Safety in Mines Research Advisory Committee

Examine the criteria for establishing the small span small pillar concept as a safe mining method in deep mines

Volume 1 of 1

Dr. FSA de Frey

Mine Engineering Consultants CC

Prof. MF Handley

RCW Webber

Research Enterprise at University of Pretoria (Pty) Ltd

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Executive Summary

De Frey in his report of mine design alternatives for the DEEPMINE project found that the small span small pillar (SSSP) concept, as a mining method for deep mines, from a rock engineering point of view, had certain positive advantages. The main advantage was the possible control of seismic activities in deep mines (de Frey: 1999).

Because it had not been used in deep hard rock mining it was decided to examine the criteria put forward as possible showstoppers by comparing the other four mining methods presently used, with the SSSP design (Vieira, Diering and Durrheim: 2000). These were longwall with strike stabilizing pillars (LSP); sequential grid method (SGM); sequential down dip method (SDD); and closely spaced dip pillar method (CSDP).

In literature studies and examination of past reports and experience special attention was given to:

Mining criteria
Production-build up to 45 000m^2 stoped per month
Rock breaking
Blasting tight ends
Cast blasting
Cleaning of strike faces.

Ventilation criteria
Ventilating on a micro basis the 15m wide back stopes
Ventilating on a macro basis including bulk and in stope cooling

Rock engineering criteria
Pillar foundation strength
Evaluate the risk of introducing SSSP as a mine design.

The SSSP mine layout was tested by the numerical model used for DEEPMINE and proved to be superior to the other methods in getting to the 45 000m^2 per month.

It was found that by using a suggested cast blasting method the mining criteria of breaking and cleaning of 15m stope faces presented no problems. The holing of wide dip raises before strike stoping is done will address the dead end back stope problem. The comparison of the SSSP method with the other 4 methods proved it to be superior from a rock-engineering point of view with the following advantages:

1. 60% of the area mined will be trouble free.
2. With backfill it will probably be possible to get up to 85% or close to this level of extraction safely in a second pass.
3. Mining will be rate-independent (can mine at a high rate) and geology-independent.
4. Average pillar stress is considerably lower for bord and pillar than for the other methods with comparable extraction ratios (See table 3.6.1).

There are still reservations regarding pillar foundation failure and a new pillar design method is proposed.

The main objection remaining would be that it has not been tried at depth in deep gold mines.

Considering the positive findings in the non-rock engineering criteria as well as the very positive indications on the rock engineering side it is recommended that the problem area of pillar strength and stability, and the possibility of a pillar run be further investigated.

Preface

Because de Frey has had a considerable amount of experience in deep gold mines he would like to share it with persons interested in this research work. It adds to the evolution of rock engineering in the deep gold mines.

He started work at Crown Mines of the Rand Mines group in 1952. At that time Crown Mines and East Rand Proprietary Mines were the deepest hard rock mines in the world. Seismic events, then commonly known as rockbursts, were almost occurring on a weekly basis.

Mining at shallow depths used pillars, ordinary timber props with headboards and skeleton mat packs for support. Where reclamation of pillars was done in old worked out areas, it was surprising to see how well these areas were kept open. Surface pillars on the outcrops of the South and Main Reef deposits kept the surface protected from caving. At the same time areas were made easily accessible for pillar reclamation, sweeping and vamping operations. Pillars on the up dip side of the tunnels in the Kimberley Reef deposit tended to slide into the tunnels on reef. The reason was the weak schist in the immediate footwall of the deposit. The pillars were therefore as effective as the strength and continuity of the footwall allowed it to be. The pillars were relatively small but where the hanginwall (h/w) and footwall (f/w) were competent, succeeded as regional and local support over fairly large spans of mined out areas.

As stoping approached depths of 1000m it was soon realised that the small pillars would not stand up to the increase in stress. Pillars tended to collapse or punch into the h/w and f/w, causing unsafe conditions. The pillars were supporting extensive spans of mined out areas. Use of small pillars was discontinued.

At depths of 2000m below surface, at Crown Mines the support of stopes consisted of timber props with headboards, 9 pointed and solid timber mat packs, waste filled matpacks and waste walls. The waste was blasted from the immediate h/w or f/w above the waste walls. At that time it was considered very effective for keeping travelling ways and escape ways open in the stoped area. This was especially so when remnants were being mined. Whether the gullies should be cut in the h/w or f/w was also debated.

By 1971 East Rand Proprietary Mine (ERPM) was the deepest mine in the world. Mining at Crown mines at depth had been stopped for economical reasons. Western Deep mine was then the second deepest. ERPM at depth was using longwall mining. The same type of support discussed above for Crown Mines was the standard, being in the same Rand Mines stable. The management of Rand Mines together with the Chamber of Mines Research Organisation were doing a lot of research into the seismic event phenomenon.

The task of rock mechanics became most important as seismic events increased as mining was taking place at greater depths.

Different types of timber mat packs such as the sandwich and composite mat packs were being experimented with. Sandwich packs consisted of alternate layers of concrete bricks and timber. The composite pack had concrete bricks placed in between the timber members of the pack. These concrete filled packs did excellent work as immediate face area support as stoping was started from original raises. As the span between the advancing faces increased the closure rate increased and the packs disintegrated and lost their effectiveness. It was clear that support at depth must never lose its ability to resist closure.

The deep mines then decided to make use of large strike stabilizing pillars to resist closure and keep the energy release rates as low as possible. Strike pillars were 20 to 30m wide on dip with spans of 200m between pillars. The aim was to still get an 85 % extraction of the reef. At first the rate of seismic events seemed to be under control. As the area extracted on strike increased it was found that rockbursts were occurring in the back areas in the vicinity of the strike stabilizing pillars. The pillar size was increased to measure 40 meters on dip. To maintain the 85% extraction the span on dip between strike pillars were increased accordingly. Seismic events were still taking place.

