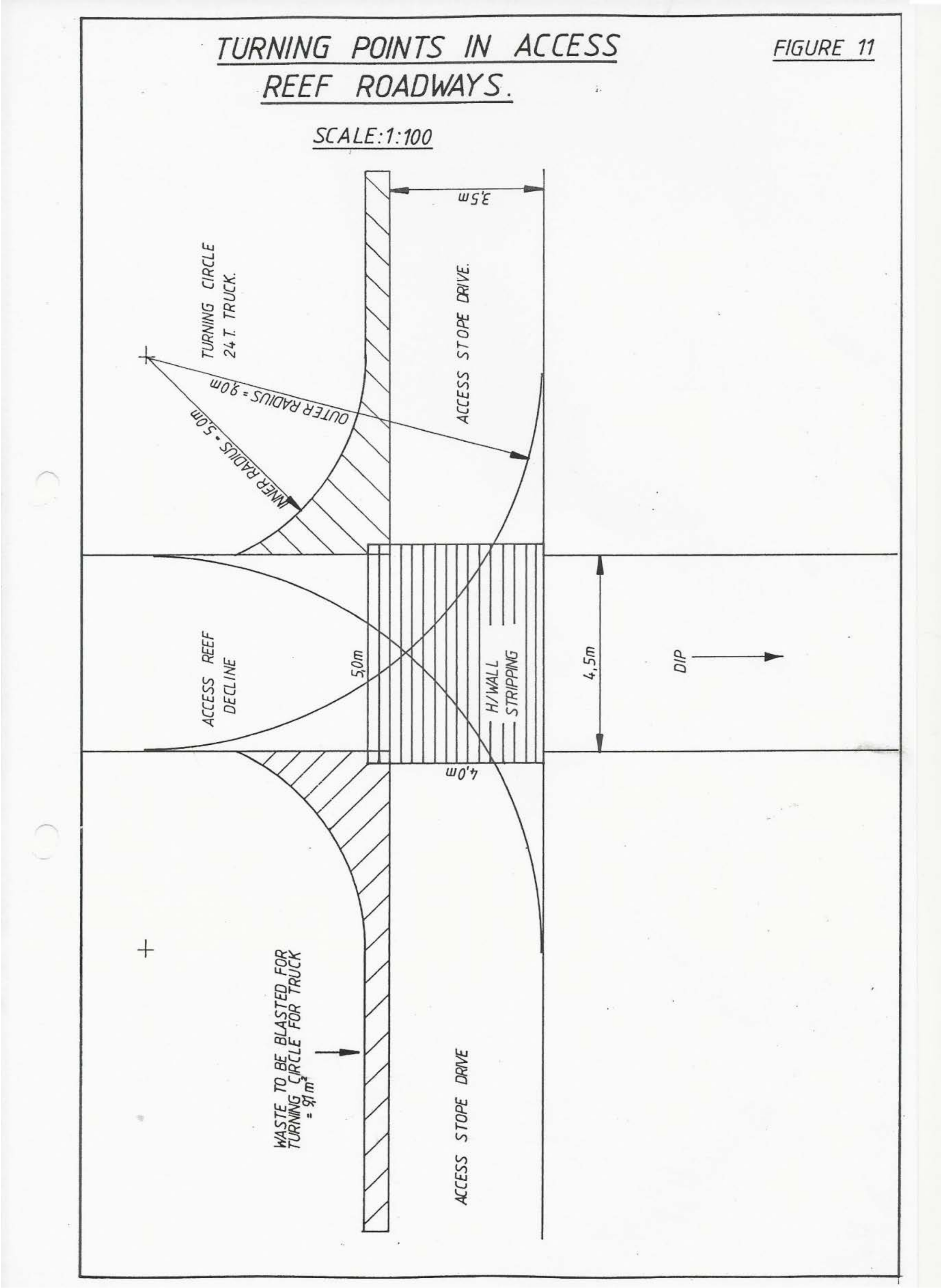

FIGURE 4.10

Turning Points in Access Reef Declines: refer to Annexure 4.2 Volume 2

CALCULATION OF DILUTION AT TURNING/TIPPING POINTS IN ACCESS REEF DECLINES: ACCOMPANYING NOTES TO FIGURE 4.10

Sidewall Blasting

Area of additional waste blasted	= 2 x 9,1	= 18,2m²
Tons of waste blasted	= 18,2 x 1,80 x 2,75	= 90 tons
Where: Average waste height	= 180cms	

Hangingwall Stripping

Area of hangingwall to be blasted (see diagram)		= 20m³
Tons of waste blasted	= 20 x 1,5 x 2,75	= 83 tons
Where: thickness of hangingwall stripped	= 1,5m	

Total waste tons blasted	= 90 + 83	= 173 tons

Total reef produced in a panel	= 40 x 150 x 1,20 x 2,75	= 19800 tons
Where: Face length	= 40m	
Advance	= 150m	
Average stoping width	= 110cm	

Theoretically for one turning point every 150 metres between access roadways

$$\text{Dilution} = \frac{173}{19800} \times \frac{100}{1}$$

$$= 0{,}87\%$$

$$= \text{(say) } 1{,}0\%$$

N.B: This dilution will only occur if the waste is trammed as reef

FIGURE 4.11

In terms of these waste dilution calculations it was also possible to confirm that there would be sufficient volume (of space) available in the worked-out areas to accommodate all the waste, blasted and trammed, from the on reef development.

Previous calculations had shown that waste dilution for any conventional mining in the same area on 95Level, where the reef was flat and with minor faulting thereby necessitating deeper gullies (3 metre depth from the top reef contact) in order to negotiate these conditions, would be 6,9%, very much the same as the 7,0% calculated above for trackless mining. It was therefore argued that waste dilution from mechanised mining need not be greater than for conventional mining and the operation of large machines when mining narrow reefs did not necessarily imply higher waste dilution. However the importance of management control over these operations cannot be over-stressed if dilution was to be controlled.

4.3.2 ***Size of Equipment and Dilution Control***

In the 90LE8 Wide Reef Project it was stated that the largest size units had been selected for the reason that the larger units would cause a reduction in working costs and it was also believed that this argument remained the same for narrow reef mining. Notwithstanding the above, in narrow reef conditions attention had to be paid to their possible effect on dilution and on the on reef roadway dimensions.

In conclusion, the largest practical size units were selected for the narrow reef project taking cognizance of the need to work within the limits of waste dilution. In other words, the selected equipment would operate in the roadway dimensions stated in the previous dilution control calculations. A further important factor in equipment selection was the use of the ejector bucket on LHD's. This issue has been discussed previously in the 90LE8 Project but in narrow reef mining it was even more important. When loading a truck with an ejector bucket the required height at the tipping point is less than that required with a conventional bucket; refer to the sketch in **Figure 4.12** showing reduced height

SKETCH SHOWING ADVANTAGES OF E.O.D. BUCKET

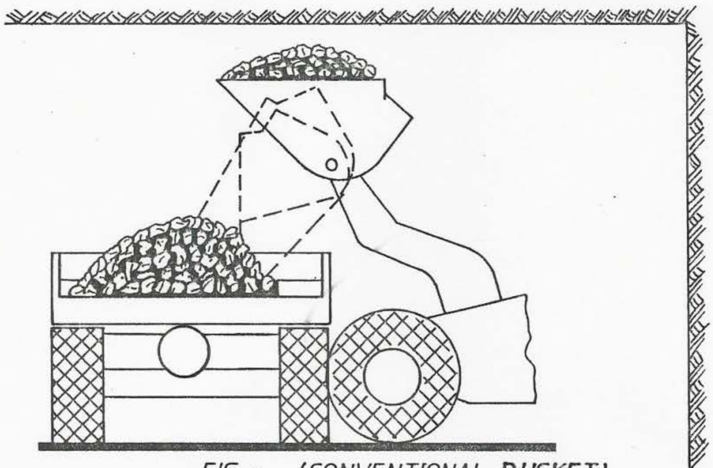

FIG. a (CONVENTIONAL BUCKET)

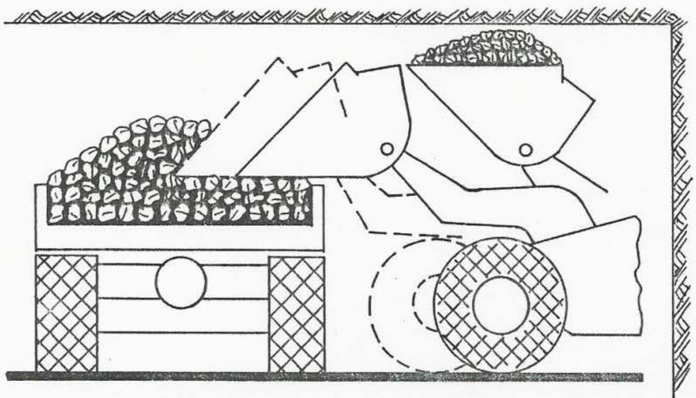

FIG. b (E.O.D. BUCKET)

N.B. 1. Roadway height in Fig. (b) is less than in Fig. (a).

2. Reach of L.H.D (with E.O.D. bucket) in Fig. (b) is greater than for L.H.D in Fig. (a).

FIGURE 4.12

Advantages of EOD Bucket: refer to Annexure 4.2 Volume 2

and therefore less waste. In addition the sketch indicates the advantage of being able to select a larger sized truck because the ejector bucket pushes the load horizontally across the truck bowl.

4.3.3 ***Workshop Strategy***

The experience gained with the introduction of equipment to the 90LE8 Project had shown that it would be better to establish a fully equipped assembly bay before allowing any trackless equipment to be stripped on surface and allowed to go underground. Refer to **Figure 4.13** for details of the 95 Level assembly bay which was planned to be used later in the project as the wash bay. With regard to workshop layout, certain footwall development work had been completed for anticipated conventional scattered mining before the planning of this project had begun and use was made of these excavations when setting out the workshop facilities. For this reason the workshop complex was not going to be ideal but would be a practical compromise in the circumstances. Refer to **Figure 4.14** for the overall layout of the workshop area and **Figure 4.15** showing a photograph of the development by conventional methods of the 95 Level assembly bay in early 1985.

4.3.4 ***Justification for the Narrow Reef Project***

The concept of a trackless operation for the narrow UEIA Reef on 95 Level at Cooke 2 Shaft, REGM had been accepted, and a capital vote approved. The justification for the project was that there would be a reduction in working costs compared to the normal practice of conventional scattered mining. The lower costs would be realised for reason that stoping costs would be less; footwall development costs would be markedly reduced; ancillary operations on footwall service levels were virtually eliminated except for the streamlined gathering haulage.

In addition, because waste development linked to the project would be minimal there would be an opportunity for additional reef hoisting to provide further revenue.

Finally, the trackless system would prove to be safer than conventional mining.

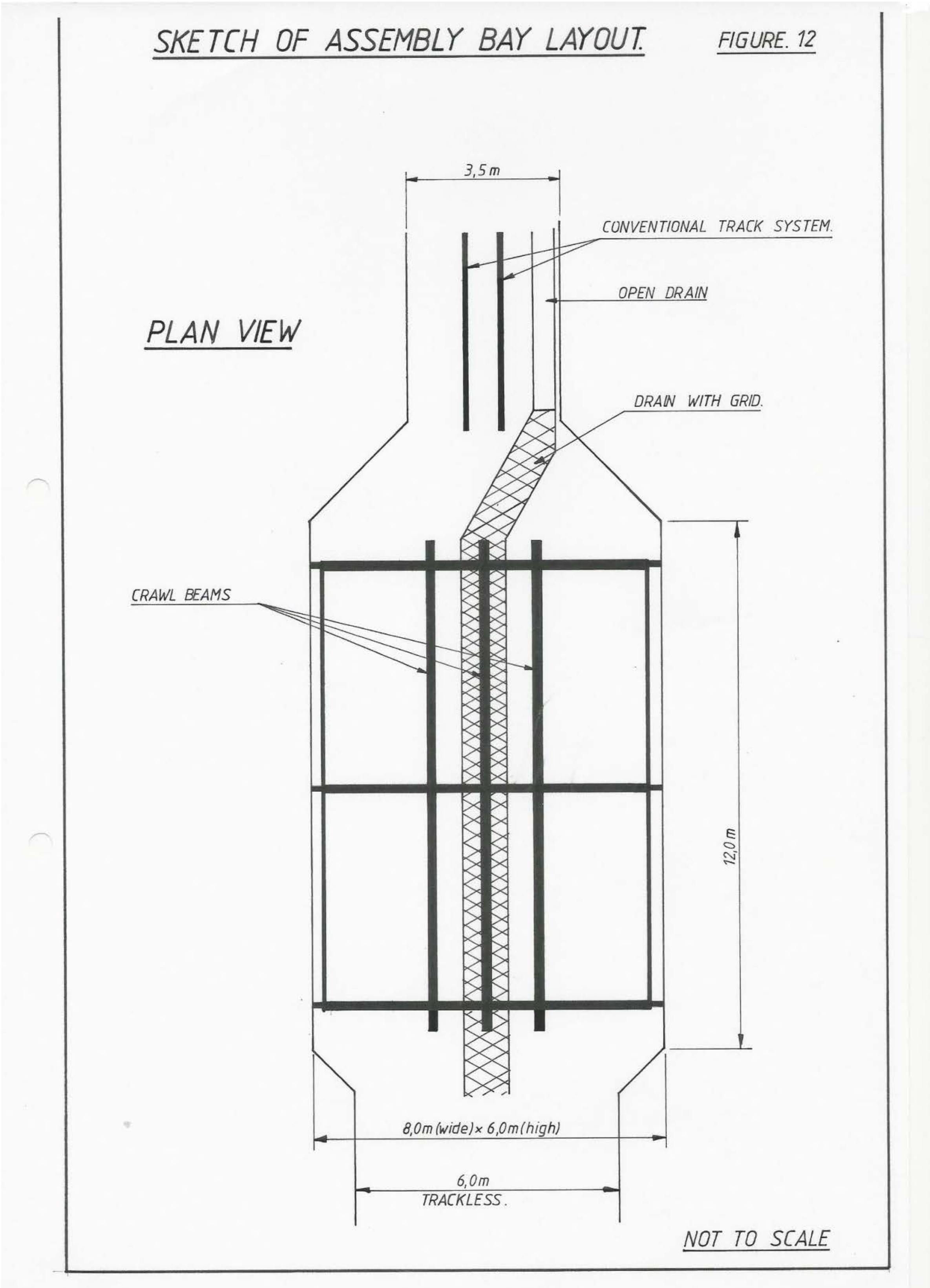

FIGURE 4.13

Assembly Bay Layout: refer to Annexure 4.2 Volume 2

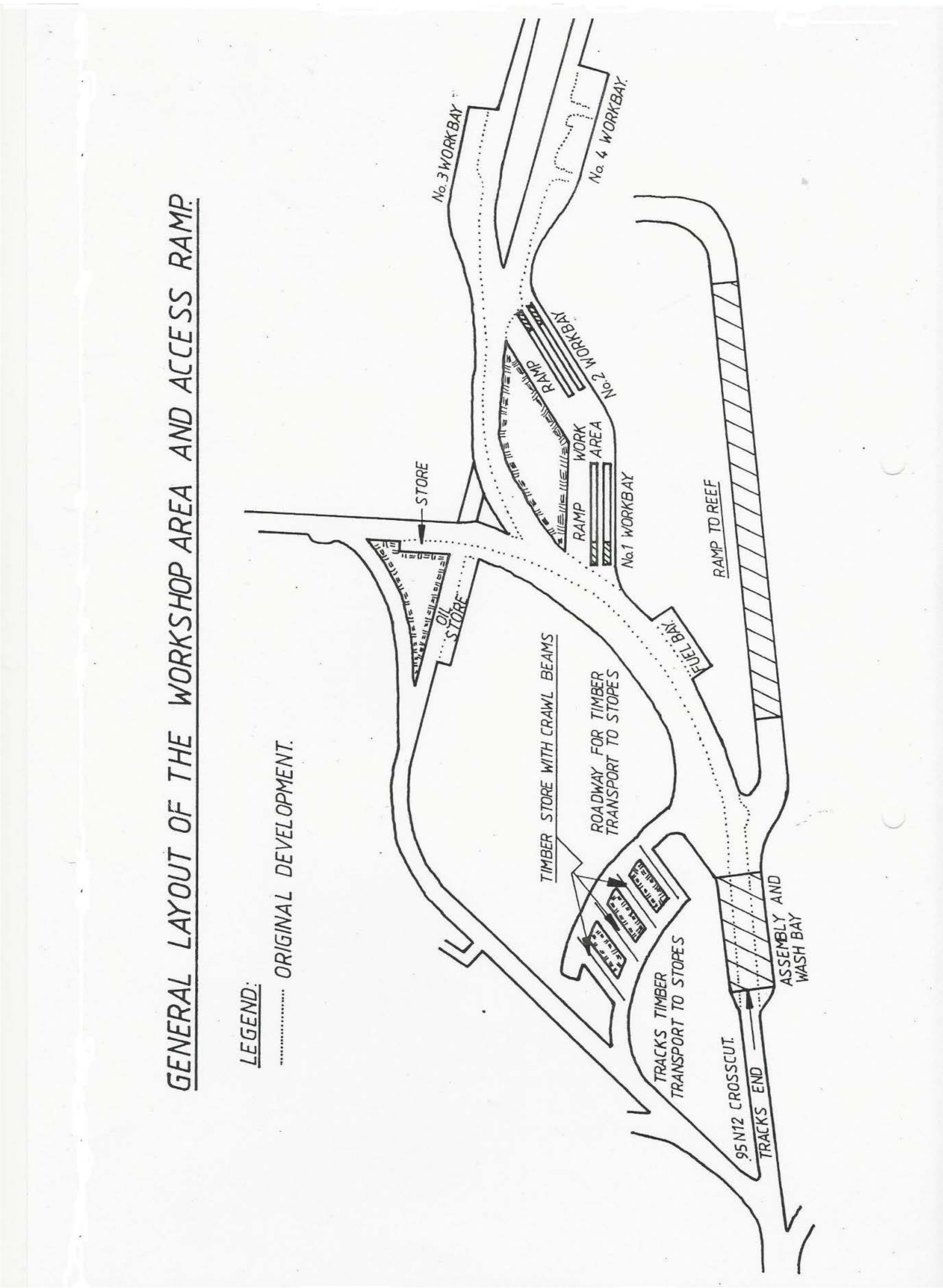

Figure 4.14

General Layout of 95 Level Workshop: refer to Annexure 4.2 Volume 2

FIGURE 4.15

Conventional Development of the 95 Level Trackless Workshop Assembly Bay: refer to Annexure 4.2 Volume 2

In due course (1986) KAR prepared and presented the paper "***Planning for a Trackless Access Stoping Operation in Narrow Reef Conditions"*** to a JCI Technical Meeting and later submitted it to the Association of Mine Managers of South Africa (AMMSA) for publication; however the paper was withdrawn by JCI from publication in the transactions of AMMSA for confidentiality reasons. Refer to **Annexure 4.2** in **Volume 2** for a copy of the paper.

4.3.5 ***Conclusions***

In late August 1985, shortly before KAR left REGM for the H.J.Joel Gold Mine, the progress on the project was such that the assembly bay was 98% complete; the ramp ex the workshops crosscut (seen in figure 4.14) had 76 metres to go to reef and would be complete by the beginning of October; development of the workshop bays were well underway; the majority of the equipment was ordered and would be sent underground when the assembly bay and the first workshop bays were completed.

This operation at Cooke 2 Shaft, REGM was to be the first major narrow reef trackless mining project in a South African gold mine and was expected to reach the planned production rate of 50000 tons/month in March/April 1987.

KAR was however not to be in charge of this project much longer as he was to be transferred from REGM to the H.J.Joel Gold Mine, a mine having been planned conventionally but where shaft sinking had not yet begun. Nevertheless, following KAR's new appointment, the H.J.Joel Gold Mine mine would soon be designed as the first totally trackless gold mine in South Africa based on the work carried out by KAR for the 95 UEIA narrow reef project at Cooke 2 Shaft; the full documentation of this change of design and the build-up to gold production at the new mine will be detailed in Chapter 5.

Before concluding this chapter on narrow reef trackless mining it is important to record the planning and work carried out for the streamlined rail haulage on 101 Level at Cooke 2 Shaft and work proposed for the exploitation of the Kimberley Reefs from Cooke 2 Shaft, REGM.

4.4 101 Level Streamlined Haulage

At Cooke 2 Shaft the major portion of ore reserves were approximately 3,5 kms from the shaft system. The need for an efficient ore clearance system serving these reserves was dictated by the planned increase in tonnage in the five year plan. In early 1984 certain work on the 101 Level so-called streamlined rail haulage, to serve both 90 Level and 95 Level, was underway. However with the geographic expansion of the workings, specifically the new trackless projects, stoping tonnage would increase significantly and the improvements to the 101 Level haulage, which were then being carried out, would be inadequate and a further upgrading (Phase 2) was therefore considered necessary by KAR.

The planned tonnage from 90 Level, 95 Level and 101 Level would increase from 3600 tons/day in mid-1984 to 5500 tons/day in the second half of 1985. In terms of the then current streamlined programme (call it Phase 1) the expected maximum tramming capacity of 101 Level would be 3000 tons/day. It was axiomatic that this capacity would not meet the requirements of the new five year plan and the planning of both trackless projects (90 Level E8 and 95 Level UEIA) could therefore be jeopardised. Thus the importance of the 101 Level haulage to the new trackless initiatives, could not be over emphasised.

4.4.1 *Constraints to the Phase 1 Programme*

The first constraint related to the quality of the track. Rail ballast consisted of run of mine development waste as was common on normal rail haulages on gold mines. This would not be acceptable for high speed tramming which would be essential to a streamlined haulage that KAR had in mind. Secondly, the rails, 30kg/metre, were worn with poor fishplate joints. Thirdly, there was no provision for any increased rail gauge on bends which was the norm of general railway design parameters; in this respect variable gauge concrete sleepers were available. Finally, track maintenance, including the control of water, was considered too labour intensive, expensive and inefficient for the required streamlined haulage.

There were also engineering considerations to overcome. The recently re-furbished trolley line locomotives had no standard components and with spares availability inadequate downtime was inevitable. In early 1984 the overall availability for the four locomotives on the level was only 52%. Also there were three different types of motors being used, each with its own traction wheel characteristics.

Finally, the Phase 1 upgrade did not provide for any improvement to remove any of these constraints.

4.4.2 ***Recommendations for the Further Upgrading of 101 Level Haulage: Phase 2***

There had to be radical changes to make the 101 Level haulage an efficient streamlined system and the proposals from KAR, in his motivational report, were as follows.

Twin Haulage System

The 101 Level was only planned to be a single haulage, again normal on multi-level gold mines at that time. However it was now proposed to introduce a twin haulage one way travelling system. A second haulage, the RAW which was already developed, would become part of the twin haulage.

Transfer Boxes

In order to ensure continuous movement of trains when loading at the transfer boxes and to avoid shunting delays, additional development would be necessary and also the use of hanging wall chutes would be constructed in the centre of roadways.

Track Installation

The track was to be lifted to provide for a minimum of 300mm thickness of ballast below the sleepers. The old ballast would be removed/vamped, using the purchased vamping Toro 150 referred to in the previous chapter, and new ballast, graded 15mm to 50mm, imported to the mine from surface: 600m^3 ballast/km of track. This new ballast would provide the necessary well drained support for the concrete sleepers and would greatly facilitate on-going alignment and levelling of the track. Finally the use of a ballast tamping machine was recommended.

Sleepers

All new trackwork would use concrete sleepers 900mm spacing and wider gauge on curves.

Rails
Where necessary existing rail ends would be cropped and fishplates huck bolted with crown welded rail joints.

Rolling Stock
The trolley line locomotives then in use could not be relied upon and were a risk to production. It was therefore proposed to acquire 15 ton trolley locomotives with 18 ton bottom discharge hoppers (6 ton hoppers were the norm on haulages). Refer to photographs in **Figures 4.16** and **4.16A** showing a new trolley locomotive and a 18 ton hopper on surface at Cooke 2 Shaft.

Labour
There was to be no general increase in labour complements but the signing on of a qualified tracklayer was considered essential.

The first motivational report submitted by KAR was dated 28 May 1984 with the final motivational report from KAR to the General Manager REGM and the Consulting Engineer JCI requesting the capital vote, submitted on 30 October 1984. Both these motivations and submissions are included in **Annexure 4.3** in **Volume 2**.

Towards the end of 1985, before KAR left for H.J.Joel Gold Mine in late August, the new upgraded 101 Level haulage was virtually complete and operational. In the photograph in **Figure 4.17** one can see the completed new streamlined haulage and **Figure 4.18** shows the Plasserail Mechanical Tamping Machine underground on 101 Level; a 'first' in a South African gold mine. Related documentation showing track standards and the cost motivation for the Plasserail Tamping Machine are all shown in **Annexure 4.4** in **Volume 2**.

4.5 Kimberley Reefs

Before KAR's departure from REGM a final motivational report, dated 30 July 1985, was submitted for the exploitation of the Kimberley Reefs at Cooke 2 Shaft. This project had been thoroughly planned in the previous months following an initial report by KAR in May 1985.

FIGURE 4.16

15 Ton Trolley Locomotive on Surface at Cooke 2 Shaft, for use on the Streamlined Haulage

FIGURE 4.16A

18 Ton Hopper, for Use on the Streamlined Haulage, Standing on Surface at Cooke 2 Shaft

FIGURE 4.17

The new 101 Level Streamlined Haulage at Cooke 2 Shaft

FIGURE 4.18

The Plasserail Mechanical Tamping Machine on 101 Level Cooke 2 Shaft

The Kimberley Reefs were a group of reefs identified immediately adjacent to the Cooke 2 Shaft system. There were various reef combinations for exploitation and it was possible that both trackless wide reef and narrow reef mining methods could be employed. At the time there were no proven ore reserves established but from extensive drilling (71 boreholes) there were possible/probable reserves (in terms of the definitions at that time) of almost 4,5 million tons at 4,33 grams/ton with a stoping width of 138 cms. There was the possibility of combining reefs to increase the stoping width at a lower grade; this would enable mechanised wide reef mining to take place.

In the report it was proposed to develop from 95 Level station to the point of reef intersection from where on reef development would be carried out. In terms of ore clearance a gathering haulage would be developed on 106 Level.

Justifications for the project were based on similar arguments as for the 95Level UEIA Project and full details of financial advantages were outlined in the motivation report. The application for the capital vote for the project was submitted to the Consulting Engineer with the report ***Exploitation of the Kimberley Reefs at Cooke 2 Shaft by Trackless Mining Methods*** by K.A.Rhodes dated 30 July 1985, one month before KAR left REGM; refer to **Annexure 4.5** in **Volume 2**.

4.6 Concluding Remarks

By the end of August 1985 the 95 Level UEIA Project, the first mechanised narrow reef mining project on a South African gold mine, was progressing well. In addition, the new upgraded streamlined haulage was in operation for both the wide reef and narrow reef trackless projects. Also approval for development of the Kimberley Reefs, starting from Cooke 2 Shaft 95 Level Station, would soon be given.

As a final comment, the revolutionary new method of mechanised trackless mining of narrow reefs in South African gold mines had commenced. This method of mining proved common in later years and is still being used today. The method first introduced by KAR at Cooke 2 Shaft, REGM in 1984/1985 is now widely known on South African mines as the hybrid system.

Now the big challenge for KAR was the design, planning and management of the first totally trackless gold mine in South Africa: the H.J.Joel Gold Mine in the Orange Free State. This new mine project will be discussed in detail in chapter 5 of this treatise.

CHAPTER 5

The Design, Planning and Management of the H.J.Joel Gold Mine

CHAPTER 5

The Design, Planning and Management of the H.J.Joel Gold Mine

In this chapter KAR will detail the work carried out by him on the design and planning of a new gold mine and the early years in the life of the mine when he was the first appointed mine manager. It is intended to set out a brief introductory narrative followed by a technical discussion of work over a three year period, from the start of the mine to the time the mine came into production and was officially opened.

5.1 Introductory Narrative

In late August 1985, when I was still the appointed Manager Mining at Cooke 2 Shaft REGM and responsible for pioneering the new trackless projects, I was called into the Head Office of JCI by the Technical Director. I was told that I was to be transferred immediately to the H.J.Joel Project which was planned as a new gold mine in the district of Theunissen in the Orange Free State. At that time REGM had a 37% equity interest in the project and it was anticipated that in early 1986 the Minister of Mineral and Energy Affairs would grant JCI's application for a mining lease.

Although there was a sense of euphoria within JCI, because after many years the company would be returning to gold mining in the Orange Free State, I was disappointed by my transfer as I believed that I still had much to complete with the trackless projects which I had started up at Cooke 2 Shaft REGM. I mentioned these doubts at the meeting with the Technical Director but the response was that it was time to move on. I did not at any time during the interview, or subsequently, receive any directive to review the design of the new mine; at that time a draft feasibility study submitted by G.W.Tregoning in April 1985 had been based upon conventional mining.

I could never be sure, and at the time I did not think about trying to find out, what the Technical Director or Consulting Engineer thought would be my reaction to having to leave REGM. After a period of two years of pioneering the use of trackless mining at Cooke 2 Shaft I was not about to abandon my efforts to advance mechanisation within the JCI Group. I had not been given any directive to review the design of the new mine with regards to a change to mechanisation but I was determined to do so. Therefore, without any mandate, I decided to commence the necessary work to set out an initial proposal for trackless mechanised mining for the new mine.

After being told of my transfer I returned to REGM and remained at Cooke 2 Shaft for the handover period to the newly appointed manager. However, during this period of approximately two weeks I moved to an isolated ‘office’ in a section of the Mine Rescue Complex, a short distance from the shaft offices. I gave instructions to be called upon only for any advice if it was needed; in other words I allowed the new manager to get on with his job of managing operations at the shaft whilst I commenced the early design work of a new trackless gold mine.

My first motivational report, dated 19 September 1985, was submitted to the Consulting Engineer and I then proceeded on a short leave.

5.2 Early Motivations for a Trackless Mechanised Mine at the H.J.Joel Project

At the end of August 1985 and going into September it was necessary to deliberate some of the basic factors which would be important for the motivation of a trackless operation at the H.J.Joel Project. The issues considered are seen below.

5.2.1 *Costs of an Operating Level*

It was necessary to estimate the costs of operating a conventional footwall haulage; this was important because a prime motivator for a trackless operation utilising the trackless access gathering haulage concept was the reduction in footwall waste development and the elimination of footwall service levels. It was therefore important to determine all the labour, both mining and engineering, required to operate a footwall haulage and, in addition, the conventional costs of maintaining a level in terms of mining and engineering stores and power costs. In fact an all inclusive cost of R/ton was necessary.

5.2.2 *Reduction of Waste Development*

For the motivation of a trackless mechanised mine design all conventional footwall levels had to be eliminated, except for the single gathering haulage, thereby saving the costs for operating footwall haulages. Footwall development costs would then be markedly reduced. Also, if waste was not generated from footwall development the hoisting of waste would be significantly less and in terms of hoisting capacity there would be potential for an increase in reef hoisting.

5.2.3 ***Gathering Haulage***

The requirements for the gathering haulage were at this time to tram the total planned tonnage by means of one train only (one locomotive plus a spare unit and hoppers of a capacity to be determined). At the tip a deceleration zone would be established to reduce brake wear and increase safety. The track layout would provide for a loop (balloon) both at the tip and at the inbye loading point. Labour would be minimal: one man operating the security tip, one train driver and an orepass boxfront attendant (continuous loading system).

5.2.4 ***Additional Reef Hoisting***

The amount of waste to be broken in the narrow reef stopes and its handling and packing in the worked-out areas needed to be assessed and also the vamping of these areas prior to the waste packing; this estimate would define the potential for increased reef hoisting.

5.2.5 ***Stope Face Work***

The mechanisation of the face drilling operations needed follow-up and trials with new electro-hydraulic face rigs; this would come in the near future. Cleaning of the face by winch was the accepted method at that time but consideration had to be given to the use of Nonel Unidets to improve throw and lessen the need for a face winch. Refer to the paper on the use of Nonel by K.A.Rhodes in Annexure 3.2 in Volume 2.

5.2.6 ***Shaft Sinking***

Another important aspect would be a revision of the shaft sinking and station development schedule in terms of the reduction in levels. Certain levels and stations could be eliminated and this would save both money and time.

All these factors (and many others) had to be assessed before completing even a first motivational report.

5.3 First Motivational Report: September 1985

The initial motivational report prepared early in September 1985 also took cognizance of the original Feasibility Study. Based on the geology of the orebody and the siting of the first two shafts to be sunk (No 3 and No 4 Shafts) it was to be seen that there would be certain major advantages for a mechanised operation against the conventional scattered mining operations envisaged in the Feasibility Study. There would be a major reduction in capital expenditure. Although for this motivation the capital expenditure would not be estimated, the main savings would fall under sinking and lining; station development; shaft system development; ore reserve development; ore/waste pass systems; hostel accommodation; housing and other surface infrastructure.

Working costs would be reduced compared to that in the Feasibility Study by at least R10/ton.

The commissioning of the shafts would be accelerated by approximately four months.

There would be a potential for an increase in reef hoisting.

Finally, there would be an expected improvement in safety performance.

An examination and some analysis of this technical report now follows.

5.3.1 *Geology*

The Feasibility Report showed that the reef was relatively undisturbed by minor faulting and the dip of the reef varied gradually between 0° and 12°. These conditions were considered favourable for the introduction of trackless equipment.

The immediate hanging wall consisted of siliceous quartzite and was considered to be competent.

However it was to be expected that water bearing fissures and dykes would be intersected and cognizance of this would be of extreme importance in the design of the mine.

5.3.2 ***Method of Mining***

It was shown in this motivational report that the waste development for a conventional scattered mining layout would be excessive. The total metres of development (for a 20 month ore reserve) were calculated at 27167 metres or 91 tons/metre of footwall development. Further, in order to exploit the reef in the area of influence of the No 3/4 Shaft systems a further 56758 metres of footwall waste development would be required. Therefore the total estimated footwall waste development to exploit the estimated ore reserve of 7,65 million tons was 83925 metres in total.

It was axiomatic therefore, at that early stage of the motivation, that a trackless operation on the reef horizon which would obviate the necessity for a development programme on seven levels but would only require a gathering haulage on one level, must be a viable alternative and such an option would be a mechanised trackless operation.

5.3.3 ***Trackless Alternative***

Four blocks of reef, defined by major faults, had been identified to be exploited from the No 3/4 Shaft system. On the first level (60 Level) a single drive would intersect the reef horizon and on 70 Level twin development ends would establish the gathering haulage to serve all four blocks. Below the main reef decline in each block a footwall service decline would be developed in waste, approximately 8 metres below the reef decline. The total footwall waste development would be 17910 metres. However, all the waste development on 60 Level would be completed as capital development during the period of mid-shaft loading (MSL) and therefore in terms of post MSL, taking cognizance of both the waste and reef mined, the estimated ore replacement factor for the trackless option would be 647 tons/metre of footwall development, a significant improvement on the corresponding conventional factor of 91 tons/metre.

5.3.4 ***General Mining Layout***

Once trackless development from 60 Level had intersected the reef horizon at the various sub-outcrops, a main reef decline and access roadways would be developed for each of the four blocks. From these roadways access stope drives would be broken away to establish the stope panels. Behind the on reef development a service decline would be developed lagging the reef development to take into account any changes in the reef such as minor faulting. The main reasons for a footwall service decline were primarily to ensure reliable intake airways off the reef horizon and for an alternative tramming roadway.

Access roadway dimensions would be 4,5 metres wide and 3,0 metres high to provide for truck loading by LHD and truck tramming. Access stope drives would be 3,5 metres wide and 3,0 metres high to allow movement of LHD's; waste generated in these drives was planned to be packed in worked-out areas in a similar manner as planned at 95L UEIA Project at Cooke 2 Shaft, REGM.

A gathering haulage on 70 Level would serve all blocks and reef hauled by truck to a single tip (orepass down to 70 Level) constructed for each block. A trolley line locomotive, only one 200 ton capacity train, would transfer all the reef to the shaft system. This gathering haulage would have a single track with balloon layouts at both the shaft and the loading points for continuous loading and tipping. Before establishment of the rail haulage, reef would be trammed back to the shaft on 60 Level by truck; refer to **Figure 5.1**.

5.3.5 ***Production Parameters***

The production from the Phase 1 of the project had been planned in the Feasibility Study at 80 000 tons reef/month. Original planning had envisaged 30 000 tons/month of waste to be generated but with the trackless option this would now be significantly reduced. It was now estimated that for the reserve down to 90 Level that at 80 000 tons reef/month footwall waste would be 5200 tons/month.

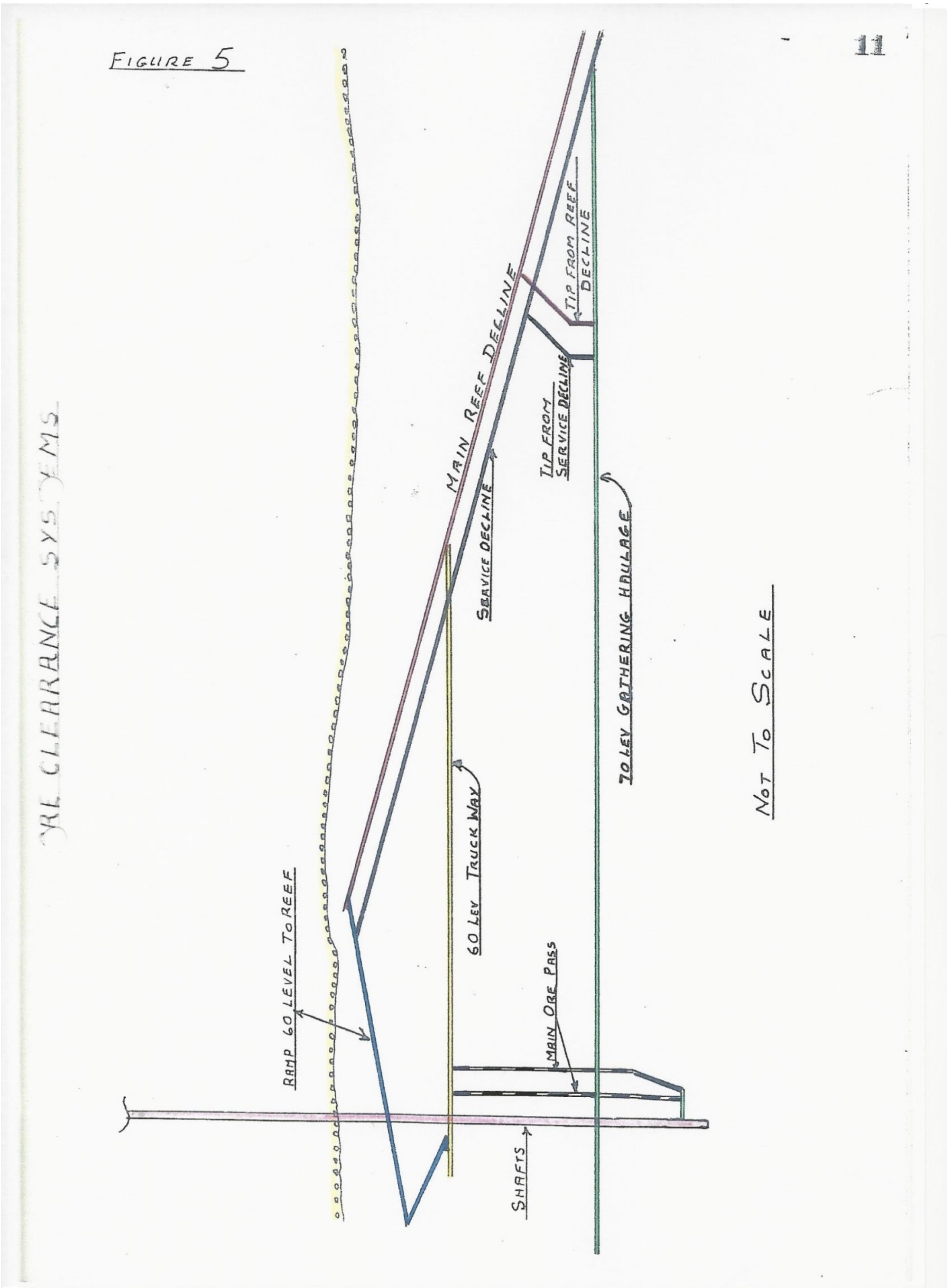

FIGURE 5.1

Ore Clearance System: refer to Annexure 5.1 Volume 3

5.3.6 ***Rock Mechanics Considerations***

In discussions with the Group Rock Mechanics Engineer there were no issues with the proposed support systems as they were very similar to the 95 Level narrow reef project at Cooke 2 Shaft, REGM. However, with regard to the planned footwall service decline it would have to be at least 15 metres below the reef horizon in order to avert the stresses induced by the reef pillars. If backfill were to be used instead of crush pillars and sticks this distance could be reduced to 5 metres.

5.3.7 ***Cycle of Operatons***

Main Reef Development

All roadways and access stope drives were to be developed using electro-hydraulic drill rigs and cleaned by 3,8m^3 LHD's into 32 ton dump trucks.

In the same manner as the 95L UEIA Project at Cooke 2 Shaft, REGM, the bottom section of the face of the access drive would be blasted first in a double cut operation. The waste would be cleaned out and trammed as waste by LHD to worked out areas when they came available. After cleaning the waste, the top (reef) cut of the drive would be blasted down. When writing this motivation it was assumed that 80% of the waste would be removed; however this was scaled back, for practical reasons, to 60% in later motivations.

In this report the first reference was made to what would become a key issue: cover drilling in main development.

Stope Drilling and Blasting

Conventional face drilling was still accepted for the motivation but the objective was always to be the development of a mechanised face rig. Also, the use of Nonel was to be further considered.

Stope Face Cleaning

The face was planned to be cleaned by face winch. However, the successful use of the (Nonel) Unidet could change this in terms of improved throw of the blast.

Stope Support
Crush pillar and stick was the method of stope support.