At that time the rapid yielding hydraulic steel prop was introduced to the mining industry. While management and maintenance controls were kept tight a considerable amount of success was experienced. This was especially the case where stopes were easily accessible for rescue operations and the restart of stoping operations. Poor management and utilisation caused the use of these props to run into disfavour. The elongate was then tried and is at the moment favoured for local support.

Backfill was tried and found to be relatively successful to help in reducing the energy release rates. The time taken to open stopes and bring them back into production after a seismic event was substantially reduced. Inability to effectively manage and control the placing of backfill is causing it to lose favour in the deep mining industry as seismic events are still taking place and causing loss of life or serious injuries.

After intensive research and analysis of the performance of gold mines for the period 1970 to 1987 de Frey came to the conclusions that mine engineers should concentrate on solving technological problems which if solved will present solutions to the rising fixed asset and production costs (de Frey: 1989). Too much time and energy was still being given to solving smaller problems that make very little contribution to the support of synergistic objectives.

The other major problem that de Frey researched was the unacceptable rate of loss of life and limb caused by seismic events in the gold mining industry (de Frey: 1993). The then Government Mining Engineer made available the reports of mine inspectors into fatalities caused by seismic events in the Carletonville mines. Having scrutinised the reports it was found that one of these mines had the most fatalities caused by seismic events. The investigation concentrated on 16 reports from this mine.

It was found that there were still seismic activities in and around strike stabilizing pillars where mining activities had taken place 2 years ago. Were the stabilizing pillars in fact retarding the process of the surrounding rock mass reaching a state of equilibrium?

The hypocentres of the seismic events were found to be ahead and above and below the reef horizons. Most of the fatalities were found to be in the immediate vicinity of the advancing stope faces. During rescue and repair operations de Frey found that the immediate h/w and f/w as well as the face itself had been ejected into the mined area. At ERPM it was found that T-props that were placed right on the face were merely kicked out and led de Frey to believe, that no support right on the face, where fatalities were taking place, could withstand the severity of a seismic event .The mine engineer therefore has to concentrate on prevention rather than on the cure. Remove or prevent the cause rather than doctor the symptoms. The objective should be to mine in such a way that the equilibrium and stability of the rock masses in the vicinity are minimally disturbed.

At the request of de Frey, Ryder by using the BEPIL computer simulation program concluded that squat pillars at great depths had a number of extremely attractive features. These were: minimal regional disturbance to the rock mass with consequently low closure, energy release rates (ERR) and field stress levels. Other mining layouts exploiting the same principles as the regular room and pillar grid are conceivable. Choice of an optimal layout would be dictated by mining considerations such as costs, cleaning, ventilation, face advance rates gully dilution and so forth. Ozbay, Ryder and Jager agreed that further research into the matter should be done (Ozbay,Ryder and Jager:1995). Esterhuizen came to the same conclusion (Esterhuizen: 1997). Leach was still unsure of what h/w and f/w conditions would be like and suggested larger sized pillars and bords (spans). As part of a report for the DEEPMINE project it was found that pillars spaced small distances apart, so as to have small spans on strike and dip, from a rock engineering criteria point of view, offered a vast amount of advantages and very few disadvantages (de Frey: 1999). The possibility of poor h/w conditions was put forward as a possible problem area (Leach: 1999).

SIMRAC suggested that the matter should be further pursued by a SIMRAC research project. A proposal to ‘Examine the criteria for establishing the small span small pillar (SSSP) concept as a safe mining method in deep mines’ was approved.

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Glossary of terms and abbreviations

°C	Degrees Centigrade
ACP	Air cooling power
kg	Kilogram
kg/s	Kilogram per second
kg/m^3	Kilogram per cubic meter
kW	Kilowatt
m	Meter
m/s	Meter per second
m^3/s	Cubic meter per second
RAW	Return air way
x-cut	Cross cut to reef
h/w	Hangingwall
f/w	Footwall
ERR	Energy Release Rate
ESS	Excess Shear Stress
FW/HLGE	Footwall haulage
EDD	Electronic Delay Detonator
cm	Centimetre
ms	Milliseconds
mm	Millimetre
UCS	Uniaxial Compressive Strength
MPa	Mega Pascal
MJ/m^2	Mega Joules per square metre
APS	Average Pillar Stress
LoM	Life of Mine

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1 Introduction

During research work done for the DEEPMINE project it was found that the use of the small span small pillar concept as a mining method in deep hard rock mines, from a rock engineering point of view, presented many advantages and very few disadvantages (de Frey: 1999). A proposal for further research work was approved by SIMRAC.

The Objective
The objective for GAP 828 was to examine the criteria for establishing the small span small pillar (SSSP) concept as a safe mining method in deep mines.

The problem statement
As gold mines are getting deeper the rate and severity of seismic events are expected to increase. The task was to prove to the mining industry that the small span small pillar (SSSP) concept as a mining method would present safer working conditions. Also, that due to fewer delays and losses, the performance of the mines could improve.

The background
During research work into the criteria for mining at depths of 3000m below surface, it was found that from a rock engineering point of view, the SSSP mining method presented advantages such as low energy release rates (ERR), low excess shear stresses (ESS), the ability to pre-develop in the footwall of the ore body as close as 25m normal to the ore body, backfilling was optional and not a prerequisite and bracket pillars were unnecessary against faults or dykes (de Frey: 1999).

The most serious obstacles foreseen were:

The SSSP concept had not been tried at depth in hard rock mining
The effect of the pillars on the h/w in the immediate vicinity of the pillars
Ventilation of the 15m dead end faces as possible back stopes
Tight end blasting of stope faces
Cleaning of the strike spans

To investigate the problems set out above the following objectives were targeted:

Do a literature survey of research and work done into SSSP mining so as to compare it with other methods of mining.

Study reports forwarded to or issued by the DEEPMINE project regarding seismic events, rock engineering, ventilation and economic viabilities of the other mining methods and layouts presently in use in the deep gold mines of South Africa

Do a similar computer simulation of the SSSP method as those done for the other mining methods investigated in the DEEPMINE project carried out by Graphic Mining Solutions International (GMSI).

The Methodology

To achieve the above objectives the following procedure was followed:

The project leader and rock engineer studied all relevant literature and material on past and present mining methods so as to establish whether the SSSP mining method is superior to the other methods from a seismic and rock engineering point of view.

The project leader, rock engineer, and ventilation engineer studied all reports forwarded to or issued by the DEEPMINE project on seismic events, rock engineering, ventilation, economics and mining methods presently being researched and practised.

GMSI used the same Iponeleng Mine computer simulation program used for the DEEPMINE project to compare the economic results of the SSSP method with those of the other methods researched for the DEEPMINE project.

2 Mine layout

It is necessary for the sake of clarity to give the details of assumptions and criteria used in the report forwarded to DEEPMINE (de Frey: 1999)

2.1 Systems criteria

The systems criteria (issues) that have been identified during workshops were:

- Rock engineering
  - Safety
  - Regional support
  - Local support
- Non-rock engineering
  - Environmental
  - Access
  - Transport
  - Mining/rock breaking
  - Economics

At workshops held by the different project leaders and the other stakeholders it was clear that from a rock-engineering point of view there were major advantages in applying the small-span small-pillar principle as a mining method.

The major advantages were:

- Major improvement in safety as a result of less seismic activities
- Scattered or concentrated mining or a combination of both could be used
- Long-wall mining using the bord and pillar stoping system would be an added advantage as this made concentrated mining possible
- Pre-development so as to determine and define geo-technical disturbances as well as ore reserves without the fear of losing tunnels is possible
- All of the pre-development and excavations within 50m and deeper of the reef horizon would be subjected to virgin stresses only
- Development-ends as close as 50m to the ore-body results in an improvement in the square metres for stoping made available for every metre developed in waste rock. This results in major savings of cost.

In spite of the above advantages the major problems (showstoppers) put forward were

- Environmental
  - Ventilating dead end panels
- Mining/rock-breaking
  - Cleaning of strike panel dead ends
  - Tight end blasting causing poor hanging wall conditions
  - Slow production build up
  -

Bord and pillar as a stoping method is discussed and evaluated in terms of the above criteria. It indicates how a production target of 45000m^2 per month could be stoped using the small pillar small span mining method whilst addressing the showstoppers.

What follows is a case study of a layout that uses bord and pillar as a method to address the suggestion and the showstoppers mentioned above.

2.2 Assumptions

Figure 2.2.1 illustrates a schematic layout to be considered having dimensions and information as set out below.

Assume that the cross-cuts (x-cuts) from the shaft on every level intersect the area to be mined in the middle of the strike distance to be mined. Three blocks will be available on strike on either side of the x-cut from the shaft. The blocks are numbered according to the level and the order in which they become available on the east and west of the shaft x-cut. For example, the first block east of the shaft x-cut, on the 3620m level below surface, will be referred to as 3620.1E and on the west 3620.1W. Two blocks per level on 6 levels on both sides of the connecting shaft x-cuts will be stoped simultaneously.

Production	45000m^2 per month
Distance below surface	3500m
Extraction	75%
Face advance per month	15m
Pillar dimensions	15m x 15m
Panel (face) lengths	15m
Relative density of rock	2,78t/m^3
Stope width	1,25m
Dip of ore body	26 degrees south
Strike of ore body	east to west
Vertical distance between levels	120m
Area to be mined	1260m on strike and 1440m on dip
Dimensions of RAW	3,5m wide x 3,5m high
Dimensions of FW/HLGE	same as for RAW
Distance between x-cuts on strike	220m
RAW to FW/HLGE x-cut	40m
X-cut FW/HLGE to reef	105m
Dimensions of boxhole	2m x 2m x 40m long
All development ends advance	2m per blast

Wide reef raises are classified as stoping.