5.3.8 ***Ventilation***

For this first motivation a ventilation report had been compiled by the Environmental Superintendent for Cooke 2 REGM. This analysis confirmed that a single main intake on 70 Level would prove effective for the ventilation of operations at the No 3/4 Shaft System (Phase 1).

5.3.9 ***Equipment***

A full preliminary inventory of equipment was listed for Phase 1 (80 000 tons/month).

5.3.10 ***Workshops***

A major workshop would be developed close to the 60 Level station and adjacent to the main tramming haulage. This workshop would provide for all the required services and repairs to all equipment.

All equipment, stripped on surface prior to transport through the restricted dimensions of the shaft (it was too late to effect any changes to the size of the shafts), would be re-assembled in the proposed assembly bay on the 60 Level Shaft Station.

Diesel fuel would be pumped underground direct to an established fuel bay.

5.3.11 ***Training***

Training had always been identified as a critical issue for the trackless mechanised mining and, before any equipment arrived at the mine, an early appointment would be the Mechanical Equipment Supervisor (MES). He would be solely responsible for the selection and training of operators and enforcing driver discipline.

5.3.12 ***Labour***

At this point in time the estimated labour complement for the mechanised operation was 1175 non-common wage scale or unskilled (NCWS) and 157 common wage scale or skilled (CWS). A comparison with the Feasibility Study showed a difference of 1975NCWS and 34 CWS in favour of the trackless option.

5.3.13 ***Safety***

Once again the safety in a mechanised operation over conventional mining was stressed. Locomotive tramming in conventional scattered mining was a major source of serious accidents and therefore, to operate a single train on a streamlined gathering haulage would greatly reduce this risk. Further, the markedly lower total labour complement of 1300 against 3300 (planned for in the Feasibility Study) would mean 2000 less people would be exposed to danger, representing a significant reduction in risk.

5.3.14 ***Justification***

It was argued that there were major cost advantages for the change to the trackless access gathering haulage concept at the new H.J.Joel Mine.

Shaft Commissioning

It was envisaged that the shaft sinking commissioning date would be brought forward by approximately four months. This would be because shaft sinking would stop at 80 Level (and not at 100 Level) thereby saving 400 metres of shaft sinking and, in addition, two of the proposed stations would not be required. Capital expenditure would thereby be markedly reduced.

Footwall Waste Development

For the trackless option total footwall waste development had been calculated at 18 000 metres as opposed to 85 000 metres for a conventional scattered layout. A difference of 67 000 metres must be considered significant and would reflect substantial cost savings for both capital and working costs.

Working Costs

Working costs would be considerably reduced due to lower stoping costs, very much lower development costs and costs of ancillary operations on footwall haulages would be virtually eliminated. It was estimated in this motivation, that a cost saving of R10/ton could be realised (equivalent in today's money terms of R120/ton).

Additional Reef Hoisting
There would be potential for additional reef hoisting because waste generated would be minimal.

Surface Infrastructure
Meaningful savings could be effected in the capital expenditure of the mine hostel at the mine and also housing accommodation in Virginia, the nearby town. In addition, there were many other aspects of surface infrastructure which could mean less capital expenditure; reduced use of compressed air, reduction in waste hoisting, use of less manpower to name just a few.

This Motivational Report entitled ***Proposed Trackless Access Gathering Haulage Mining Operation at the H.J.Joel Project*** by K.A.Rhodes was submitted to the Project Manager for the Consulting Engineer on 19 September 1985. It is now attached as **Annexure 5.1** in **Volume 3**.

The response to this report by the Consulting Engineer, dated 23 September 1985, can be seen in his memorandum to the Technical Director and is seen overleaf.

To sum up then, in a period of less than one month from the start of planning, the first motivation for a trackless mine at the H.J.Joel Project had been received favourably by the Consulting Engineer and had been forwarded to the Technical Director.

5.4 Follow-up and Second Motivational Report: October 1985

After the submission of the initial motivational report (Report No 1 dated 19 September 1985) a meeting was held on 20 September with the Consulting Engineer. At that meeting approval was given for KAR to work on a follow-up report (Report No 2) which would detail development and stoping schedules, shaft sinking programmes, life of mine schedules, working costs and capex estimates. It was requested that this report should be made available for discussion on 28 October 1985; the report was submitted on that date by KAR. This second report provided for significantly more detailed information and the major additions were as follows.

Johannesburg Consolidated Investment Company, Limited

GOLD AND URANIUM DIVISION

GHSB/rfm

23rd September, 1985.

MEMORANDUM TO : MR. H. SCOTT-RUSSELL.

A report by Mr. K.A. Rhodes dated September 19th, 1985 on a proposed trackless mining operation vs a conventional gold mining layout is attached.

Whilst a number of details need to be hammered out and notwithstanding the limited experience of this method in South African gold mines the proposal has a great deal of merit.

I would recommend that as soon as Mr. Rhodes returns from a 10-day leave spell he devotes his full attention to detailing the mine layout using a T.A.G.H. system. The exercise would include the scheduling of shaft sinking, development, stoping, working costs and capex etc. Supporting documentation from the various Head Office consultants would accompany the final recommendation.

Arrangements are currently being made to construct a model of 60 and 70 levels layout in order to more fully comprehend the ramifications of the proposed system.

Report submitted
28/10/85

G.H.S. BAMFORD.
Consulting Engineer.

cc Messrs. G.Y. Nisbet
G.P. Wanblad
R.C. Bertram
Coetsee
G.H.S. Bamford

5.4.1 ***Life of Mine Schedules***

A life of mine schedule (known as Option 9) was completed in the same format as for the Feasibility Study (Option 3). Both these schedules were over a span of 31 years.

5.4.2 ***Capex Estimates***

A comparison of factorised capex estimates had been made for conventional mining and trackless mining by the Capital Projects Control (CPC) Department of JCI and these were in favour of the mechanised operation by a difference of more than R45 million for Phase 1 of the mine, in today's terms (2014) this would be over R500 million.

5.4.3 ***Shaft Sinking Programme***

A revised bar-chart for the commissioning of the No 3 Shaft and No 4 Shaft systems indicated that the completed programme could be brought forward by four months. Although it was not necessary in terms of the mechanised option to sink No 3 Shaft below 80 Level it was still recommended, and provided for in the revised programme, that sinking and equipping should continue to 100 Level in order to provide for a fall-back position if company policy dictated later that mining should revert to conventional methods.

5.4.4 ***Shaft Equipping***

At No 3 Shaft it was envisaged that the shaft would be equipped as initially approved. However, at No 4 Shaft it was recommended that a large single cage be installed in order to provide for the movement of large equipment through the shaft, thereby obviating major stripping and slinging in the shaft as had previously been the case at the trackless operations at REGM's Cooke 2 Shaft.

5.4.5 ***Station Layouts***

Detailed station layouts had been set out for both 60 Level and 70 Level. On 60 Level provision was made for a workshop with total facilities for the maintenance, breakdowns and overhaul of trackless equipment; a separate re-assembly bay in close proximity to the No 3 Shaft which would be re-positioned in terms of the new proposal for a large cage at No 4 Shaft; main tipping arrangements for trucks; material storage and handling arrangements; a bus terminal from where the transport of people direct to the face would take place. On 70 Level there would be main tips for reef and waste and a balloon rail system for continuous tramming; workshop facilities for electric trolley line locomotives; a 25 ton hopper repair bay; Plasserail workshop and a store.

5.4.6 ***Dilution***

In estimating the dilution, it had now been assumed that 60% of the bottom waste cut, in the double cut method in both access reef declines and access stope drives, would be trammed as waste and 40% of the waste cut would be trammed as reef.

Therefore:

Dilution in Access Reef Declines (ARD's) $= \frac{40 \times 22{,}28}{19800} \times 40\%$

= **1,80**%

In terms of the above

Channel width	= 1,20 metres
Panel length	= 40 metres
Waste portion of ARD per metre advance	= 22,28 tons
Dilution	= 40%

Ore reserve tons generated for 40 metres of ARD development

= 19800 tons

Dilution in Access Stope Drives (ASD's) $= \frac{17{,}32}{132} \times 40\%$

= **5,25**%

In terms of the above

Channel width	= 1,20 metres
Waste tons in ASD per metre advance	= 17,32 tons
Dilution	= 40%
Face tons blasted in ASD per metre advance	= 132 tons

Total dilution is therefore = 1,8% + 5,25%

= 7.05%

These calculations were virtually identical as for the dilution calculations set out by KAR for the 95L UE1A Project at Cooke 2 Shaft, REGM; refer to chapter 4. In effect the calculations showed once again that the use of large size equipment working on a narrow reef horizon did not imply an acceptance of excessive dilution.

In a supporting contribution to the report from the JCI Group Surveyor there was general agreement for the dilution calculations given in the report for both trackless and conventional mining.

5.4.7 ***Capital Equipment***

The list of capital equipment had been revised with the main changes being an up-grade in sizes of the LHD’s and the introduction of roofbolters.

5.4.8 ***Labour***

There were only minor changes to the labour complements. New complements were as follows with previous figures in brackets, NCWS 1137 (1175) and CWS 159 (157).

5.4.9 ***Working Costs***

Working costs for the conventional option in the Feasibility Study had been R65/ton for the production rate of 80 000 tons/month; the mechanised option was estimated at R54/ton at the same rate of production.

5.4.10 ***Further Investigations***

The report set out certain aspects of the proposed option that required further investigation in the immediate future, but it was stressed that they were not areas of concern; these issues are briefly recorded below.

No 4 Shaft Cage

The finalisation of the cage and winding arrangements at the No 4 Shaft which would accommodate large components and sub-assemblies of trackless equipment and also the identification of the individual components for each machine which would be handled by the cage.

No 1/2 Shaft System

Consideration would be given to a single large diameter downcast shaft to handle large equipment in No 1 Shaft and the possible future conversion of No 3 Shaft to an upcast shaft which would obviate the necessity for No 2 Shaft.

Backfill
An evaluation of the stope support systems described in the report by the Group Rock Mechanics Engineer and a determination of the necessity for a backfill plant.

Material Handling
A study was to be carried out of the material handling arrangements from the stores to the underground workings through the shaft system

Explosives
There was a need for an evaluation of options for the delivery of explosives to the shaft, which would include the possibility of direct delivery by AECI from its future factory in Virginia.

Stoping Cycle
The preparation of stoping layouts and detailed cycles (blasting and cleaning) if a decision had to be made on the use of backfill; stope layouts without crush pillars would facilitate face cleaning and could reduce the number of electro-hydraulic rigs for ASD development.

Water Control
The preparation of detailed layouts for the pumping of water from the workings to the shaft station, taking cognizance of the use of trackless equipment on the reef horizon.

During October and November 1985 work continued on the motivation of the project which culminated in further draft reports being prepared (Reports No.3 and No.4) and a final motivation report of 23 January 1986 (Report No.5).

5.5 Final Motivation Report: January 1986

Following on from the meetings with the Consulting Engineer in September and October 1985 when the first motivational reports for a trackless operation at the H.J.Joel Project were discussed, a directive was approved that a report which would provide for a final recommendation should be made available before the end of January 1986; this report could then be submitted to the Board of JCI for their approval.

The first draft final report was submitted on 02 January and provided for life of mine schedules, capex estimates and a technical report.

5.5.1 ***Life of Mine Schedules***

In the report three life of mine schedules with different strategies were discussed; these options were known as 9A, 9B and 9C.

Option 9A

In this option the grade would be equalised as soon as possible .To meet this parameter it would be necessary to sink conventionally the No 1/2 Shaft system immediately in order to develop the north-eastern portion of the lease area where the lowest grades could be expected.

Option 9B

This option would delay the sinking of the No 1/2 Shaft system until first revenue in June 1988, this option had financial advantages.

Option 9C

This third option would delay the sinking of No 1/2 Shaft system until the last possible date to ensure continuous steady state input to the plant. This would be the worst case for equalisation of grade and would also be a risk to continuity of production.

In effect the recommendation in the final report would be Option 9B, soon to become known as Option 10 in the final plan for the project.

5.5.2 ***Capex Estimates***

The final capex estimates for the 120 000 tons/month mine were R738,8 million and R659,9 million for the conventional and trackless alternative respectively (in 2014 money terms R8,5 billion and R7,6 billion respectively). At that point in time these estimates were awarded an 80% confidence concept rating, indicating that at that early stage there would still be aspects which could cause scope changes to the project. Refer to Capital Expenditure Project reports on the following pages extracted from Annexure 5.2 in Volume 3**.**

5.5.3 ***Technical Report***

Some important technical aspects of the final report which represented changes from the previous motivation reports should be examined.

Geology

There was only one economic reef horizon, the VS5/Beatrix Composite Reef situated at the base of the Eldorado Series. The reef was displaced by a number of North-South trending faults with maximum throws of the order of 70 metres. Relatively little minor faulting was expected and the dip of the reef was generally flat. Such conditions favoured the use of mechanised equipment operating on the reef horizon. Water bearing fissures and dykes would be encountered based on the experience of the neighbouring Beatrix Mine, but no detailed information was yet available at the time of this report.

Reserves

The accepted life of mine reserve was 34,8 million tons at a gold grade of 6,8g/ton.

Shaft System

Sinking of No 3/4 Shaft system would be carried out as planned. Access to the reef would be on 60 Level with gathering haulages on 70 Level and 90 Level. At No 3 Shaft, equipping would remain unchanged but at No 4 Shaft provision would be made for a large cage to allow for the movement of equipment through the shaft and detailed planning had commenced for this change.

The site of the No 1/2 Shaft system remained unchanged. Access to the reef horizon from these shafts would be on 110 Level with gathering haulages on 130 Level and 150 Level. A large cage would be installed at No 1 Shaft.

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JOHANNESBURG CONSOLIDATED INVESTMENT COMPANY LIMITED
TECHNICAL SERVICES DIVISION
CAPITAL PROJECTS CONTROL DEPARTMENT

CAPITAL EXPENDITURE
PROJECT SUMMARY REPORT

COMPANY : H.J. JOEL PROJECT - 120 000 t/m MINE
SECTION : PHASE 1 - 80 000 T.P.M.
PHASE 2 - 120 000 T.P.M.
OPTION : TRACKLESS MINING vs CONVENTIONAL MINING
OPTION 10

BASE DATE 85.01.01
UNESCALATED
R 1000's

COMPARISON ESTIMATE SUMMARY

120 000 TPM MINE

	CONVENTIONAL (Updated 3B)	TRACKLESS (Option 10)
Phase 1		
No.3 Shaft	144 317	132 874
No.4 Shaft	28 279	30 883
Infrastructure	161 940	131 860
Plant (80 000)	85 109	85 109
Sub Total	419 645	380 726
Phase 2		
No.1 Shaft	191 284 *	161 941 *
No.2 Shaft	14 881	14 881
Infrastructure	73 358	67 197
Plant (+ 40 000)	35 220	35 220
2nd outlet	4 431	-
Sub Total	319 174	279 239
TOTALS	738 819	659 965

PRELIMINARY COST ESTIMATE (P.C.E)

OVERALL CONTINGENCY 12%

CONCEPT CONFIDENCE 80%

Difference in capital requirement is R78 854 000 in favour of trackless mining.

Both Options - Start sinking Nos. 1 and 2 Shafts 6/'88

* = Refrigeration Plant included

8

28

JOHANNESBURG CONSOLIDATED INVESTMENT COMPANY LIMITED

TECHNICAL SERVICES DIVISION

CAPITAL PROJECTS CONTROL DEPARTMENT

CAPITAL EXPENDITURE
PROJECT SUMMARY REPORT

COMPANY : H.J. JOEL PROJECT - 120 000 t/m MINE
SECTION : PHASE 1 - 80 000 T.P.M.
PHASE 2 - 120 000 T.P.M.
OPTION : TRACKLESS MINING vs CONVENTIONAL MINING
OPTION 10

BASE DATE 85.01.01
UNESCALATED
R 1000's

C O N C E P T   C O N F I D E N C E

To : Project Manager H.J. Joel Project - J. Coetsee
for the Consulting Engineer Engineering Services - G.P. Wanblad

From : Production Manager H.J. Joel Project - K.A. Rhodes

Date : 30th December 1985

Subject : CONCEPT CONFIDENCE - H.J. JOEL PROJECT - 120 000 TONS PER MONTH GOLD MINE, TRACKLESS AND CONVENTIONAL MINING

The individual concept confidence for the following disciplines for both a conventional operation and a mechanised operation are as follows:

Mining	80%
Underground Engineering	80%
General Mine Infrastructure	80%

Therefore the overall concept confidence for both conventional and trackless mining is 80%,which by definition, indicates a warning that the scope and process are not yet firm and the possibility of change occurring is still strong.

The major uncertainties in the scope definitions are :

a) The scheduling of the sinking programme for the No.1/2 Shaft system which affects the method of sinking.
b) Detailed engineering design of the mid shaft loading arrangements for 60 Level and 70 Level have not yet been finalised.
c) The evaluation of the stope support systems (detailed in the report from the Group Rock Mechanics Engineer) and the determination of the necessity for a backfill plant must still be carried out.
d) Detailed layouts of the pumping arrangements from the workings to the main shaft pump stations have not been finalised.
e) The final requirements for the Training Centre are still to be determined.

K.A. RHODES
Production Manager, H.J. Joel Project

AGREED AT 80%

Date 7/1/86

J. COETSEE
PROJECT MANAGER
H.J. JOEL PROJECT

AGREED AT % 80%

Date 07/01/86.

G.P. WANBLAD
CONSULTING ENGINEER
ENGINEERING SERVICES

Station Layouts

Detailed station layouts had been completed for 60 Level, 70 Level and 90 Level. Main tipping arrangements had been set out on both 60 and 70 Levels for dump trucks, including the temporary requirements for mid-shaft loading (MSL) with the necessary rock passes.

The main workshop facilities were to be situated on 60 Level with workbays on 70 Level for the MSL development phase.

All arrangements, for material handling and for the transport of personnel onto the reef horizon, would be concentrated on 60 Level.

On 90 Level arrangements for the streamlined continuous rail haulage would be established, also the necessary workshops for trolley line locomotives, 25 ton hopper repair bay and a Plasserrail workshop. Similar arrangements would be duplicated at the respective stations at the No 1/2 Shaft system.

Main Development

All development to the various sub-outcrop portions of the reef in the four target blocks (A, B, C and D) would take place from 60 Level. The development layout remained unchanged from that set out in the first motivational report. The total footwall waste development for the reserve in the four blocks had now been determined to be 22000 metres for a reserve of 12,65 million tons, more than half of which would take place in the footwall service declines; details of this development are detailed below.

	Metres
Access ramps	1300
60 Level Access Roadways	1700
70 Level Gathering Haulage	3100
Orepasses ex 70 Level	500
90 Level Gathering Haulage	3200
Orepasses ex 90 Level	500
Footwall Service Declines	11700
Total	22000

This total of 22000 metres for the trackless mining proposal would now compare with 110000 metres of footwall development required for the development of the same ore reserve if conventional mining methods were to have taken place.

In terms of the overall development planning the two shaft systems would be linked and development of the No 1/2 Shaft system would be based on the trackless access/gathering haulage concept set out at the No 3/4 Shaft system.

Development scheduling would have to take into account cover drilling constraints in terms of which the maximum advance in any development end would not exceed 100 metres per month; this rate of 4 metres per day with drilling bays at 30 metre intervals would allow for nine days to drill the cover holes before the end would be out of cover.

Rock Mechanics Considerations

Following discussions with the Group Rock Mechanics Engineer it had been decided to support the stopes with grout packs; at that time such a system was being used successfully at the Rustenburg Section of Rustenburg Platinum Mines. The immediate face area would be supported by 40 ton hydraulic props.

Ventilation

The development schedule and production build-up and the general mine layout, including detailed stope layouts, had been fully discussed with the Group Ventilation Engineer and detailed supportive documentation had been provided by him.

Main Development on the Reef Horizon

Access reef declines would be spaced at 150 metre intervals (limiting the LHD tramming distance to 150 metres) with stope faces 40 metres between the centres of the access stope drives. Notwithstanding the cover drilling programmes taking place in advance of the development, 6 metre pilot holes would be drilled with every round by the electro-hydraulic drill rig. The development of all main access roadways would be carried out by the double cut method with the waste being blasted separately from the reef.

With respect to the access stope drives, such drives would not be developed as a development end; all reef would be blasted on the face and footwall lifting would be practised. The waste would be loaded out and packed in worked-out areas as previously planned.

It was assumed initially that stope drilling would be conventional but it was expected that a hydraulic face rig would be considered at the time when stoping commenced.

Dilution

In estimating the dilution from access reef decline development it was confirmed that 60% of the waste blasted in the double cut operation would be trammed as waste and the remaining 40% trammed as reef; this being an acceptable practical assumption. Refer to **Figures 5.2** and **5.2A** showing photographs of double cut mining.

In the access stope drives in the proposed footwall lifting or benching operation it was also assumed that only 60% of the blast would be taken out as waste and the remainder trammed as reef. In order to avoid loss of reef (reef trammed as waste) it would be necessary to carry out this benching operation in advance of the stope face and therefore between the two immediately adjacent faces; in this respect refer to **Figure 5.3**. The total dilution had been previously calculated at 7,05% for all development. However, in addition to ongoing development it would be necessary to establish turning and passing points for dump trucks and also tipping points (LHD into truck) at the intersections of access reef declines and access stope drives. The total waste generated from these sources was shown to be less than 1% (theoretically calculated in **Figure 5.4** and **5.4A** to be 0,69%). However, this would only be dilution if this waste was allowed to be trammed as reef.

The waste dilution calculations in this final motivational report were approved by the Group Surveyor who also provided a supporting document to the report.

FIGURE 5.2

Double Cut Mining: Waste Cut Blasted with Reef in Hangingwall

FIGURE 5.2A

Double Cut Mining: LHD Below Reef in Hangingwall

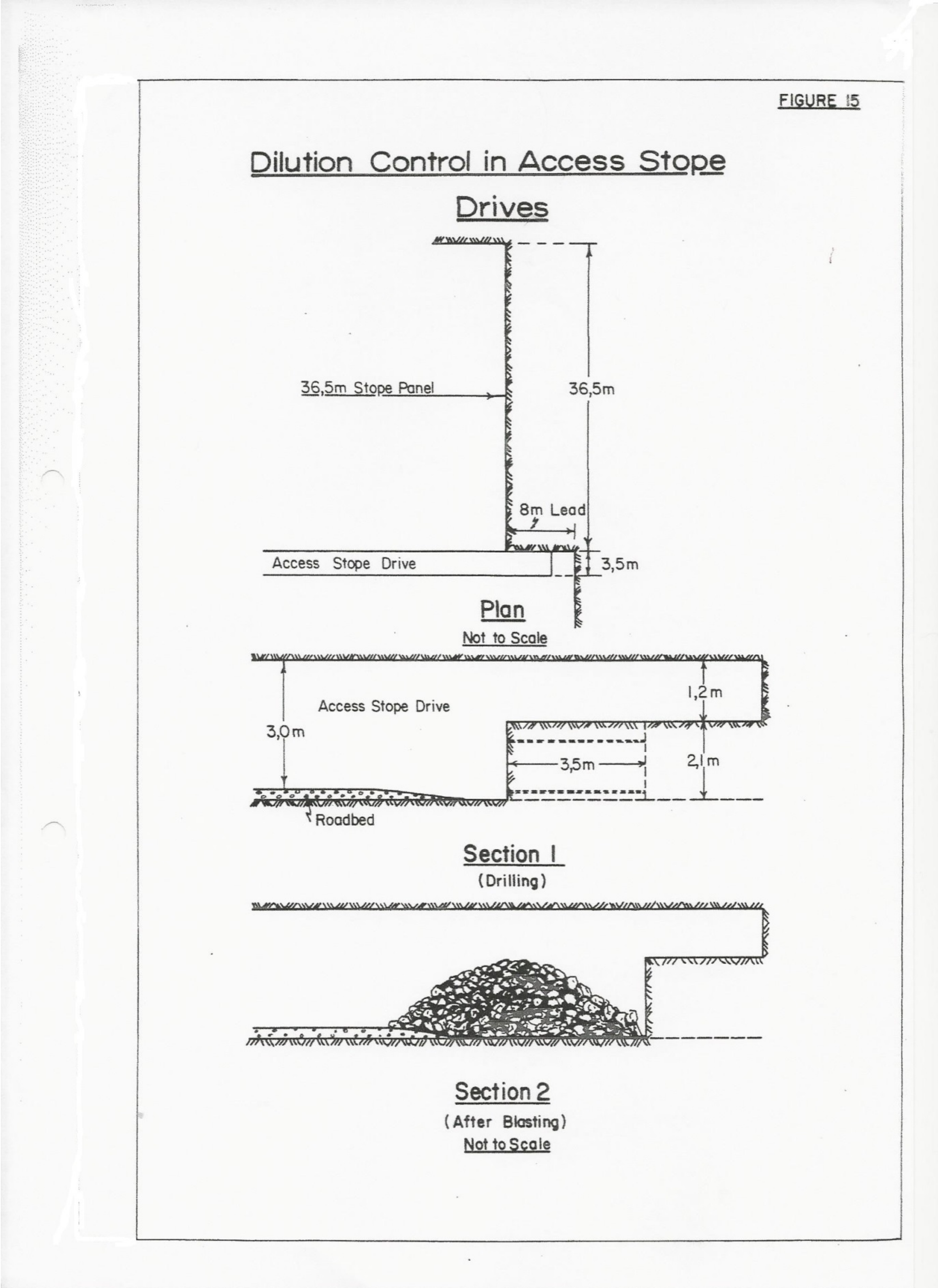

FIGURE 5.3

Dilution Control in Access Stope Drives: refer to Annexure 5.2 Volume 3

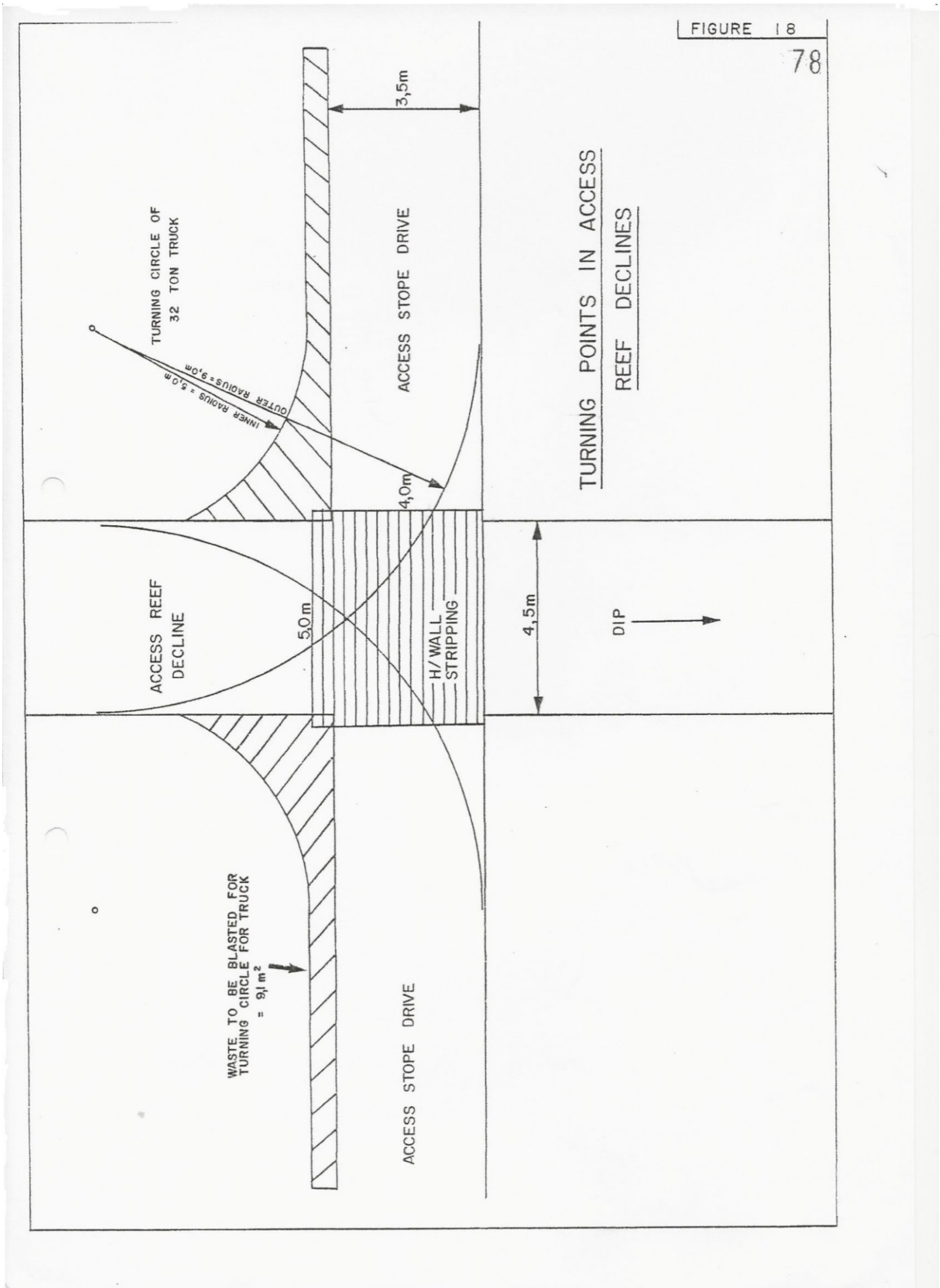

FIGURE 5.4

Dilution at Tipping Points in Access Reef Decline: refer to Annexure 5.2 Volume 3

ACCOMPANYING NOTES TO FIGURE 18

CALCULATION OF DILUTION AT TURNING/TIPPING POINTS IN ACCESS REEF DECLINES

A) Sidewall Blasting

Area of additional waste blasted	= 2 x 9,1	= 18,2m^2
Tons of waste blasted	= 18,2 x 1,60 x 2,75	= 80 tons
Where : Average waste height	= 160 cms	

B) Hangingwall Stripping

Area of hangingwall to be blasted (see diagram)		= 20m^2
Tons of waste blasted	= 20 x 1,5 x 2,75	= 83 tons
Where : thickness of h/wall stripped	= 1,5m	

Total waste tons blasted	= 80 + 83	= 163 tons
Total reef produced in a panel	= 40 x 150 x 1,43 x 2,75	= 23600 tons
Where : Face length	= 40m	
Advance	= 150m	
Av. stope width	= 143cm	

As 163 tons of waste is produced at turning points for every 23 600 reef tons

Dilution $= \frac{163}{23600} \times 100$ $= \underline{0{,}69\%}$

N B : This dilution will only occur if the waste blasted is trammed as reef.

FIGURE 5.4A

Dilution Calculations Related to Figure 5.4: refer to Annexure 5.2 Volume 3

Equipment

The detailed inventory for the equipment required for 80000 tons/month (Phase 1) was set out in the report.

Units	No of Units Required
LHD 4,3m^3	8
Electro-hydraulic Drill Rig	8
Roofbolter	2
32 ton Dump Truck	10
Utility Vehicle	5
Land Cruiser	14
Impact Breaker	4
Grader	1
Bulldozer/Grader	1
Winches 37kW	50
Personnel Transporter (busses)	3
Explosive Vehicle (underground)	1

The numbers of the primary units were calculated in terms of the following analysis.

LHD's

LHD's were required for development, stoping and waste packing operations. The production capacity of an LHD (and a truck) was calculated from a basic formulae which evolved from the following parameters.

Production capacity in tons/minute is tons trammed by LHD ÷ total cycle time of the unit in minutes.

In terms of the above:

Tons trammed/minute = 0,85 x L

Where 0,85 is the utilisation of the machine and L is the carrying capacity of the LHD or 7 tons in the bucket of a 4,3m^3 LHD.

Total cycle time of the unit can be split into two elements where T is that part of the cycle to load, manoeuvre and tip and the tramming portion of the cycle is $(2 \times D) \div S \times \left[\frac{1000}{60}\right]$

Where D is the one way tramming distance in metres, S is the average return trip speed of the LHD in kph, 1000 is the conversion factor from kilometres to metres and 60 is the conversion factor from hour to minutes.

Therefore, the production capacity of the unit (P) is:

$$P = 0{,}85 \times L \div T + \left[\frac{2 \times D}{S \times \frac{1000}{60}} \right]$$

$$\text{or } P = 0{,}85 \times L \div T + \left[\frac{2 \times D}{S \times 16{,}67} \right] \text{ tons/minute}$$

$$\text{or } P = 60 \times 0{,}85 \times L \div T + \left[\frac{2 \times D}{S \times 16{,}67} \right] \text{ tons/hour}$$

$$\text{or } P = 51 \times L \div T + \left[\frac{2 \times D}{S \times 16{,}67} \right] \text{ tons/hour}$$

The monthly production of the unit can be estimated from the available hours in the month using the definitions stated below.

$$\text{Availability} = \frac{\text{Total hours} - \text{Engineering downtime}}{\text{Available hours}} \times 100\%$$

$$\text{Utilisation} = \frac{\text{Hours worked (metre readings)}}{\text{Total hours} - \text{Engineering downtime}} \times 100\%$$

The assumption for these calculations for both availability and utilisation was in general 85%.

The relevant calculations for the LHD of 7 tons capacity were therefore as follows; utilisation is built into the above formulae.

For Development:

Maximum one way tram (D)	= 300 metres
Dump, manoeuvre and tip (T)	= 5 minutes
Average return trip speed of LHD (S)	= 5kph

Therefore: $P = 51 \times 7 \div 5 + \left[\frac{2 \times 300}{5 \times 16{,}67} \right]$ = 29 tons/hour

Therefore production rate/day is calculated at
29 x 15 hours x 0.85 (availability) = 370 tons/day

The LHD was assumed to clean 2 rounds (130 tons each)/shift
Total duty /day is (2 x 2 x 130) = 520 tons/day

Therefore number of LHD's required for development = 1.4 (say 2)

For Stoping:

Average tramming distance one way (D) = 75 metres
Dump, manoeuvre and tip (T) = 3 minutes
Average speed for return trip (S) = 5kph

Therefore $P = 51 \times 7 \div 3 + \left[\frac{2 \times 75}{5 \times 16{,}67} \right]$ = 74 tons/hour

Total production rate/day is calculated at
74 x 15 hours x 0.85 (availability) = 943 tons/day

Daily production from stoping is
80000 ÷ 23,5 = 3400 tons/day

Therefore number of LHD's required for stoping = 3.6 (say 4)

For Waste Packing:

Footwall lifting waste tons to be packed per month was based on twenty five panels being worked at any one time, each advancing 20 metres per month. At a stoping width of 120 cms the total waste tons to be packed was calculated to be of the order of 6400 tons/month assuming that only 60% of the total tons blasted would be packed.

One way tram (D) = 500 metres
Load, manoeuvre and dump (T) = 6 minutes
Average speed for return trip (S) = 5 kph

Therefore $P = 51 \times 7 \div 6 + \left[\frac{2 \times 500}{5 \times 16{,}67}\right]$ = 20 tons/hour

Production capacity/day is calculated at
20 x 15 x 0.85 (availability) = 255 tons/day
Waste tons to be packed/day is
6400 ÷ 23,5 = 272 tons/day

Therefore number of LHD's required for waste packing = 1.1 (say 2)

The total number of LHD's (4.3m³ capacity) for the project was therefore estimated to be 8.

Trucks

Using the same formulae as for LHD's and the following assumptions:
Tramming distance one way (D) = 1000 metres
(grade 8° with passing points every 200m)
Average speed (S) = 5kph
(estimated at 5kph to allow for empty trucks stopping at passing points)
Load, manoeuvre and tip (T) = 21 minutes
(LHD loading 4/5 passes)

Truck capacity (L) is assumed to be 29 tons (32 ton rated truck)
$P = 51 \times 29 \div 21 + \left[\frac{2 \times 1000}{5 \times 16{,}67}\right]$ = 33 tons/hour

Production capacity/day is calculated at
33 x 15 x 0.85 (availability) = 420 tons/day

Total production per month is 80000 tons reef and 12000 tons waste = 92000 tons/month
or = 3915 tons/day

Total number of trucks therefore = 9.3 (say 10)

Drill Rigs

In determining the number of drill rigs, cognizance had been taken of the following parameters.

At steady state production, 500 metres of access stope drives advance would be necessary; 25 panels advancing 20 metres/day. Assuming that one rig would drill three rounds of footwall lifting per day of 3 metres advance, this would require say three rigs.

It was also estimated for this report that 300 metres of development in reef declines and footwall service declines would be necessary in a month. It was assumed that two rounds per day would be drilled by a rig; this would necessitate three rigs for increased tramming distance between development ends.

In addition to the above, two additional drill rigs would be required for the development of the 70 Level gathering haulage.

In total it was decided to plan for a total of 8 electro-hydraulic twin boom drill rigs.

Roofbolters

At that time it was estimated that two roofbolters would be necessary for development operations.

Utility Vehicles

The five UV's planned for were predominantly flat-bed vehicles with a crane for material handling. Specialist vehicles such as a dedicated explosives vehicle, a grader and a bulldozer would be additional and were also provided for.

Personnel Vehicles (Land Cruisers)

At the time, the Land Cruiser was the favoured vehicle for general work and in this inventory it was planned for fourteen such units. The necessity for this type of vehicle in a mechanised operation could never be over emphasised. These vehicles would be constantly used for transporting small mining crews, engineering artisans, explosives, spare parts, some mining stores and for overall supervision. The breakdown of the units planned for were as follows.

		No of Units
Mining work		4
Engineering – two per level		4
Survey/sampling		2
Senior management		1
Underground Manager		1
Mine Overseers		2
	Total	14

Busses

On any one shift it was expected that of the order of three hundred persons would need to be transported from the terminal (at the shaft station) to the inbye workings and this would require four cages (75 persons per cage). Therefore it was planned for three busses, of 75 person capacity, to be used for this work. It was realised that three busses would be adequate as the first bus would have returned to the station before the arrival of the fourth cage.

Engineering Considerations

The workshop which was to be constructed on 60 Level would provide for the total maintenance and overhauls for the complete fleet of equipment and would be fully equipped and operational before stoping operations commenced. The assembly bay to be established would be equipped before any trackless mining equipment went underground.

All mining managers and supervisory mining personnel would be fully committed to the support of the engineering discipline for the maintenance of equipment, and would demand driver discipline to ensure that the maintenance of the equipment could be carried out strictly in accordance with the relevant planned schedules.

Initially all trackless equipment necessary for the development of the mine would have to be stripped on surface prior to going underground. The proposed large cage in No 4 Shaft would later obviate the necessity for this major stripping operation; however, the large cage would only be installed after the commissioning of the No 3/4 Shaft system.

Labour
The estimated underground labour complements for the Phase 1 of the project (80000 tons reef/month) were relatively unchanged from the earlier reports: 159 CWS and 1132 NCWS. The surface complement had been assumed by factorisation of the Feasibility Study complement to be 132 CWS and 650 NCWS. The total complement was therefore 291 CWS and 1782 NCWS. In terms of Phase 2 (120000 tons reef/month) the total complement for CWS and NCWS would increase to 386 and 2459 respectively.

Working Costs
The cost difference in favour of the trackless option was confirmed at R11/ton (again in 2014 money terms this would be more than R120/ton); of the R11/ton, R8/ton would be related to development. The total mine costs were fixed at R54/ton and R65/ton for the trackless and conventional options respectively.

The final motivational report ***Proposed Trackless Access Gathering Haulage Mining Operations at the H.J.Joel Project*** by K.A.Rhodes, dated 23 January 1986, can be seen in **Annexure 5.2** in **Volume 3**.

5.6 **Formal Approval of Trackless Mechanised Mining at the H.J.Joel Project**
Following submission of the final report, a presentation was made by K.A.Rhodes to the Executive Committee of the Board of JCI on 30 January 1986. Included in this presentation was a summary comparing the conventional and trackless mining methods and this can be seen in **Figure 5.5**.

The relevant extract from the minutes of the Executive Committee on 30 January 1986 is included on the following pages. The main recommendation being '***that all further planning associated with the Joel Mine be based on trackless mechanised methods'.***

5.7 **Shaft Sinking and Mid-Shaft Loading**
Pre-sinking of both No 3 and 4 Shafts commenced in August 1985; these operations at No 3 Shaft were completed in late October and at No 4 Shaft in early November of the same year. Pre-sink was carried out to a depth of 45 metres below the collar of both shafts.

NOTE TO THE EXECUTIVE COMMITTEE

EXCO MEETING - 30 JANUARY 1986

COMPARISON OF CONVENTIONAL AND TRACKLESS MINING SYSTEMS

		Conventional Mining System	Trackless Mining System
1.	Capital expenditure R1 000's (Phase 1 and Phase 2 Totals)	738 819	659 965
2.	Working costs		
	(Phase 1 - 80 000 tons reef/month)	R65/ton	R54/ton
	(Phase 2 - 120 000 tons reef/month)	R57/ton	R49/ton - reducing to *R47/ton
3.	Total labour		
	(Phase 1 - 80 000 tons reef/month)		
	a) Skilled	356	291
	b) Unskilled	3 900	1 782
	(Phase 2 - 120 000 tons reef/month)		
	a) Skilled	473	386
	b) Unskilled	4 800	2 459
4.	Tons/Underground employee		
	(Phase 1)	32,9	75,1
	(Phase 2)	38,8	79,1
5.	Tons/Total employee		
	(Phase 1)	25,8	46,8
	(Phase 2)	30,3	51,0
6.	m^3/Total employee		
	(Phase 1)	9,4	17,0
	(Phase 2)	11,0	18,5
7.	Total metres of footwall waste development; No 3 & 4 Shaft area of influence	110 000	22 000
8.	Total tons milled (Phase 1 and Phase 2)	38,12 million	40,00 million
9.	Recovery grade grams/ton (Phase 1 and Phase 2)	5,5	5,4
10.	Total gold produced Kg	209 495	216 195
11.	First reef tons	April 1987	April 1987
12.	Metallurgical Plant		
	1st Module	May 1988	January 1988
	2nd Module	September 1988	April 1988
	3rd Module	July 1992	June 1991
13.	Tons stockpiled before milling commences	125 900 tons	127 000 tons

In addition to the above, an improved safety performance is confidently predicted for the mechanised option.

* Lower working costs estimated when only No 1 and No 2 Shafts working.

FIGURE 5.5

A Comparison of Conventional and Trackless Mining Methods

Johannesburg Consolidated Investment Company, Limited

GROUP SECRETARIAL SERVICES DEPARTMENT

MJM/bl

6th February 1986

MEMORANDUM TO : MR G H S BAMFORD
MR J COETSEE
MR P J CRONSHAW
MR R L MENNE
MR K A RHODES
DR F J P ROUX
MR G W TREGONING

The following is an extract from the minutes of meeting of the Executive Committee held on 30th January 1986 - for your information :-

H J JOEL GOLD MINING COMPANY LIMITED - FINANCIAL ASSESSMENT OF MECHANISED MINING AND CONVENTIONAL MINING OPTIONS - (GHSB, JC, PJC, RLM, K A RHODES (KAR), FJPR and GWT present)

A memorandum from VGB dated 28th January 1986 and annexures thereto to the effect that the Technical Services Division had produced a revised plan based on the adoption of trackless mining methods (previous Exco approval to proceed with Joel had been based on the adoption of conventional mining methods) and that both the Gold and Finance Divisions which had conducted financial assessments of the two options had come to the conclusion that the relative advantages of the trackless mining method over the conventional mining method were such that the trackless method of mining should be adopted for Joel, was considered.

With the aid of a model of the Joel mine, slides, together with a summary comparing conventional and trackless mining systems, KAR reported in some detail on how mechanised mining operations would be conducted and indicated in essence that the trackless method offered, inter alia, a higher investment return, lower overall capital expenditure, and, largely as a result of a much smaller labour complement and considerably less footwall waste development, a significant reduction in working costs. KAR also pointed out that an improved safety performance was predicted for the mechanised option.

In reply to a question from GHW, HS-R and GHSB reported that a test stope (for mechanised operations) was being established at Cooke I, that mechanised operations were conducted at Cooke II and that it was not envisaged that the "learning curve" would constitute a problem insofar as the Joel mine was concerned. Attention was also drawn to the fact that some of the machinery required in a mechanised option was complex and that provision had been made for the necessary workshops and maintenance staff with a view to ensuring that a high level of machine/equipment availability was maintained.

HS-R stressed that it was absolutely essential that waste dilution be closely controlled, and indicated that the success of our mechanised mining operations would depend to a large extent on the degree of success achieved in limiting dilution to a minimum.

The following recommendations were then approved :-

(i) that all further planning associated with the Joel mine be based on trackless mechanised methods,

(ii) that planning be implemented to allow for the potential expansion of the mine from 120 000 to 160 000 tons per month, and

2/.....

uf

H J JOEL GOLD MINING COMPANY LIMITED - FINANCIAL ASSESSMENT OF MECHANISED MINING AND CONVENTIONAL MINING OPTIONS - (GHSB, JC, PJC, RLM, K A RHODES (KAR), FJPR and GWT present) (Contd.)

(iii) as a contingency, in the event that implementation of the mechanised option was not cost efficient, that the mining plan make provision for reverting from mechanised methods to conventional mining methods.

As a consequence of the adoption of the recommendations referred to above, it was agreed that PJC should submit a revised financing plan to Exco. At the request of GHW, it was also agreed that the necessary report to Exco should incorporate details in regard to the new breakeven prices for operating costs and operating costs plus capital expenditure.

Mike

M J Meyer
Secretary

Main sinking started on 06 January 1986 after the erection of the two headgears, refer to photograph in **Figure 5.6**.

Following equipping of both shafts for mid-shaft loading (MSL), trackless development started on 60 Level on 30 December 1986. On 70 Level, trackless development started in July 1987 following the equipping of the MSL between 60 Level and 70 Level. At the end of December 1987 shaft sinking and the associated development had been completed at No 3 Shaft and stripping had commenced from the shaft bottom up the shaft. These stripping operations were completed to surface in February 1988 and the use of the service cage, which had been operating in that shaft to support the MSL development operations, was then lost to MSL development. This left MSL development operating under single outlet conditions from No 4 Shaft and only in May 1988 could it be expected that a single cage facility would be made available in No 3 Shaft.

Up until this time (in effect the end of April 1988) 4314 metres of trackless development had been completed using the MSL installation. In fact, when the No 3 Shaft had been completely commissioned in late 1988 for rock hoisting, men and material handling, more than 6000 metres of MSL development had been carried out simultaneously with the sinking operations and equipping programmes. The result of this was to bring forward the first reef production by one year.

This project, which involved the sinking of two shafts with its associated station development on four levels concurrently with MSL development on two levels, was extremely complex. The interface of these operations carried out on a seven day week basis necessitated a total ‘hands-on’ style of management for it to succeed. However, it also had to be remembered that the H.J.Joel Gold Mine was the first gold mine in South Africa to be designed as a trackless mine from the outset, with only minimal lead time, as has been described in the narrative to this chapter; this alone had required innovative planning and management controls. In addition to these factors, management had to be aware of the need to avoid an inrush of water from the deep underground aquifer, the constant dangers of methane, the complications of the ventilation systems when operating MSL concurrently with sinking, and the obvious necessity at all times for the safety of persons working in the shafts. During this period another aspect which proved important to

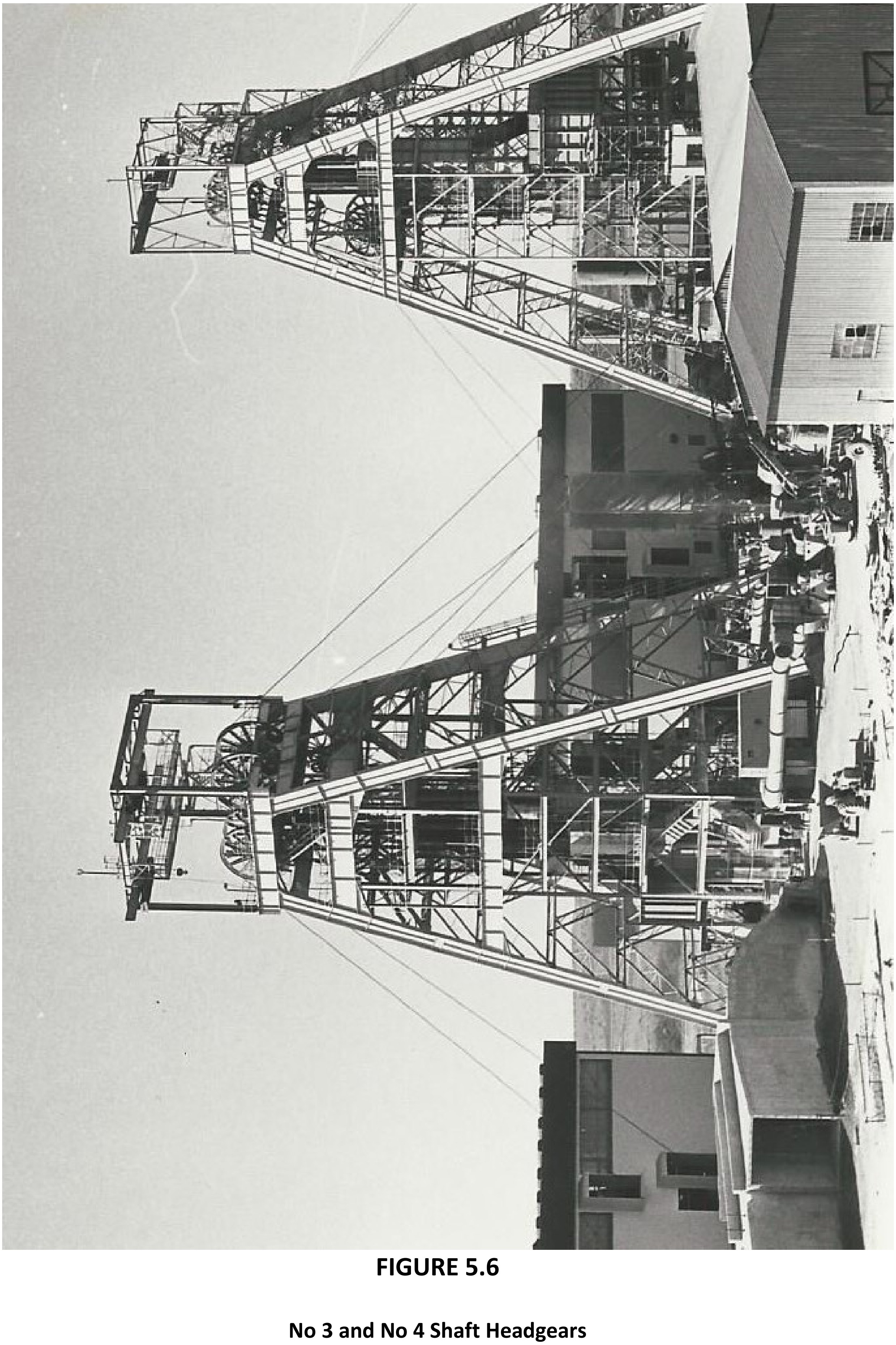

FIGURE 5.6

No 3 and No 4 Shaft Headgears

manage was the turnover of personnel, not only in shaft sinking but also for MSL development; work on a new developing mechanised mine was totally different from that on an established conventional gold mine and it proved difficult for new employees.

For a full description of shaft sinking and MSL operations at the mine refer to the technical paper, ***"Shaft Sinking and Mid-Shaft Loading Operations at H.J.Joel Gold Mine, Orange Free State, South Africa"*** by K.A.Rhodes, attached as **Annexure 5.3** in **Volume 3.** This paper was published in the transactions of the Institution of Mining Engineers in the United Kingdom, The Mining Engineer, in August 1988.

5.8 Early Mine Development

Throughout 1987 and 1988, during the early development and build-up of production at the mine, there were many investigations, optimisations and technical exercises carried out and, in addition, a substantial learning curve had to be overcome by the mining and engineering personnel, most of whom were experiencing trackless mechanised mining for the first time. Some of these issues can be discussed here.

5.8.1 *Rock Mechanics Considerations*

As more knowledge of underground strata conditions became available during early development operations at H.J.Joel Mine and also information from the neighbouring Beatrix Mine, it was necessary to review the support system for the mine. The changing conditions at the mine would lead to the consideration of various options. Originally it was considered that timber props and yielding reef pillars would be the support system and later the use of timber props and grout base packs was favoured. Nevertheless it was stated in the October 1985 motivational report that the use of backfill was to be investigated.

In terms of a technical evaluation of the effectiveness of all the above options it became evident that a backfill method would provide the best support system for the mine, this taking cognizance of the potential for inrushes of water as mining would be taking place in a deep aquifer. There would also be a need for regional pillars but to a certain extent this would be provided for by fault losses due to the increasing number of N-S faults being encountered in the early development work.

Design work therefore commenced for the introduction of uncemented backfill material over the total stoped out workings.

5.8.2 ***Ventilation***

In Phase 1 of the operation, final calculations for the main fans' duties, taking cognizance of the trackless equipment planned for, would be 350 – 360 m^3/sec of air at the density of 1kg/m^3. The expectation of hot fissure water could cause relatively high wet bulb temperatures which would necessitate that heat tolerance testing facilities would have to be made available. In addition, the start of Phase 2 of the mine, where operations would be in the northern deeper part of the mine, would dictate the installation of a refrigeration system.

5.8.3 ***Structural Geology***

In 1988 it became clear that the immediate area of the mine being developed from the No 3/4 Shaft system was significantly more affected by faulting than was at first thought, specifically N–S faulting, which was to break up the mining area into smaller blocks. In addition, the dip of the reef was much steeper than had been originally expected. This improved understanding of the geology would certainly have necessitated an increase in development if conventional mining had been planned for, and the advantages of trackless mining, when negotiating geological faults between mining blocks, was therefore clearly proven.

5.8.4 ***Cover Drilling***

At the time of the original Feasibility Study it was known that the neighbouring Beatrix Mine had intersected water fissures on dykes and faults and it was anticipated that water intersections would occur at H.J.Joel Mine. However, no specific information had been gained from exploration drilling. As elsewhere in the Free State Goldfields the area of the project was characterized by the confined (deep) Ventersdorp and Witwatersrand aquifer which is overlain by the relatively impermeable Karoo sequence. The free ground water table in the Karoo sequence was measured prior to the commencement of work at H.J.Joel Mine and the average below ground level was 20 metres. Prior to the commencement of mining at Beatrix Mine in 1980 it is believed that the pressure head in the deep aquifer was about 110 metres below ground level. However as mining operations expanded at Beatrix this water level continued to drop. This was to be expected but in early 1986 when the Beatrix Mine was close to being flooded following a major water intersection it was decided to drastically intensify cover drilling and cementation during shaft sinking at H.J.Joel Mine and this continued during MSL development operations. This was a very necessary step to take as only limited pumping capacity was in place during the early sinking and development phase. In order to significantly reduce the risk of any inundation, it was decided to drill the cover round from the face thereby causing development of the end to stop while cover drilling was taking place. Thus the development in such an end was restricted and thereby partially negated the advantages of trackless mechanised development. The rate of advance in development ends was further reduced by the necessity for diamond drilling which was much slower than percussion drilling; however, this was decided for safety reasons when more control could be exercised in the event of striking water. Only in late 1988, when the deep aquifer had been de-watered to well below 60 Level, did cover drilling operations revert to percussion drilling, carried out simultaneously with development.

5.8.5 ***Methane Gas***

Associated with the large quantities of fissure water was methane, which was present in solution and later released into the underground workings. Emissions of methane would occur during pumping operations and also when de-watering caused a lowering of the water table of the deep aquifer. Under these conditions, as was generally the case in the southern Free State Goldfields, certain regulations which applied to fiery mines were also made applicable to the H.J.Joel Mine.

In terms of the directive from the office of the Chief Inspector of Mines in Virginia, it was necessary for the mine manager to compile a Methane Manual which was both comprehensive and had wide implications for the operation of trackless equipment. In the preparation of this manual the mine manager was assisted by the document “Flammable Gas in Metal Mines, A Guide for Managers” dated October 1989 and based on the original Guide published by the Association of Mine Managers, Orange Free State Branch in 1973. Since the issue of the original guide cognizance had been taken of the revised regulations which prohibited any work when the flammable gas concentration was above 1%, thereby necessitating the mandatory use of accurate flammable gas detecting instruments. Refer to **Annexure 5.4** in **Volume 3** for the contents of this manual and see Section 2 which is specifically relevant for trackless drill rigs in use at the mine.

On a brief technical visit to the United Kingdom in August 1987, KAR observed a control system at Selby Colliery whereby the underground environment was being monitored from a surface control room. At that colliery, at every working face, a sensor head was transmitting to surface the concentration of methane in the general body of the air, and outbye of the face, a further monitoring device was recording the roof layer. This system for a gassy mine was considered invaluable and would later be introduced by KAR for the H.J.Joel Mine.

5.8.6 ***Stope Face Drilling and Blasting***

It had been assumed in early motivations that the stope face would be drilled conventionally with pneumatic jackhammers. However, by 1988 it had become a commitment to introduce a hydraulic rig (the Stomec stope rig) to improve the efficiency of the project.

The blasting system, which still had to be finalised, would incorporate delay detonators for improved throw and to minimise cut-offs; nonel or magnadets would be the choice.

5.8.7 ***Dilution Control***

The operation of large trackless machines in narrow reef conditions was always going to be challenging. It had been shown that waste dilution from the trackless mining method should not be greater than for a conventional operation. It had also been shown that the theoretical volume available for packing in worked-out areas would always be greater than the volume of rock to be blasted in development operations hence proving that it was theoretically possible to pack all the broken waste, notwithstanding that it had been planned to only tram 60% of the total waste to worked-out areas. However, it had been estimated that if all the waste from on reef development was allowed to be sent as reef to the mill then the waste content of tons milled would be 18%. The necessity to exercise control over waste tramming was therefore obvious to everybody.

5.8.8 ***Optimisation Exercises***

During the early period of shaft sinking there were opportunities for KAR to consider optimisations to the mine plan. One such proposal was to develop a ramp between 60 Level and 70 Level thereby eliminating the need for a workshop on 70 Level and making use of the 60 Level main workshop complex. The cost savings of this proposal were R100 000, which in today's 2013 terms would be R1,2 million. There were also non-quantifiable benefits to be gained from this proposal, mainly that engineering maintenance supervision would be improved due to the concentration of all services in one workshop. Also the overall supervision of the mine would be improved by a connection between the two levels for supervisors' vehicles; in the original plan access between the two levels was by means of a vertical shaft only.

This proposal, submitted to the Consulting Engineer, motivated by the cost saving and supported by the Capital Projects Control (CPC) Department, was approved; refer to memorandum dated 16 June 1986 and supporting document from CPC in **Figures 5.7, 5.7A and 5.7B.**

Further proof of the flexibility of the trackless method of mining was an exercise related to 70 Level operations. In terms of the Option 10 Plan, 70 Level was planned as a gathering haulage with trackless access on 60 Level. Geological information which came to light in early 1987 caused 70 Level to have a dual purpose: a gathering haulage by means of dump trucks and as a trackless access to a portion of one of the blocks (Block A) also accessed by 60 Level, this being necessary due to a fault having divided Block A. The rail gathering haulage was then planned for 90 Level.

H.J. JOEL GOLD MINING COMPANY LIMITED

RECEIVED 23 JUL 1986 23/7/86

MEMORANDUM

T.R.S.

Looks good to me – do you have any comments?

4/7.

Ref: KAR35

TO: MR G.H.S BAMFORD
CONSULTING ENGINEER

FROM: K.A. RHODES

DATE: 16 JUNE 1986

SUBJECT: OPTION 10 OPTIMIZATION

In terms of the Option 10 Motivation Report a workshop would be provided on 70 Level Gathering Haulage.

It is now proposed in an optimization plan to develop a ramp between 60 Level and 70 Level and eliminate the workshop on 70 Level ; all back-up services for vehicles operating on 70 Level will be provided by the workshop complex on 60 Level.

In addition it will not be necessary to construct an explosives store on 70 Level (in addition to 60 level).

The cost savings envisaged in this proposal are in excess of R100 000 (refer to attached note ex C.P.C Department).

Further to these cost savings there are certain major non-quantifiable benefits from this proposal as follows:

(a) Engineering services will be concentrated in a single workshop complex thereby improving engineering maintenance supervision.

(b) Overall supervision of the mining operations will be improved by the connection between 60 Level and 70 Level (previously no travelling way was planned between the two levels).

It would be appreciated if you could indicate your approval of this optimization.

K.A. RHODES
PROJECT MANAGER

c.c N E W
J C P
B C G

FIGURE 5.7

Optimisation Proposal for Ramp from 60 Level to 70 Level

FOR THE ATTENTION OF:- MR. K. A. RHODES
PROJECT MANAGER
N.T. JOEL G.M. Co LTD.

NOTE TO MR W.R.WOOLLS 21/5/86

COST VARIATION - DELETE WORKSHOP ON 70 LVL AND REPLACE WITH INCLINE BETWEEN 70 AND 60 LEVELS (REVISED TO LATEST INFORMATION FROM MR. K. RHODES 20/5/86

70 LEVEL WORKSHOP

DEVELOPMENT WORKSHOP	60m x 10m x 5m = 3000m³		
	30m x 10m x 5m = 1500m³		
FUEL BAY	12m x 4m x 4m = 192m³		
EXPL STORE	12m x 6m x 3m = 216m³		
FUSE STORE	6m x 6m x 3m = 108m³		
	5016m³	@ R42	210672
CONCRETE TO WORKSHOPS			133500
CRANE SUPPORT STRUCTURES			85000
CRANES			45000
ELECTRICS			20000
WORKSHOP EQUIPMENT (50%)			125000
FUEL TANK & MISC (ALLOW)			10000
			R629172

RAMP 4,5m x 3m x 700m (KR)

DEVELOPMENT 9450m³ @ R46/m³	434700
ROOF BOLTING TO DEVELOPMENT	32000
STEEL SUPPORT TO 20% OF DEVELOPMENT	25000
CONCRETE DRAIN	21000
ELECTRICS	5000
	R517700

COST SAVING R111 472

NOTE:
1) ABOVE BASED ON DEVELOPMENT BY MINE
2) BASE DATE PRICES -JULY 1985
3) NO ALLOWANCE FOR ANY CHANGE TO SIZES OR EQUIPMENT AT 60 LEVEL WORKSHOP.

FIGURE 5.7A

Cost Variation: Cancel Workshop on 70 Level and Develop Ramp from 60L to 70L

JOHANNESBURG CONSOLIDATED INVESTMENT COMPANY, LIMITED

TECHNICAL SERVICES
DIVISION

CAPITAL PROJECTS CONTROL DEPARTMENT

M E M O R A N D U M

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Chrono

TO :		
	MANAGER - C.P.C	
	or GROUP CAPEX MANAGER - C.P.C	

FOR : THE PROJECT MANAGER - MR. K.A. RHODES

FROM : THE PROJECT CONTROLLER - MR. I.S. BROWN

DATE : 27th June 1986

SUBJECT : H.J. JOEL GOLD MINING COMPANY LIMITED
OMISSION OF 70L WORKSHOP

The omission of 70L Workshop necessitates the inclusion of a service ramp between 70L and 60L in order that 70L equipment can be serviced etc., in the 60L workshops. The estimated value of the aforementioned Change of Scope is a saving of R100 000.

NOTE: a) Base date of Costs - July 1985

b) Development by Mine

c) No variation to 60L Workshop

IAN S. BROWN
PROJECTS CONTROLLER - H.J. JOEL

ISB/jme/2331J

FIGURE 5.7B

Cost Saving: Ramp Optimisation Proposal

5.8.9 ***Equipment***

As the project gathered momentum in 1987 and into 1988 there were certain matters to consider with regard to equipment selection.

Size of Equipment

The equipment selected for the H.J.Joel Project was detailed in the Option 10 motivation report and was agreed to. In fact, the same size of equipment had been recommended and accepted for the 95L UEIA narrow reef project at Cooke 2 Shaft, REGM, previously discussed in Chapter 4. When taking cognizance of the known geology and structure of the orebody at the time of the initial motivation report, the selection of the equipment for the H.J.Joel Mine had been well considered. However, in those areas where the reef had been found to be generally narrower than expected it had been necessary to downsize some of the equipment on the reef horizon. Nevertheless, such a decision complied with the general principle of selecting the largest size machine possible, always taking cognizance of roadway dimensions and the possible effect of dilution.

Rigs

Six standard face rigs, capable of drilling a 3,8 metre hole, had been ordered. In addition two face rigs, with telescopic chain feeds capable of drilling a 2,8m roofbolt hole in a single pass in a height of 4,8 metres, had also been ordered; the same machines were also able to drill a 3,2 metre face hole. Early experience had shown that for the successful operation of electro-hydraulic drill rigs constant attention had to be given to numerous factors, some of which are identified below.

It was necessary to establish a standard procedure for boom movements during the drilling of a round and once determined it was essential to exercise discipline over the operators. Such a typical sequence is seen in **Figures 5.8** and 5.**8A** for the two boom drill rig; left hand boom (green), right hand boom (red).

The use of check list procedures by rig operators was vital with the necessary follow-up inspections by supervisory officials. In this respect the mechanical equipment supervisor was a key person.

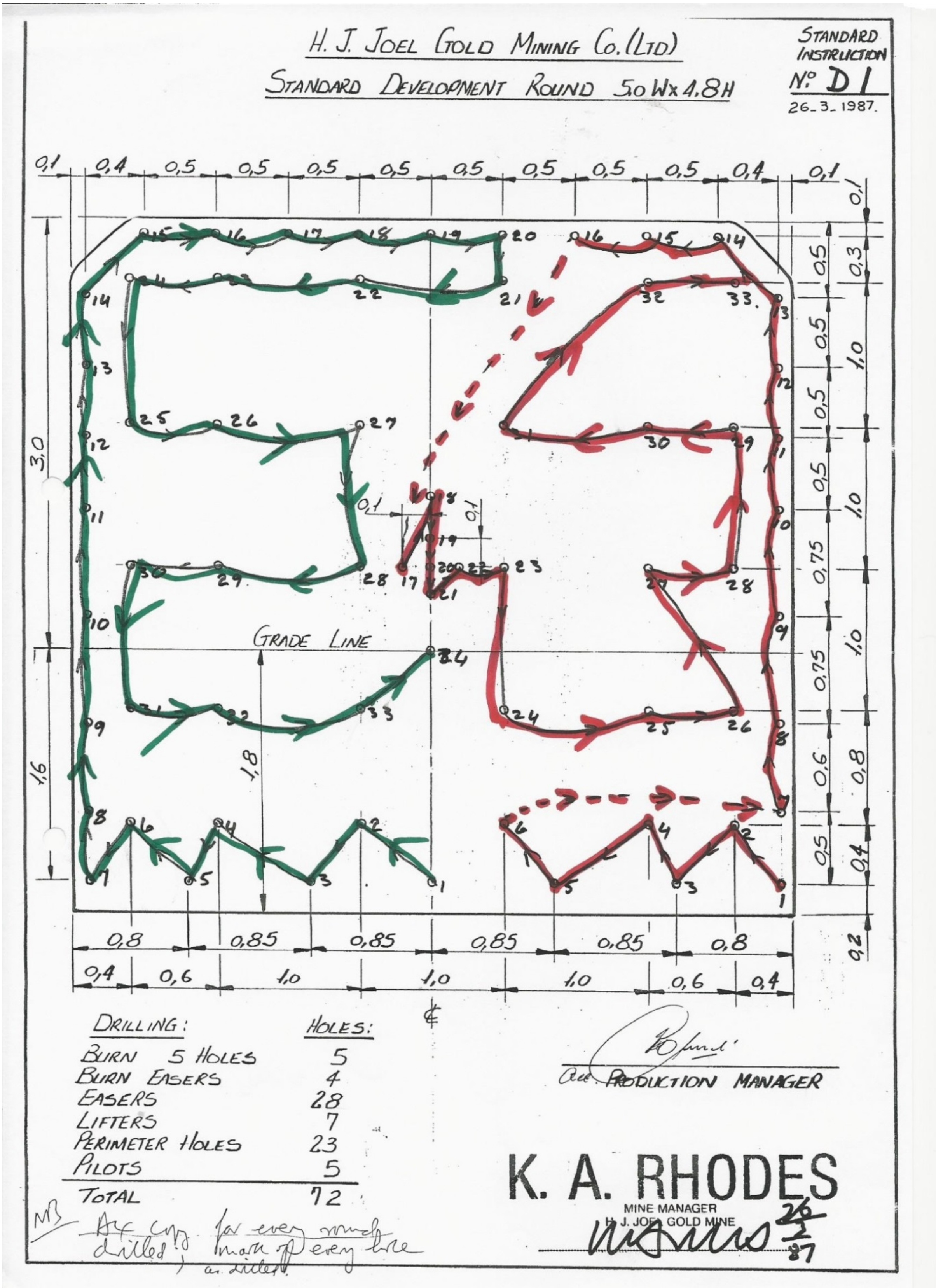

FIGURE 5.8

Typical Boom Sequence for a Two Boom Drill Rig

FIGURE 5.8A

Boom Sequence for Two Boom Rig with Details of Cut

It was important that supervisory staff, including senior management, had to be aware of the more important supervisory points in order to exercise proper control over the operation; failure to master these skills would cause costs to spiral out of control and reduce the drilled metres per shift.

It had become clear that poor operator performance, whether due to limited skills or plain abuse of the machine, would reduce drill string life markedly. Particular attention had to be given to hole collaring and bit removal. Incorrectly adjusted pressures would also exacerbate the problem.

It was also important to monitor drifter performance, the single most costly component of drill rig costs.

Roofbolters

Two dedicated roofbolters had been ordered, each with two booms to be able to drill a 1,8 metre roofbolt hole in a height of 3,3 metres in a two pass system; the second boom with a hanging basket enabled a person to change rods and install the bolt.

Face rigs were not used for roofbolting operations with the sole exception of the telescopic rigs, previously referred to, which had been introduced purposefully for the development of the high workings close to the shafts, for example in the workshop area.

The introduction of an automatic roofbolter had proved difficult due to the necessity to install the standard approved 2,7 metre long roofbolt. This same problem of simple geometry had occurred at the Cooke 2 Shaft 90 Level E8 Project and the same two pass roofbolters were introduced there and were still working. However, after two years of consideration and discussions with a specific OEM it had become possible to recommend a dedicated automatic roofbolter (a Robolt) which would be able to operate in the footwall service declines and ramps without necessitating any additional height. This new machine, operated by one man, would vastly improve the safety of the operation: all the functions necessary to drill the hole and install the bolt would be carried out safely by remote control with the operator in the cabin or under a safety canopy.

LHD's/Trucks

In terms of LHD requirements a later decision was taken to purchase two 7m³ units, the remaining six units being 4,6m³. This decision to opt for the two larger units was motivated by the (initial) long tramming distances from MSL waste development on 60 Level before the introduction of trucks was possible; the timing of the completion of the main tip being the crucial factor. The 7m³ machine proved to be a workhorse on both 60 Level and 70 Level. In fact, in terms of size the 7m³ unit was marginally narrower (bucket width) than the 4,6m³ machine. Refer to photographs of 7m³ LHD on surface and underground in **Figure 5.9** and **5.9A**.

The 4,6m³ LHD's and the 24 ton trucks operated well together, specifically in main development work. However, it was realised that when narrower reef than planned for originally had been encountered it had become necessary to reconsider the size of equipment on the reef horizon in order to control dilution.

Refer to photographs of 24 ton truck at the 60L main station tip in **Figures 5.10** and **5.10A**.

5.8.10 *Workshops*

Construction of the main workshop on 60 Level was carried out through 1987 and 1988, refer to **Figure 5.11** for a sketch of the original workshop layout. Refer to photographs in **Annexure 5.5** in **Volume 3** showing the workshop both under construction and partially completed.

5.8.11 *Technical Audits*

From the outset of operations, arrangements were made with the responsible OEM's to carry out technical audits of their equipment and also for them to assess operators' skills. Drilling audits were particularly important in order to maintain performance and for the control of drill rig costs, including drill string. Also, every opportunity was taken to gain technical knowledge from OEM specialists when they were visiting South Africa from overseas.

FIGURE 5.9

7m³ LHD on Surface

FIGURE 5.9A

7m³ LHD Working Underground

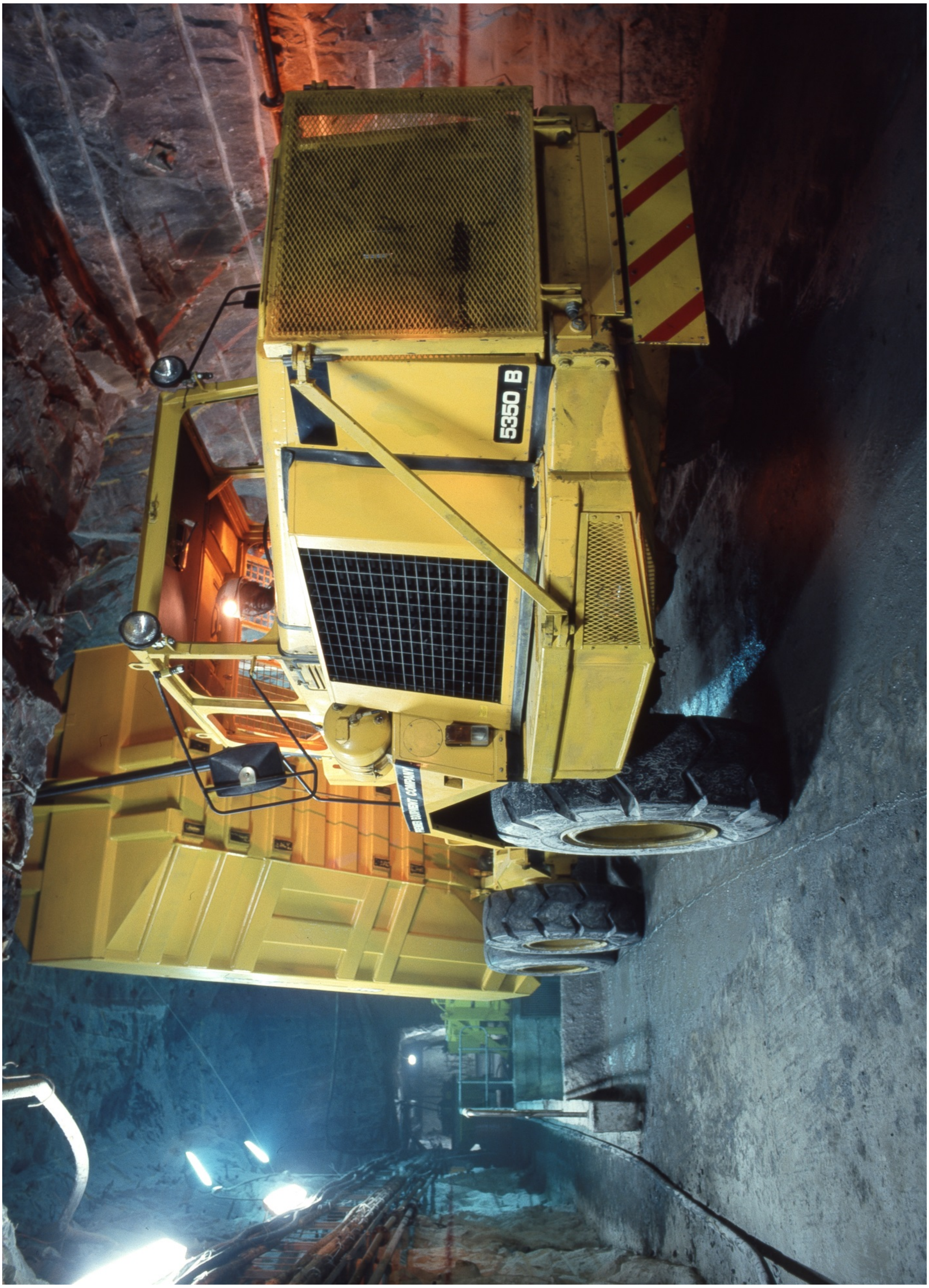

FIGURE 5.10

24 Ton Truck at 60 Level Main Station Tip

FIGURE 5.10A

24 Ton Truck Tipping at 60 Level Main Station Tip with Impact Breaker

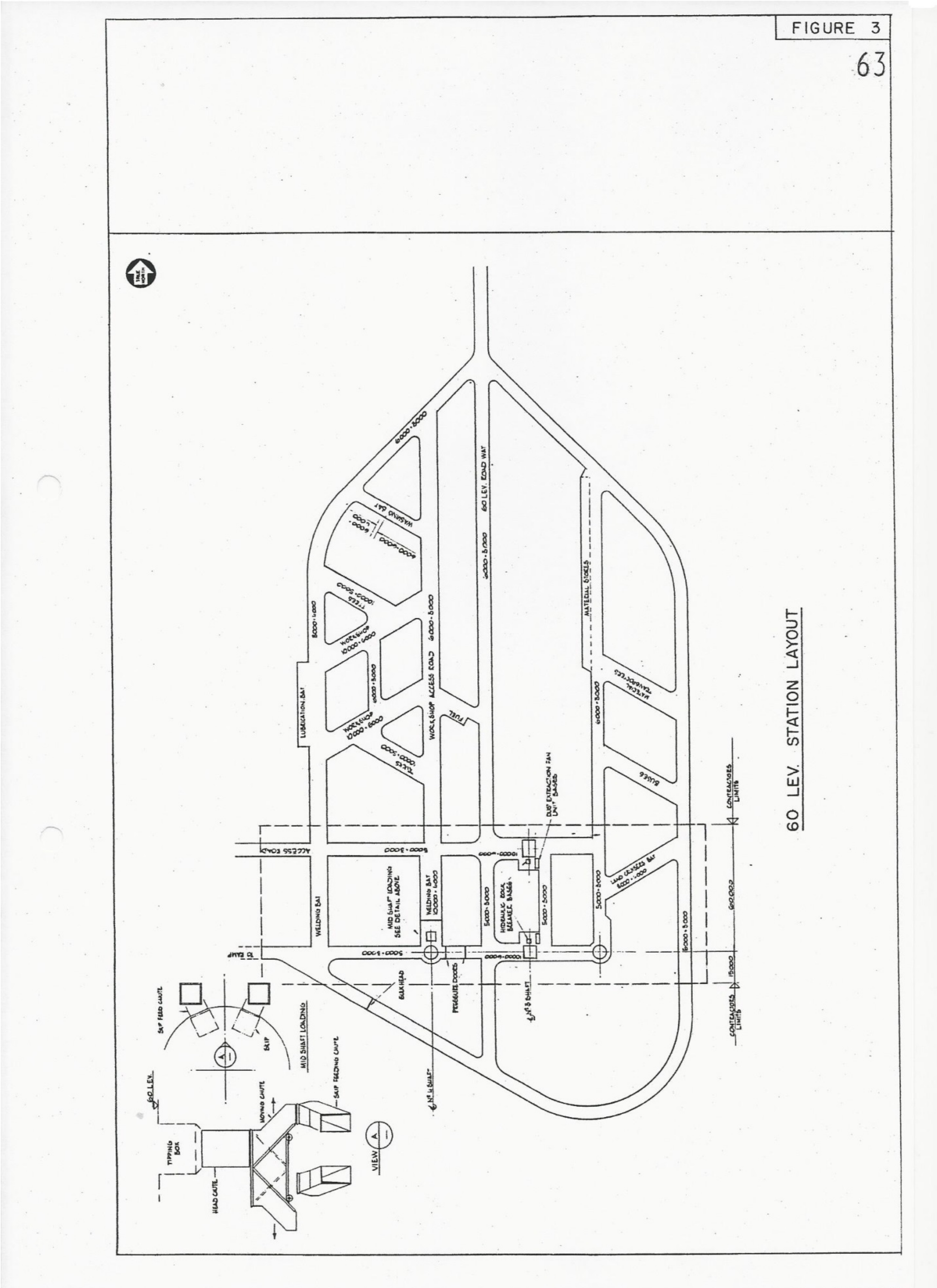

FIGURE 5.11

Original Workshop Layout on 60 Level Station: refer to Annexure 5.2 Volume 3

5.8.12 *Damage*

Damage and abuse to equipment is not uncommon throughout the mining industry world-wide, particularly in underground mines and where operators' skills are only considered to be less than average. At a new project like at H.J.Joel Mine damage could have been particularly high and it was therefore extremely important to exercise control from the outset. Firstly a damage investigation procedure was agreed upon; this investigation was initiated by the responsible project engineer and followed through by the MES.

Following the investigation a report would be forwarded to the responsible manager and resident engineer before being finally signed off by the mine manager. Full documentation of every incident, with the relevant approved disciplinary action which had been taken, was being kept by the mine manager in a damage book for future reference and action. Damage and abuse of equipment can be extremely costly to an operation and, more importantly, the loss of production caused by such damage is never easy to quantify but is undoubtedly very significant.

It was for these reasons that damage control was an early key issue and which also highlighted the importance for training programmes for operators, supervisors and management and the need for the MES to exercise driver discipline from the beginning; in fact, the appointment of the MES was made before the delivery of any equipment to the mine.

The early appointment of the MES was based on the experience of KAR at Cooke 2 Shaft, REGM where such an appointment had been made for the first time. This decision was vindicated as soon as trackless development commenced at H.J.Joel Mine. Because of the inexperienced staff and the difficulties of recruitment, and the fact that operators and supervisors had only limited, if any, experience with the concept of trackless mechanised mining, there was significant damage to equipment from the outset specifically in the first year of operations in 1987. During 1987 the mine was considering the recruitment of operators from Prieska Copper Mine in the Northern Cape of South Africa which was winding down its operations towards the end of its life. It was interesting to learn from Prieska that in the first twelve months of an operator's appointment disciplinary action for damage and abuse of equipment was very high but it fell away markedly after that period. Therefore, damage could be expected to be high early in the mine's development programme. However, without the discipline exercised through the appointment of the MES from the outset and a damage investigation programme driven by KAR as the mine manager, damage could easily have spiralled totally out of control. As it was, in 1987 damage was still occurring almost daily, as reflected on the daily reports from the MES.

5.8.13 *Recruitment and Training*

Undoubtedly the biggest challenge to the success of the mine, in the first two years of mine development, was the recruitment and retention of personnel, both mining and engineering, at operator and artisan level and also in supervisory positions.

Recruitment

From 1987 the major issue at H.J.Joel Mine, which affected the level of skills, was recruitment. The mine was a long way from JCI's gold mining operations in the Transvaal and it proved very difficult to persuade even a limited number of officials and supervisory staff to transfer to the Orange Free State. Also, for similar reasons it was not easy to recruit people with any experience from other mines. Even after recruitment it proved difficult to retain skills: in the first year of operations there was a turnover of 60% of production supervisory staff, both mining and engineering.

It was also difficult to recruit new employees with the right basic qualifications to be trained as operators. All new candidates for training had first to undergo psychometric testing before being accepted and the failure rate at the beginning was disturbingly high; see overleaf a letter, sent to the mine manager from the JCI appointed industrial psychologist and via the JCI Senior Personnel Officer based at Battery Reef Training Centre, which refers to the difficulties of recruiting candidates for trackless equipment operator training.

During 1987 and early 1988 attempts were made to recruit operators from far afield in South Africa, from Prieska Copper Mine nearing the end of its life and also from the du Toits Kloof

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INDUSTRIAL STAFF TESTING SERVICES
(PTY) LTD.

REG. NO. 73/06604/07

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~~Wynberg~~
~~Sandton~~

Noted. KAN 9/4/87.

16th March, 1987

RECEIVED
07 APR 1987

Mr. S. McLuckie
Senior Personnel Officer
Battery Reef Training Centre
Randfontein Estates.

Dear Mr. McLuckie,

Could you ask the Joel Officials if it is possible to obtain older applicants for the TM^3 operator jobs? Sixty per cent of this last batch were between 19 and 23 years old and fifty percent of them had never had a job of any kind.