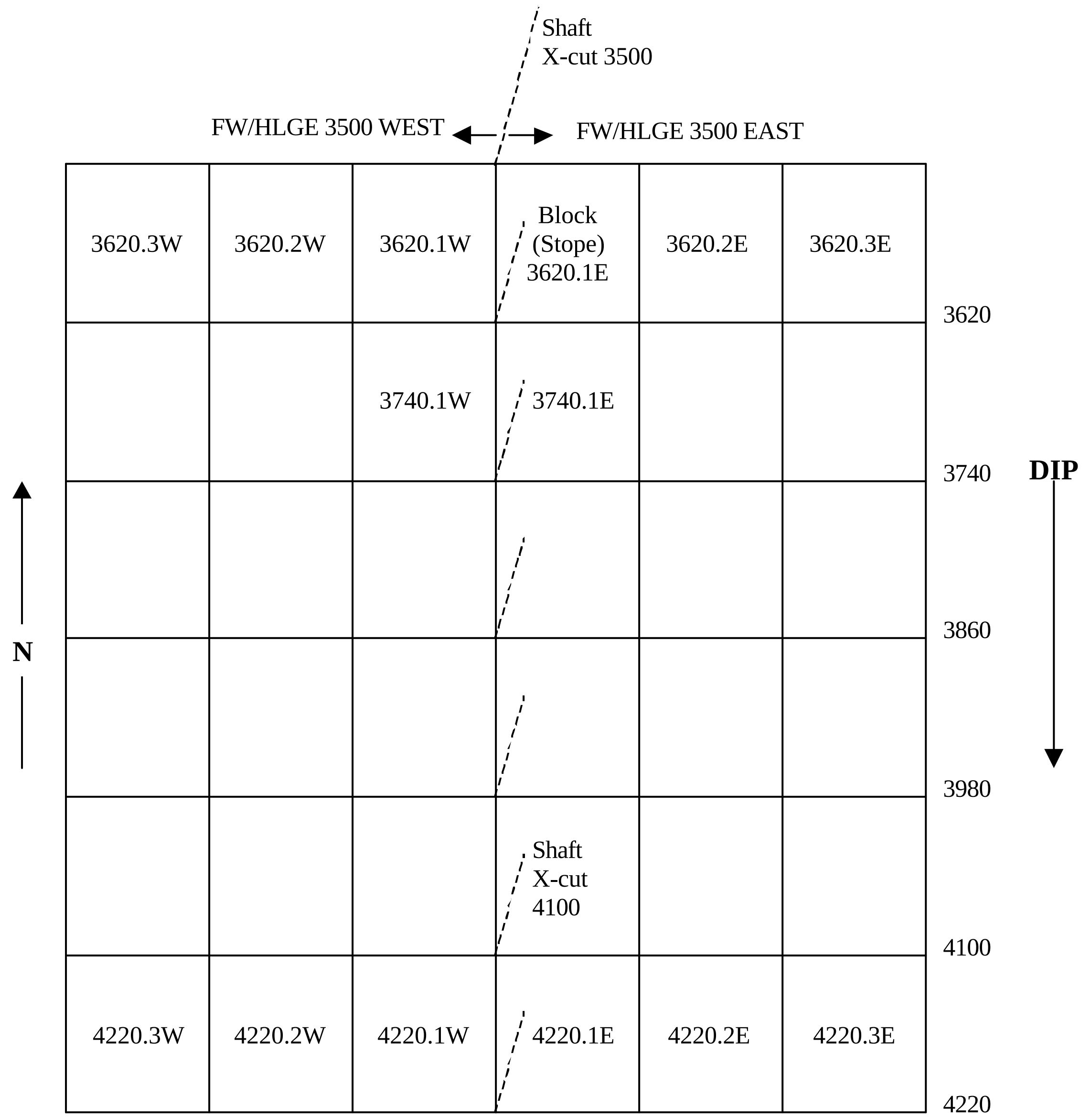


Figure 2.2.1: Schematic layout of area to be mined.

2.3 Development

The development detail and schedule is set out below.

2.3.1 Development layout

Figure 2.3.1 illustrates the general layout of a block of ground measuring 210m along strike and 240m along the dip on the plain of the reef.

All ends are pre-developed to prepare for stoping operations. The development required per block consists of the following:

RAW	220m
FW/HLGE	220m
RAW to FW/HLGE x-cut	40m
X-cut FW/HLGE to reef	105m
Four boxholes	160m
Total	745m

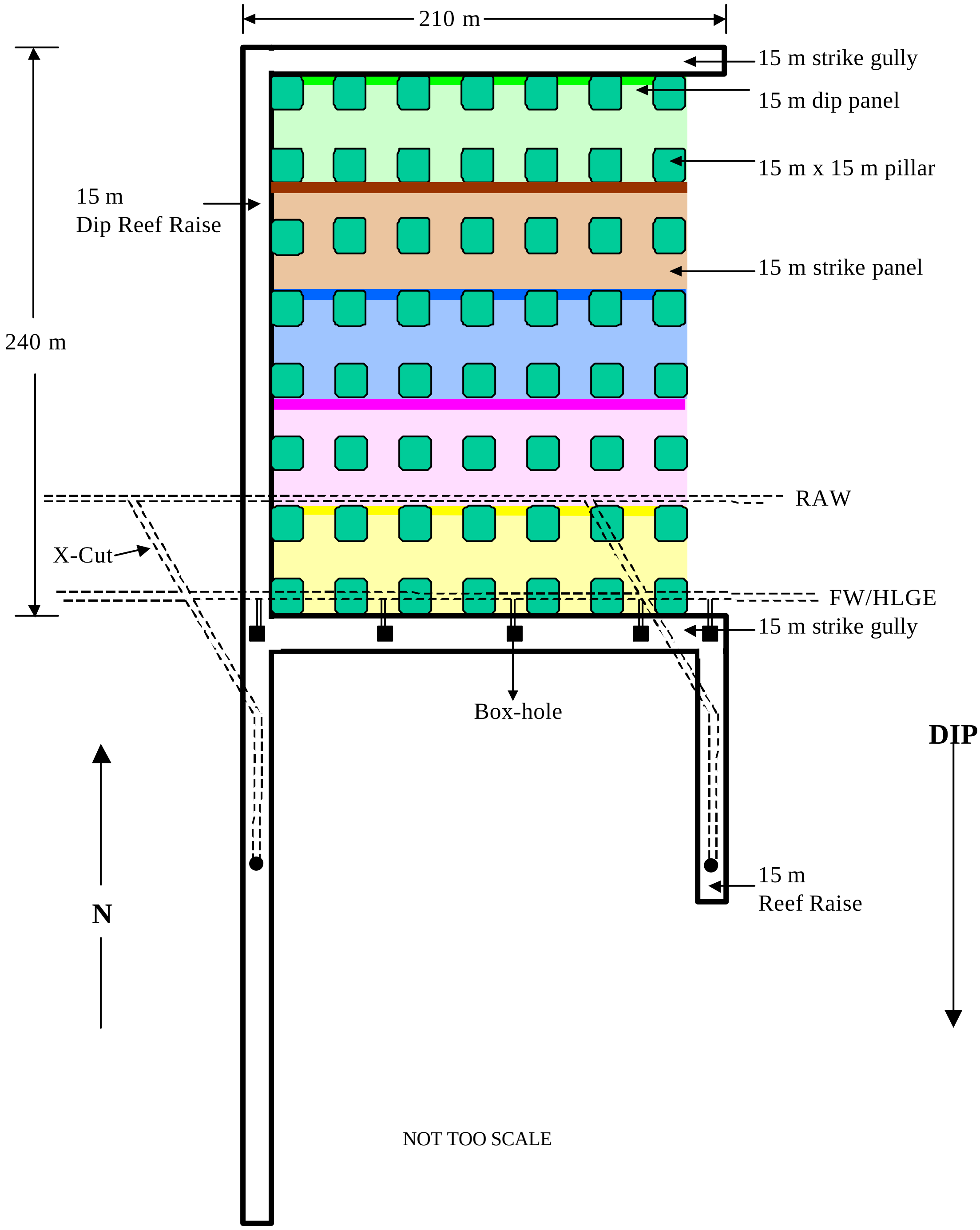


Figure 2.3.1: General layout of an up-dip stope block.

Figure 2.3.2 illustrates a cross-section through one level and Figure 2.3.3 the general development and stope layout over 3 levels.

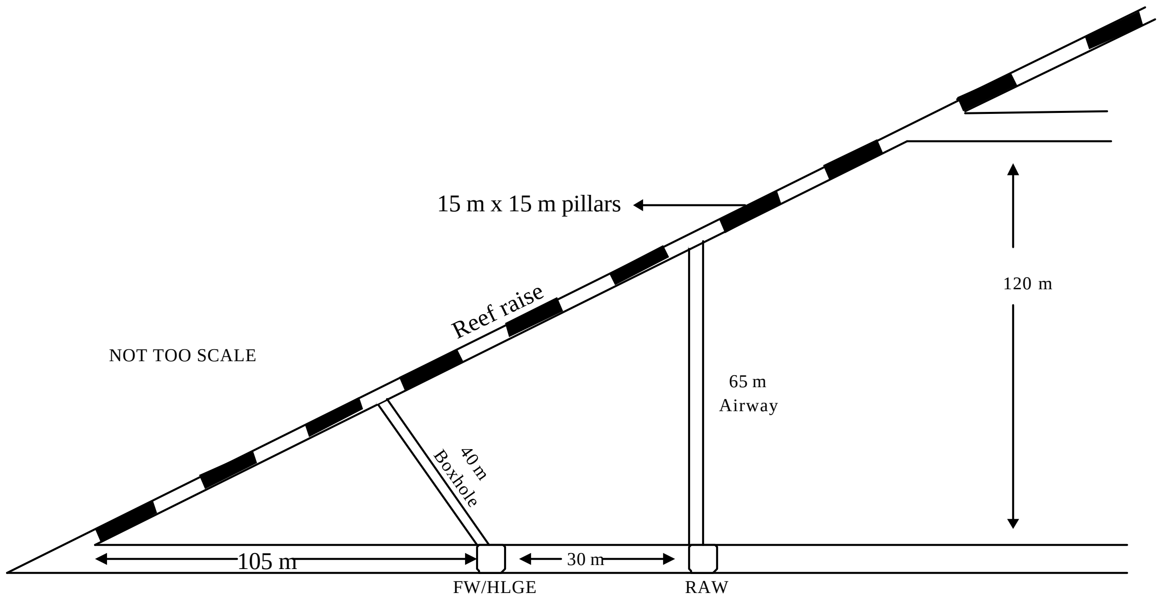


Figure 2.3.2: Cross-section through a level.

A block consists of 50400m^2 ore of which 37800m^2 stoped, represents 75% extraction.

The 745m of waste development makes available 37800m^2 of ground to be stoped, which means that 1 m development on waste provides 50.74m^2 of reef for stoping.

The 745m development on waste produces 21700 tons of waste rock. In its broken form it will, as a result of the swelling factor, fill a space of 21700/1.6 = 13562m^3 At a stope width of 1,25m an area of 37800m^2 will make available a space of 37800m^2*1,25m = 47250m^3. Waste from 745m development will then fill (13562/47250x100) 28% of the stoped area.