Most of the young people are very bad accident risks for a number of reasons, ranging from lack of confidence, to inability to shoulder resposibility, to carelessness or even downright recklessness. I know this from bitter experience in Putco where we eventually had to raise the minimum age limit to 25 and finally to 26. With this sort of experience behind me I have had to fail a whole lot of this last batch of applicants. In fact, between the MTB tests and the TAT, only 7 out of 29 passed. This makes the whole exercise very costly and unproductive.

With older applicants, especially ones with some working experience, even if it isn't on the mines, the pass rate would be much higher, as can be seen from the fact that six of the seven PASSES had worked somewhere.

Yours sincerely,

Lyn Shaw.

MRS LYN SHAW
INDUSTRIAL PSYCHOLOGIST

PRODUCTION MANAGER	NEW	✓
RESIDENT ENGINEER		
MANAGER F AND A		
PLANNING MANAGER	BCS	✓
CHIEF GEOLOGIST		
CHIEF SURVEYOR		
MINE PERSONNEL MANAGER	EBO	✓
CHIEF SECURITY		
ENVIRONMENTAL SUP.		

Action MPM
Asst P.O.(H)

ē return to KAN

Tunnel Project which was nearing completion and where KAR had visited in 1986. A recruitment drive for artisans was also directed overseas with some limited success. Artisan skills was a particular problem as there were few skills available and to some extent labour brokers had stepped in to commandeer their skills; therefore the mine was also employing contract artisans through these companies.

Training

There was the need to train operators and supervisors working for the first time on a highly mechanised operation which was totally different and more highly demanding than work on a conventional gold mine.

At any new trackless mechanised mining project skills training is vital for operators, artisans, supervisors (both mining and engineering), engineers and managers. In this respect all suppliers of equipment (OEM's) had the responsibility to provide training programmes related to their specific equipment, these programmes being part of any package deal when purchasing an OEM's equipment. One such programme for supervisors (including managers and engineers) was compiled by the OEM supplying drill rigs. The theoretical part of this programme was set out in a series of simple drawings which identified components and also good and bad practices; this short training course proved highly beneficial and is considered of such importance that a copy of the course, with KAR's handwritten notes (in red), is attached as **Annexure 5.6** in **Volume 3.**

The importance of this supervisory training could not be over emphasised. Reports from the MES were constantly referring to instructions being given to the operators by too many people and the danger there was that untrained supervisors were giving instructions to trained operators, causing driver frustration. Another issue which often lead to conflict was that operators had a pre-conceived mind-set that their job was to drive a machine and at first they did not accept that if the machine was under maintenance or breakdown, they would be required by their supervisors to do other work, such as working on the improvement of roadbeds.

The initial training of operators following the selection procedures was the responsibility of Battery Reef Training Centre (BRTC) at Randfontein; this facility was set up by JCI for trackless equipment training for the Group's gold mines.

As with supervisory staff it was proving difficult to stabilise the complement of operators. In the first year there was a high absenteeism, particularly on weekends and Mondays; there was also a high desertion rate. In many cases, where operators were new to mining, they were unable to adjust to underground working conditions. Even after the first nine months of development work there was a deficiency of 25 operators of a planned complement of 100 and as operators were classified upwards from in-training through 'C', 'B' to 'A' it was significant there existed no 'A' category operators. In an attempt to improve on this situation a revised procedure for engaging and initial training was devised in discussions between the MES and BRTC. The crux of this proposal was that initial tests would take place at the mine and if satisfactory candidates would go to BRTC for full testing only. If the candidate passed both tests he would return to the H.J.Joel Mine for a period of say three months to work as an engineering or mining helper, in other words, a candidate operator who would be monitored by the MES on his attitude, presence at work (time and attendance), willingness to learn and work, self-discipline and generally prove himself worthy to be trained as an operator. Following a successful interim period at the mine he would return to BRTC for operator training.

5.8.14 *Standards*

Juxtaposed with the training programmes was the necessity to develop standards for the operation and maintenance of trackless equipment. It must also be realised that H.J.Joel Mine was a totally new greenfields operation and all procedures and job procedures were required to be set out, not only those relating to trackless equipment. The total number of such standards signed off by KAR in the early years, were in the hundreds, relating to shaft sinking, MSL procedures, ventilation, engineering procedures, in addition to all the trackless mining requirements. In fact, if they were all to be included in this record of work it would warrant a manual in itself. For examples of some very early managerial instructions related specifically to trackless mining, refer to **Annexure 5.7 in Volume 3** for directives forbidding the cannibalization of equipment, any breaches of which would be severely dealt with; general instructions for the operation of trackless equipment; special instructions to LHD drivers clearly stating that the LHD was to be used for cleaning operations only; a directive how to manage roadbeds including the control of water. These instructions were typical of the time, being hand-written by KAR, the mine manager; later directives and standards were of course more formalised.

A supervision report was also designed at that time specifically for trackless development which would assist front line supervisors.

5.8.15 *Management, Men and Morale*

Mining managers, firstly, had to have the required engineering technical knowledge in order to be able to manage and direct mechanised operations. Secondly, mining managers had to be committed to the engineering function. Mining managers were responsible for the operations or production whilst engineering managers were responsible for the maintenance and therefore the availability of the machines. Both had their independent responsibilities but they had to work as a team and it was the job of KAR to ensure that this happened. Wherever possible, advantage was taken for senior officials to visit Cooke 2 Shaft REGM to view progress and improve their knowledge and even to criticise where they thought it necessary; these visits were followed by reports which were distributed at senior level. Audits by the OEM's (and other audits by overseas experts) were scrutinised, discussed and action taken; even audits carried out for Cooke 2 Shaft, REGM followed the same procedure. All these reports and the relevant discussions were part of the objective of acquiring technical knowledge.

With a committed higher management team it was necessary to improve morale throughout the organisation from the top down to the lowest level. There were certain key stages: convince the men that the plan could be achieved; continuously talk to the men in order to motivate for a higher performance; maintain high standards of work and carry out major construction work on day shift only, under senior supervision; inform crews of performance and encourage competition between crews (use information boards for daily progress, for example, record the fastest round); maintain equipment in first class condition. It was also vital to have motivational briefings to emphasise objectives and safety standards. These would be month end gatherings for motivation and morale building. The address by the mine manager would explain overall monthly performance and would always motivate and strive for a higher performance.

The development of a new mine, and this mine was the first of its kind in South Africa, developed from farmland in the middle of the northern OFS, was not going to happen without some degree of autocratic management. Notwithstanding, KAR did introduce a concept typified by an acronym: ARA. ARA meant ***authority*** (delegated down within defined parameters), with the commensurate ***responsibility*** which makes one ***accountable*** for one's action. What KAR was attempting to put in place was a clear cut line of command where one's authority was known and when decisions were taken at all levels within defined parameters to suit the circumstances of a newly developing trackless gold mine.

5.9 Trackless Mining Symposium 1988

In February 1988 at the Trackless Mining Symposium held in Johannesburg and initiated by the Association of Mine Managers of South Africa (AMMSA), the paper "***The Design of a New Trackless Gold Mine***" by K.A.Rhodes, Mine Manager, H.J.Joel Gold Mining Company Limited, was presented and later published in the transactions of AMMSA. A copy of this paper is attached as **Annexure 5.8** in **Volume 3**.

This paper was awarded the AMMSA medal for the best paper presented at the symposium.

5.10 Postscript to Chapter 5

On 21 October 1988 the H.J.Joel Gold Mine was officially opened and the first gold pour ceremoniously carried out. The opening of the mine took place only three years after the first motivational report had been submitted for a change in the design of the mine from conventional mining to a trackless mechanised mining method. It had therefore taken only three years to change what existed originally as two farms, Leeuwbult and Leeuwfontein, into a producing gold mine, the first trackless gold mine in South Africa.

The mine was named after Jim Joel. He retired as chairman of Johnnies (JCI) in 1962 and at the time of the opening of the mine he was still alive at the age of 92 years. H.J (Jim) Joel was the last significant family link with the company founded by Barney Barnato in 1889 which was registered as the Johannesburg Consolidated Investment Company Ltd.

CHAPTER 6

Establishment of the Waterval Mine, Rustenburg Platinum Mines

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Establishment of the Waterval Mine, Rustenburg Platinum Mines

In this chapter KAR will relate to his involvement in the Waterval Project, a new platinum mine part of Rustenburg Platinum Mines. The chapter will also define his work as a mining consultant on the introduction of low profile mechanised equipment for narrow tabular reefs prior to the establishment of the Waterval Mine.

6.1 Introduction

At the end of 1995 KAR formed his own single member consultancy, KAR Mining Consultant cc and worked extensively as a consultant to Anglo American Platinum Limited (Amplats). This work was mainly focussed on the use of trackless mechanised equipment in narrow reef platinum orebodies. A most important milestone in this respect was an investigative report into the use of trackless mechanised mining in narrow reef stope widths; this report being submitted to Amplats in June 1999. What followed from this investigative report was the establishment of the Waterval Platinum Mine at Rustenburg Platinum Mines. The mine has since been re-named Bathopele Mine, but for the purpose of this exposition the name Waterval has been retained as all relevant documentation refers to Waterval Mine.

6.2 Availability of Low Profile Trackless Equipment in 1999

In the year 1999 there was seen to be a need to improve productivity and reduce operating costs at Amplats' mines and, therefore, the objective of the investigative report referred to above was to consider the application of trackless mechanised equipment in a mining width of 1,5 metres; this was later to be amended to 1,8 metres. Indeed, that perception in 1999 is even more valid today in 2014 than it was then as operating costs on platinum mines continue to escalate while metal prices stagnate.

It was intended to split this investigation into two phases. Phase 1 was the investigative survey to identify low profile drilling and loading equipment capable of working in narrow reef widths; a subsequent phase would be to define methods of mining and examine their viability at Amplats' mines. The investigation necessitated holding discussions with all original equipment manufacturers (OEM's) as to their current available equipment and also to find out if they were intending to pursue the development of low profile equipment. In this respect a matrix of drilling and loading equipment was systematically set out in the report. From this matrix it could be concluded that there was at that time only limited equipment available which would be capable of working in a planned mining width of 1,5 metres. Machines capable of working in narrow widths were identified as a drill rig, manufactured by Tamrock in France, which was intended to be used on South African chrome mines; at the time four machines had arrived in South Africa with the first rig sent to Millsell Chrome Mine. GHH were also manufacturing LHD's in South Africa and Germany. It was also learnt that low profile equipment was operating on KGHM's copper mines in Poland. The existence of low profile equipment working at KGHM's mines was clearly significant to this project.

The recommendations of this report were then set out.

6.2.1 To carry out a hands-on visit to KGHM's mines in Poland.

6.2.2 Subject to a positive report on KGHM's operations, it was then a recommendation to define mining methods and explore the viability of all possible trackless options.

6.2.3 It was also necessary to define the requirements and costs of operating trackless equipment in a 1,8 metre mining width at a new mine such as Waterval.

A copy of the investigative report ***An Investigation into the Availability of Low Profile Trackless Mechanised Mining Equipment for Narrow Stope Widths*** *by K.A.Rhodes dated June 1999 is attached as* ***Annexure 6.1*** *in* ***Volume 4.***

6.3 **<u>Visit to KGHM's Mines</u>**

In terms of the recommendations of the investigative report, submitted to Amplats by K.A.Rhodes, a visit to Poland took place in October 1999. Accompanying K.A.Rhodes on this visit were senior officials of Amplats. Also on this visit discussions relating to LHD's were held with GHH in Germany and in discussions with Tamrock in Austria it was confirmed by Tamrock that they had set their sights on a full range of low profile capital equipment. However, it was in Poland at KGHM's Polkowice – Sierozowice mines that full scale trackless mining in narrow reef conditions was seen (but not down to 1,5 metres). However, LHD's in use had insufficient carrying capacity and were therefore considered unsatisfactory.

Nonetheless, the overall general conclusions to be drawn from the visit to Poland was that it was now important to obtain proposals from the major OEM's for a full suite of equipment, able to operate in a mining width of 1,8 metres with a maximum machine height of 1,4 metres. The visit therefore provided the opportunity for KAR to initiate a new trackless mining design for Waterval Mine.

Brief notes on the visits to these mines and mine companies, ***Notes on the visit in October 1999 to Germany, Austria and Poland,*** compiled at the time by K.A.Rhodes, are attached as **Annexure 6.2** and can be seen in **Volume 4**.

6.4 Waterval Platinum Mine

In October 2000 the contract for the project management of Amplats' new Waterval Mine was awarded to Townsend Van Der Walt and Partners, Consulting Engineers (TWP) and KAR was asked by Amplats to be mining consultant for the project. Notwithstanding that a feasibility report had been completed by Amplats, this was an opportunity to design and consult on a new platinum mine which would employ trackless equipment from the outset.

The mining method would be room and pillar operating on the UG2 Reef horizon on full dip of not more than 10° . This would be the first major operation by Amplats to exploit the UG2 reef; up until then there had been only limited mining of the UG2 reef at other shafts at Rustenburg Platinum Mines.

6.4.1 *Access to Mine*

The control budget estimate (CBE) for the Waterval Mine provided for two decline systems (East and West). However, it was believed by KAR that in order to reduce technical risk to manageable levels it was necessary to sink three decline systems (East, Central and West). With reference to **Figure 6.1**, the arguments for this proposal were set out as follows.

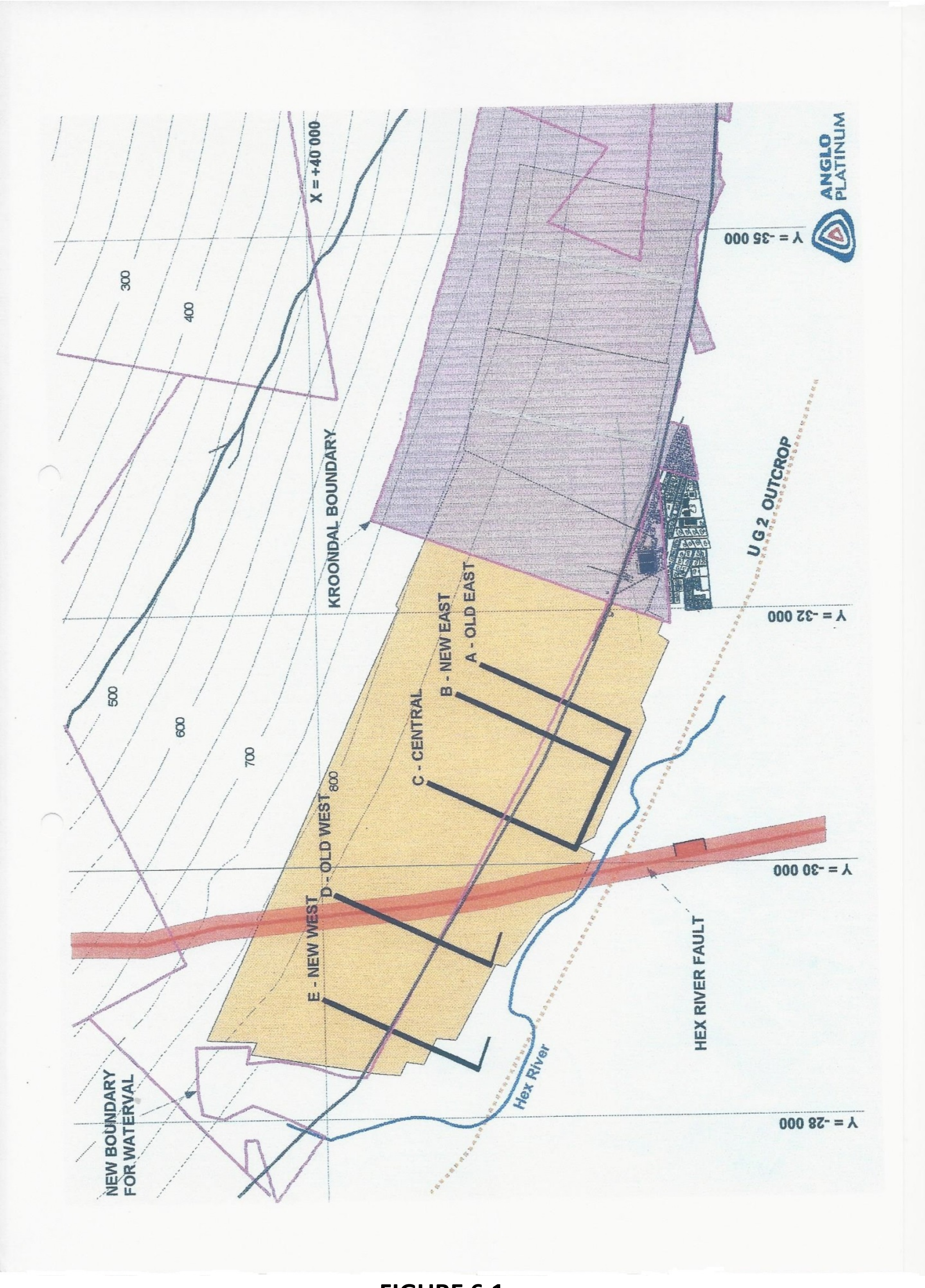


FIGURE 6.1

Three Decline Systems: East, Central and West

West Mine

At the West Mine it was considered most improbable that any production panel mining would take place beyond the Hex River fault (refer to Figure 6.1). The reason for this was based on experience of mining the Merensky Reef and more recently the experience gained on the UG2 Reef at the nearby Paardekraal Shaft. It was also considered most likely that approximately 50% of any mining operations at the West Mine would be affected by rolling reef conditions where amplitudes of 4 to 5 metres could be common; this opinion was also based on recent experience at Paardekraal Shaft. It should be recorded that KAR had wide experience of rolling reef conditions on the Merensky Reef as manager of Townlands Shaft in the late 1970's. Taking cognizance of these factors KAR recommended that the West Mine decline development be moved west by approximately 500 metres (again refer to Figure 6.1).

Although previously it had been assumed that total mine production would be split equally between East and West Mines, it was now recommended that the West Mine's planned production should be (say) one third only.

East Mine

Following from the recommendations for the West Mine, it then followed that the East Mine should be planned for an output of approximately two thirds of total mine production. However, in order to limit any technical risk it was further proposed to develop two decline systems east of the Hex River fault. There were several reasons for this proposal: two declines would accelerate the opening up of the area; there would be an increased geographical exposure to geological information related to the best mining cut (UG2 only or UG2 + Leader in the hanging wall of the UG2); two major points of attack would reduce the risk of any sudden loss of face, specifically due to pothole activity which could be significant; improved flexibility in terms of face availability which was important for trackless mechanised operations.

The access options and change of scope design parameters, intended to be used for capex estimates, were compiled and submitted by KAR on 19 November and 27 November respectively; see ***Waterval UG2 Project Access Options and Change of Scope Design Parameters*** by KAR, in **Annexure 6.3** included in **Volume 4**.

6.4.2 ***Access Development***

The Waterval UG2 Project CBE and also the subsequent mining development enquiry document had provided for three on reef access declines at each decline system. KAR did not believe that this was the best practical way to develop a mine based on the room and pillar layout; it was considered that additional access decline roadways were necessary (to act as 'ledging' roadways) in order to accelerate the opening up of production stoping sections off the main development. This recommendation to open up the mine by means of five on reef decline roadways as proposed by KAR was agreed to by Amplats. Refer to **Figure 6.2** for general layout of on reef mine development.

6.4.3 ***Mine Design***

The normal UG2 chromitite has a width or thickness of the order of 0,65 metres to 0,85 metres. The lower contact of the UG2 occurs above a pegmatoidal pyroxenite of approximately 10 – 30 centimetres in thickness, which is underlain by norite , generally in excess of 10 metres. The hanging wall to the UG2 is a feldspathic pyroxenite of varying thickness. The overlying strata of pyroxenite contains several chromitite layers and the first hanging wall chromitite above the UG2 is generally a substantial chromitite seam, typically 20 – 30 centimetres in thickness and is known as the UG2 Leader; its location relative to the UG2 is of great significance. There are other chromitite occurrences in the hanging wall, including what is known as the 'triplets'.

The location of the UG2 Leader above the main UG2 Reef dictates the mining width. In general at Waterval Mine the middling between the UG2 Reef and the UG2 Leader was too great and would normally preclude the mining of both together. Therefore it was planned to mine only the UG2 Reef with footwall (norite) waste to make up the mining width to 1,80 metres, a stoping width considered necessary for total mechanisation of the project; refer again to the investigative report, compiled by KAR, on the

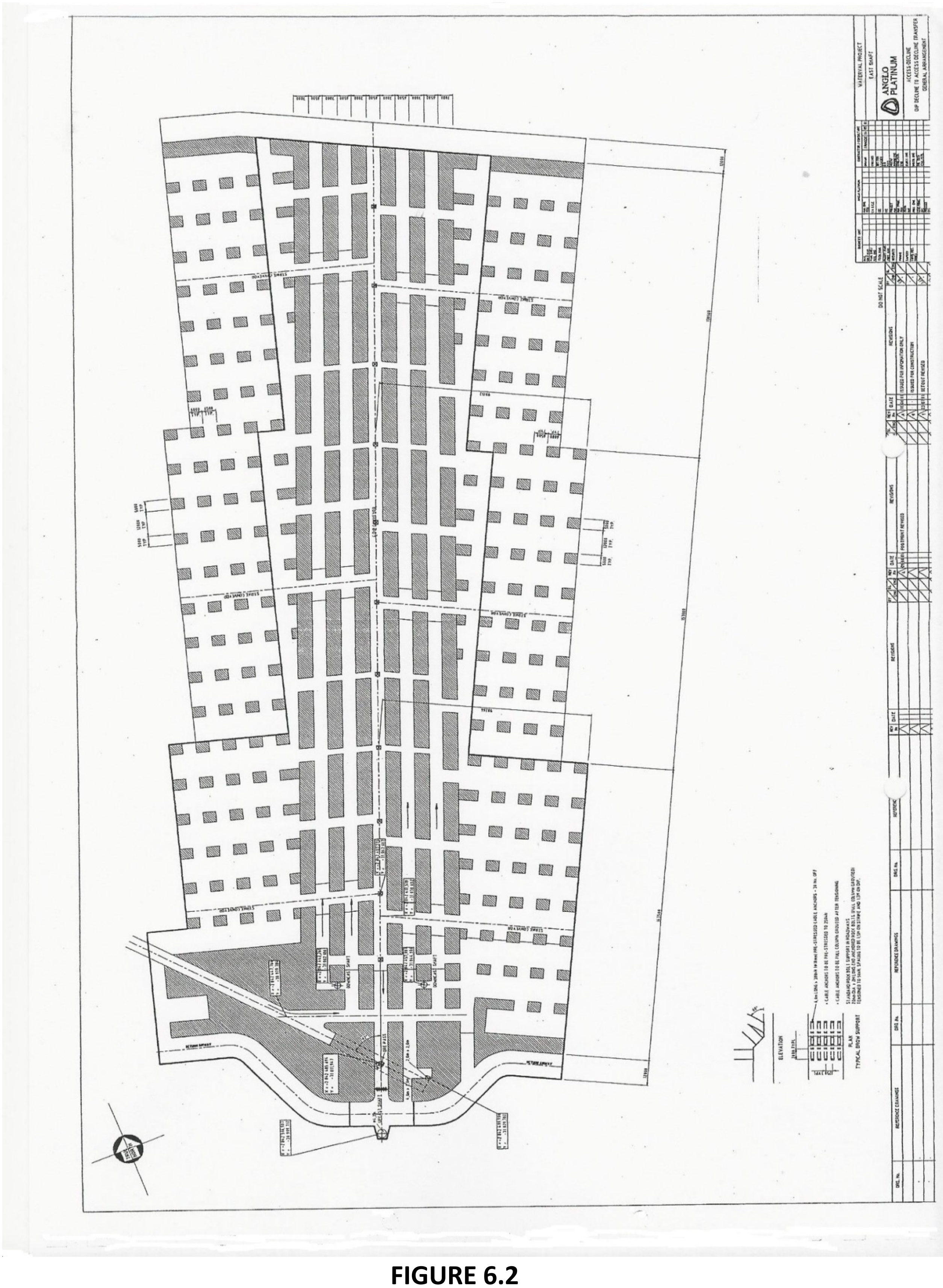


FIGURE 6.2

General Layout of On Reef Mine Development

availability of low profile machines for narrow stoping widths.

Room and pillar was the choice of mining method working on a full dip of 9°, this dip being generally consistent. It was now considered feasible to operate mechanised equipment on a true dip (with production panels on strike) of up to (say) 10° and therefore it was not deemed necessary to use the stepped room and pillar configuration as was adopted for the 90 Level E8 Project at REGM in the mid-1980's (refer to Chapter 3). All mining operations would be mechanised: face drilling, loading, roofbolting, charging up operations and transport. Transfer of reef to surface would be by means of conveyors.

Production Parameters

The planned UG2 reef production was for 140 000 tons/month.

Assuming

UG2 S.G.	= 4,0
UG2 Reef channel width	= 0,80 metres
F/W waste S.G.	= 2,9
Waste mined	= 1,00 metres
Then	
M^2 reef mined	= 140000 ÷ (4,0 x 0,8)
	= 43750m²
Tons waste	= 43750 x 2,9 x 1,0
	= 126875 tons
Therefore total tons broken	= 266875 tons

In fact it was assumed that 270 000 tons would be broken in total per month (140000 tons UG2 reef + 130000 footwall waste).

An important issue in the early design stages was the amount of waste to be separated and left underground and this will still be discussed.

Mine Access

As previously stated, there would be three decline systems. However, in establishing the mine only two declines would initially be developed; East and Central. The West Mine would only be developed at a later date. Each decline would have a conveyor and space for the movement of trackless machines. During the sinking of the two access declines it was planned to develop muckbays to a depth of 10 metres in order to reduce LHD tramming distances (LHD cleaning direct to surface) during the face cleaning cycle, thereby enabling face drilling and final rock clearance to take place simultaneously.

Generalised Mining Layout

All operations were on the reef horizon, the only exceptions being the main access declines and the necessary reef transfer arrangements at the bottom of each main decline.

The general opening up of the mine would be by means of a five road development (changed from three roadways) from which production sections would take place on strike. Main roadway dimensions were planned at 6,50 metres wide x 1,80 metres high with the central conveyor roadway being 2,0 metres high. Production panels were to be turned off at 5° above strike from the main development.

Panel Geometry

The main parameters of the room and pillar layout were rooms at 14 metres wide; initially pillars would be 6 metres on strike and 5 metres on dip but increasing with depth; pillar holings were planned to be 6,5 metres on dip, theoretical extraction being 87%. There were no barrier pillars planned for.

Cycle of Operations

Mining would take place on two shifts of 10 hours for six days a week.

The full suite of equipment would operate in a height of 1,80 metres. In terms of the aforementioned investigative report by KAR, such a suite of equipment would be available from major OEM’s. The basic cycle parameters would provide for single boom face drill rigs which would drill a round of not less than 3,2 metres; blasting with emulsion explosives; loading out of the face by a 6 ton low profile LHD; roofbolting using low profile single or double boom rigs with both manual rod handling and bolt installation.

Full reports from the responsible consultants for rock engineering and ventilation were submitted later. A draft skeleton document outlining the mining design criteria ***Waterval UG2 Project: Mining Design Criteria*** by K.A.Rhodes, dated June 2001, is attached as **Annexure 6.4** in **Volume 4.**

6.4.4 *Equipment Requirements and Equipment Selection*

The tender document called for a fleet of equipment comprising 18 (6ton) LHD's, 14 single boom drill rigs and 14 roofbolters; these requirements were based on the following calculations.

LHD's

Using the formulae for performance (P)

$$P = 51 \times L \div T + \left[\frac{2D}{S \times 16{,}67} \right]$$

Where:

L = 6 ton capacity
D = one way worst tramming distance of 100 metres
T = loading, manoeuvring and tipping of 3 minutes
S = average speed of LHD at 6kph

Therefore

$$P = 51 \times 6 \div 3 + \left[\frac{2 \times 100}{6 \times 16{,}67} \right]$$

= 61 (say) 60 tons/hour

For 280 hours/month working time the tonnage of 270000 tons/month would require 16 units. If it is assumed that performance on development down dip will be marginally less than in stope panels it was prudent to assume for 18 units.

Therefore number of 6 ton LHD's required was 18.

Drill Rigs

In a 14 metre room (and 6,5 metres split) the tons/blast generated was calculated as follows:

Room width	= 14 metres
Reef thickness	= 0,8 metres
S.G. of reef	= 4,0
Split width	= 6,5 metres

Waste section = 1,0 metres
S.G. of waste = 2,9
Advance/blast = 3,0 metres
Therefore:
Room tons/blast = (14 x 0,8 x 4,0 x 3,0) + (14 x 1,0 x 2,9 x 3,0)
= 256 tons/blast
Split tons/blast = (6,5 x 0,8 x 4,0 x 3,0) + (6,5 x 1,0 x 2,9 x 3,0)
= 118 tons/blast

If it is assumed that the guaranteed penetration rate of the rig was 2,5 metres/minute and there were 89 holes in a room and 49 holes in a split and also assuming a 40 second interval for boom movements between holes, then the time for drilling a room is as follows:

Room $= (89 \times 3{,}2 \div 2{,}5) + \left(89 \times \frac{40}{60}\right)$
= 173 minutes

If it is further assumed that it would take 5 minutes for a set-up (there would be two in a 14 metre room) and 15 minutes to tram between rooms, then total time for a round in a room
= 173 + (2 x 5) + 15
= 198 minutes

In a similar calculation the total time to drill a split is therefore:

Split $= (49 \times 3{,}2 \div 2{,}5) + (1 \times 5) + (1 \times 15) + \left(49 \times \frac{40}{60}\right)$
= 115 minutes

Therefore tons generated/minute from the drill rig are (256 ÷ 198) and (118 ÷ 115) or 1,28 and 1,02 for a room and split respectively.

The average tons/minute for rooms and splits is adjusted in the ratio of 2,5 to 1 (number of rooms for one split). Average tons/minute can therefore be calculated at 1,20 and the average tons generated in a month per drill rig, assuming 6 hours availability in a shift and 47 shifts/month, can be estimated at 20304.

For a total production of 270000/month, the number of drill rigs is calculated to be 13,3 (say 14).

Therefore the required number of drill rigs was 14.

Roofbolters

The estimated m² of ground exposed for 140000 tons reef production/month = 140000 ÷ (0,8 x 4,0)

= 43750m²

The roof bolt pattern was determined to be 1,5 metres x 1,2 metres. Therefore the number of roofbolts required/month

= 43750 ÷ (1,2 x 1,5)

= 24305

Assuming 10% additional bolts installed

= 26736 (say 27000)

Therefore number of roofbolts required in a shift assuming 47 shifts/month = 575

It was assumed that in a 6 hour shift that a roofbolter could install 42 bolts and therefore requirements would be 13,6 (say 14).

Therefore the required number of roofbolters was 14.

In March 2001 an enquiry document was issued for the supply of trackless mechanised mobile (TM3) equipment for the Waterval UG2 Project and all tenders were received from the main OEM's on 03 April 2001. In order to make the final recommendation on the choice of TM3 equipment, KAR deemed it necessary to follow a logical and systematic strategy to the selection process.

Preliminary Matrix

Following presentations by the OEM's a preliminary matrix was prepared to enable the number of options to be reduced. After a scrutiny of this preliminary matrix it was then decided to concentrate on the main suppliers; in principle this matrix was set out as in **Figure 6.3**.

During the preliminary stages of the selection process it became apparent that, although roofbolters were both available and being developed for narrow width mining, the selection of any specific roofbolter would best be deferred until certain trials and investigations had taken place. There were some sound reasons for this decision. The viability of rotary drilling had to be tested when drilling in the pyroxinite hanging wall and it was only expected that drilling trials would be complete by the end of that year (2001).

PRELIMINARY EQUIPMENT SELECTION MATRIX GUIDE

Supplier	LHD	Face Rig	Roofbolter
1			
2			
3			
4			
5			
etc			

For each OEM a symbol was recorded from the following selection

Notes

A (green) = Available to work in less than 1,80 metres mining height and currently operating in either South Africa or elsewhere in the world.

N/A (red) = Not available

D (blue) = Designed for and under construction with availability this year.

E (black) = Eliminated

FIGURE 6.3

The assessment of the use of Swellex was being made by the Amplats Rock Engineering Consultant and the development of a specific roofbolter to take advantage of the use of Swellex was also being considered.

One OEM had introduced its first roofbolter to Bleskop Shaft and another OEM's roofbolter was being assembled prior to trials.

In terms of the above it was decided to defer a final decision on a mechanised roofbolter until the end of 2001.

Second Matrix

A second matrix was used for the final adjudication of the remaining OEM's tenders. This matrix considered three aspects: costs, technical aspects and planned deliveries.

In terms of costs both capital costs and maintenance contract costs, projected for four years, were compared. These costs provided only for maintenance labour, spares, staff and administration. They did not include any provision for tyres, fuel, greases, oils, bucket lips, drill string or machine operators' costs.

The technical adjudication part of the matrix considered engine capacity kW, bucket carrying capacity m^3, axle capacity, tyres, tramming capacity in tons, height (with canopy), ground clearance, machine length and width, mass tons and power/mass ratio.

Finally, consideration was given to earliest available delivery dates.

Final Selection

The option of choosing only one OEM for both LHD's and face rigs was also considered as it could then be possible to negotiate a more favourable maintenance contract agreement.

At the end of a technically exhaustive process the recommendation by KAR, accepted by Amplats, was to select both LHD's and face rigs from Atlas Copco: the ST 600LP LHD and the

Boomer 281L 1SL face rig. Both these machines can be seen in photographs in **Figure 6.4** and **Figure 6.5**.

With regard to the roofbolter, later in the year it was decided to purchase the Boltec SL, making the total fleet Atlas Copco.

6.4.5 ***Blast Design***

The application of trackless equipment in narrow reef conditions in a room and pillar operation must necessitate mining waste. In other operations in the Bushveld Igneous Complex (BIC), specifically in chrome mines and also at Kroondal Platinum Mine (immediately downdip of Waterval Mine), scalping of waste underground was common. In these operations the waste portion of the mining cut is the pyroxenite middling, typically between the LG6 and LG6A on chrome mines and the UG2 and the Leader at Kroondal. In all these operations there is little or no drilling in the pyroxenite middling but only in the chromitite, above and below the middling. In practice, due to a closely spaced near vertical jointing, the pyroxenite middling breaks into blocks that allows scalping at the tipping point. However, the lithology at Waterval was such that the middling between the UG2 and Leader was too great to provide for a viable cut of both seams. It was therefore necessary to include footwall waste with the UG2 in the cut and as such required intensive drilling of the footwall waste portion in order to break the round.

The CBE document for Waterval had assumed that 45% of the waste could be scalped at the tipping point and packed underground. Notwithstanding, KAR argued that this was not practical, primarily because the footwall waste of the UG2, being intensely drilled, would be highly fragmented and be almost impossible to separate as there would be no marked difference in appearance of the chromitite and the footwall pegmatoidal norite. Therefore, 45% segregation could not be achieved and it would be sensible to plan for 0% and target (say) 10% with the possibility of 5% being achieved.

After trials, conducted by a blasting consultant at RPM's Bleskop and Boschfontein Shafts, the results showed that for a 1,80 metre

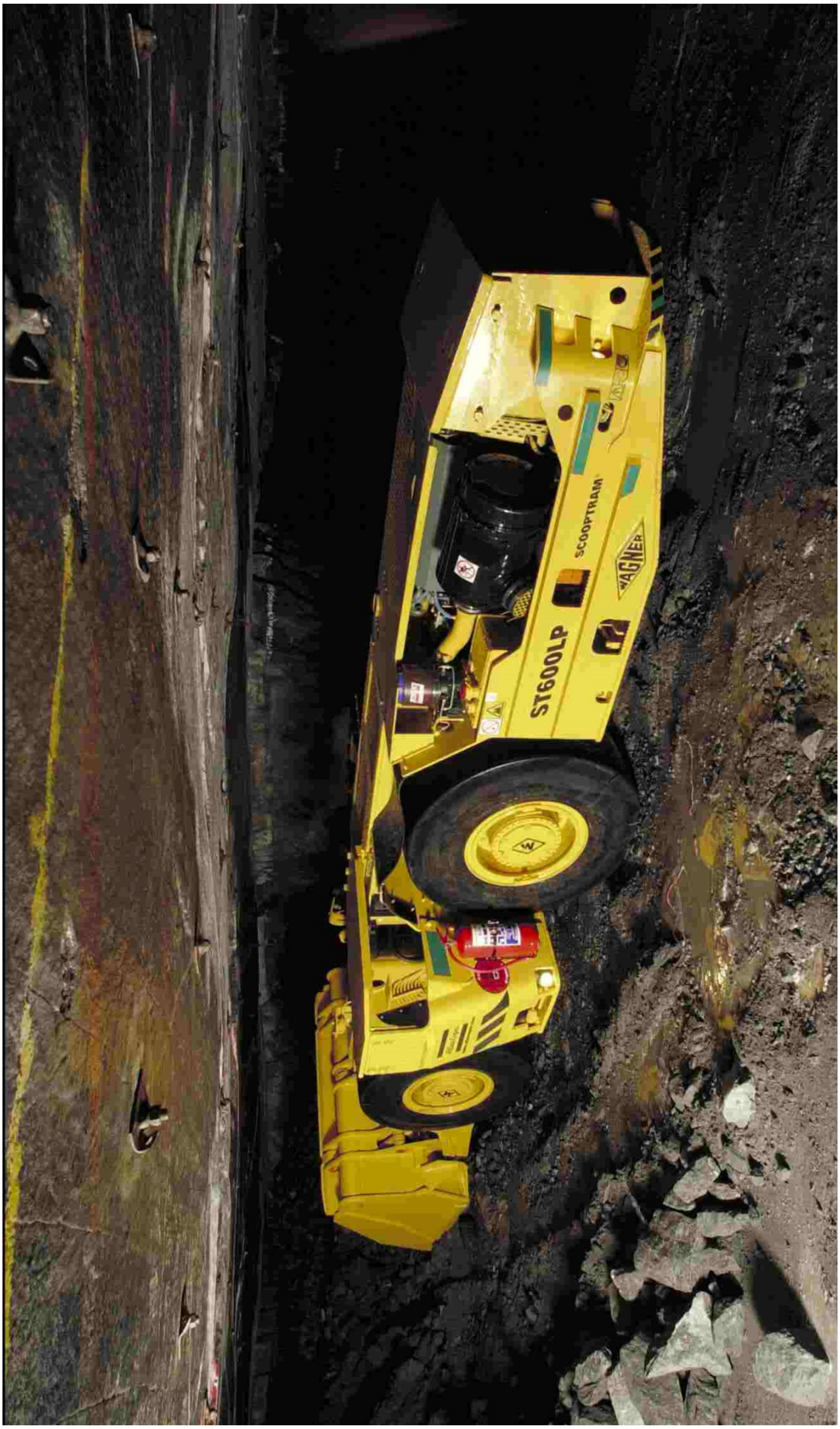


FIGURE 6.4

Low Profile LHD

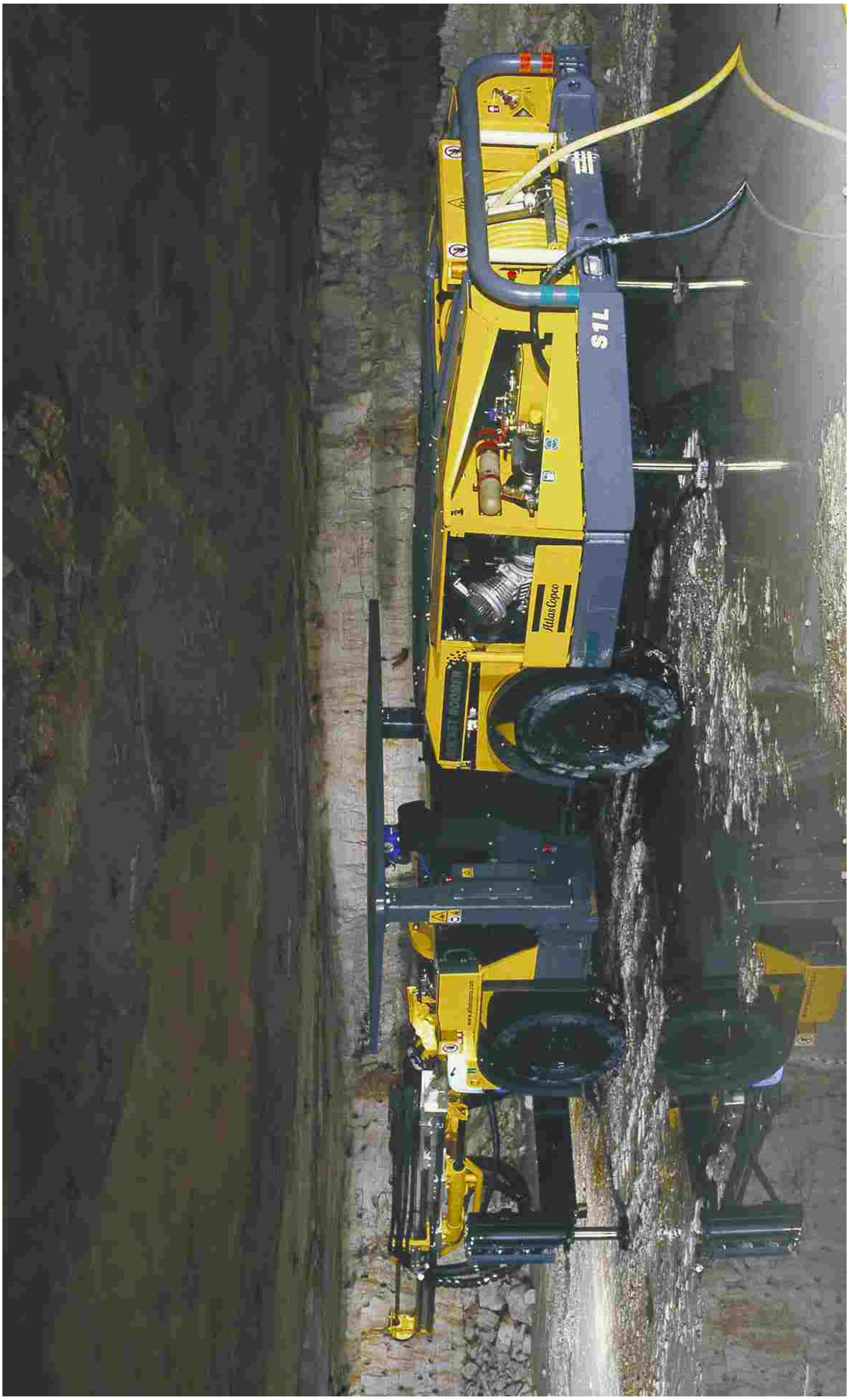


FIGURE 6.5

Low Profile Drill Rig

cut with the UG2 only and footwall waste being mined, only 2.5% of the blasted rock would not pass through a 300mm grizzley. Therefore the trials proved that the CBE document which estimated waste segregation to be 45% for the project was flawed and it also confirmed that the recommendations of KAR to plan for 0% segregation had proved correct. Refer to a report by A.J.Rorke titled ***Waste Fragmentation in Stoping UG2: Waterval*** dated 10 December 2000; in **Annexure 6.5** in **Volume 4.**

6.4.6 ***Face Availability***

For the production tonnage of 270 000 tons/month with no waste packed, and making an assumption that face advance in a room would be 35 metres/month representing one blast every two days or four shifts. Advance per month is 23,5 (days) ÷ (2 x 3,0 advance) = 35,25 (say) 35 metres.

The number of rooms required to achieve the planned call of 270000 tons/month (all blasted rock sent to the mill) is therefore 270000 ÷ (T x 35) where T = tons blasted/room/metre advanced.

Assuming SG of UG2	= 4,00
F/W Pyroxenite/Norite SG	= 2,90
UG2 channel width	= 0,80 metres
F/W waste	= 1,00 metres
Average SG of face	$= \dfrac{(0,80 \times 4,00) + (1,00 \times 2,90)}{1,80}$
	= 3,39
Tons/metre advance in a room	= 14 x 1,80 x 3,39 x 1,24
	= 105,9
Say T	= 105

Where Room width	= 14,00 metres
Room height	= 1,80 metres
Average SG	= 3,39

A conversion factor of 1,24 accounts for extra tonnage blasted in strike holings

The number of rooms required to be available at all times is therefore calculated as follows:

Number of rooms = 270000 ÷ (105 x 35)
= 73,4

However, in order to make provision for faces having to be stopped, primarily for potholes or for any other reason, it was prudent to plan for an additional 50% available face (73,4 x 1,5) or 110 rooms which could relate to twelve production sections of nine roadways. In terms of pothole activity, the UG2 Reef is known to descend below its normal footwall horizon and come to rest on a lower horizon; these depressions are known as potholes. The depth of the pothole can vary and they are common at RPM's mines. Although in certain cases some potholes can be mineable, potholes will disrupt mining operations and face will be lost while development takes place around the affected area. Potholes at this project could represent up to 20% of the mining area but their occurrence is both random and erratic and it was therefore necessary to plan for an additional (say) 50% face availability.

6.4.7 ***Engineering Maintenance***

It was always the intention to enter into an agreement with the principle OEM for a maintenance contract. In later years KAR was to establish a maintenance action plan which could be adapted for any underground mine and this will be further examined in Chapter 7.

6.4.8 ***Rock Clearance***

When the Waterval Project was being planned for the generally accepted method for rock clearance in semi-mechanised room and pillar operations in the chrome mines of the BIC, utilising LHD's for face loading, was by conveyor. No recommendations to change this system to truck haulage were made by KAR for Waterval Mine; notwithstanding that trucks had been introduced by KAR, for the first time, at the trackless projects at REGM and H.J.Joel Gold Mine, described in previous chapters. At the same time that the Waterval Mine was being established, KAR had been planning the East and West Boschfontein Mines for Amplats and these operations were planned for truck haulage.

The reason that no real consideration was given at that time by KAR for ore clearance by truck was because the Waterval Mine was a totally on reef operation where the channel width of about 80cms had already been diluted more than 100% to enable mechanised operations to take place. Nonetheless trucks could have been utilised in a single trucking roadway from an LHD loading point to surface or to a single decline conveyor and direct to surface. The hanging wall of this roadway would have had to be carried above the UG2 Reef horizon to expose the UG2 Leader (thereby gaining ongoing knowledge of the exact location of the UG2 Leader) in order to enable sufficient height for at least a 30 ton capacity truck to operate. In hindsight this could have been a better option. The operation would have been much simpler, particularly in the early build-up to steady state production, because there were excessive delays in the installation of the section strike conveyors and also with the new Stamler feeders where the most serious issue was big rocks causing long delays at the tip leading to increased LHD cycle times.

Immediately following the commissioning and hand-over of the Waterval Mine, KAR had advised the Board of Impala Platinum Mines to plan for trucks at their new operations at Ngezi in Zimbabwe; today, ten years later, Ngezi is still successfully operating trucks from underground to surface in a similar room and pillar operation.

6.5 Contractual Factors

The development of a new mine from scratch is no mean feat. There will always be issues, problems and challenges in the early stages and in the build-up to steady state production. It is not common for any mining engineer to have had the experience of starting up a new mine. Nonetheless, KAR has been fortunate, as a project manager and a mine manager, for the start-up of the Otjihase Copper Mine in South West Africa forty years ago; for the total design and management of the H.J.Joel Gold Mine in the Orange Free State, fully described in Chapter 5; and as the mining advisor and project consultant for the Waterval Mine at Rustenburg.

The Waterval UG2 Project was developed from the detailed design stage to potential steady state production in just two years. In terms of such a build-up period, the project must be seen in general terms to have been an overwhelming success. Notwithstanding, it was considered by KAR that the programme could have been further reduced and brought to the point where full production would have been realised earlier. There were specific issues which had an effect on the contract programme. Whilst it is not intended to interrogate the contract performance in any detail, reference can be made to two issues: development advance and site management. It must be stated from the outset that following the selection of the contractor, certain layouts were changed by KAR from the mining design originally documented in the feasibility study. A major difference in the development layout, motivated by KAR and agreed to by mine management, caused the development footprint to change from a three roadway layout to five roadways.

At the tendering stage the contractor submitted that they would achieve 18 metres advance at each decline on a daily basis, utilising a suite of equipment of their own. In order to maintain the same sinking rate over the five roadways it was agreed to supply the contractor with an additional suite of equipment provided by the mine. Theoretically this could have provided for a total daily advance of 30 metres. However, the target daily advance was agreed to be 25 metres by both the contractor and project team. Daily advances never achieved 25 metres however and the target was systematically scaled down to 17 metres per day as the project neared the end of its life. In reality, the average total daily advance for life of the project was of the order of only 12 metres at each decline.

In early 2002, when it became clear that the revised target of 25 metres in the footprint at each decline was unachievable by the contractor, KAR initiated indabas primarily between the project team and the contractor, by means of a specific objective action plan (SOAP). This technique had been developed and used before on many occasions by KAR on trackless operations and projects; refer to ***Guidelines for Specific Objective Action Plans***, attached as **Annexure 6.6** in **Volume 4**.

With regard to the lower than planned performance by the contractor at Waterval, many factors were identified at the indabas but probably the five most important of these were as follows.

Advance/Blast had proved unacceptable. Following the indabas, audits by a blasting consultant recommended that more attention had to be paid to drilling the round correctly with a concentration on the cut and also the use of stemming.

The Stamler Feeders had given problems primarily due to large rocks causing long delays at the LHD tipping point. In a response to an audit on the Waterval conveyor system, KAR wrote a Note for the Record, see Fi**gure 6.6**, specifically relating to the LHD requirements planned for in the design of the mine; refer to the calculation of the production capacity of the LHD (previously seen in this chapter) and shown again in **Figure 6.6A**.

Poor Ventilation Conditions in the contractors' area of responsibility were constantly experienced due to ventilation curtain brattices and ventilation doors being damaged in the updip areas of the mine being developed independently by the mine company; this caused intolerable conditions for the contractor in the downdip development.

A Lack of Control of Water from the upper production sections was causing flooding of the downdip development almost daily. On previous operations managed by KAR, an important directive was to develop production roadways below strike in order to force the pumping of water out of these faces and not to allow water from drilling operations to flow by gravity to the downdip development thereby causing flooding; this policy was not accepted by the mine management. This was a glaring example of the mine's inability to control service water in the production workings of the mine, with water from their mining operations (immediately updip of the contractor) flooding the downdip development roadways being developed by the contractor; through to the completion of the footprint the mine was unable to manage this issue.

Shift Changeovers required an interrogation with the focus on communications between shifts, communications during the shift and the organisation of work on any specific shift. In December 2001 KAR undertook a technical trip to Western Australia and on all the mines visited, there was in use a shift changeover system

NOTE FOR THE RECORD

ADI FRITTELLA AUDIT ON WATERVAL CONVEYORS

As requested I submit the following comments on the Adi Frittella audit on the Waterval underground conveyor system; these comments relate specifically to loading by LHD onto strike conveyors.

The LHD (6 ton machines) loading rate calculated when the equipment fleet was determined was 60 tons/hour. The operating hours per month/LHD can be estimated at 250 hours, this being equivalent to 15000 TPM/LHD. At 270 000 TPM broken (planned production requirement) this equates to 18 LHDs (6 ton capacity) and this was the LHD requirement in the original tender document which was issued at that time. Further to the order for 18 LPST600 LHDs placed with Atlas Copco an additional 4 similar units were ordered. Therefore the total fleet now at Waterval is 22 with one extra unit for training.

If it is assumed that 22 units produce 270 000 TPM at 250 hours/month/unit then the production rate can be taken as 49 tons/hour; this loading rate being significantly less than the original calculation above.

It is to be understood however that the above does not take into account any double handling of rock in terms of waste scalping at the strike feeders.

For any further clarification please contact the writer.

Ken Rhodes
Tel: 083 309 0787
e-mail: rhodes@kar.co.za

Problems with feeders onto conveyor — BIG ROCKS
∴ better to use trucks

KAR
October 2002

FIGURE 6.6

A Note for the Record by KAR: LHD Requirement

Calculation of Production Capacity of LHD's when Loading onto a Section Conveyor Feeder

In the following formulae

$$P \quad = \quad 51 \times L \div T + \left[\frac{2D}{S \times 16,67}\right]$$

Where

P, is the production capacity in tons/hour

L, the carrying capacity of the LHD, is 6 tons

D, the one way tramming distance is 100 metres

T, loading, manoevering and tipping time is 3 minutes

S, the average speed of the LHD is 6kph

Therefore, $P = 51 \times 6 \div 3 + \left[\frac{2 \times 100}{6 \times 16,67}\right]$

= 61 tons per hour, (say) 60 tons per hour

FIGURE 6.6A

which had been in use for about five years. In that system, specific instructions for the oncoming shift (two changeovers a day) were written onto a board, usually by the manager in charge. All workers on the oncoming shift were present at this briefing and at the end of the meeting, which lasted about twenty minutes, a copy of all the instructions written on the board were given to all the shift workers (copies being printed directly off the board). Notwithstanding that South African mines, even mechanised mines, are far more labour intensive than those in Australia, such a changeover shift meeting, which could include say for example supervisors and primary equipment operators, would be extremely advantageous to any new trackless mining operation.

Unfortunately, despite the identification of these major issues the contractor did not improve on their performance. Throughout the period of the contract the contractor continued to make repeated senior personnel changes, both at site manager and master sinker levels, and the new appointees were generally recent entrants to the company. There can be no doubt that repetitive changes of site management by a contractor on a project employing trackless mechanised equipment, which requires hands-on control, can only be destructive to the project.

6.6 Postscript to Waterval

As stated in the above paragraphs, execution of the project could have been better. However, the starting up of any new mine will always have its problems (or challenges if one prefers) but that is to be expected in any new operation; there is no textbook guide available to start up a new mine.

However, there are some very clear principles learnt by KAR when building a new mine or even for any new project. In the opinion of KAR it is important during the start-up of a new mine for all persons to be able to see clearly the line management structure, to understand who is the responsible person in charge and also to accept that an autocratic style of management may be needed at times. It can only be re-iterated that the building of a new mine is totally different from the ongoing management of an established mine.

At Waterval Mine the company made the decision to appoint an outside consultancy to manage the project. This approach became 'fashionable' from the late 1990's. Thus there were three parties: mine management, the project team and the contractor all working on one project, with the project team controlling the contractor. When the development footprint had opened up face room on strike, the mine became involved with mining and at the same time the contractor, managed by the project team, carried on with the downdip development. Under these circumstances experience has shown that this can prove suicidal. This approach was alien to KAR as his experience at Otjihase Copper Mine, Cooke 2 Shaft, REGM and H.J.Joel Gold Mine had shown the necessity for a clear line management structure with one competent person, with mining qualifications, in overall charge of the mine personnel and any mining contractors. At the Waterval Project, KAR was the technical advisor and consultant to the project but was not the manager, as had been the case at REGM and H.J.Joel Mine. Nonetheless, there was credit to be gained by the establishment of the Waterval Mine less than four years from the investigation by KAR into the availability of low profile trackless equipment: in that time Waterval had become the first fully mechanised trackless platinum mine in the BIC.

CHAPTER 7

Other Trackless Mechanisation Projects, Proposals and Trials

1998 - 2008

CHAPTER 7

Other Trackless Mechanisation Projects, Proposals and Trials: 1998 - 2008

Since K.A.Rhodes formed KAR Mining Consultant cc in 1995, KAR has been involved in the design and trackless mechanised mining consultancy work for numerous mines. In some cases KAR was the project manager or worked as part of project management and in others KAR acted as the consultant for these projects or trials. Notwithstanding that KAR's consultancy experience in the last twenty years has been associated predominately with mines employing various trackless mechanised mining methods, including wide orebody and massive mining operations, this chapter will focus primarily on KAR's work on narrow reefs in South African platinum mines up to 2003. This exposition will cover the work carried out for Amplats on their new projects, at that time, in the Rustenburg area and in addition certain trials with new mining techniques during the same period. Discussions will relate to the planning of the Styldrift Mine where it was intended to utilise high powered tunnel boring machines to access the orebody; the development of Boschfontein East and West Mines by the hybrid method with large capacity trucks for ore clearance to surface; trials with long hole stoping methods; a project to develop a reef raise with a tunnel boring machine at Bafokeng Rasimone Mine.

In addition, this chapter will refer to the work carried out later by KAR as the project manager of a gold mine in Ethiopia and also in the development, over several years, of a maintenance action plan to be used in the management of trackless operations in general.

7.1 Planning for the Styldrift Mine

In mid-1999 KAR was requested by Amplats to carry out the initial mining planning and design work for the Styldrift Project; the new mine would be contiguous with Amplats' Bafokeng Rasimone Mine, situated in close proximity to the Sun City Magaliesberg Complex in the North Western Province of South Africa.

It was intended that this planning work would lead eventually to a controlled budget estimate (CBE) for the project, following the completion of a series of preliminary cost estimates (PCE's).

7.1.1 ***Early Concepts***

At the start of this project a series of low accuracy PCE's were carried out, specifically in terms of mine access options. The original conceptual plan had been a conventional multi-level vertical shaft access to exploit the narrow 0,9 metre Merensky Reef. However, geological data suggested that the area where the transition from Rustenburg facies type reef to Zwartklip facies type reef occurred there was a 'broad band' of reef traversing the property located more or less in the middle of the farm, see **Figure 7.1**. This area was conservatively estimated to contain some 55–65 million tons of reef at not less than 1,5 metres to more then 2,0 metres wide (possibly 2,5 metres wide). In terms of this transitional change of facies and the overall flat dip of the reef a change of mining method, from a conventional method with excessive footwall development, to a trackless mechanised method on the reef horizon had to be considered by KAR. Notwithstanding that it would be necessary for further geological drilling, the shaft system design changed from a multi-level station layout to an effective single level station, in order to access the reef horizon by trackless methods. Preliminary cost estimates for this change showed an improvement in the viability of the project.

Following this decision it was then proposed for this trackless option that access to the mine should change from a vertical shaft to a decline layout. There were many advantages for a trackless mining method with a decline layout: direct access to the mine for vehicles without the need for stripping and re-assembly of equipment underground; improved easy access for supervisors and management; equipment readily removed to surface; direct access by vehicle for maintenance personnel when breakdowns occur; easy delivery of spares. In fact the PCE showed a lower capital cost and working costs, with a reduction in time for the build-up to full production. Various options were considered for developing the declines and in July 1999 KAR proposed, for the first time, to access the Styldrift Mine by tunnel boring machine (TBM). On 10 August 1999 a decision was made to proceed only with the TBM option for the CBE; see **Figure 7.2** for copy of an original Note for the Record. In terms of this decision, two declines were planned for: a 6,5 metre diameter tunnel equipped

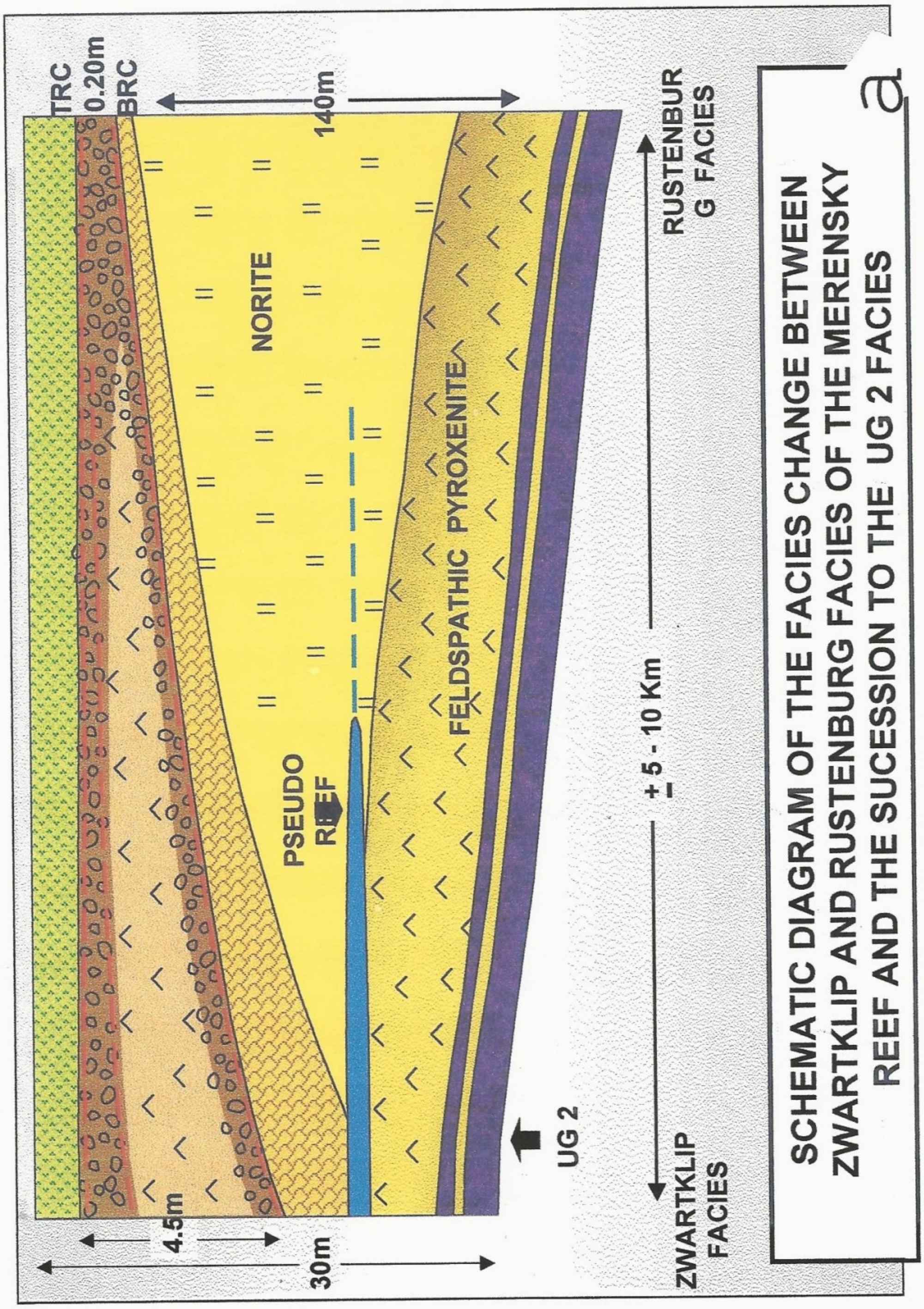


FIGURE 7.1

Facies Change between Zwartklip and Rustenburg Facies of the Merensky Reef

STYLDRIFT CBE

NOTE FOR THE RECORD

A BRIEF DISCUSSION WAS HELD ON TUESDAY, 10 AUGUST 1999

TO REVIEW THE OPTION OF TBM'S AT STYLDRIFT

PRESENT	:	B.R. Beamish	-	Amplats
		F.W. Horn	-	Amplats
		J.M. Halhead	-	Amplats
		A.J. Field	-	Amplats
		A van Jaarsveld	-	Amplats
		R.W. Hieber	-	Amplats
		G Chunnett	-	Amplats
		J.J.A. Botha	-	Amplats
		N.J. Townshend	-	TWP
		K.A. Rhodes	-	Mining Consultant
DISTRIBUTION	:	All above plus		
		L.C. Pretorius	-	Amplats
		J Dreyer	-	Amplats

1. K Rhodes explained the advantages of TBM's and the improved IRR and production profiles.

2. B Beamish asked what the disadvantages were. No major disadvantages have been identified.

3. A line of surface geological holes will be required along the TBM route. G Chunnett to provide cost estimate.

4. Conveyor TBM layout to be reviewed to accommodate vehicles as well. This will allow one way traffic.

5. J Botha recommended that primary crushing underground be reviewed. Hydraulic conveying should also be considered, but not for CBE purposes.

6. Conclusion

   It was decided to proceed only with the TBM CBE option. Shaft work to be discontinued and prepared for filing.

N.J. TOWNSHEND
PROJECT MANAGER

FIGURE 7.2

A Note for the Record: Decision to Proceed only with the TBM for the CBE

with a man-riding conveyor belt and with vehicle access and a second 5,0 metres diameter tunnel for vehicle access, thereby providing for separate up and down traffic. This concept proved to have an improved IRR for the project.

7.1.2 ***Mine Design***

Following the decision to opt for a trackless mine with access declines developed by TBM's, mine design commenced and was completed in October 1999 for CBE costing purposes. The initial basic mine design parameters were set out by KAR; refer to **Figure 7.3.** More detailed aspects of the mine design follow.

Geology

The broad band defining the transition zone on the Merensky Reef horizon, previously referred to, had been assumed to have an economic mining width of 200 cms, well in excess of the normal conventional mining best cut of 90 cms. With an estimated dip of the reef at 10°, the deposit would lend itself to trackless mechanised mining methods.

The resource had been estimated at about 78 million tons at 2,0 metres wide with a 23% geological loss and 15% overall loss for pillars. At that point in time it had been proposed to drill an additional ten holes to increase the confidence of the planning of the project. Nonetheless, the existing geological data justified the planning of a mechanised mine.

Production Parameters

All the previous PCE exercises had assumed a steady state production rate of 230000 tons per month and this was accepted for a CBE; this would envisage a life of 28 years for the mine.

Mine Access

A twin decline system would access the mine by means of TBM's at a dip of 11°. The conveyor decline was planned to be driven at 6,5 metres diameter for a length of 5480 metres. In addition to rock transport the conveyor would be planned for man-riding facilities at the main shift times. Service vehicles would be able to travel alongside the conveyor. The service decline, with a diameter of 5,0 metres for a length of 5473 metres, would provide

STYLDRIFT BASIC MINE DESIGN

The Styldrift Project is planned as a totally mechanised mining operation in a reef mining width of 2,0 metres. Access to the mine will be a twin decline system.

Mine Access
The two declines will be developed by tunnel boring machines (TBM's) at a declination of 11°.

Conveyor Decline
This decline will be bored with a diameter of 6,5 metres for a length of 5480 metres. In addition to the transport of reef out of the mine, the conveyor will provide for a manriding facility at main shift times.

Service Decline
This decline will have a diameter of 5,0 metres for a length of 5473 metres and will provide for vehicle access to the mine for material and equipment and also for personnel during the shifts. The availability of a service decline direct from surface to the underground workings has major advantages for any trackless mining operation.

The new generation of TBM's, known as 'high performance' machines are capable of rapid rates of advance and it is envisaged that these declines will be completed within a year of their commencement (approximately 450 metres advance in each decline per month).

General Mine Layout
All mine development will be carried out on the reef horizon with the exception of a main level footwall infrastructure which consists of a trackless equipment workshop, material handling facilities, pump station and reef transfer arrangements. Following completion of this footwall development, which will take place during the build-up to full production, no more waste will be sent out of the mine.

Mining Method
The method of mining selected for this project is mechanised room and pillar mining. Mining will take place in two stages. During primary mining on advance the percentage extraction will be 68%; in secondary operation on retreat pillars will be reduced in size and the final percentage extraction within panels will be 91%.

Panel mining will be cyclical. All drilling operations will be carried out with mobile drill rigs; face rigs and roofbolters. Broken ore will be trammed from the faces by LHD onto the tail end of panel conveyors and conveyed to underground silos by trunk conveyors.

Mine Planning
The total mining operation has been scheduled and the mine can be expected to be in full production at the rate of 230000 tons per month reef, four years after the decision is given to proceed with the project.

K.A.Rhodes: October 1999

FIGURE 7.3

vehicle access to the mine for materials and for personnel during the shift when man-riding on the conveyor would not be possible. It was conservatively expected that both the declines would be completed in one year at an average sinking rate of 450 metres/month. Although these advances may have seemed high, it had to be emphasised that the TBM technology was well proven, even in the most extreme conditions, and it was possible to place a high level of confidence in the decision to use TBM's which were of a new generation and known as 'high performance' machines.

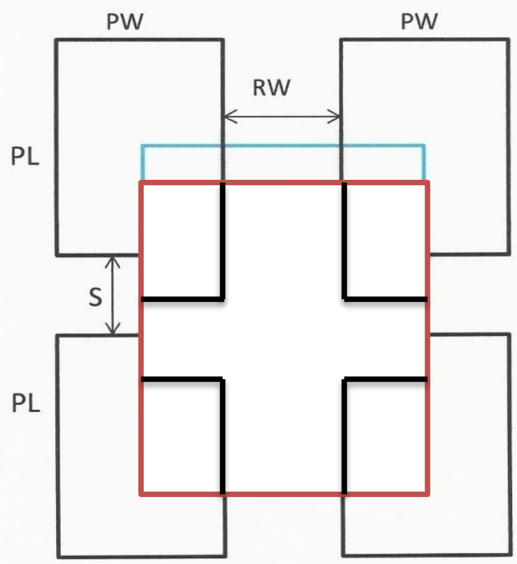
Rock Engineering Considerations

There were specific rock engineering parameters defined for this project which were based on the practical requirements of a room and pillar operation. Rock engineering modelling work had confirmed these recommendations.

In terms of the mining depth and the necessity to optimise extraction and in order to take full cognizance of safety requirements, it was recommended that barrier pillars should surround the mining sections. These were planned at 15 metres wide with mine panels of 215 metres. Mining operations would therefore be compartmentalized thereby eliminating the risk of an uncontrollable pillar run throughout the mine. Extraction would take place in two stages. A sequence of primary and secondary extraction had the distinct advantage that pillars developed during the primary operation on advance would be larger than final requirements and thus would represent a higher factor of safety.

During primary mining the room span was recommended to be 15 metres and pillars 15 metres x 10 metres with access holings 4 metres wide. During secondary extraction on retreat, pillars would be reduced to 4 metres x 10 metres, in effect crush pillars. Maximum extraction would be achieved during the secondary operation because it would take place on retreat. The percentage extraction during the primary mining would be only 68%, with final secondary extraction (after the secondary phase) being calculated at 89.5% (say) 90%. Refer to **Figure 7.4** and **Figure 7.5**.

Primary Extraction



Primary Extraction = $\frac{(PW + RW) \times (PL + S) - (PW \times PL)}{(PW + RW) \times (PL + S)}$

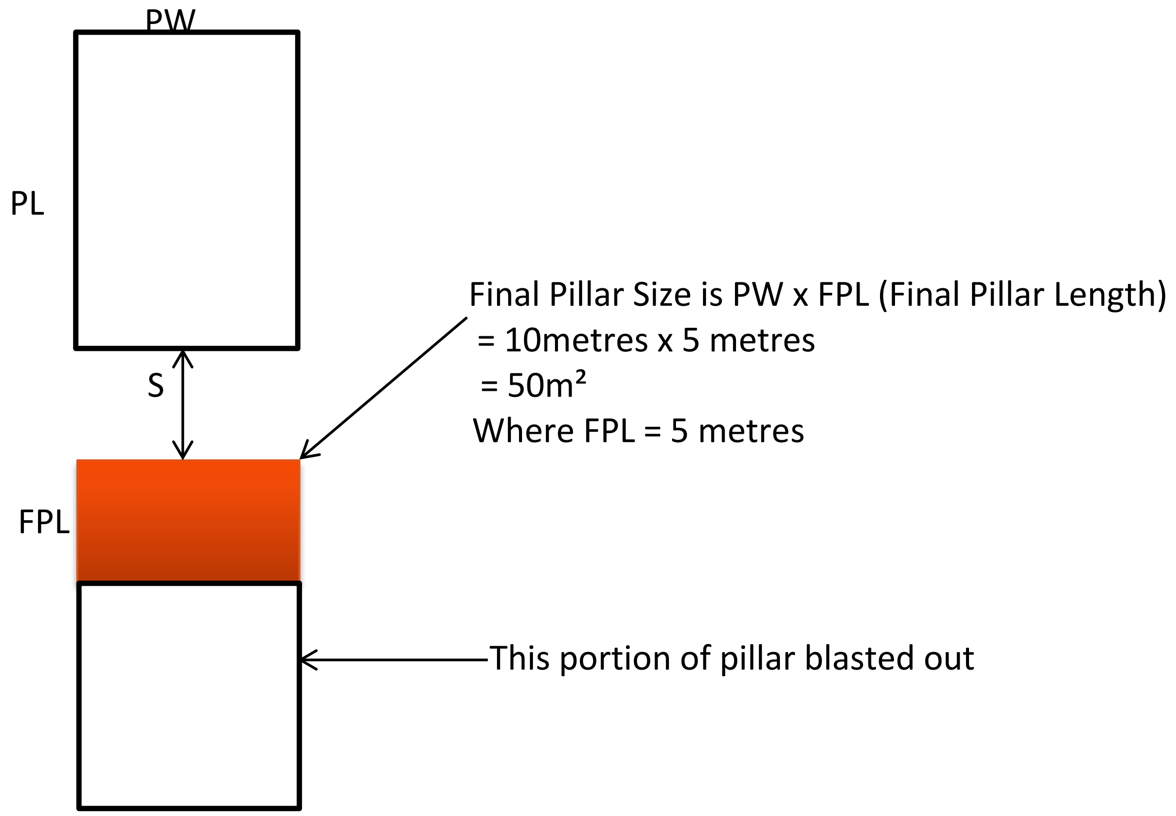
Where
Pillar width (PW) = 10 metres
Pillar length (PL) = 15 metres
Room width (RW) = 15 metres
Split (S) = 4 metres

% extraction = $\frac{(10 + 15) \times (15 + 4) - (10 \times 15)}{(10 + 15) \times (15 + 4)}$

= 68%

FIGURE 7.4

Secondary Extraction



Final Extraction (after secondary mining)

$$= \frac{(PW + RW) \times (PL + S) - 50}{(PW + RW) \times (PL + S)}$$

$$= \frac{(10 + 15) \times (15 + 4) - 50}{(10 + 15) \times (15 + 4)}$$

$$= 89{,}5\%$$

FIGURE 7.5

Ventilation Aspects

The use of a large fleet of trackless equipment underground at relative depth could be expected to necessitate a large quantity of ventilating air. It was also not acceptable to course air from one panel to the next adjoining panel and, therefore, dedicated return airways (RAW's) would have to be established from the outset.

The quantity of ventilation required to satisfy all the criteria for the project, including double shift multi-blasting stoping operations (fixed time blasting at the end of the shift), was calculated to be 750 kg/sec. In terms of this quantity, ten main intake roadways with eight RAW's would be required. In order to ensure acceptable quality of ventilation at the faces it would require that all intake air into a panel be forced along the flank roadways of a panel thereby causing high velocities of air in a restricted number of roadways. This would avoid any heat build-up which would inevitably occur from slow moving air passing over a larger number of roadways; this policy being mandatory due to the expected high virgin rock temperatures. Jet fans were to be used close to the working faces to ensure adequate face velocities. Refer to **Figure 7.6**.

It was expected that due to a high virgin rock temperature and the use of diesel driven equipment that 15mW of refrigeration from bulk air cooling would be necessary from the outset.

Provision was made for all the required health and safety matters: gas detection and monitoring, self-contained self-rescuers, refuge bays and fire suppression systems on all mobile equipment.

General Mine Layout

All development was planned to be on the reef horizon with the exception of any main level infrastructure, with all footwall waste development being completed during the build-up phase.

Footwall infrastructure would include workshops to provide for ongoing maintenance of all the equipment and all other ancillary requirements. Material storage bays would be provided for in the

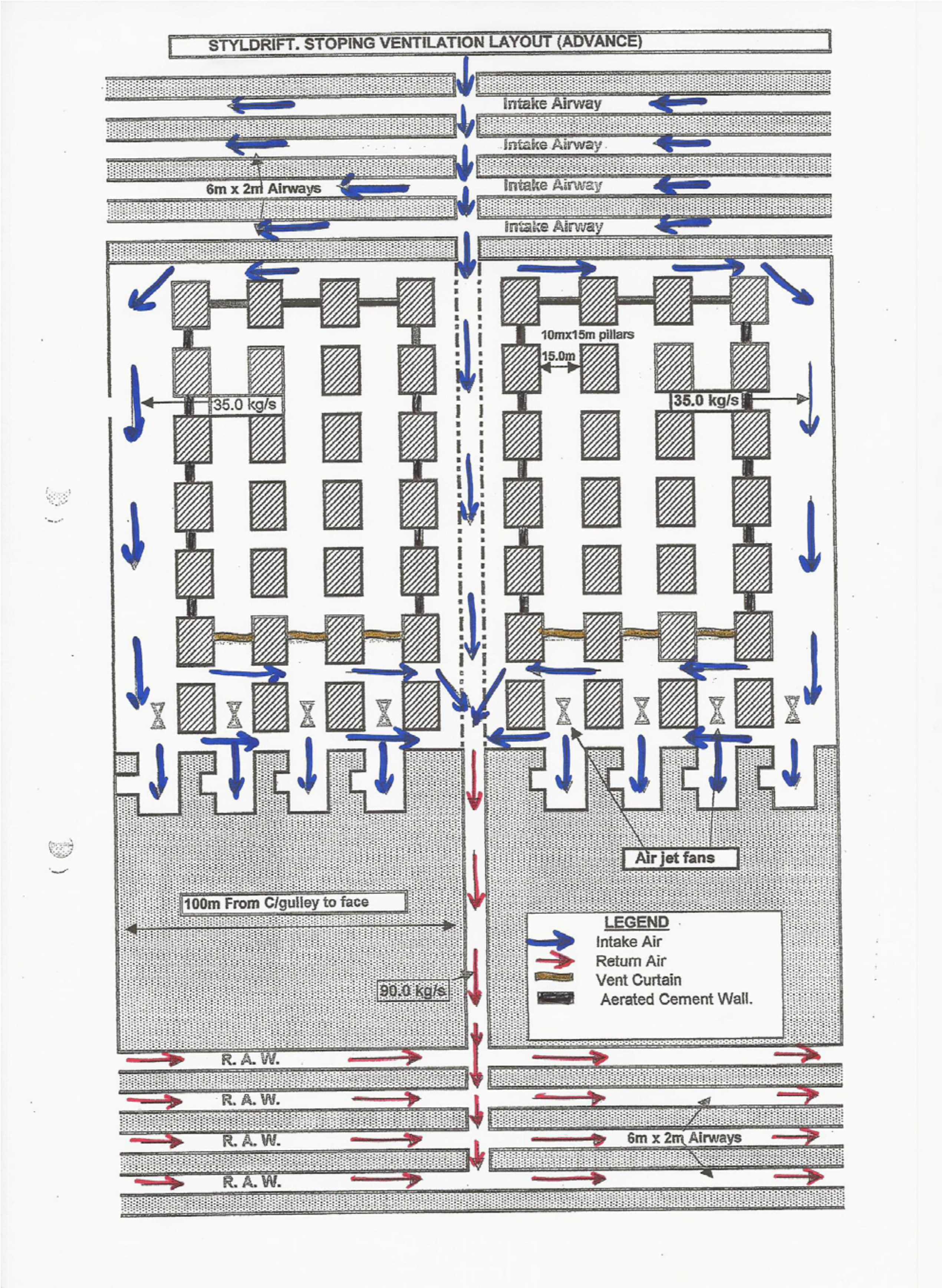


FIGURE 7.6

Stope Ventilation

underground layout, with the necessary service access roadways for utility vehicle transporters. Provision was made for vertical settlers and clear water dams and a pump station.

From a centrally situated orepass on the reef horizon, reef would be conveyed to three silos with a live storage capacity of about 5000 tons or 50% of the daily call. From loading stations below the silos, reef would be transferred by sacrificial conveyor to the main decline conveyor and then to surface.

Mining Layout

The main on reef development was to consist of 18 roadways on dip and strike (all 6 metres x 2 metres); 10 intake and 8 RAW's.

Panels would be developed off the strike development. Mining would take place downdip in a room and pillar layout with a mining width of 2,0 metres. A panel would consist of nine rooms (eight pillars).

Cycle of Operations

Operations were to be cyclical operating with a full suite of trackless equipment: face rigs, roof bolters, LHD's.

All face drilling was planned to be carried out by low profile electro-hydraulic single boom rigs. Length of round would be 3,5 metres with hole diameter of 41-43mm. Preliminary drilling patterns had been designed.

Blast designs were conservative as specific site conditions were not known; this was also because of the necessity to ensure good muckpile conditions for LHD loading and further, to eliminate serious bottlenecks at the panel tipping point due to large rocks. Ore fragmentation was therefore critical and blast designs were such that 50% of the broken rock was expected to be less than a range of 70 – 100mm with maximum lump size of 300mm. Notwithstanding the success of ANFO over many years in the mining industry, because of the focus on mechanisation and improved labour efficiencies it was proposed to use emulsion explosives. Some of the more important advantages of emulsion explosives are that emulsion is non-explosive and cannot be detonated until it is sensitised during the charging process, which means it can be transported as material stores; improved face advance was likely and it would be possible to achieve an advance of 0,5 metres greater than that of ANFO; charging cycles are markedly reduced; emulsion is water proof which was an advantage for down dip mining; because its density can be changed during the charging process there is increased flexibility in blast design, for example, perimeter holes could be charged at a lower density. All blast designs provided for the use of shock tube assemblies.

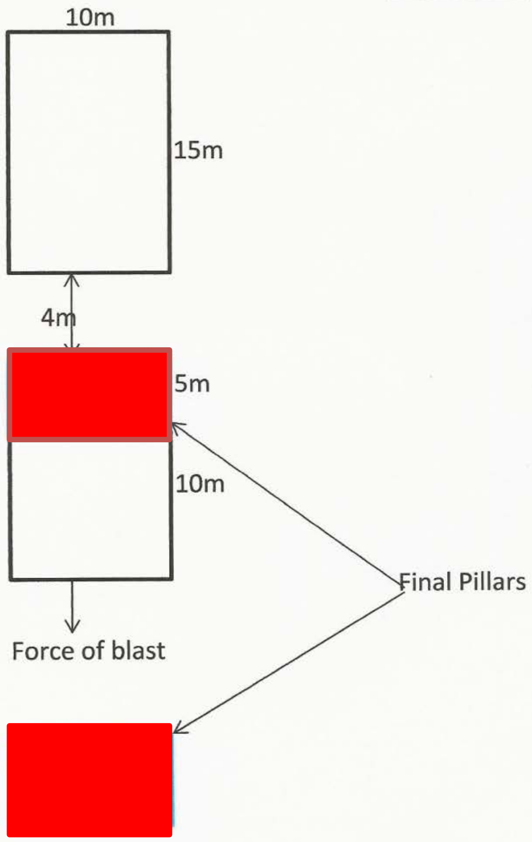
Cleaning would be carried out by low profile 3,3m³ LHD's with 6 ton carrying capacity. The LHD would tip onto the end of an advancing panel conveyor standing in the centre of a panel, with the average one way travel distance of the order of 75 metres. It can be stated at this point that this project was being planned at the same time KAR was the consultant for Waterval Mine where trucks were not being planned for; previously discussed in chapter 6. Therefore, trucks were not being considered for Styldrift at this stage of planning.

In panel mining support was designed to be on a 1,5 metre x 1,5 metre pattern with 1,5m long x 16mm diameter roofbolts. All roofbolt holes would be drilled by a roofbolter with manual installation.

Although no waste was planned to be mined from the cut there would be waste generated from the hanging wall, because of a necessity to blast for conveyor/material roadway crossings and air crossings, and also at tipping points; such waste would be packed in worked-out areas.

When secondary extraction occurred there would be certain changes to the cycle. Drilling of pillars would be carried out by a long hole production drill rig and blasted in a single blast (±620 tons). No support would be installed and cleaning of the blasted reef would take place with remote controlled LHD's. Refer to **Figure 7.7**.

Final Blast



Final Blast against downdip pillar to contain most of the rock (final cleaning by remote controlled LHD ie Torotel kit).

Tonnage blasted = 10m x 10m x 2m(S/W) x 3,1(SG) = 620 tons

FIGURE 7.7

Dilution
In terms of a mining cut of 2,0 metres without waste, theoretically there would be no planned for dilution. Waste would therefore only be generated as a result of bad mining practices.