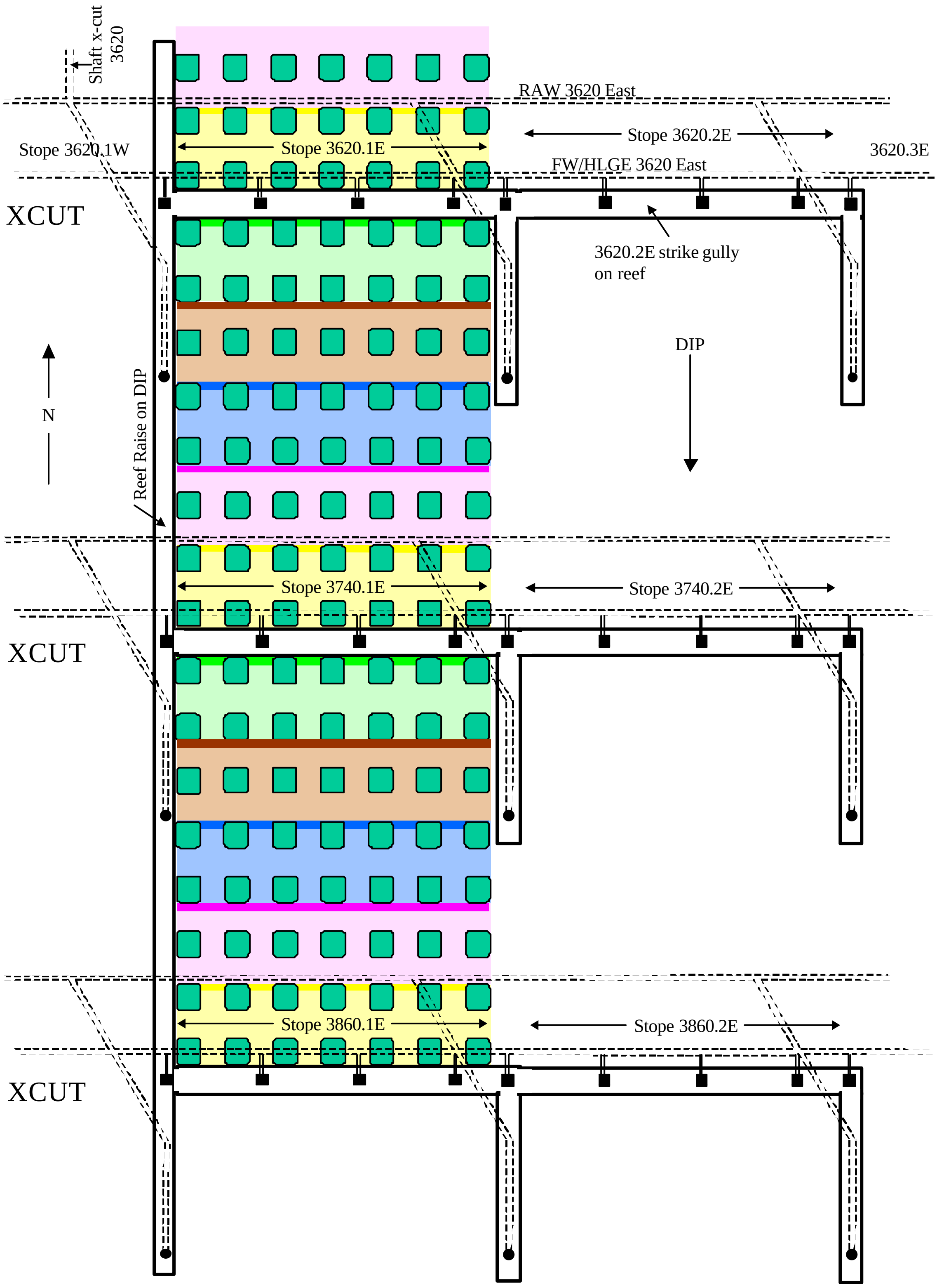


Figure 2.3.3: General development and stope layout over three levels.

2.3.2 Development schedule

Figure 2.3.4 illustrates the schedule for development and stoping operations and Figure 2.3.5 represents a schematic layout of the sequence.

2.3.2.1 Preparation of block 3620.1East

The details for this block are set out in Table 2.3.1 below.

Table 2.3.1 The details of block 3620.1East**.**

	Metres	Days	Cumulative Days
X-cut 3620.1E to FW/HLGE	40	20	20
FW/HLGE to X-cut 3620.2E	220	110	130
RAW to X-cut 3620.2E	220	110	112
X-cut 3620.2E from FW/HLGE to reef	105	53	183
Reef raise 3620.2E to strike gully	85	43	226
Four box holes and strike gully on reef	160	110	226

Reef raise 3620.1E will be wide raised from the 73rd day, for a distance of 240m, taking 120 days and will therefore be completed on the 193rd day.(See figures 2.3.4 and 2.3.5)

The 3620.1E stope strike gully and connecting box-holes will be completed between the 116th and 226th days. Block 3620.1e will therefore be ready for stoping operations after 226 days from the time that the x-cut from the shaft intersects the reef horizon.

The 5 stopes below 3620 level will also be ready after 226 days/24 days/month=9.4 month say 10 months. These 6 stopes, together with the 6 on the west down to 4220 level will therefore be ready for stoping after 10 months.

2.3.2.2 Preparation of block 3620.2East

The details of block 3620.2East are set out in Table 2.3.2.

Table 2.3.2 The details of block 3620.2East.

	Days	Cumulative Days
FW/HLGE from X-cut 3620.2E to X-cut 3620.3E	110	240
X-cut 3620.3E from FW/HLGE to Reef	53	293
Reef raise 3620.3E to strike gully	43	336

No reef raises above the strike gully are required as the stoped area of 3620.1E will be available as a return-airway. Stope 3620.2E together with the other stopes down to 4220 level on the east and west of the x-cuts will thus be available for stoping after 336 days. It would therefore take 336-226 = 110 days from the time that one block is ready to stope to the time that the next block on strike could be ready for stoping. Block 3620.3E together with the other blocks on the 3E and 3W line down to 4220 level should therefore be ready for stoping on day 446.

Shaft x-cut 3620 to RAW	Complete
X-cut RAW to FW/HLGE	20 days
RAW to X-cut 3620.2E	112 days
FW/HLGE to X-cut 3620.2E	130 days
3620.2E X-cut from FW/HLGE to Reef	183 days
3620.2E reef raise to strike gully	226 days
Block 3620.1E	Stoping starts on 226 days and ends after 586 days
FW/HLGE from 3620.2E X-cut 3620.3E x-cut	240 days
3620.3E X-cut from FW/HLGE to Reef	293 days
3620.3E reef raise to strike gully	336 days
Block 3620.2E	Stoping 336 days to 696 days
FW/HLGE from 3620.3E X-cut 3620.4E x-cut	350 days
3620.4E X-cut from FW/HLGE to Reef	403 days
3620.4E reef raise to strike gully	446 days
Block 3620.3E	Stoping to 803 days