Equipment
It must be remembered that immediately before the design, by KAR, of Waterval Mine had been completed, KAR had issued the report on the availability of low profile equipment for narrow width mining. It was therefore to be expected that, during the course of the detailed design stage of the Styldrift Project, equipment would be available for mining in a width of 2,0 metres (the mining cut at Waterval Mine was planned at 1,8 metres). It was also at this time that the visit by KAR and some of Amplats' senior managers to KGHM's mines in Poland took place.

The preliminary fleet of equipment had been determined by KAR and can be summarised below.

Type of Equipment	No of Units
LHD's (6 ton)	16
Face Drill Rigs	16
Roofbolters	10
Production Long Hole Rigs (conversions only)	(4)
Grader	1
Personnel Transporter	20
Material Transporter	12
Crane Truck Transporter	2
Decline Bulk Transporter	3
Total	80 + (4)

Commentary on this fleet is below.

LHD's
Notwithstanding all the previous arguments in these chapters regarding the choice of the largest units, taking cognizance of the mining width, there could be arguments for the use of an increased size of LHD in the main development where a 2,0 metre height had to be maintained; such a unit could have been 4,6m³

capacity. However, in stope panels a low profile LHD of 6 tons capacity was assumed. Ejector buckets would also be required for tipping on to the tail end grizzley of the conveyor without the necessity for hanging wall stripping.

In determining the LHD requirements in the stope panels, the following calculations were made.

Tons/hour: $P = (51 \times L) \div T + \left[\frac{2D}{S \times 16{,}67} \right]$

In this case the following assumptions were made:

Load, manoeuvre, dump:	T =	3 minutes
One way tram (conservative):	D =	100 metres
Average speed:	S =	6kms/hour
Carry capacity of LHD:	L =	6 tons

Therefore $P = 51 \times 6 \div 3 + \left[\frac{2 \times 100}{16{,}67 \times 6} \right]$

$= \underline{61 \text{ tons/hour}}$

If production of 230000 tons/month was split into 200000 tons from stope panels and 30000 from development sections and if the estimated working hours of the LHD was 280 hours/month, then from production panels the LHD requirements would be 200000 ÷ (280 x 61) = 11,7 (say 12).

From development, due to face availability restrictions, tons/hour trammed by LHD's would be less due to a lower utilization and we could therefore assume the tonnage would be (say) 10000 tons/month from one LHD.

Development LHD requirements are therefore $\frac{30000}{10000} = 3$

Therefore total LHD's = 12 + 3 = 15

For conservatism assume 16 LHD's.

Face Drill Rigs

In determining face drill rigs requirements the following parameters were used.

Drilling penetration rate = 1,5 metres/minute

Time between holes	=	40 seconds (0,67 minute)
Time between set-ups	=	5 minutes
Time to tram between faces	=	15 minutes
Hole length	=	3,2 metres

Calculation of the time from a typical 'total' room which includes the four metre heading in advance of the face with two shoulders at 5.5 metres and a 50% availability of a split. Headings and Shoulders have total number of drill holes at 87.

Therefore, time to drill face	=	$87 \times \frac{3,2}{1,5} + (87 \times 0,67)$
	=	244 minutes
Set-ups are 3 x 5	=	15 minutes
Move	=	15 minutes
Total	=	274 minutes

The split between pillars requires 33 holes.

Time to drill split	=	$\frac{33 \times 3,2}{1,5} + (33 \times 0,67)$
	=	93 minutes
Set up	=	5 minutes
Move	=	15 minutes
		113 minutes

Therefore total time	=	274 + 113 x 0,5
	=	330 minutes
For a total of 331 metres		
Metres/minute	=	1,0

In determining the tons/metre drilled, the following parameters were used:

Advance for 3,2 metre round	=	3,0 metres
SG	=	3,15

Therefore tons are calculated as follows

Tons (Headings)	=	4m wide x 2m highx3x3,15
	=	76 tons
Tons (Room shoulders)	=	5,5 metres/shoulder x 2 x 2 x 3 x 3,15
	=	208 tons
Split (same as heading) x 50%	=	38 tons

Total tons	=	<u>322 tons</u>
Tons/metre drilled/minute	=	322 ÷ 331
	=	<u>0,97</u>

In a nine room panel with eight splits, tons and metres drilled are as follows:

		<u>Tons</u>
Headings 9 x 76	=	684
Shoulders 9 x 208	=	1872
*Splits 8 x 76 ÷ 0,5	=	<u>304</u>
Total	=	<u>2860 tons</u>

*Splits are generally 50% of the available time

		<u>Metres Drilled</u>
Headings 9 x 33	=	297
Shoulders 9 x 54	=	486
*Splits 8 x 33 x 0,5	=	<u>132</u>
Total	=	<u>915 metres drilled</u>
	=	915 x 3m effective advance
	=	2745 metres
Tons/metre drilled	=	<u>1,04 tons/metre</u>

Assuming 0,97 metres drilled/minute, 1,04 tons/metre drilled, 280 hours in a month:

Tons generated for a rig	=	280 x 0,97 x 60 x 1,04
	=	16950 tons/month
say	=	16000tons/month
Rigs required 230000÷16000	=	14,3

or matching the suite of 1 LHD = 1 Rig, then rig requirements would be 16 same as for LHD.

Therefore the number of Face Rigs required was 16.

<u>Roofbolters</u>

The arguments for the availability of roofbolters was the same as for the Waterval Project

In determining the number of roofbolters required, the theoretical number of roofbolts to be installed in a month would be calculated as follows.

Area mined per month in m^2 = $\frac{230000}{2 \times 3,15}$

Where:

Monthly tonnage = 230000

Mining height = 2,0 metres

S.G. of reef = 3,15

Therefore:

Area mined/month = 36500m^2

At a roofbolt pattern of 1,5 metres x 1,5 metres the number of roofbolts/shift = 36500 ÷ (1,5 x 1,5)

= 16200/month

or = 345/shift

Therefore:

Number of roofbolters, assuming 42 bolts to be installed in a shift

= 8,2 say 9

However, there would be a necessity to install additional bolts for faults, slips or generally poor ground conditions.

Therefore number of roofbolters required was 10.

Trucks

It was not planned for the use of trucks at steady state production and in hindsight this was probably a mistake. However, during a period in the initial development of the footwall infrastructure it would have been required to employ large 50 ton capacity trucks to clear waste rock up the service decline before permanent transfer arrangements could be established.

Pumping

The mining was down dip room and pillar which meant that water would have to be pumped from the face when the drill rig was working. The philosophy was that every drill rig had to be allocated a portable electric pump which had to operate when drilling was taking place. Water from the panel would then be pumped to dams situated in the strike development and from there through boreholes and launders to the clarifier.

Logistics

In this respect logistics is intended to cover rock clearance and transport of materials and people.

Following the transfer of reef by LHD to the panel conveyor (or development to the main trunk conveyor), reef would be transported finally to surface: from the panel conveyor onto the strike conveyor and by trunk conveyor into a common orepass feeding the main decline conveyor. All conveyors were planned at 1050mm.

The service decline was to be the main arterial into the mine for materials; bulk transporters would carry materials, engineering spares, bulk chemicals for blasting operations and containers to the material bays on the main level and from this point, section transporters would carry all materials and equipment direct to any part of the mine. This concept, although simple in principle, required detailed planning input at the project management stage in order to fully streamline the operation. The total requirements for materials can be seen in **Figure 7.8**.

In the original design concept persons would be transported down and out of the mine by manriding conveyor; this would primarily operate for people at fixed shift times. At other times vehicles would operate in both declines for supervisory and service personnel. However, at a later stage when detailed planning was done this changed with the decision to develop a single TBM decline; in terms of this decision changes were made to the means of transporting persons underground and these will be discussed later in this chapter.

Engineering Maintenance

The main workshop complex was to be developed underground on the main footwall level and would provide for all aspects of trackless vehicle maintenance: planned and scheduled maintenance; major repairs and breakdowns; lubrication; fuel supply; welding bays; electrical workshop; tyre management; stores and offices.

It was always the policy to enter into a maintenance service agreement with the appointed OEM and, at that time, in discussions with one specific OEM the following six levels of performance type of contract were available and it is considered worthwhile to briefly define these options.

TABLE OF UNDERGROUND MATERIAL REQUIREMENTS

Material
Assume 20 days/month to be conservative

Material	Calculations	Daily Requirements
Explosives	Based on emulsion explosives; 4,64kg/hole (Rorke Report) and the estimated 65060 holes/month, monthly usage is 65060 x 64 ÷1000 tons = 301 tons/month say 15 tons/day **Note: these are chemicals and are not subject to any legal requirements in terms of explosives.**	15 tons + accessories
Roofbolts	Calculated in costs estimates at 18871/month or say 950/day	950 roofbolts
Conveyor Material	34 metres advance/month in sections and 30 metres/month in main intake development and main returns. Therefore total conveyor advances are (34 x 6) + 30(2+2) = 324 or 17m/day say 20 metres/day	20 metres of conveying material, structure + belting (x2)
Pipes	350 metres/month; air, water, pump columns = 18 metres/day say 20 metres/day	20 metres (x3)
Ventilation	Vent. Columns exhaust 1015mm: 150 metres/month in main Development or 7,5 metres/day. 3-4 lengths/day. Fans/Jet Fans : estimated at 20% of fans under repair equals 1/day say 2/day up and down therefore 2 x 2 = 4 movements/day	3 – 4 vent pipes 4 fans
Cables	120 metres/month main HT cable plus 200 metres/month LT cable Say 350 metres/month or 20 metres/day of cable. 1 – 2 drums/month to go underground.	1 – 2 drums cable **per month**
Engineering Spares	Trackless spares every day to underground workshop	
Small Mining Consumables	2 containers/day to underground loading bays	

FIGURE 7.8

***Spare Parts Contract**: t*he contract secures availability, delivery and cost of spare parts to keep the equipment running.
***Component Exchange Contract**: c*ritical spare components are stocked at site and charged as required.
***Preventive Maintenance Contract**:* the OEM makes scheduled site visits based on equipment usage and supplies the service parts and carries out periodic maintenance according to fixed charges.
Cost and Availability Contract: The OEM would provide guarantees on operating costs and mechanical availability of the equipment; this would include cost per hour, cost per drilled metre etc. The OEM would supply full supervision and maintenance personnel.
Cost and Productivity Contract: this contract would, in addition to the cost and availability contract, provide for productivity guarantees of the purchased equipment.
Performance Agreement Contract: in this contract the OEM would provide the equipment including guarantees related to the costs and productivity with financial rewards or penalties which have been agreed to by both parties. Equipment is up-graded, re-built or replaced at the OEM's discretion.

The above contracts provide for increasing responsibilities and commitments from the OEM and the choice of level of contract needs to be determined by the mine company before delivery of equipment.

For the estimation of working costs for the Styldrift CBE it was assumed that a cost and availability contract would be agreed to and the estimated costs per hour of the major capital equipment had been provided by the OEM.

Fuel would be sent, in measured bulk quantities, from a surface bulk storage tank to the underground workshop fuel station by pipeline.

Trackless Management
At this time KAR was giving consideration to a specific objective action plan (referred to in the previous chapter as a SOAP), for the management of a trackless mining operation; the action plan to incorporate all the factors related to the availability of the

equipment (which is the responsibility of the underground maintenance discipline) and the use of the availability of the equipment (this being the responsibility of the mining production discipline).

Schedules

The development of the mine was completely scheduled following the planned completion of the TBM declines through to steady state production and the key dates, based on the assumption that a decision to proceed with the project was given before the end of 1999, were as follows.

Date	Action
January 2000	Start project management and design, including the refurbishment of the TBM's
October 2000	TBM's start boring
November 2001	TBM's withdrawn from mine
January 2002	Start underground development work
October 2002	Ventilation and rock transfer arrangements complete
November 2002	Commence on reef development
January 2004	Full production rate of 230000 TPM reef achieved

Labour

Preliminary planned labour complements for the underground mine were estimated as follows.

Mining (production)		216
Mining (other)		83
Logistics		182
Supervision		19
Services		160
	Total	660

Therefore, tons (reef) per underground employee was say 350.

Working Costs

Operating costs were estimated and for this exercise it was assumed that an OEM maintenance service agreement would be agreed to. A breakdown of costs is below.

	Rands/ton Milled
Mining (in panel stoping and development)	44,38
Logistics (transport and conveying)	6,88
Supervision	1,74
Services	6,49
CARA (Abnormal Capital Allowance)	4,35
Power	9,00
Refrigeration	8,00
Total	80,84

The above figures were submitted in September 1999 for CBE purposes but at the beginning of 2000 certain adjustments had to be made; these costs were finalised in terms of the following.

September 1999

CBE costs estimated by KAR at R80,84/ton.

January 2000

Following discussions with the Amplats Finance Division the September 1999 costs were escalated to R85,05/ton.

June 2000

Amplats Planning Department considered the estimated costs could be grossly underestimated; R120/ton was stated as possible costs, an increase of 40%. Using the Anglo American Technical Services costing model, the audited costs arrived at were R88,23 in January 2000 terms; this a mere 3,7% greater than those costs set out by KAR for the same date. See **Figure 7.9** for copy of an original memorandum on the subject.

July 2000

In July 2000 terms the R85,05/ton (R88,23/ton) was escalated to R91,61/ton.

9-JUL.'00(WED) 16:15 AMPLATS GEOLOGY DEPT FAX:+27 11 3735107 P.001

MEMORANDUM

To : **Mr. Len Pretorius**

From : **A.J.Raubenheimer & J.A.Wood**

Date : **20th June, 2000**

Subject : **Shaft Head Cost Estimate for the proposed 230Ktpm Styldrift Mine.**

Following concerns raised by the Business Development & Planning Department regarding the accuracy and lack of supporting documentation for the Shaft Head Working Cost Estimate for Styldrift, all operating costs were re-examined with the use of the A.A.T.S. Costing Model.

The original model was substantially debugged and upgraded to incorporate a much larger set of input parameters and calculations. Much of the data made available from the various sources were interrogated, verified and agreed to. Numerous over- and underestimations were found and corrected. Clarification meetings were also held with Mr. Ken Rhodes, the mining consultant responsible for the mining design.

On the basis of the best available information, the unit shaft head cost calculated comes to **R88,23 / ton milled** in January 2000 money terms. It therefore appears that this cost determination is only some R4,00/ton higher than the original (Sept. 99) estimate. i.e. within the 10% C.B.E. specification.

It is therefore jointly recommended by the writers that this Shaft Head Cost be used in further DCF analyses of the proposed mine.

A.J.Raubenheimer **J.A.Wood**

FIGURE 7.9

Memorandum on Shaft Head Cost Estimates

The new Shaft Head Costs Summary was then as follows.

		R/ton Milled
Mining Production		51,99
Logistics		7,97
Supervision		1,92
Services		7,48
CARA		5,43
Power		11,83
Refrigeration		4,99
	Total	91,61

For a further breakdown of these costs refer to **Annexure 7.1** in **Volume 4** titled: ***Breakdown of Working Costs for the Styldrift Project in July 2000 Terms***.

The above would probably have risen to R100/ton by March 2001. However, it would then have been necessary to re-interrogate the costs from a zero base, specifically in terms of exact wage scales and those costs affected by the devaluation of the rand in 2000 and its effect on equipment imported from outside the country.

7.1.3 **Change of Concept in Terms of Access by TBM**

In terms of a CBE optimisation exercise an alternative option for a single larger diameter decline to be developed by TBM was considered against the decision to bore two declines.

Initially it was thought the diameter of the single decline would need to be 9,1 metres and would have to provide for a conveyor for rock clearance and manriding (at shift times), material transport, access for trackless machines and off shift personnel transport. Further, with only a single decline it would be necessary to establish a second outlet; the downcast shaft for bulk air cooling would have to be equipped.

There were to be changes made to the mine access concept, based on a single TBM decline. This exercise carried out by KAR, showed that there were perceived technical risks to manriding on a long decline conveyor and a change was made.

Manriding

The single decline exercise had highlighted the need to review the transport of persons by manriding on the decline conveyor. Manriding on a conveyor 5500 metres long represented an excessive travelling time in and out of the mine; at the maximum legal belt speed of 2,5 metres/second one way travelling time would be 36 minutes. In order to reduce the time to an acceptable period it would require the belt speed to almost double to 4 even 5 metres/second. However travelling at this speed would introduce risks. Firstly, alighting at these speeds could be considered dangerous. Although on a visit to German coal mines KAR had experience of belt speeds of 3,5 metres/second planned for, an adjacent decelerating conveyor belt had to be installed at the alighting platforms to enable persons to get onto before finally stepping off onto the stationary platform. Secondly, high speed travelling allows very little time for a person to adjust from a travelling position on the conveyor to an alighting position, at which time that person must be standing.

These risks caused KAR to consider an alternative method for the transport of people.

Alternative Means of Transport for Persons

The most obvious way for persons to access the mine would be by vehicle down the decline. In this way persons working in the same area or working together, such as panel crews, could be allocated their own vehicle and drive directly to their working place, thus ensuring that the whole crew arrives together. Other more general workers could be transported in multiple carrier vehicles which would then be available during the shift for materials, as in a cassette system. These arguments were considered to be the most practical for the project. Nevertheless, this would require an increase in the number of personnel vehicles from 20 to 50.

Rock Clearance

The conveyor would carry reef out of the mine, as in the CBE, with no manriding facility. The new proposal would now provide for the conveyor to be installed in the crown of the decline thus enabling the full width of the circular decline to be used for passing traffic, thereby avoiding bottlenecks in a long decline.

However with the conveyor slung in the crown and situated above moving vehicles travelling below on the invert, in order to avoid any large rocks falling off the conveyor and causing possible injuries to persons, it was proposed to establish a crushing station at the bottom of the mine.

In order to finalise the diameter of the single decline a more definitive exercise had to be carried out; see the arguments briefly stated in **Figure 7.10** and refer to **Figure 7.11** for a cross section of the single tunnel in its final condition.

Second Outlet Provision

The CBE had provided for a twin decline system with two means of egress. A single decline arrangement would then necessitate planning for a second outlet and this would be provided for in the downcast ventilation shaft. It was proposed that a safe and effective hoist such as a rack and pinion Alimak system would meet this requirement. The use of the system would obviously be very infrequent and there would therefore be no need for a sub bank for the flow of bulk cooled air from the refrigeration plant.

Material Transport

Bulk carriers would transport material and emulsion explosives down the decline.

Ventilation Requirements

There would be no change to the ventilation planning. Intake air down the decline would be discharged directly into the upcast shaft after ventilating the footwall development complex, requiring only 100kg/second at an air speed of only 3 metres/second in the single decline; refer to **Figure 7.12** for a Note for the Record issued by KAR.

Production Build-up

The primary motivation for the development of the mine by means of TBM’s was an accelerated build-up to full production

6.1 SINGLE TBM

For: Lower Capex US$18,7m (-15,6m).
Walkway for people away from traffic.

1

Against: One way traffic.

- Bottlenecking.
- Risk of blockage in decline.
- Possible delays in and out of mine in case of emergency.

Programme delayed by up to 2,5 months.

7,0 SINGLE TBM

For: Lower capex US$23,9m (-10,4m).
Conveyor on side of tunnel.

2

Against: One way traffic.

- Bottlenecking.
- Risk of blockage in decline.
- Possible delays in and out of mine in case of emergency.

Possible restrictions to people walking.

Programme delayed by up to 2,5 months.

7,62 SINGLE TBM

For: Lower capex US$24,6m (-9,7m).

3

Two way traffic.

- Bottlenecking eliminated.
- Minimal risk to total blockage in decline.
- Access to mine by vehicle virtually assured at all times.

Walkway for people away from traffic.

Against: Programme delayed up to 3,5 months.

FIGURE 7.10

Arguments to Finalise the Diameter of the Single TBM Decline

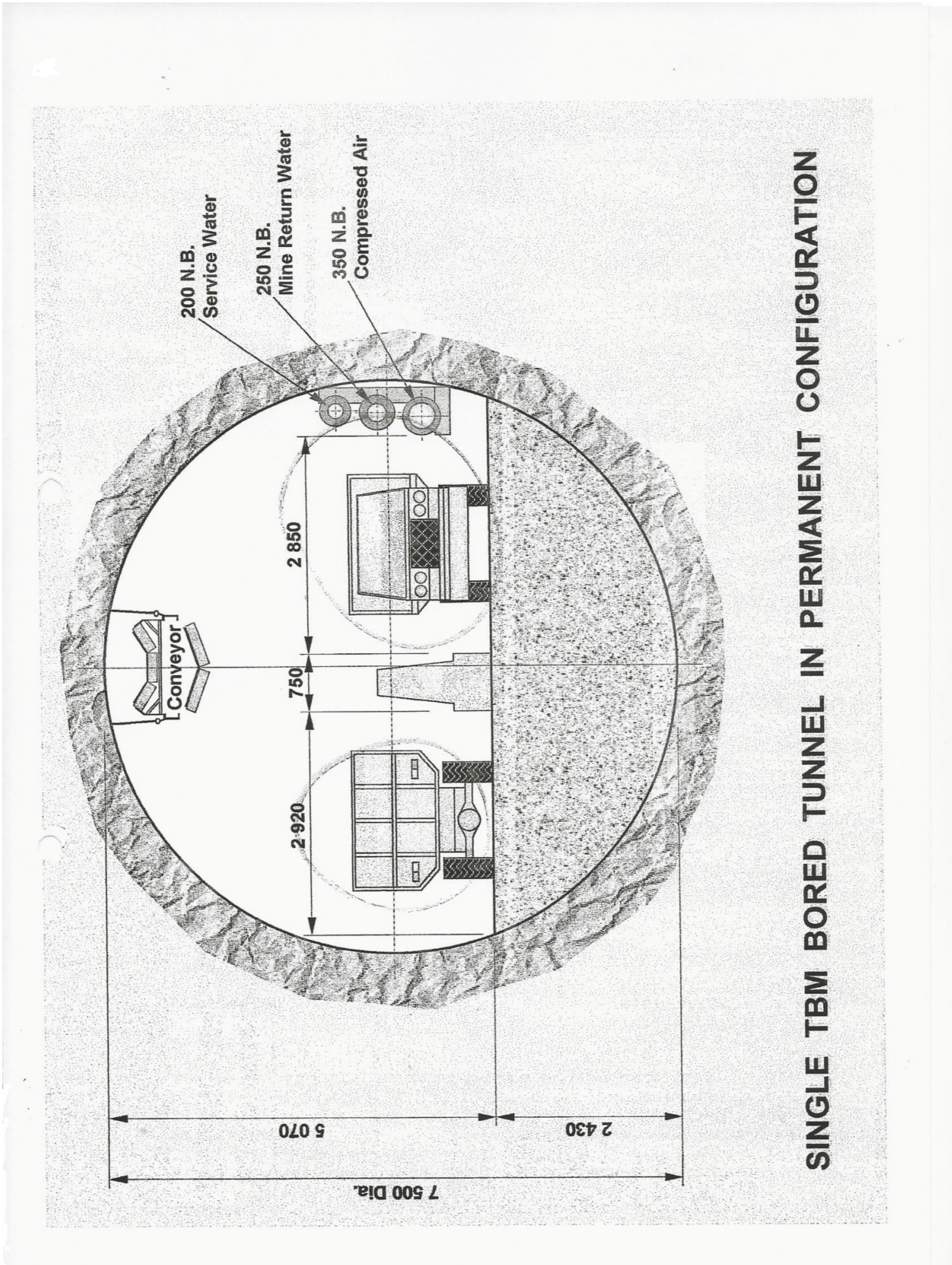


FIGURE 7.11

Cross Section of Single TBM Decline in Final Condition

STYLDRIFT PROJECT

NOTE FOR THE RECORD

With reference to the Styldrift Ventilation and Cooling Infrastructure Design Criteria document, it is stated that the dedicated downcast shaft will supply the cooled air (650kg/s) for the production area while the ancillary ventilation requirements (100kg/s) for the footwall infrastructure, which includes material bays, workshops, reef transfer arrangements and pump station will be supplied via the two TBM shafts.

It has however now been decided that access to the mine will be through a single decline of 7,62m diameter. The open area of the decline, after construction of the roadbed, is calculated at 32,6m² and therefore the air speed will be 3,0 m/s and the maximum velocity over the belt (belt speed 2,5 m/s) is therefore 5,5 m/s which in terms of ventilation planning parameters is acceptable.

This matter has been discussed with the Amplats Ventilation Consultant.

03/08/00

FIGURE 7.12

Note for the Record on Ventilation Design Criteria issued by KAR

and, therefore, it was imperative that this advantage was not lost. More detailed planning suggested that there could be an increase in the time to ramp up to steady state production of about two months. However, some advantage could be gained if the conveyor could be carried close behind the TBM and waste trucking to surface might then be eliminated; in the CBE, trucking of waste from development would take place in the Service Decline while the Main Decline was being equipped. With this method there would be no loss of production build-up time.

Final Proposal

The single access design factors can be summarised: underground crushing of reef; direct transport of persons underground by 'land cruiser' type vehicles; transport of materials by bulk carrier to underground storage bays; bulk transport of emulsion explosives (chemicals) to underground silos; fuel pipeline from surface to underground bulk storage tanks; second outlet provision at the downcast ventilation shaft by means of Alimak hoist; no change to CBE ventilation planning.

7.1.4 *AATS Review*

In July 2000 a technical audit was carried out by Anglo American Technical Services (AATS) on the Styldrift Feasibility Study and CBE. In terms of mining, it supported the use of a TBM to access the workings by driving declines and not sink vertical shafts. It also approved the selection of a mechanised room and pillar method of mining.

In conclusion it found no technical issues which could jeopardise the technical success of the project. However, the review did suggest that more geological information would improve the mine design and planning. Some comments on issues raised at these audit meetings and on the responses by KAR to a memorandum from the Technical Director of Anglo American Corporation are seen in **Annexure 7.2** in **Volume 4**.

7.1.5 ***TBM Project***

In February 2000, prior to the change of concept for access design, a technical visit was made by KAR and other responsible engineers to the Lesotho Highlands Water Project. The objective was to see a TBM operation and have discussions with TBM engineers.

Visit to Mohale Tunnel

The Mohale Tunnel was a joint venture by Concor/Hochtief and was being developed from both ends by separate TBM's; total length would be 32 kms. There were certain matters discussed during the visit which related directly to the Styldrift Project; a few of the more important issues were as follows. Firstly, there was an immediate need to carry out surface drilling along the line of the TBM route in order to gain detailed information for geological and rock classification; refer to **Annexure 7.3 Volume 4**, for a general description of the geology along the TBM route. Secondly, it was recommended that during the refurbishment, by the appointed contractor, of the selected TBM the client should monitor the work; this conclusion follows from experience at Mohale when problems occurred with the cutterhead and main thrust bearing of the TBM. Thirdly, the matter of heat build-up during TBM tunnelling, specifically with high-powered machines, needed to be further understood and it would therefore be necessary for the Group Ventilation Consultant to make a 'hands-on' visit to a typical site. Finally, it was recommended that the enquiry document should only be issued after the pre-qualification discussions had taken place. Refer to **Figure 7.13** for a view of the Mohale TBM and also refer to **Annexure 7.4** in **Volume 4** for ***Notes on a Visit to Lesotho Highlands Water Project, 17-19 February 2000***; notes prepared by K.A.Rhodes.

Pre-Qualification of Tenderers

The Pre-Qualification documentation was issued on 13 July 2000 and was received back on 27 July 2000, with a short list prepared by 10 August 2000. On 26 September 2000 it was decided to enter into negotiations for a contract, with a JV known as the Hochtief/Concor/Statkraft/Cementation Joint Venture, to become known as Platun JV. In early 2001 Platun JV proposed that a selection process for a TBM be carried out, which would include visits to TBM's in Europe and the USA. Five TBM's were evaluated but the preferred machine was an Atlas Copco Robins MK27 – 3360 which was practically new and therefore the risks of material fatigue were unlikely. It is common with TBM boring that a unit is

Mohale Tunnel
Tunnel Boring Machine

FIGURE 7.13

identified somewhere in the world and is then re-furbished to the requirements of any new site. In this case the TBM would have to be fitted with a new cutterhead and also, due to the decline being bored at 11°, a new back-up system would have to be designed and manufactured. The re-furbishment time would be 40 weeks. In order to secure long lease equipment times it was necessary to reach consensus on contract terms and prices. The project duration, from placement of order for the TBM until completion of the work, was predicted to be 25,6 months.

Shortly following the visits overseas to select a TBM, a detailed technical document was prepared by Platun JV which included details of the back-up system and a risk analysis report.

Technical Proposal by Platun JV

Briefly, the length of the tunnel would be 5500 metres with a tunnel diameter of 7,62 metres, the vertical curve radius of 2000 metres and the inclination of the tunnel 0,3° - 11°. Excavation would be by the open hard rock TBM method. Tunnel cross sections during sinking and permanent condition are shown in **Figure 7.14** and **7.14A** respectively, both taken from Platun JV's Report of April 2001. Although the rock to be bored could be classified as very hard to extremely hard, vertical and sub-vertical jointing of the rock and their orientation in relation to the line of advance was expected to be favourable for the TBM penetration rate.

The TBM would be without a shield; a shielded machine would hinder rock support close behind the face. Installed power of the machine was 2500kW. The unit would be equipped with a probe drilling rig, grouting equipment, roofbolt drill rig, shotcrete platform, steel arch erector and pumping facilities.

The back-up system would provide for fire detection and suppression system, chilling system to cool air on the TBM, rescue capsule, conveyor extension to accommodate the permanent belt conveyor behind the TBM and gas measuring instrumentation with automatic shutdown of the TBM.

STYLDRIFT DECLINE PROJECT

PlaTun's Report
April 2001

1. INTRODUCTION

1.1 General

The tunnel is to be bored in the North West Province of South Africa near Rustenburg, in the Bushveld Igneous Complex as the Styldrift decline shaft for a new platinum mine development adjacent to BRPM. The tunnel will provide access to the ore body and to the underground infrastructure.

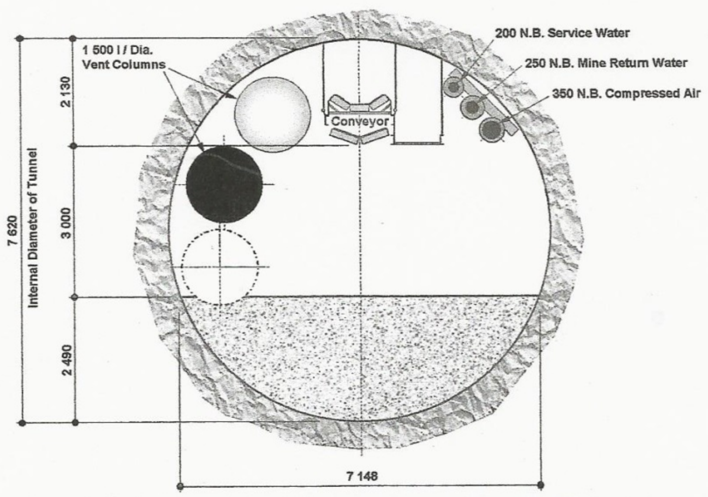
Brief Project Description:

Tunnel Data: Internal Diameter	7,62	m
Total Length of Tunnel	5,500	m
Tunnel Vertical Curve Radius	2,000	m
Tunnel Inclination	0.3 – 11.0	degrees

Excavation Method:

The excavation method foreseen is by open hard rock TBM method.
Rock support will be installed to the Employer's requirement.

Typical tunnel cross section including backfilled roadway - see diagrams below.

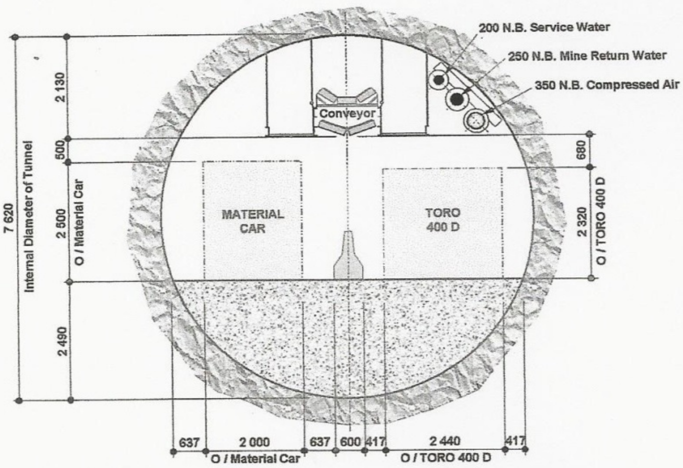


CROSS SECTION THRO. DECLINE - 7,6m Dia. SINGLE TUNNEL SINKING CONDITION

6 April 2001

FIGURE 7.14

Cross Section of Decline in Sinking Conditions



CROSS SECTION THRO. DECLINE - 7,6m Dia. SINGLE TUNNEL PERMANENT CONDITION

The Employer:

ANGLO PLATINUM MANAGEMENT SERVICES (PROPRIETARY) LIMITED, acting as Agent for and on behalf of: RUSTENBURG PLATINUM MINES LIMITED BAFOKENG RASIMONE PLATINUM MINE

The Employer's Consultants:

TOWNSHEND VAN DER WALT & PARTNERS (TWP)

1.2 Prequalification and Anglo Platinum's Order placed with PlaTun for Professional Consulting Services

TWP, on behalf of the Employer, called for prequalification in July 2000.

PlaTun Joint Venture consisting of:

HOCHTIEF AG	Leader
CEMENTATION MINING	RSA
CONCOR HOLDINGS (PTY) LTD	RSA
NCC TUNNELLING, Norway	Previously Statkraft Anlegg, Norway

submitted their prequalification document on 27th July 2000.

As a follow-up to the PQ, PlaTun submitted a proposal to carry out certain preliminary investigations with the aim of shortening the lead time for a contract of this nature.

FIGURE 7.14A

Cross Section of Decline in Permanent Condition

In terms of predicted performance the average long term weekly production time would be 5,79 days: boring would take place Monday to Saturday with maintenance and probe drilling taking place on Sunday. The penetration rate would be in the range of 1,2 to 2,0 metres/hour with a maximum possible rate of 4,0 metres/hour. Long term penetration rate was expected to be 1,7 metres/hour at 39% production per day. Average advance was predicted at 11 to 18 metres/day with an average of 92,1 metres/week. The project duration was planned for 111 weeks (25,6 months); 40 weeks for TBM refurbishment, manufacturing and assembly on site with 69 weeks boring and 2 weeks for tunnel finishing. It was also planned to leave the TBM underground on completion of the tunnel but the back-up system would be removed.

TBM Project Progress from 2001

In May 2001 continuous discussions took place between the Amplats' Project Team and Platun JV which culminated in meetings in London in May-June 2001. These meetings had the main objective of coming to an agreement on a contract at an acceptable cost and to have a clear understanding of the type of contract which was being proposed: the New Engineering Contract (NEC). By July 2001 a draft contract document had been compiled with a detailed Schedule of Responsibilities.

However, in 2001 Bafokeng Rasimone Mine (BRPM) became a part of the project and by late 2001 the Royal Bafokeng Nation became involved for the first time with their technical representatives, the Minerals Corporation. Following the intervention by the Royal Bafokeng Nation KAR became less involved with the Styldrift Project.

In April 2002 a risk assessment was carried out over two days by Snowden Consultants. In the opinion of KAR (who was a participant in the workshop) there were two relevant issues in the final report from the workshop. Firstly, there existed the risk of not getting the shift underground down the decline in an acceptable time frame if the mining method had to be changed from trackless mining to conventional mining: a lack of flexibility for the decline system transport arrangements for any marked increase in labour complements. It was correct to say (by KAR) that the decline access was motivated by the choice of the mining method: the advantages of a decline from surface for a trackless mining method have been set out many times before. The downcast shaft was to be equipped only for an emergency, in terms of the need for a second outlet. Secondly, there was a recommendation to carry out a further risk assessment of the TBM method of accessing the mine in terms of time and cost. KAR was not aware of this particular exercise ever being done.

By the end of that year, December 2002, with KAR no longer the lead consultant, a new option had been proposed and accepted which still retained a TBM for access (without a conveyor) and vertical shafts (East and West Mines) for men, materials and rock hoisting. Also a new enquiry for tunnel boring was then issued but tenderers were also invited to tender alternatives to a TBM.

After an intervening period of several years, it appears from media publications, that the final feasibility study on Styldrift was approved in September 2008. Access will be by a twin vertical shaft system to a depth of 740 metres. The method of mining will be both mechanised and conventional. Production will still be 230 000 tons/month and is planned to commence in 2015, with steady state in 2018.

7.1.5 ***Postscript to Styldrift***

KAR began the work of the design of the Styldrift Mine in mid-1999 and continued for nearly three years. The project was reviewed by Anglo American Corporation and the conclusion in their report published in August 2000 stated “No major technical issues were found that would jeopardise the technical success of the project”. Nevertheless, by the end of 2001 things changed dramatically with the intervention of the Royal Bafokeng Nation’s technical advisors.

If the project had gone ahead as originally planned Styldrift would have been the first trackless underground mine in South Africa developed from surface by TBM.

7.2 Boschfontein East and West Mines

In late 2001 KAR had completed a design for the Boschfontein East and West Mines at Amplats' Rustenburg Section, based on the room and pillar method. This work was being carried out at the same time that the Waterval Mine was being designed and developed; this has been discussed in chapter 6 of this treatise. However, a coarse comparative exercise was carried out by KAR on an alternative design for the Boshfontein Mines which assumed a hybrid method of mining. In terms of this exercise the hybrid design was considered to be marginally more viable and consequently a new controlled budget estimate (CBE) was prepared which was based on the hybrid layout. The main factors which influenced this decision to prepare the new CBE and change to a hybrid concept were an accelerated build-up to steady state production and a significant improvement in head grade.

7.2.1 *Hybrid Philosophy*

At this point in time it would be appropriate to discuss hybrid philosophy. The hybrid concept (although not named as such) was first introduced to South African mines by K.A.Rhodes (KAR) at the Randfontein Estates Gold Mine in 1984; refer to chapter 4 for a full description of how this evolved. In 1984 the geology of the reef on 95 Level at Cooke 2 Shaft (REGM), in relation to the footwall development grid as it was, had suggested to KAR that a rapid access onto the reef horizon with a 'super' gathering haulage on a deeper level would be favourable for the introduction of trackless mechanised mining to the narrow flat dipping reef. Following the successful introduction of this type of mining at REGM, KAR designed, managed and commissioned the H.J.Joel Gold Mine based on the same method although the geology of the reef there would normally have been associated with conventional multi-level footwall development; H.J.Joel Gold Mine has been fully described in chapter 5 of this exposition.

Since then hybrid mining has been introduced into certain other operations in South Africa but has not generally been accepted in the mining industry. In simple terms, the hybrid design juxtaposes trackless mining methods (for main development, cleaning and ore clearance operations) with a conventional labour intensive narrow reef stoping layout. The retention of conventional stoping in the design cannot be considered ideal as it does not provide for the elimination of hand-held rock drills. However, in the mid 1980’s it was always considered, and expected, that a mechanised drill rig would be developed and be able to take over from hand-held rock drills; refer to the discussion of the development of the Stomec Rig in chapter 4.

A further aspect of hybrid mining is the use of trucks, and this is now discussed for narrow reef mining specifically at the Boschfontein East and West Mines, which were designed by KAR.

7.2.2 ***Ore Clearance by Truck***

In terms of any ore clearance system for a hybrid design there is several options: LHD into trucks, LHD onto conveyors or even LHD cleaning into orepasses with direct loading into trucks. At the H.J.Joel Gold Mine LHD’s loaded into trucks which transported the ore back to the vertical shaft for hoisting. At the Boschfontein East and West Mines the hybrid design, set out by KAR, provided for trucks to travel direct to surface by means of the developed ramp system.

However, at that time (2002) conventional wisdom at Amplats did not accept the concept of direct tramming to surface by truck for a new shallow mine: ore had to be conveyed to surface. Notwithstanding, it was agreed with KAR that a conveyor installed to surface could be served by a single tipping point underground for trucks which would move deeper into the workings every few years as tramming distances increased. In this concept the mine would be opened up with trucks tramming direct to surface until it was the right time to establish a conveyor.

In fact, a comparative exercise, led by KAR, was carried out at the end of June 2003 which considered the three options for ore clearance at the Boschfontein West Mine: trucking directly to surface; trucking to a single fixed tipping point underground onto a conveyor; trucking to a vertical rock hoisting shaft. For the purpose of that exercise the life of the mine (approximately 15 years) was split into three phases in time. The model developed for the study indicated that trucking of rock to surface should continue after the initial development of the mine until such time that it would no longer be viable to operate trucks to surface, which in the study was in the second phase (up to 10 years). In terms of this exercise, it was recommended that any installation of a conveyor should be delayed, thereby deferring significant capital expenditure.

Visits to base metal mines, mining massive orebodies, by KAR in April 2002 had confirmed that it was a common practice to tram ore by truck to a single tipping point underground and then by conveyor to surface. It was also accepted practice that trucks would travel up full to the tip and down empty back to the workings. The conclusions of the above exercise and the visits to base metal mines to see in practice the use of trucks on ramps, supported KAR's decision to include, in the design of the Boschfontein Mines, truck tramming in ramps both to surface and later to a common tipping point.

For completion of the above discussion, the study referred to above is included (without the appendices) as **Annexure 7.5** in **Volume 4**: ***Boschfontein West Mine Rock Transport Investigation***, by K.A.Rhodes, June 2003.

7.2.3 ***<u>Summary of Mine Design</u>***

In summary, the underground mine design for both Boschfontein East and West Mines was based on a hybrid layout: conventional stoping operations with development (except raising on reef) carried out with trackless equipment.

<u>Production</u>

The mines were planned for 150ktpm and 100ktpm at the West Mine and East Mine respectively.

<u>Design Parameters</u>

Underground development was planned to take place from twin ramps developed from surface. During the build-up to full production all broken rock was to be transported to surface by truck. At steady state conditions reef would be trammed by truck to a single underground tip and crusher station before being sent by conveyor out of the mine. Cleaning and tramming of all rock would be carried out by 18 ton capacity LHD's and 50 ton trucks.