***Figure 2.3.4**:* Schedule of development and stoping.

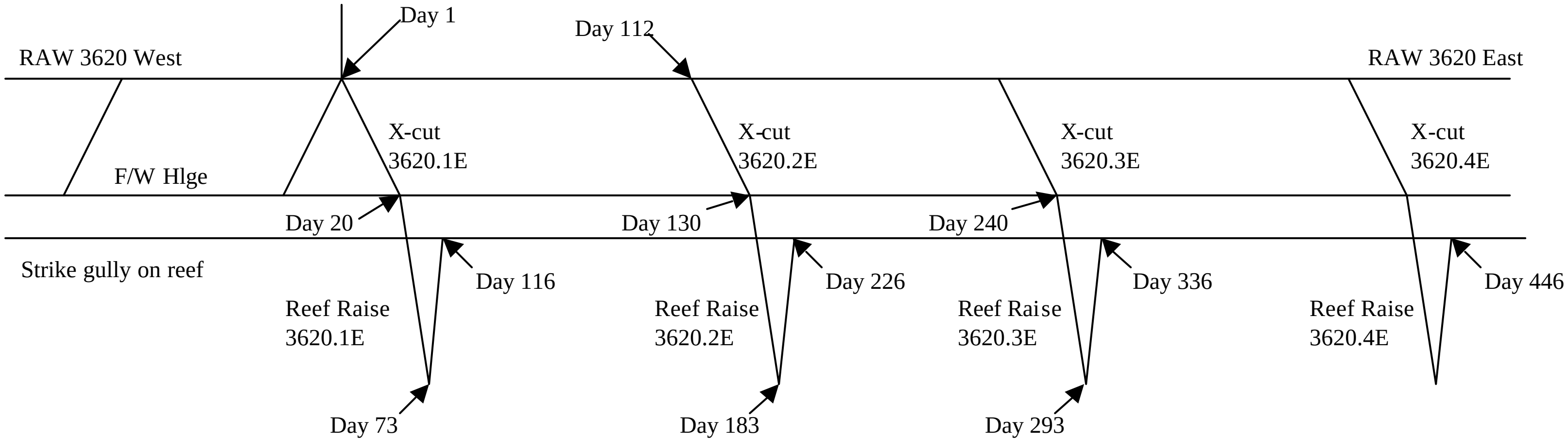


Figure 2.3.5: Schematic diagram of the development schedule

2.4 Stoping

The stoping procedure is discussed next.

2.4.1 Stope layout

Figure 2.3.1 shows the layout on reef of a block of ground measuring 210m on strike and 240m on dip. Assuming a face advance of 15m per month the block will be depleted in 15 months. Producing 37800m^2 of reef in 15 months requires a production of 2520m^2 per month per stope (block). Producing 45000m^2 per month would require 18 blocks being mined at the same time. Figure 2.3.3 illustrates 3 stopes on dip being stoped simultaneously.

2.4.2 Stoping schedule

Figure 2.3.4 sets out the time schedule for mining out three consecutive blocks on strike.

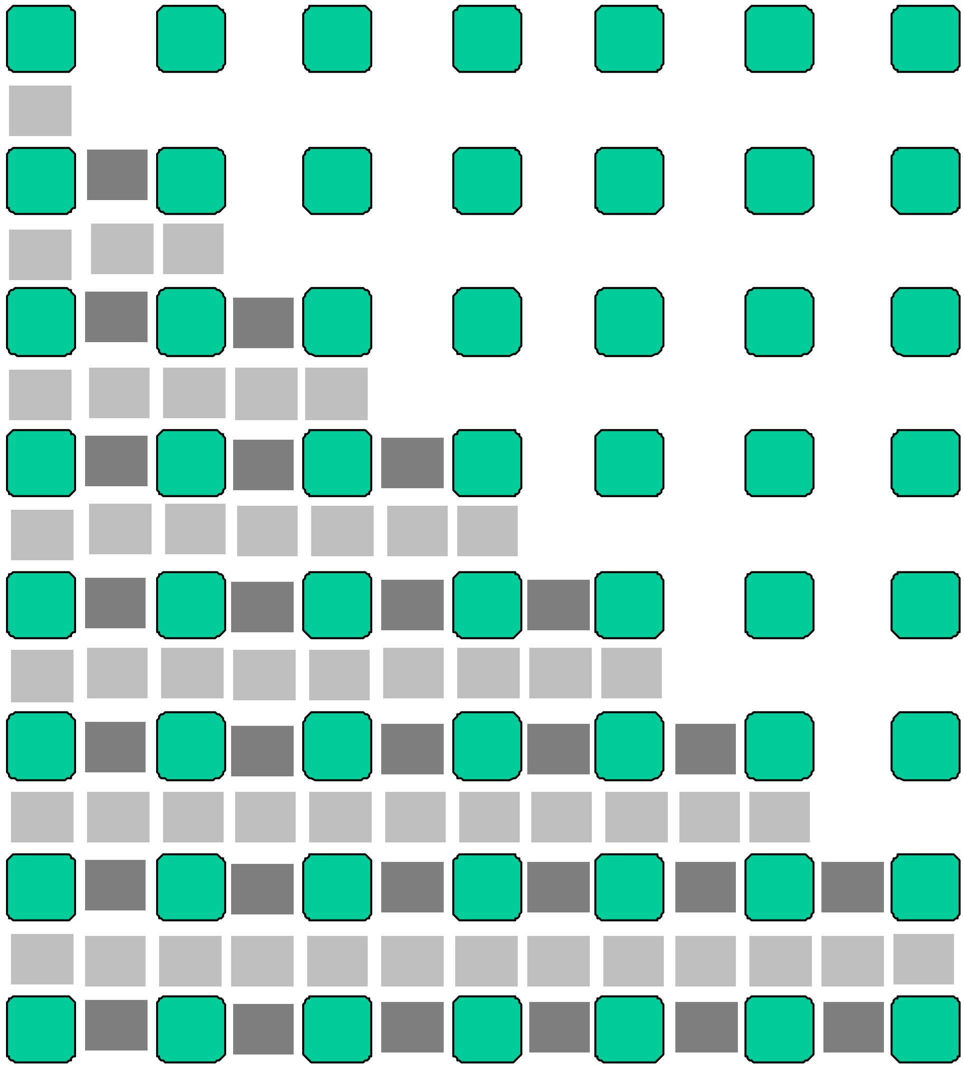
Block 3620.1E will be available for stoping after 226 days/24=9.4 months say 10 months
Block 3620.2E will be available after 336 days/24=14 months.
Block 3620.3E will be available after 446 days/24=18.6months

A new block on strike can be ready every 5 months.
A stope will be mined out every 15 months from the time that it is ready.

Block (stope) 3620.1E is depleted after 25 months from the time that development was started from the shaft x-cut intersection of the RAW.
Block 3620.2E will be out after 30 months and 3620.3E after 35 months. The same procedure will be taking place west of the 3620.1E line and on the levels below. The area of 1260m on strike and 1440m on dip will, after say 3years, be mined out.

It is important to note that blocks 3620.1East, 2East and 3East from day 446 could be stoped simultaneously

Figure 2.4.1 illustrates the overhand method of stoping .The only difference from the up-dip method is that it would take longer before the face reaches full capacity. In a high percentage payable area where the bottom of a long-wall is ahead of the upper faces the necessity for the raising from the x-cut-reef intersection to the strike gully falls away. The reason is that the top of the lower overhand face has already stoped passed this position. Panels mined up-dip are shown as horizontal lines and panels mined on strike are shown as vertical lines.