A complete suite of trackless equipment would be utilised in all big end access development: twin boom drill rigs, roofbolters, LHD's, trucks, UV's.

Stoping was planned on a downdip layout with stope backs 190 metres. Stope faces were 28 metres long with a planned stoping width of 0,9 metres. For the downdip layout scraper winches would clean down the reef raises ('box per panel') to muck bays from where LHD's would load directly into trucks.

Grade and Dilution

At the Boschfontein Mines the most important motivation for the hybrid layout over a room and pillar layout, as designed for Waterval, was an improvement in head grade. The UG2 Reef would be mined at minimum width and all development rock (both reef and waste) would be sent out of the mine as waste and not to the mill.

7.2.4 ***Final Comment on Hybrid Philosophy***

The retention of labour intensive stoping and the necessity for the manual operation of rock drills for face work is the most significant disadvantage of any hybrid layout; this has been the case since its first introduction by KAR at REGM in the 1980's. The need to mechanise face operations and eliminate conventional rock drill operators in narrow reef stopes had always been the objective of KAR, firstly in the 1980's on JCI's gold mines and later in the 1990's at Amplats' platinum mines. However in the early 2000's, at the time that Amplats' Rustenburg UG2 expansion programme was being planned for, there had been no commitment to such a new method even though trials had been planned, designed for and carried out on Amplats' sites at Union Section and Rustenburg Section; these trials will be discussed further in this chapter.

7.3 **Long Hole Stoping Trials**

There has always been an obvious necessity to improve face productivity by means of a proven alternative to the present conventional drilling and blasting of narrow reef stopes on the gold and platinum mines of South Africa. In terms of this need, early discussions with a leading OEM were initiated by KAR as far back as 1993, when working as a consulting mining engineer for JCI. Later in 1996 KAR, as an independent mining consultant, completed a project assignment for Amplats related to long hole drilling techniques for the blasting of stopes on their platinum mines.

7.3.1 ***Background***

Although the drilling of long holes in stoping operations is common practice in massive orebodies and indeed in steeply dipping narrow vein mining conditions world-wide, it has never established itself in the narrow flat dipping tabular orebodies found in South Africa. Nonetheless, trials with long hole drilling have been carried out over the years on South African gold mines. A study of the Association of Mine Managers' Transactions reveals that experimentation took place on gold mines in the Orange Free State in 1958/59. These trials, although they were limited, are perceived from the relevant technical papers to have been successful. Further documentation of any subsequent trials cannot be found in later transactions of the Association of Mine Managers. It is therefore inconclusive as to why long hole production stoping methods were not further developed. A possible answer may be seen in the contribution by J.P.Andrew, to a paper by R.P.Plewman from Harmony Gold Mine, when he stated that "no matter what method is used, the limits of the method are decided more by what can be handled than what can be broken". This is an important conclusion to be borne in mind when designing any new method.

In all of these early trials the dip of the reef was between 17° and 20° with a stoping width of 1,2 metres to 1,5 metres. Drilling was carried out by bar rigged machines with hole lengths of up to 60 feet (20 metres). The questions asked of the method by KAR in 1996 were what might have been asked forty years before, and were related specifically to the dimensions of the access drive for a drill rig to operate; the accuracy of the hole to be drilled; the effect of a heavily charged blast; the comparison of costs to conventional; the overall productivity of the new method and would it be more viable.

In July 1996 K.A.Rhodes presented to Amplats his project report ***Long Hole Drilling Techniques for Blasting Stopes in Narrow Reef Conditions on Platinum Mines***, this complete report can be seen as **Annexure 7.6** in **Volume 4**. In his conclusions KAR recommended that Amplats should carry out trials at Union Section; details of this proposed trial were set out in the report. It was further recommended that trials should be carried out at a second site in order to develop the concept.

This report of 1996 had demonstrated that the long hole stoping system (LHS) was capable of being developed to a technically proven level which could be economically viable. If these trials could prove successful the overall objective would be to introduce the new method to a 'greenfields' operation, when the design of the mine could support the new technology from the onset; in principle, similar to the establishment of the H.J.Joel Gold Mine in the 1980's, fully documented in chapter 5 in this exposition.

7.3.2 ***Union Section Trial***

Immediately following the submission of the above technical report, KAR proposed that a trial be carried out at Union Section (RPM). It was proposed to split the trial into two separate phases: a pre-stoping test drilling programme and stoping trials on a selected panel.

The methodology for the test drilling would focus on optimum hole length in respect of accuracy; drill steel confirmation; bit size; collar procedure taking cognizance to not exceed 100 bar percussion pressure; the use of tubes. During the full period of this trial all drilling operations were to be under the 'hands-on' supervision of a drill master supplied by the OEM (Tamrock).

In terms of the stoping trials on a panel, the objectives were to determine the drilling accuracy of the long holes (follow-up to the test drilling programme); determine throw blasting characteristics; assess the effect of the blast on conditions in the stope, specifically the hanging wall; to optimise burdens and hole spacings; to ascertain drilling and blasting costs; to establish overall costs, to some degree of confidence, for the method.

Prior to these proposals, in fact as far back as 1993, significant test work had been carried out by the same OEM, this work being initiated by KAR. This work, carried out at Tamrock's Test mine, specifically focussed on drilling accuracy and the conclusions at that time were that tube drill steel (with 64mm bit) had to be used at a percussion pressure of not more than 100 bar in order to achieve the objective of a hole deviation of not more than 100mm over a length of 20 metres (0,5% accuracy); reference Reef Mining Project, Tamrock Technology Centre Research, dated 03 December 1993. Refer to **Figure 7.15** showing graphs of drill hole deviation at variable percussion pressures.

In 1998 KAR submitted his final proposed study for long hole drilling stoping at Union Section Declines; this submission set out a conceptual layout.The long hole system (LHS) as proposed would not require persons to work in the stope panel and, therefore, no in stope support would be planned for. For this requirement the span between pillars was restricted to 18 metres; the Group Rock Mechanics Consultant for Amplats provided supporting comments to the proposal.

Key production parameters were identified and it was expected that the long hole drill rig would drill 5000 metres in any month and with a burden of 100 cms between the two rows of holes the monthly production from one rig would be 2500m^2. The necessary stope development required to support the project's production parameters was also defined as 427 metres per month for a monthly production (reef) of 26000 tons.

Refer to **Annexure 7.7** in **Volume 4** for ***A Conceptual Study for Long Hole Stoping at Union Section Decline*** by K.A.Rhodes, dated 02 November 1998.

Notwithstanding, before the trials commenced there were some concerns from Union Section which were responded to in a brief document. A copy of these concerns and the responses set out by KAR can be seen in **Volume 4**; refer to **Annexure 7.8, *Union Section Concerns Related to the Long Hole Stoping Method***, dated 30 November 1998.

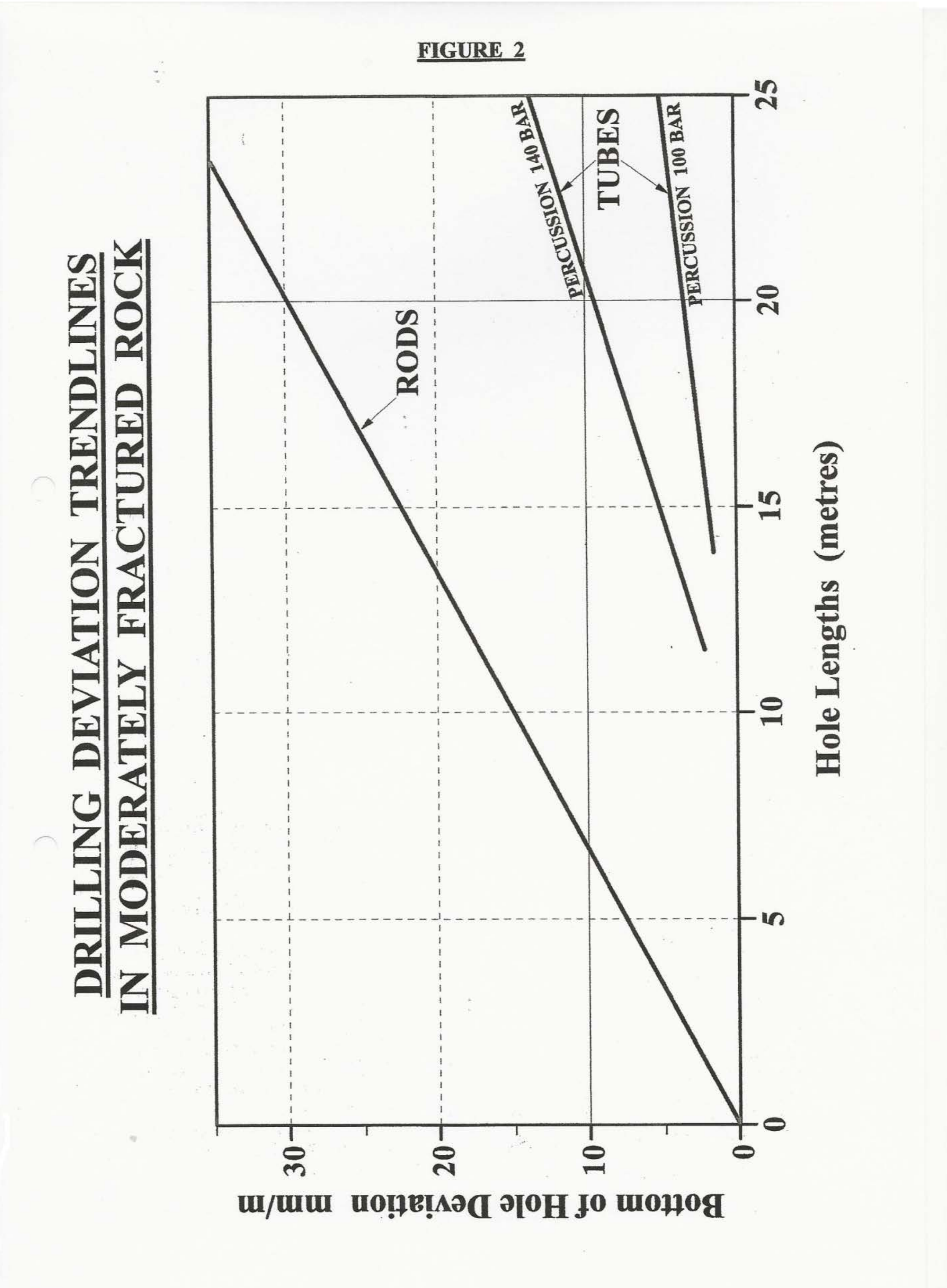


FIGURE 7.15

Drilling Deviation of Variable Percussion Pressures: refer to Annexure 7.7 Volume 2

In summary, the results from the trials could not be classed as an overall success in that the production rig in use, being primarily a machine designed for long hole drilling in large open stopes, was not the ideal machine for long hole drilling in narrow flat tabular reefs. It was therefore believed that a new low profile drill rig, at that time being developed for South African chrome mines, could be designed specifically for long hole drilling in the narrow platinum reefs. This was one of the machines identified by KAR in his investigation report prior to his planning of the Waterval Mine.

7.3.3 ***Boschfontein Trial***

Following the Union Section trials, in 1999 KAR held discussions with the management of Rustenburg Section (RPM) on the possibility of carrying out LHS trials at their Boschfontein Decline Section. Boschfontein had been developed as a trackless hybrid project which had then been abandoned but there were some pillars available for extraction and also there were areas that had been developed by trackless equipment, making it favourable for LHS trials. Subsequently, for a six month period in 2000 LHS trials did take place at Boschfontein. The objectives at Boschfontein were similar to those at Union Section with one major difference in that a much reduced stoping width was targeted (65cms or less) as the Merensky Reef channels at Rustenburg Section are extremely narrow compared to the 1,5 metre UG2 Reef at Union Section. The re-designed low profile rig was used in the trial to drill 15 metre holes with the same accuracy as planned for at Union Section of 0,5% (75mm maximum deviation). Refer to **Figure 7.16** for the general stope layout.

In overall terms this trial was considered to be a success, in that expectations had been met specifically in terms of stoping width control; accuracy of drilling long holes; very good hanging wall and footwall conditions in the stoping area; no necessity for persons to enter the stoped out workings. In terms of these findings, KAR submitted a further motivational report for the continued use of LHS. The conclusion in that report stated that a reduction in costs of 3,8% was likely which, coupled with an improvement of grade of about 6%, could result in an overall improvement in profitability of the order of 10%. It was clearly a safer method than that of conventional stoping and it had to be a definite move towards the

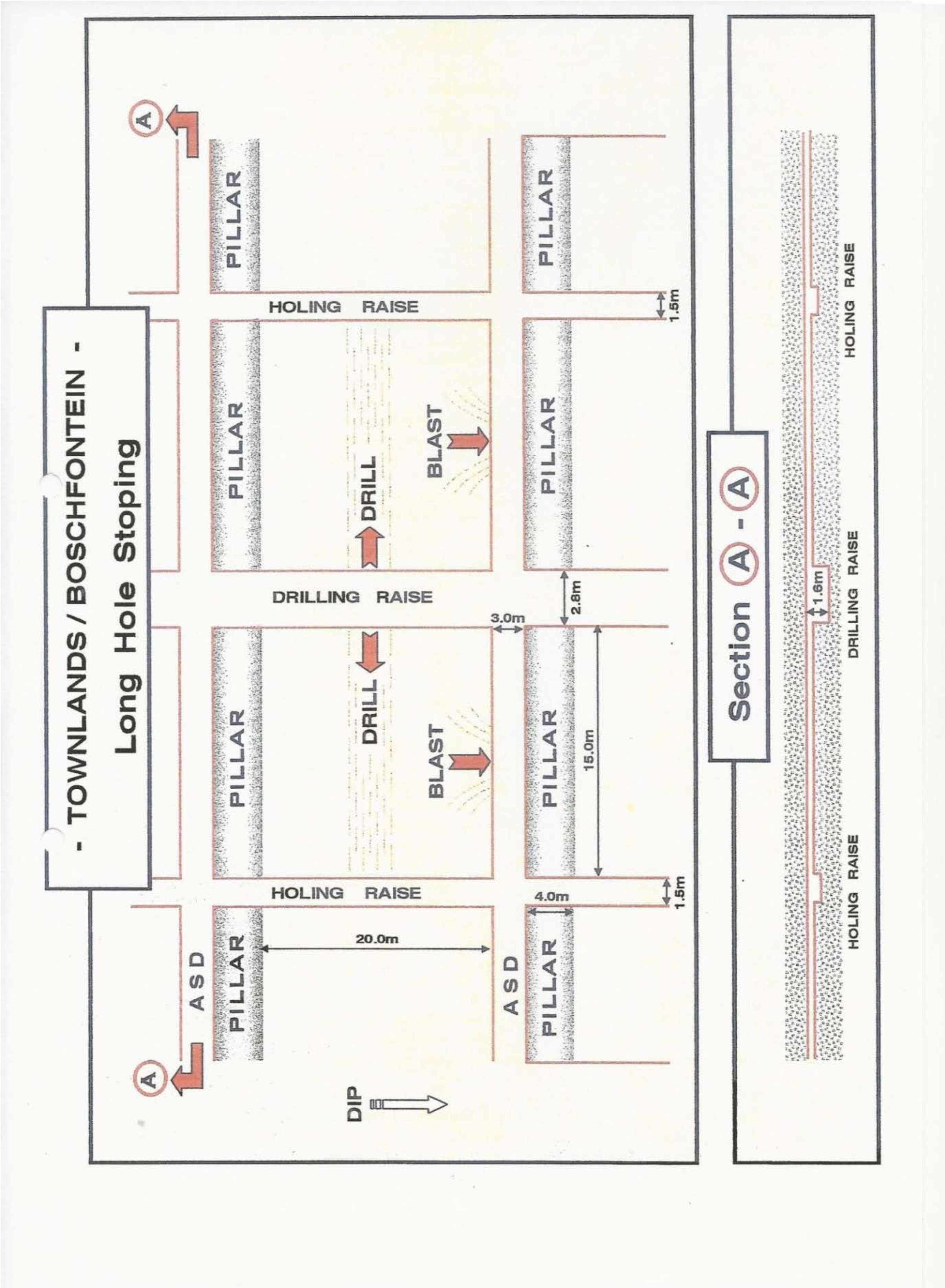


FIGURE 7.16

General Stop Layout for Long Hole Stoping Method

ultimate goal of total mechanisation. Therefore it was axiomatic that if any improvement in the viability of stoping operations could be achieved it would be considered worthwhile to pursue the concept. The recommendation therefore was to continue with the project and extract the pillars on the Merensky Reef and subsequently to prepare a more definitive feasibility for the mining of the UG2. Notwithstanding the different geotechnical conditions on the two reef horizons, it was proposed that the Merensky Reef layout would be assumed for the UG2 Reef subject to confirmation (or otherwise) by the Amplats Rock Engineering Consultant. A copy of this motivation entitled ***Preliminary Report on Future Long Hole Stoping Operations at Boschfontein Shaft, Rustenburg Section, Rustenburg Platinum Mines*** by K.A.Rhodes, dated 02 August 2000 is seen in **Annexure 7.9** in **Volume 4.**

At the same time, in order to further motivate the use of LHS, KAR set out a conceptual proposal for a modified type of long hole stoping, defined as resue long hole stoping (RLHS); the mine targeted was Bafokeng Rasimone Platinum Mine (BRPM). This method was intended to counter the uncontested argument that LHS in the trials was development intensive. In this proposed change to the LHS, long holes would be drilled in the waste below the reef and the waste would be blasted away from the advancing face allowing the reef to be blasted down in a separate operation and then loaded out. The proposed method allowed for an increased working height for machines to operate on the reef horizon but would allow the waste to be left behind. It had the advantage of being less reliant on the drilling accuracy, possibly tolerating greater deviations in longer holes, and grade would be improved over the standard LHS method. There was no interest in a feasibility study for the method and therefore no follow-up took place. The concept of the method was drawn up by KAR in late 2000 and the proposal ***Mechanisation Options for Bafokeng Rasimone Mines***, by K.A.Rhodes is attached as **Annexure 7.10** in **Volume 4.**

Nonetheless, in spite of the success of the above trials the potential for LHS as an alternative mechanised mining method has not been realised on platinum mines.

However, as a postscript to this section on long hole stoping, it can be recorded that in recent years there has been two gold mining companies prepared to introduce LHS as the production mining method on their operations. Firstly, Central Rand Gold who intended to mine out reefs left over 100 years ago at their Consolidated Reef Mine. Unfortunately they had to cease underground mining shortly after commencement of operations but this was not directly due to the method of mining. Secondly, Great Basin Gold Burnstone Mine closed down in 2012 and were therefore unable to prove the method conclusively.

To sum up, the overall conclusion regarding LHS is that until a proven viable rock cutting method for stope face operations can be developed it is axiomatic that trials of LHS should be pursued on a wide enough scale to determine whether it is a viable alternative to conventional stoping and therefore an enhancement to hybrid trackless mining.

7.4 Tunnel Boring Machines

The planning for the access decline development of the Styldrift Mine by TBM has been discussed previously in this chapter. This section will now focus primarily on the tunnel boring of a reef raise project managed by KAR. However, before this discussion it is relevant to this section to discuss the project report on the BorPak boring system, prepared by KAR in 1999.

7.4.1 *BorPak Boring Machine*

In 1996 KAR prepared a project report for Amplats which recommended the purchasing of a BorPak 1500 blind borer for the development of stope connections (reef raising and orepass development).

The first trials with the prototype BorPak had taken place in the late 1980’s and the first production machine was introduced to the mining industry at the Las Vegas Mining Show in 1992. The BorPak was an automatically operated blind boring machine which could bore excavations varying between 1,2 metres and 2,5 metres in diameter. In effect it was a mini tunnel boring machine in the way it operated; there was no drill string or any need for a pilot hole. The machine was stabilised by packers, inflated against the walls of the excavation, which absorbed the thrust and torque of the machine, refer to **Figure 7.17**. The machine could easily be transported through the workings by rail haulage even in a conventional mine; refer to **Figure 7.18** and **Figure 7.19**. In orepass development, cleaning of cuttings would be by gravity whereas in flat inclinations (reef raises) it would be necessary to use a vacuum system; refer to **Figure 7.20**. Only three machines had been in operation at the time of the project report but the technical aspects were believed to have been proven; KAR had the advantage of seeing the BorPak in operation at the Hartley Platinum Mine in Zimbabwe in May 1996. The BorPak was capable of drilling an accurate hole up to 300 metres in length whereas other blind borers were restricted to 100 metres due to their limitations in accuracy; therefore it had the potential to bore a conventional reef raise at Amplats' mines. In effect the project investigation showed that the BorPak was capable of developing a stope connection (reef raise and orepasses) in a time of six months less than by conventional means; refer to **Figure 7.21**. This was therefore an opportunity for a more rapid build-up of face where required at an existing mine or in the build-up to steady state at a new mine.

The report indicated a financial justification for the BorPak machine and recommended the purchase of a rail mounted BorPak for its immediate introduction at any existing shaft where a more rapid opening up of face would be advantageous or at any new greenfields mine. Unfortunately the recommendation was not acted upon. A copy of the project report ***The Application of the BorPak Boring System for Raise Development in Platinum Mines***, by K.A.Rhodes, dated February 1997, is attached as **Annexure 7.11** in **Volume 4**.

7.4.2 ***Bafokeng Rasimone Tunnel Boring Project***

On 05 September 2000 Amplats signed a contract with Bomar Tunneling and Underground Contractors (Proprietary) Limited (BOMAR) for the boring of a reef raise at Bafokeng Rasimone Platinum Mine (BRPM); the length of the raise was envisaged as 450 metres in length but in fact it proved to be 340 metres. KAR was the appointed project manager.

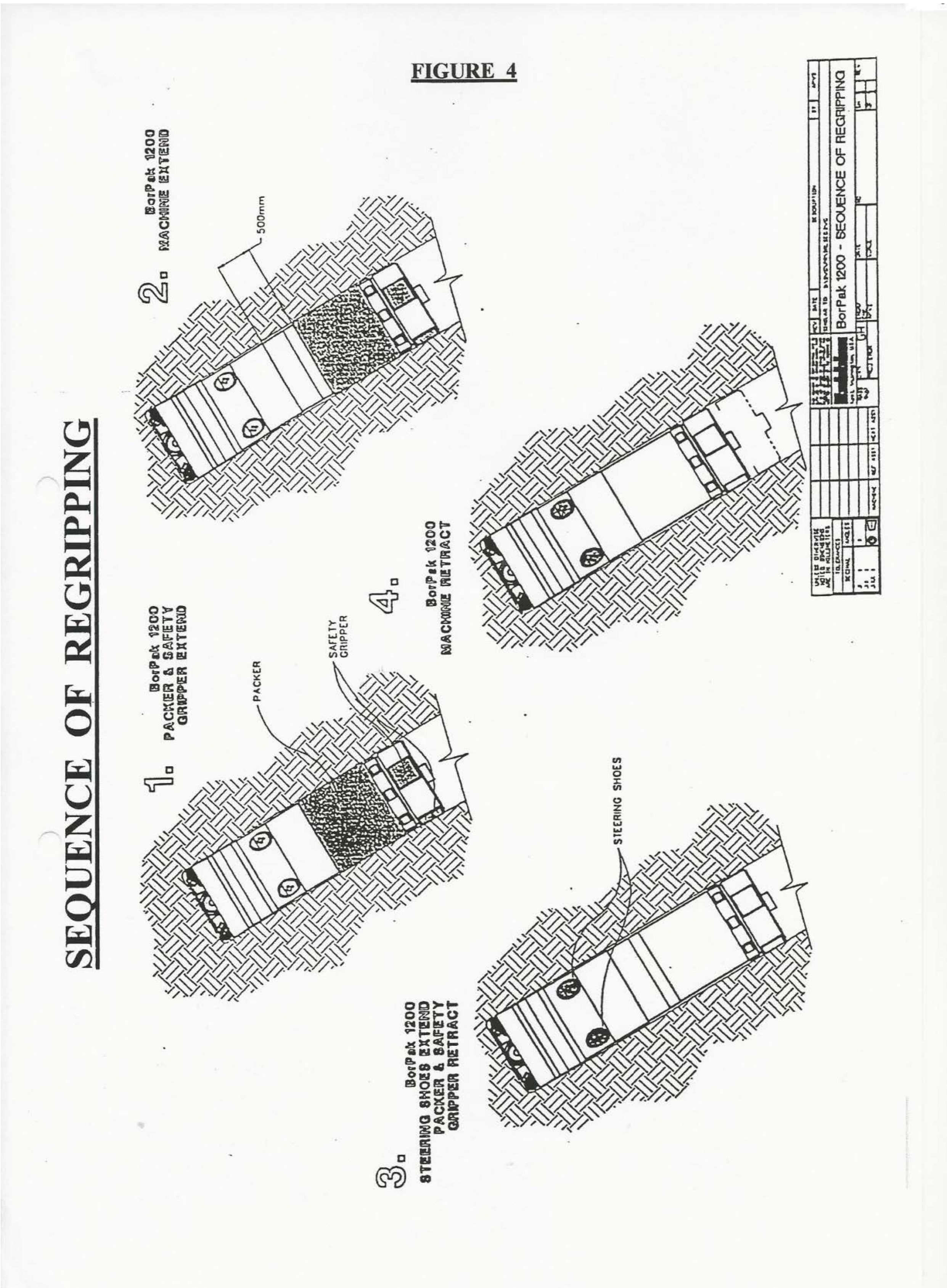


FIGURE 7.17

BorPak: Sequence of Re-Gripping: refer to Annexure 7.11 Volume 4

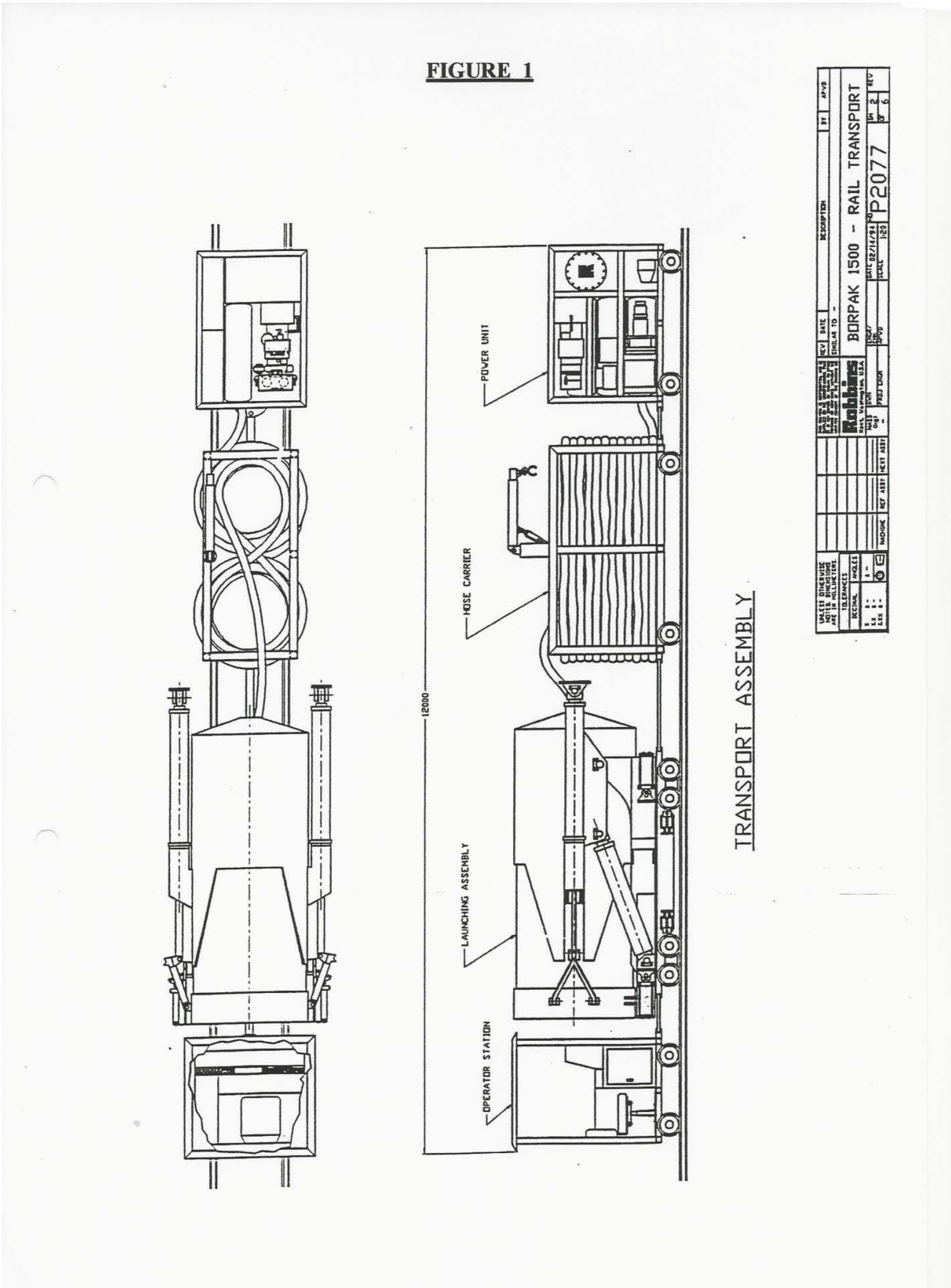


FIGURE 7.18

BorPak on Rails: refer to Annexure 7.11 Volume 4

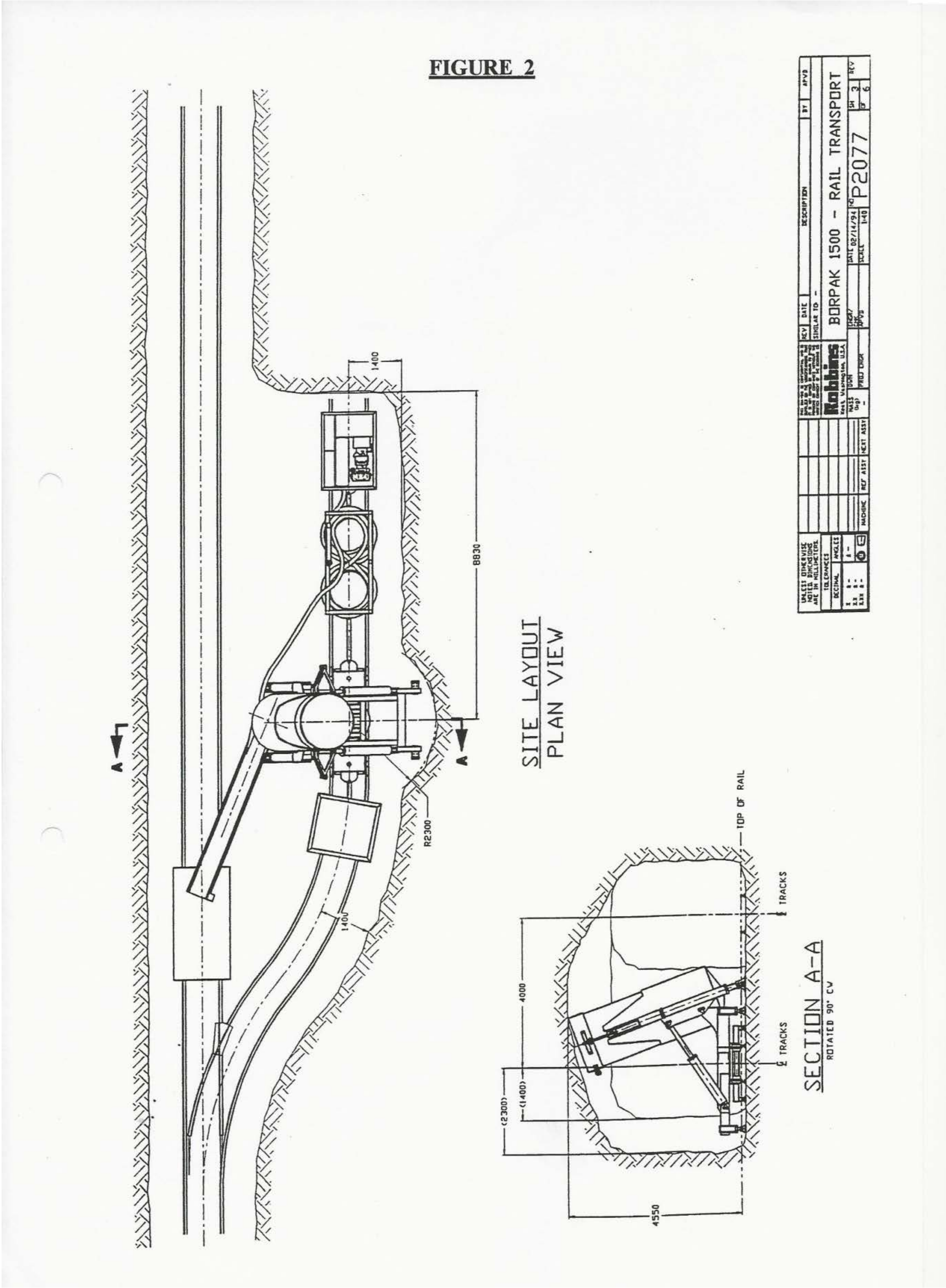


FIGURE 7.19

Rail Transport of BorPak Underground: refer to Annexure 7.11 Volume 4

FIGURE 5

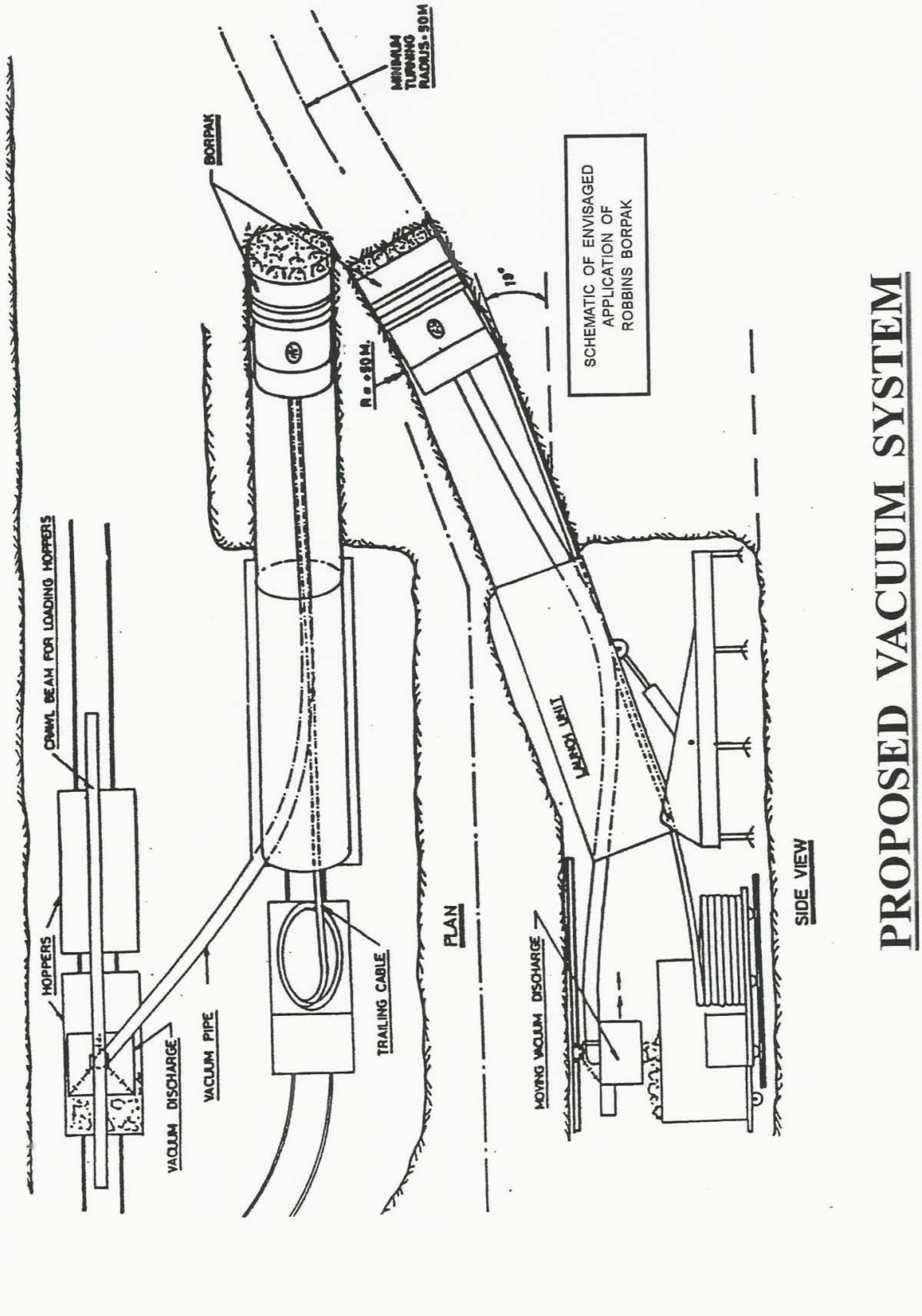


FIGURE 7.20

Proposed Vacuum System for BorPak: refer to Annexure 7.11 Volume 4

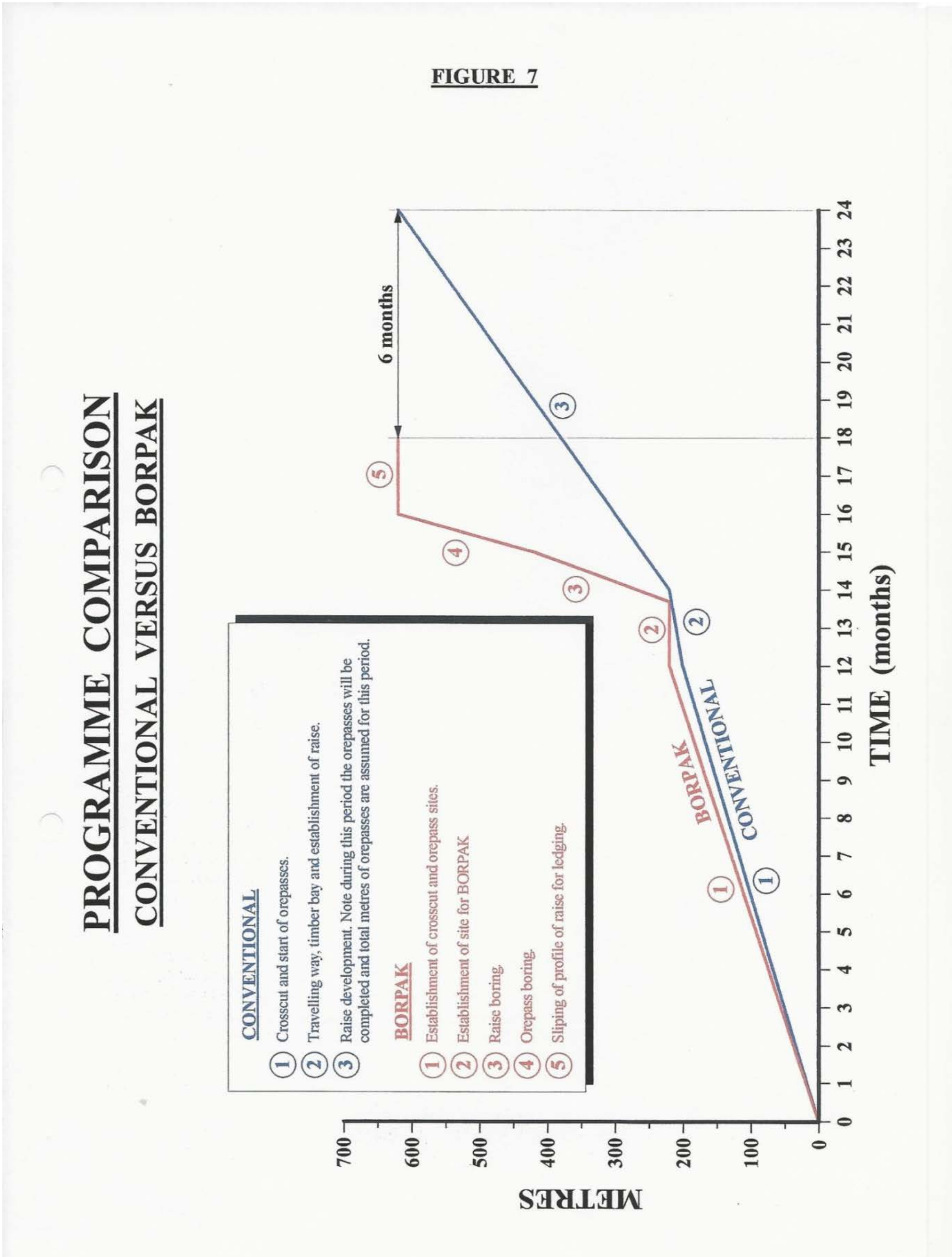


FIGURE 7.21

A Programme Comparison of Conventional and BorPak: refer to Annexure 7.11 Volume 4

Background

The use of a TBM is common practice in civil engineering projects around the world but with few exceptions the mining industry has failed to take advantage of this proven technology. Amplats perceived the need to accelerate the rate of reef development for new mines, such as the BRPM Project then being developed; this being the background for the signing of the contract with BOMAR working in association with the Robbins Company in the USA.

Machine Details and Scope of Work

The TBM used for this project was a Jarva MK6, manufactured in 1972 and the scope of the work was to establish a reef raise 340 metres in length. It was generally expected that the TBM would bore at an inclination of plus 14° and therefore cleaning of the rock chippings could not be gravity assisted. It was planned for cleaning of the raise, immediately behind the machine, to be carried out by scraper winch, the return sheave wheel being attached to the back of the TBM. However, this did not prove to be an ideal cleaning system.

Ventilation Considerations

The ventilation arrangements consisted of an exhaust system which proved effective, notwithstanding initial problems primarily due to interference by the scraper winch arrangements. It was mandatory to equip an inflammable gas detection sensor as close as possible to the cutter head; this sensor was calibrated to cut off the power to the TBM when the methane concentration reached 1.4% in the vicinity of the sensor head. The system was checked at weekly intervals and, further, all personnel were trained in gas detection and operators carried a methanometer at all times. These precautions were necessary as methane gas is not uncommon on platinum mines.

Problems and Delays

The TBM started boring from a launching platform on 13 December 2000 and holed through on 15 June 2001.

Learning Curve

Initially it had been expected that the actual boring of the 340 metre reef raise would have taken about two months; this turned out to be highly optimistic and it actually took six months (22 days lost for holidays). However, for this new project there was a long learning curve with many problems.

From the outset the operators experienced difficulty in the steering of the machine. This lack of skills caused the TBM to hole into an orepass with a loss of 24 days; it had been planned for the TBM to pass over this excavation. The presence of a Robbin's technician, sent for from the USA, resulted in an improvement in the ability of the TBM to climb steeper.

Fouling of the services in the raise by the scraper arrangements caused delays; such problems would demand a different cleaning system for future flat dipping raises.

There were also numerous delays due to mechanical and electrical failures of the TBM.

Towards the end of the project three intersections of methane occurred, causing the TBM to shut down automatically.

Performance
Approximately one month was lost due to holiday breaks, therefore the boring phase took five months. The average rate of penetration (ROP) was 0.47 metres per hour. This was considered slow but rates were affected by potholes, steering problems, an inefficient cleaning system and a low powered TBM.

In the later stages of the project the best results occurred: best advance in a day 13.8metres; best advance in a week 47.6 metres; best advance in a month 104.7 metres (last month); best ROP in a shift 1.33 metres/hour. From a total of 3208 hours, boring time was 716 hours (22,3% TBM utilisation). A summary of delays can be seen in the pie chart on **Figure 7.22**.

Lessons
There is no doubt that the project proved successful in spite of the choice of an old (1972) TBM.

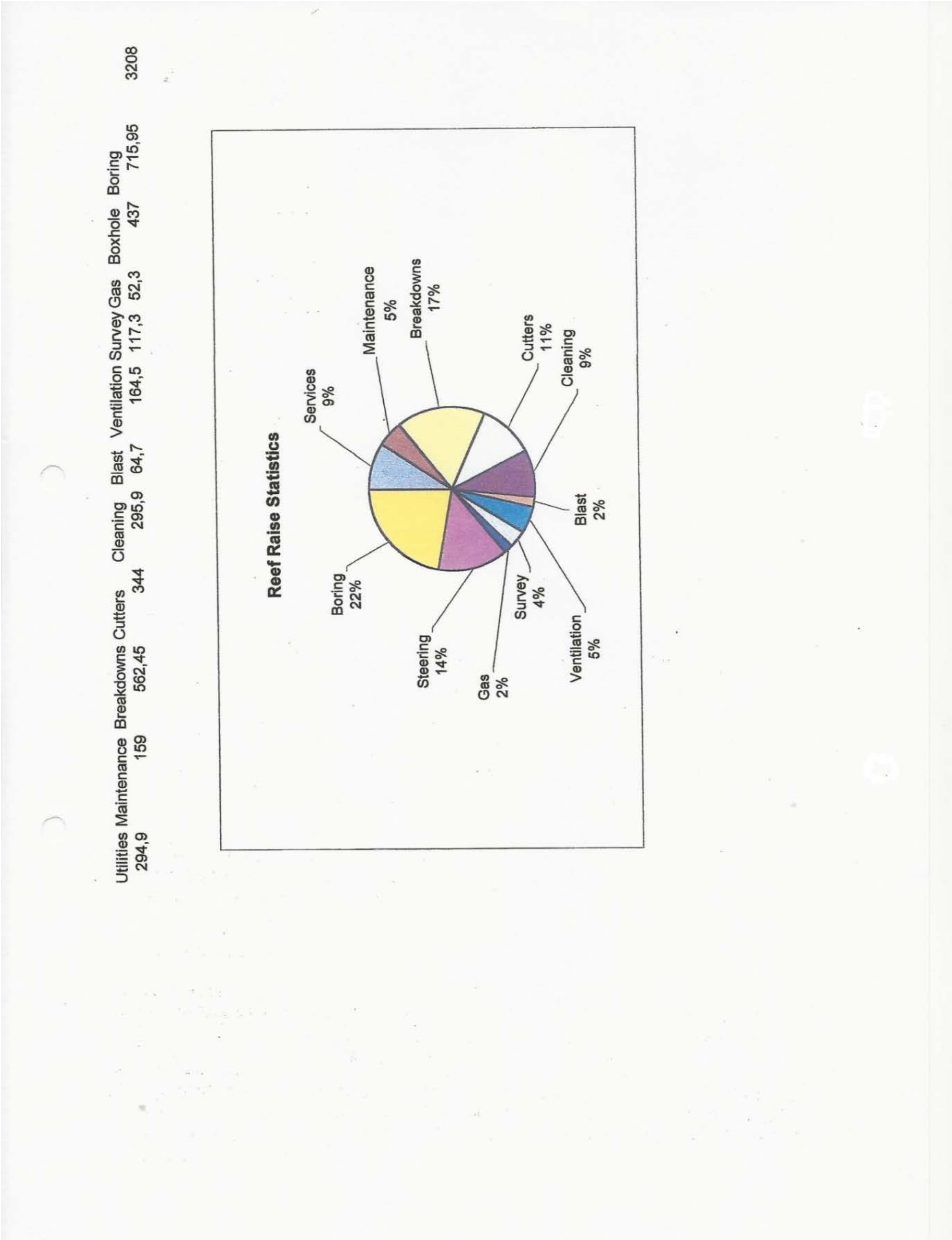


FIGURE 7.22

TBM Project at BRPM: Summary of Delays

When scrutinising the penetration rates of the different rock types the best results were achieved when boring on reef and when the reef plane was the least disrupted, specifically by pothole occurrences.

Optimum advance would never be achieved with scraper winch cleaning and it was realised that it would be necessary to design for an alternate system; possibly conveyors or a vacuum system. The installation of services had to be interrogated further, the use of a scraper did not allow for boring to take place when services were being installed.

The need for 'hands-on' supervision and management from the outset was a pre-requisite for such a project; the performance improved markedly after the TBM technician/advisor from Robbins became involved. The splitting of the day into three shifts, one for maintenance on dayshift and the following two shifts for boring, proved successful.

TBM Project Summary Report
A detailed description of this project has been documented in a paper presented to the 6th International Symposium on Mine Mechanisation and Automation at the Sandton Convention Centre in September 2001. This paper was prepared by KAR assisted by Peter Horrell, a TBM consultant. This paper can be seen in **Annexure 7.12** in **Volume 4**; "***Reef Development with a Tunnel Boring Machine on a South African Platinum Mine***" by M.Stander, K.Rhodes, P.Horrell, D.Sammons, G.Harrison, J.Dean. (Mr Stander presented the paper as the mine manager of BRPM.)

However, a summary report written by KAR and Peter Horrell in December 2001 sets out in greater detail additional project costs and those mistakes which would not want to be repeated at future projects. In addition, the report details the pre-planning requirements considered necessary for any TBM project. These comments are far-reaching and cover issues not included in the above mentioned technical paper. The summary report is included in **Annexure 7.13** in **Volume 4**; ***Summary Report on the Use of a Tunnel Boring Machine for Reef Development at Bafokeng***

Rasimone Platinum Mine by K.A.Rhodes and P.Horrell, December 2001.

To sum up, the TBM used in the trial was manufactured in 1972 and its design, with its back-up system, was not considered ideal for reef development. Notwithstanding the long delays when boring and other problems, there was the belief that a new TBM could be designed specifically for reef development, with a more flexible steering system in order to negotiate changes in reef, that would be able to achieve a high rate of advance, up to say 400 metres/month. It was therefore expected with some confidence that reef development could and would be carried out with TBM's in the future. Unfortunately this has not been the case to date.

7.5 Post 2003

During the period 2003 to 2008 KAR continued to consult on mechanisation projects for mines exploiting narrow reefs; typical examples were the hybrid method for Eastern Platinum Mines' Barplats Mine, the Snap Lake diamond mine in North-West Canada and a proposed new uranium mine in India.

7.5.1 *Legadembi Gold Mine*

It was during this period that KAR was the Project Manager for the Legadembi Gold Mine in Ethiopia. Legadembi Gold Mine, situated near Shakisso in the south of Ethiopia, had been operating as a surface mine for several years and it had become necessary to develop an underground mine.

The mining method chosen by KAR was a horizontal cut and fill method and operations would be fully mechanised. Although open stoping methods would be commonly selected for wide steep orebodies, at Legadembi there was a need for selective mining which would necessitate strict control over the drilling and blasting operations. In addition, open stoping methods would create high, near vertical, stopes where instability of the hanging wall would represent a high risk of dilution which was unacceptable for the mine. The proposed horizontal cut and fill method would enable stoping operations to be confined to the payable limits of the orebody.

At the time that the underground mine design was being carried out it was known that about half the extent of the main orebody had a payable width of 30 metre and geotechnical work had established that a maximum stable span would be only 14 metres. In terms of the selected mining method it was therefore necessary to design for a modified room and pillar layout within each cut being mined; pillars would be necessary to ensure that 14 metre spans were not exceeded. Therefore a modelling exercise had to be carried out to determine the best way in which the mined out rooms and pillars would be left.

In the horizontal cut and fill system drilling was to be carried out flat on each cut. In this manner a horizontal slice would be taken out with the face being mined as a brow (planned for at 4 metres) with a 2 metre high slot left below for the full length of the stope. This slot provided for a space for the broken rock and also served as a ventilation airway in the stope. All operations were to take place on waste fill. In this manner mining was similar to a wide reef room and pillar operation but which was repeated for each horizontal slice in a series of vertical lifts in the orebody. Refer to **Figure 7.23** for details of a stope cross section.

Various stope layout options were examined, for which in addition to a maximum room span of 14 metres, were also based on recommendations of both pillar and bay dimensions of 5 metres. The final recommendation was for a central room of 14 metres with staggered pillars 5 metres wide and 5 metre wide bays mined out either side of the central room. The details of these exercises to determine the preferred option for the room and pillar layout are the crux of the paper "***Design of In Stope Pillars in Cut and Fill Mining for a Gold Mine in Ethiopia"*** written by K.A.Rhodes and T.Rangasamy, delivered at the 5th International MassMin Conference in Lulea, Sweden organised by the Lulea University of Technology, the paper being published in the transactions of that conference; refer to **Annexure 7.14** in **Volume 4.**

7.5.2 ***Maintenance Action Plan***

Reference has been made in this chapter to a maintenance action plan for trackless equipment. After many years of managing and consulting for trackless mechanised mining projects and mines, in

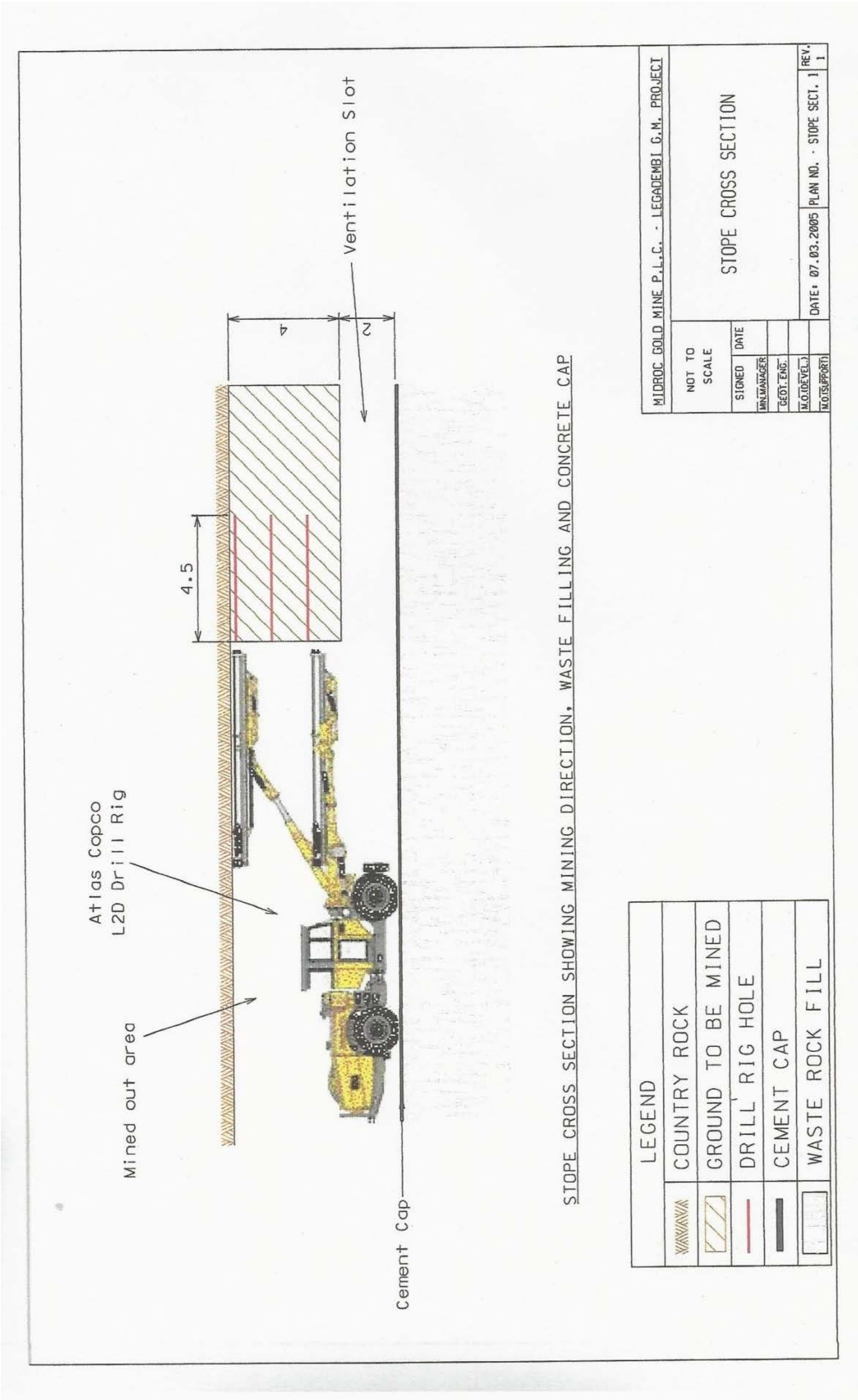


FIGURE 7.23

Stope Cross Section: Horizontal Cut and Fill

2008 KAR completed a guide for use in trackless management; this guide would be applicable to all mechanised methods of mining, notwithstanding that the focus of this exposition is on near flat tabular reefs. The importance of the engineering function for the success of any trackless operation has been stressed in this treatise. The establishment of a maintenance plan becomes even more critical when considering the change from conventional mining to trackless methods on gold and platinum mines in South Africa and with this in mind the details of the guide ***A Maintenance Action Plan*** by KAR 2008 is included as **Annexure 7.15** in **Volume 4.**

7.6 ***Conclusions and Summary***

In this chapter the initiatives by KAR to pioneer the use of TBM's for development on new platinum mines has been discussed. In the case of the Styldrift Mine, after nearly three years of detailed planning the project was aborted following the involvement of the Royal Bafokeng Nation. At that time, contractual negotiations between Amplats and a contractors' consortium, Platum JV, were at a very advanced stage and planning for the boring of a large diameter decline had been completed. Without the intervention of the Bafokeng Royal Nation the Styldrift Project, which had been reviewed and approved by Anglo American Corporation, would have been the first underground mine in South Africa to be developed from surface by TBM.

At BRPM a reef raise was bored by an old TBM as a trial, which under the difficult circumstances described in this chapter was still a success. However, no follow-up took place. Previously, in the late 1990's, attempts to promote the use of a mini tunnel borer (the BorPak) also failed.

Over a five year period KAR continued to advance the LHS method of stoping on narrow platinum reefs. Successful trials were carried out but the method still remains untested under production conditions in order to prove (or otherwise) its viability.

The use of high capacity trucks for the start-up of new shallow decline mines and typically Amplats' Boschfontein Mine has been described. It has been shown that trucking to surface can continue for an extended period until it becomes more viable to introduce a decline conveyor, with the trucks continuing to tram to a single tipping point. The advice given by KAR to Impala (Zimplats), after the Boschfontein proposals, to utilise trucks to tram ore from the stopes direct to surface has proved to be a success at their platinum mines at Ngezi in Zimbabwe.

The Legadembi Gold Mine, where KAR was the responsible project manager for a new underground mine design, has been referred to in this exposition; this project gave KAR the opportunity to use his experience over many years in the mechanisation of tabular reefs, to plan for an innovative design in a near vertical orebody.

Finally, a detailed guide for the maintenance of trackless equipment, compiled by KAR over several years, has been referred to in this chapter and is included in Annexure 7.15 in Volume 4.

CHAPTER 8

Some Final Thoughts on the Mechanisation of Narrow Tabular Reefs in South Africa: Past, Present and Future

CHAPTER 8

Some Final Thoughts on the Mechanisation of Narrow Tabular Reefs in South Africa: Past, Present and Future

The purpose of this chapter is to set out briefly what could be the future for the mechanisation of narrow reefs on the gold and platinum mines of South Africa, taking into account what has been achieved in the past and what is now the present status of mechanisation on these mines.

8.1 Past Experience

Trackless mechanised mining operations commenced on the South African gold mines in the early 1980's; these first projects have been described in chapters 3, 4 and 5 of this treatise. Prior to this date limited trials had taken place with LHD's for cleaning operations in strike gulleys on a few gold mines. However, it was only at Randfontein Estates Gold Mine (REGM) in the period 1983 to 1985 that full suites of trackless equipment were introduced for the first time to tabular reefs in a gold mine in South Africa. In 1988 the H.J.Joel Gold Mine was commissioned as the first totally trackless gold mine in South Africa. This early work on the mining of narrow reefs by trackless mechanised mining methods (TM3) was pioneered by K.A.Rhodes at REGM and the H.J.Joel Gold Mine.

Following this pioneering work by KAR, the Technical Director of the parent company (JCI) approved the introduction of numerous other TM3 projects at all JCI's gold mining operations. By 1986 JCI had, in either operation or had placed orders for, a total of 103 LHD's, 103 drill rigs, 44 trucks and 135 UV's; these numbers were stated by the then Technical Director of JCI, Mr.H.Scott-Russell, in his keynote address to the Trackless Mining Symposium, Association of Mine Managers of South Africa (AMMSA), February 1988. Mr.G.W.Futcher, President of AMMSA at the time of the above symposium, wrote in his foreword to the transactions of that symposium that the symposium "***has put together the steps that have been taken by the gold mining industry towards entering a new era of mining technique***". The statement reflected how gold mines in South Africa, at that time, had lagged behind the advances gained in mechanisation by other metalliferous mines (and coal mines) around the world; this also being typical of metalliferous mines (other than gold) and coal mines in South Africa. There was of course good reasons for the South African gold mines failing to capitalise on advances made with mechanisation world-wide: one had to take cognizance of the narrow tabular reefs mined in South Africa which do not compare with orebodies mined elsewhere in the world. Nevertheless, it is important to quote again from the then President of AMMSA in his foreword to the aforementioned symposium transactions: "***The introduction of trackless mining into South African gold mines has not been easy. No doubt many years of research development will be necessary before fully integrated trackless mining methods are utilised throughout the industry. The decade of the eighties has demonstrated without a doubt that trackless mining methods can be used in South African hard rock mining involving narrow tabular ore bodies***". Unfortunately this was not to prove to be the case as sufficient time was not given for further development of mechanisation on South African gold mines; as from the beginning of the 1990's trackless mechanised mining in narrow reef in gold mines lost favour to the entrenched conventional methods of the previous hundred years. There were many reasons for this failure to follow through with mechanised mining. Some of the more important barriers to success were escalating costs of spares from overseas, difficulties in recruiting qualified artisans and probably more importantly management did not have the necessary knowledge of mechanised mining and there was a lack of determination to change.

It can only be agreed with the then President of AMMSA that it would have taken many years to establish TM3 as a viable alternative throughout the South African mining industry. In this respect it is worthwhile reflecting on the time required to change from shovel cleaning to scraper winch cleaning on gold mines. In 1931/32 scrapers were first introduced for packing waste into old worked out stopes and also for cleaning flat raises and winzes and by 1933 they were being used increasingly in stopes. It was only in 1937/38 that the use of scrapers in stopes, which was almost standard equipment in the flat East Rand mines, was being extended to the steeper stopes of the Central Rand. In 1952 scrapers continued to be in general use and by 1955/56, with the exception of a single mine in the Orange Free State, scrapers were in overall use for cleaning stope faces. It was also at this time in 1955/56 that scrapers were beginning to gain favour for cleaning strike gullies as opposed to hand tramming.

Acknowledgement is given for the above information to the paper "Stoping Practice on the Transvaal and Orange Free State Goldfields" by Beck, Henderson, Lambert and Mudd published in the Transactions of the Seventh Commonwealth Mining and Metallurgical Congress, 1961. If one takes cognizance of a period of twenty five years when these changes were taking place, from hand cleaning with a shovel to the use of scrapers, it is axiomatic that it would take many years to totally revolutionise the mining method for narrow reef stoping. At the time of the late 1980's/early 1990's there was clearly no overall determination in the industry generally to strengthen the case for mechanisation; it was too easy to remain with the established methods and there was no incentive to change as there was not the pressure from increasing labour costs as there is today. Nonetheless, there is no reason to not believe that the original hybrid method, first introduced in 1984 at Cooke 2 Shaft, REGM by KAR (refer to Chapter 4) and then later employed by KAR in the design of H.J.Joel Gold Mine, could not have been converted to a totally mechanised method with the introduction of the electro-hydraulic face rig replacing the manually operated compressed air jackhammers. Even as far back as 1977 a crawler-mounted face drill rig underwent trials at Rustenburg Platinum Mines; refer to the paper "The Demag Stope Drill Rig" by van der Meulen and Harrison, in the Transactions of the Association of Mine Managers of South Africa 1976/77. The initial indications then were that the machine had a 'definite potential' and its drawback could be overcome by new design and techniques. It was however expensive to run but it was still expected to have a significantly lower operating cost when the potential of the machine was fully developed. It is therefore the belief of KAR that with the necessary determination a mechanised face rig (the Stomec), previously referred to, could have been developed nearly three decades ago to be effective and viable in order to transform the hybrid system to a fully mechanised trackless operation. Similar rigs in very low profile form have been developed for the so called XLP operations on the platinum mines in South Africa; this initiative will be discussed later.

With only limited experience of hybrid mining taking place in South Africa, from the mid 1980's to the turn of the century, mechanisation of narrow reefs has been focussed on the extra low profile (XLP) equipment developed from about 2000. Early trials began at Lonmin Platinum Mines with a room and pillar layout which was changed to a breast mining layout with on reef development. In 2004 Lonmin decided to mechanise all new shaft projects specifically at Hossy Shaft although Saffy Shaft was also included. However, the production build up did not materialise and in 2008 there was a change of strategy at Lonmin to reduce the scale of XLP mining in order to allow the method to prove itself on a significantly reduced scale. At the same time Saffy Shaft began the process of 'de-mechanisation' leaving only Hossy Shaft where limited XLP mining is still taking place. Trials with XLP equipment have also been ongoing in recent years at Amplats' Batophele Mine (previously Waterval Mine).

In summary, mechanisation of narrow reefs, where the focus shifted from gold mines to platinum mines in the 2000's, has not proved viable to date with XLP equipment operating in very low stoping widths; although the method has proved far safer it cannot yet compete in terms of productivity with conventional mining.

8.2 <u>**The Present**</u>

Currently attention is concentrated on the platinum mines in South Africa where a strike on the mines of Amplats, Impala and Lonmin has recently dominated the industry. This strike has just been settled after exactly five months, the longest strike ever in the mining industry. The strike, called by the Association of Mineworkers and Construction Union (AMCU), was in fact a follow through from the Marikana tragedy in late 2012; since Marikana, AMCU have taken over as the major union on the platinum mines in the Rustenburg area from the National Union of Mineworkers (NUM) who had previously held that position for decades.

It would be relevant to consider the build-up to the events at Marikana in 2012. The tragedy was preceded by a strike of workers from Lonmin's mines where a demand for a basic entry wage of R12500 per month was demanded, an increase of more than 100% which clearly would be untenable for any mining company in South Africa utilising labour intensive methods of mining. This strike had been spearheaded by underground rock drill operators (RDO's), a class of labour which was believed many years ago to be declining, emphasising the need to mechanise face drilling operations wherever possible. New trainees were not being made available; as an example in the experience of KAR, the position of spanner assistant to the RDO had been discontinued around 1990. With a shrinking source of new RDO's it was obvious to everyone that the time would come when the platinum mining companies would be 'held to ransom' by the RDO's. Ongoing human relation problems and disputes over bonus payments had been a clear indication for a long time that a confrontation, between mine management and the large complement of RDO's in labour intensive mines, would be inevitable. It could be argued that this in effect was what occurred at Marikana in 2012 and what then led to the longest strike in the history of mining in South Africa. This then was the background to the Marikana tragedy where thirty four miners lost their lives in a confrontation with police, notwithstanding that twelve other persons died in the weeks leading up to the final incident, amongst them police officers.

The question to be asked in retrospect is whether the confrontation at Marikana was unavoidable or not. If a more determined effort had been given to the mechanised drilling of stope faces many years before, could the outcome have been different. It was ironical that the confrontation had occurred at Lonmin's mines where only a few years previously (2004) Brad Mills, the CEO at that time, had said that all future mining operations would be mechanised, this strategy being reversed in 2008 following his departure. Present thinking at Amplats appears to be that their older mines at Rustenburg could be sold off in the short term and they would introduce mechanisation, where they considered it practicable to do so, on any newer shallow orebodies now being developed. At Implats they are currently considering some type of mechanisation at their new shafts in the Rustenburg area but their new development at Leeuwkop may be mechanised or they would consider abandoning that new project: refer to **Figure 8.1**, Business Day lead article March 28, 2014. Any planned mechanised operation at Leeuwkop would reduce the labour from (say) 10000 for the conventional mine to less than 3000 for the mechanised alternative.

8.3 <u>The Future</u>

In terms of any future mechanisation of narrow reefs on South African mines, the focus will be on the platinum mines. In the last few decades the South African gold mining industry has been decimated. In 1961 South Africa produced almost two thirds (65,6%) of the free world's gold, and today it is only a tenth of that. Notwithstanding, since 2010 Anglo Gold Ashanti (AGA) have developed partnerships with other organisations, known as the AGA Consortium, to develop the means of

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FTSE-JSE indices	Close	% ch
All Share	47380.98	▼ 0.57
Top 40	42663.18	▼ 0.65
Findi 30	57931.02	▼ 0.58
Resources 20	55145.42	▼ 0.98
Goldex	1431.51	▼ 3.08

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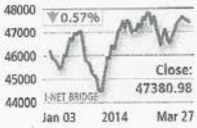

Bonds/forwards	I-Net Bridge	% ch
R186	8.38	▼ 1.05
R207	8.07	▼ 0.95
3-mth NCD spot	5.73	▲ 1.21
R/$ (6-mth)	11.029	▲ 0.41
R/$ (12-mth)	11.390	▲ 0.40

Currencies	I-Net Bridge	% ch
R/$	10.601	▲ 0.79
R/£	17.594	▲ 0.67
R/€	14.561	▲ 1.13
$/€	1.374	▲ 0.33
¥/$	102.200	▼ 0.28

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BERTRAND BADRÉ Global recovery depends on infrastructure Page 10

24hours

Summit in a quandary over rejected Grace Mugabe visa

A SUMMIT of European and African leaders due to be held in Brussels next week was in a degree of jeopardy yesterday over a rejected visa request by Zimbabwe President Robert Mugabe's wife Grace and other last-minute arguments. "We are waiting to hear if the AU decides to go ahead and postpone the summit," Aldo Dell'Ariccia, the EU ambassador in Harare, said. Page 5

First MeerKAT antenna launched

THE Minister of Science and Technology yesterday launched the first antenna of the MeerKAT telescope, taking SA and partners closer to building the Square Kilometre Array. Page 4

Anglo 'on path to recovery'

ANGLO American, reviewing global assets to boost profit after replacing its CEO, has started correcting years of "self-harm", says US financial research company Sanford C Bernstein. Page 11

Tencent weighs on Naspers share

NASPERS, Africa's largest media firm, needs to convince investors it will grow profit if SA's second-worst performing stock this month is to curb a slide led by its stake in Tencent. Page 11

Acucap-Sycom deal announced

CONSOLIDATION in the JSE's R250bn listed property sector continues to hot up, with the long

No state contracts for (

CAROL PATON
Writer at Large

THE African National Congress's (ANC's) investment arm, Chancellor House, should not hold direct government contracts and should avoid all conflicts of interest in future, ANC treasurer-general Zweli Mkhize told a panel discussion on political party funding hosted by the Institute for Security Studies (ISS) in Cape Town yesterday.

While Chancellor House recently divested its stake in Hitachi Power Africa, which held a R38.5bn contract with state-owned enterprise Eskom, the investment company holds stakes in several companies that have government contracts or operate in government-regulated industries such as mining.

Dr Mkhize is the third high-ranking ANC official over three terms of office who has promised to restrict the activities of the party's investment vehicle, to bring it in line with basic ethical principles.

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Implats may mechanise or abandon key project

Strike-hit miner could cut labour force by thousands at Leeuwkop

CHARLOTTE MATHEWS
Senior Writer

IMPALA Platinum (Implats) might consider abandoning its new R11bn Leeuwkop platinum

Amcu wants wider strike

Frc
afte

FIGURE 8.1

Business Day Report on Implats' New Leeuwkop Mine

exploiting their deep level resources by advanced automation technology. This would be achieved by boring out the reef and introducing backfill of high strength. However, there is a huge gap between the present labour intensive methods employing unskilled workers and any future automation which would require a highly skilled labour force working at a depth of five thousand metres. The AGA Consortium would envisage minimum underground labour complements, albeit highly skilled, which clearly will be at variance with the need to create jobs and obviously unpopular with a unionised labour force.

In terms of platinum mining, the industry goes back almost one hundred years and there is no reason to believe that it will not continue for many more years to come. Platinum mining began in the 1920's on the Eastern Limb of the Bushveld Igneous Complex (BIC) from dunite pipes before production on the Merensky Reef began ramping up in the Rustenburg Region from 1925 to 2000. Since 2000 the UG2 has been mined extensively on the Western Limb. Nevertheless, the Eastern Limb is now again the favoured target area notwithstanding the push to expand surface mining operations in Limpopo on the Platreef (Amplats' Mogalakwena Mine). The focus for mechanisation on the narrow tabular orebodies is the wider UG2 Reef (as opposed to the narrower Merensky Reef) and in the opinion of KAR mechanisation could be introduced at Amplats' Amandelbult Mine or any new operations on the Eastern Limb, Implats' new shafts in the immediate Rustenburg area and Lonmin's operations at Marikana. Only recently, with a change of CEO's, has Lonmin announced a policy of 'de-mechanisation' and therefore only Amplats and Implats are considering mechanisation at this point in time.

The hybrid method, introduced originally by KAR at Cooke 2 Shaft REGM in 1984, started with the disadvantage of still employing compressed air driven jackhammers on the stope face but with the anticipated changeover to mechanised electro-hydraulic face drilling with the Stomec rig. However before the Stomec rig could be fully developed trackless mining operations were generally aborted on the South African gold mines. Since the XLP trials at Lonmin and Amplats' Batophele Mine the mechanised face rig is now an option. In 2008 KAR was involved in the trials at Amplats' Amandelbult Mine with crawler mounted XLP equipment working in steep dipping conditions of 18° to 20° in a UG2 stope width of about 1,5 metres and the face rig and roofbolter performed well; these machines were developed by Atlas Copco, the same company that built the Stomec all those years ago; refer to the photographs of the drill rig and roofbolter in **Figure 8.2** and **8.3** respectively. In addition Sandvik have been operating XLP equipment at Lonmin for over ten years and more recently at Batophele. The mechanised face rig is therefore available and it is the opinion of KAR that such equipment should be introduced using the hybrid layouts first planned thirty years ago. Also, face cleaning can now be done by dozers, recently proven in the XLP trials, and therefore any final clean-up from the stope face (after the throw blast) can easily be carried out by dozers.

The reason XLP equipment was introduced in the first instance was the need to minimise dilution, which is always a concern with mechanisation in narrow reef workings. However, dilution from 'big end' access drives and roadways can be controlled (as can be seen in chapters 4 and 5 of this treatise). Therefore, if it can be assumed that when full mechanisation, including the use of a mechanised stope face rig, is employed at similar stoping widths as for conventional mining then there is every reason to believe that such a method, a type of advanced hybrid method, which is now totally mechanised would prove viable. At Amplats' Batophele Mine where the UG2 channel width is 0,85 metres the cost of producing platinum is below the average break-even price (working cost and maintenance capex), as can be seen in **Figure 8.4**, a bar graph generated by J.P.Morgan in January 2013. If the Batophele Mine is cost effective and the UG2 channel width at Lonmin's operations and at Amplats' Amandelbult are 1,25 metres and 1,55 metres respectively, then it must be a consideration to introduce mechanisation to these mines, in a room and pillar operation as at Batophele Mine. It also has to be noted that in the Figure 8.4 bar graph the costs at almost all Amplats' underground mines, Implats' mines and Lonmin operations at Marikana lie above the break-even price; all these operations, with only minor exceptions, employ conventional labour intensive methods. Notwithstanding that the information in the bar graph may not be definitive now, it still reflects the ranking of the various producers' operations and the conclusions remain the same.

To sum up, for the immediate future it would make good sense for the major platinum producers to introduce mechanisation at selected operations, as discussed. The introduction of mechanised mining to these operations will result in a loss of jobs, mainly unskilled labour, but

FIGURE 8.2

Remote XLP Crawler Mounted Drill Rig

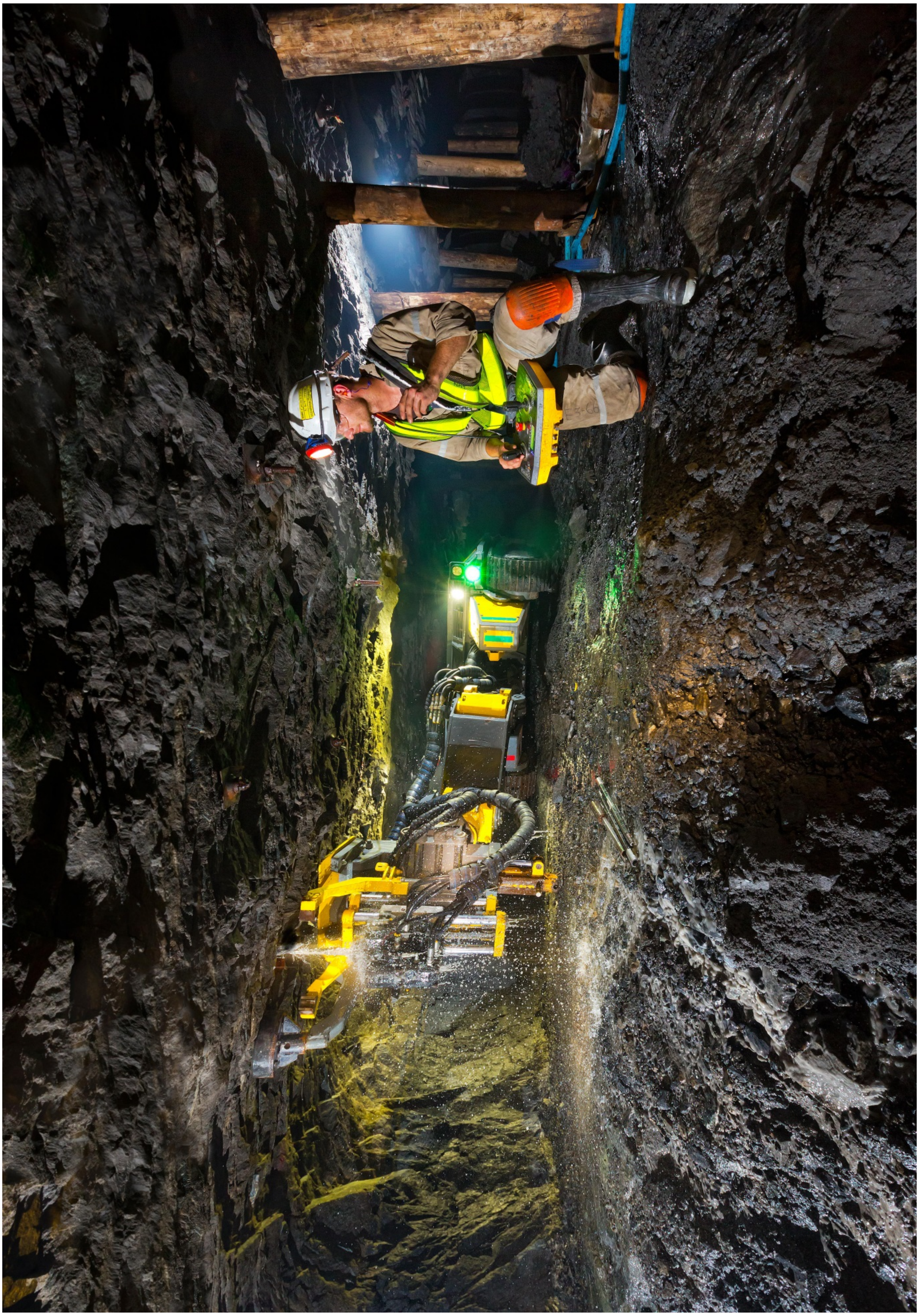

FIGURE 8.3

Remote XLP Crawler Mounted Roofbolter

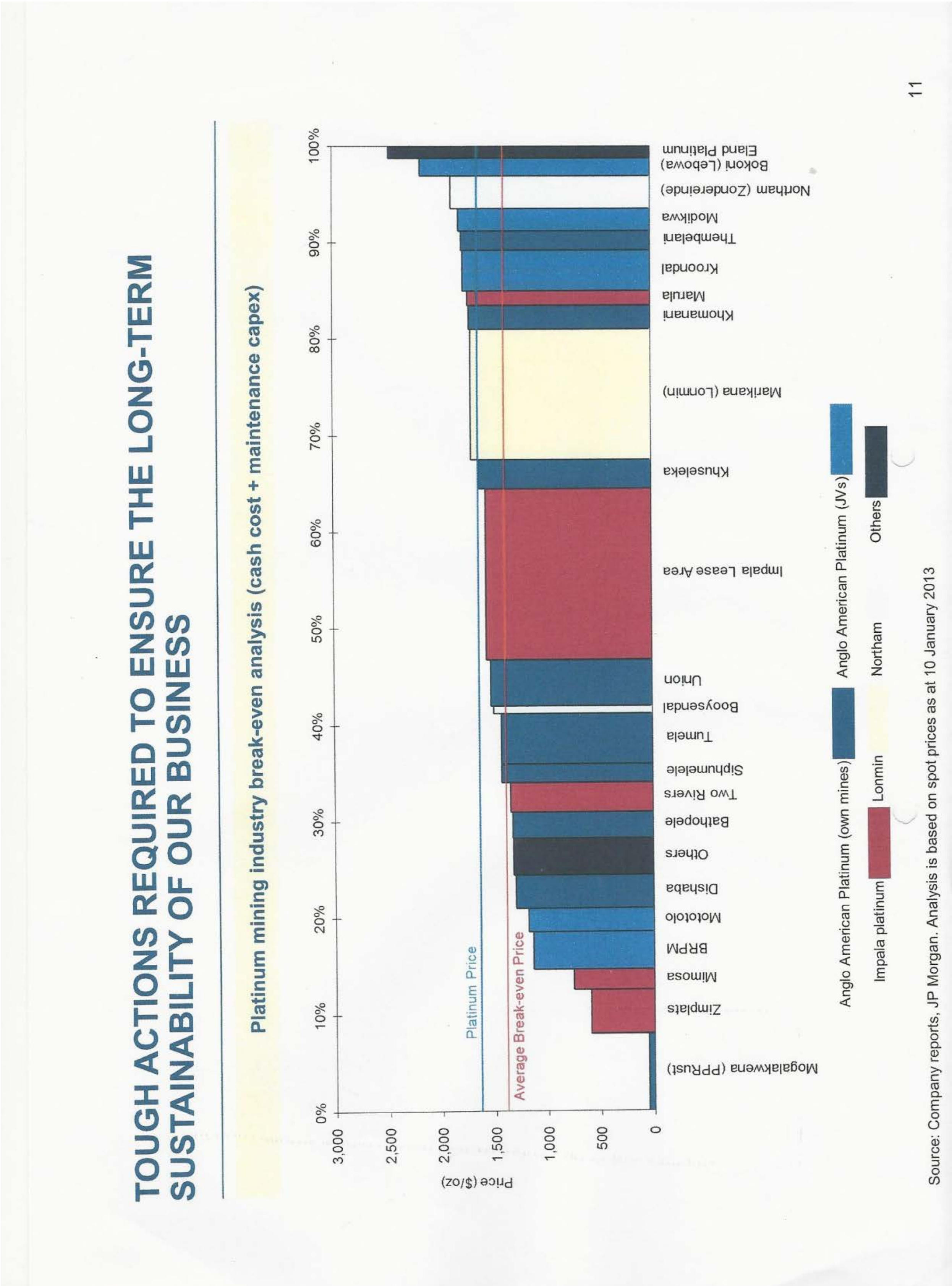

FIGURE 8.4

Bar Graph of the Platinum Mining Industry Break-Even Costs

it will generate opportunities to establish a more skilled labour force earning higher wages and bonuses. It will also generate profit for the companies which in turn will consolidate existing jobs and may even create new jobs. The year 2014 appears to be the watershed for underground platinum mining and it is believed that it is now the right time for a further serious attempt to introduce mechanisation at certain operations where the employment of CONOPS (continuous operations), for instance, could also be advantageous to any future mechanisation programme.

K.A.RHODES
May 2015