NOT TO SCALE

Figure 2.4.1: The overhand stoping layout.

2.4.3 Stope drilling and blasting

Figure 2.4.2 illustrates a drilling pattern for advancing the stope face 0,625m per blast. Sixty-four holes drilled at an angle of 37 degrees to the face will cast the rock back for a distance of 8m to 14m from the face. Two machine operators and 2 machine operator assistants will be able to drill the 85m in a shift.

As there was a certain amount of doubt as to the application of the suggested method John Truter of African Explosives Limited was approached to investigate the proposed method. He could not see why this method would not give the expected throw (cast) blasting. His report follows.

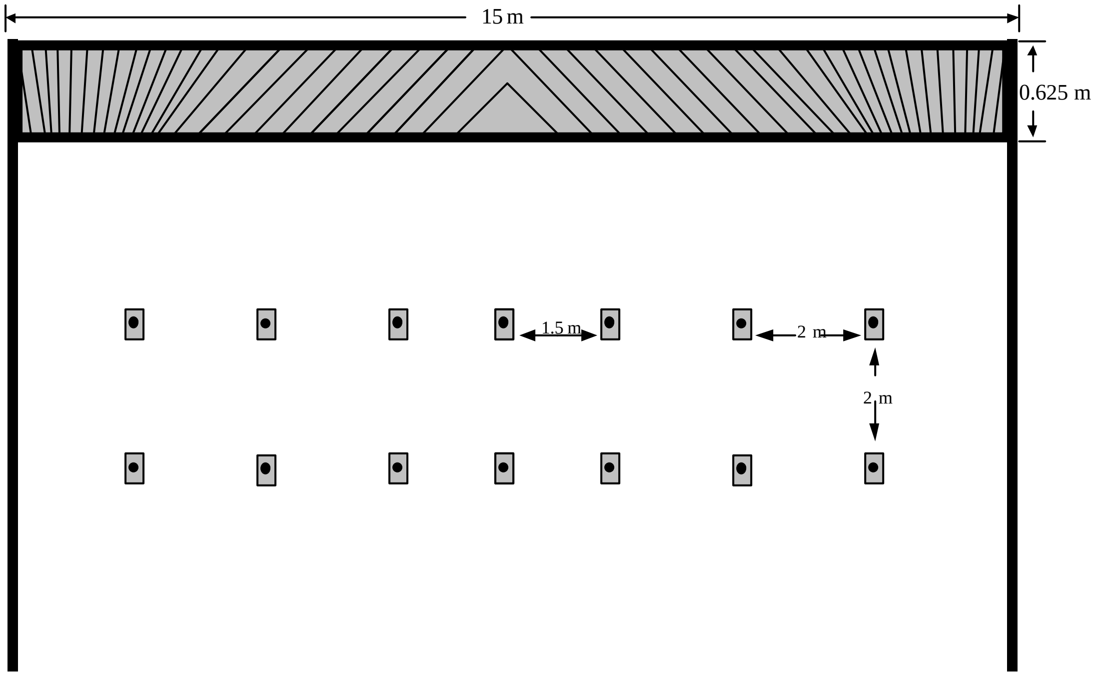


Figure 2.4.2: Drilling and blasting panels for 15m panels.

With due consideration to your report we would like to note the following:

1. **Advance per blast** – two options have been considered
   0.625m which equates to 15m/month over 24 shifts &
   0.89m that equates to 90% of the effective hole length of 99cm.

2. **Drill pattern** – a normal box pattern should give a better throw of the broken rock and would also make the connection of shock tube products easier. If hand held drilling machines were used the operators would have one set-up per two holes and would be inclined to produce better results.

3. **Type of fuse** – the casting of the broken rock away from the face will necessitate the use of an initiation system that has a very good sequential blasting profile. The best products in this range are the EDDs (Electronic Delay Detonators) while shock tube products will have to be tried and tested to check their results in this application. Inter-hole delays should be in the order of 125ms.

4. **Charge length** – the 37-degree angle of the drill holes will mean that the collar area need not be charged as normal. We suggest that the bottom 66cm of the hole be used for charging up with explosives in all hole lengths from 1.05 to 1.1m.

5. **Burden** – the average equivalent burden works out to be 60cm, which would normally be acceptable, but trials should be done to ensure that this is effective on the rock mass being mined.

6. **Explosive charge** – using a charge length of 66cm, a hole diameter of 34mm and a compaction of 21%, Powergel 813 (25*200) will require 4 sticks of explosive which will result in powder factors of 1.95 & 1.37kg/m^3 for the two different advances/blast.

7. **Stemming** – good quality stemming should be used in all cases and the length should be plus 50cm.

8. **Blasting costs** – using costs of R10 per shock tube unit, R20 per EDD and R0.55 per explosive cartridge will result in costs of about:

Table 2.4.1: Explosive costs.

Costs	**Costs (R/m²) for 0.625m advance/blast**	**Costs (R/m²) for 0.89m advance/blast**
Shock Tube	65	46
EDD	118	83

According to Eric Vascatto of Selective Blast Mining, holes drilled at right angles to the face, using a selective blasting system, will be able to cast rock back for a distance of 8m. This would mean that face preparation could commence almost immediately after the re-entry period. It would also clear the face back far enough so that face scraping will not be necessary.

2.4.4 Stope cleaning

All cleaning will be done during the drilling shift. Scraper-winches will clean the up-dip panels. Strike panels will require little cleaning by scraper-scoops as most of the rock will be cast on strike into the dip scraper lines.

Figure 2.4.3 Illustrates a strike gully with dimensions of 1m in width by 1m depth connecting the box-holes. Cleaning of the strike gully will be done by snatching of the ropes of the scraper winches used for down-dip cleaning.

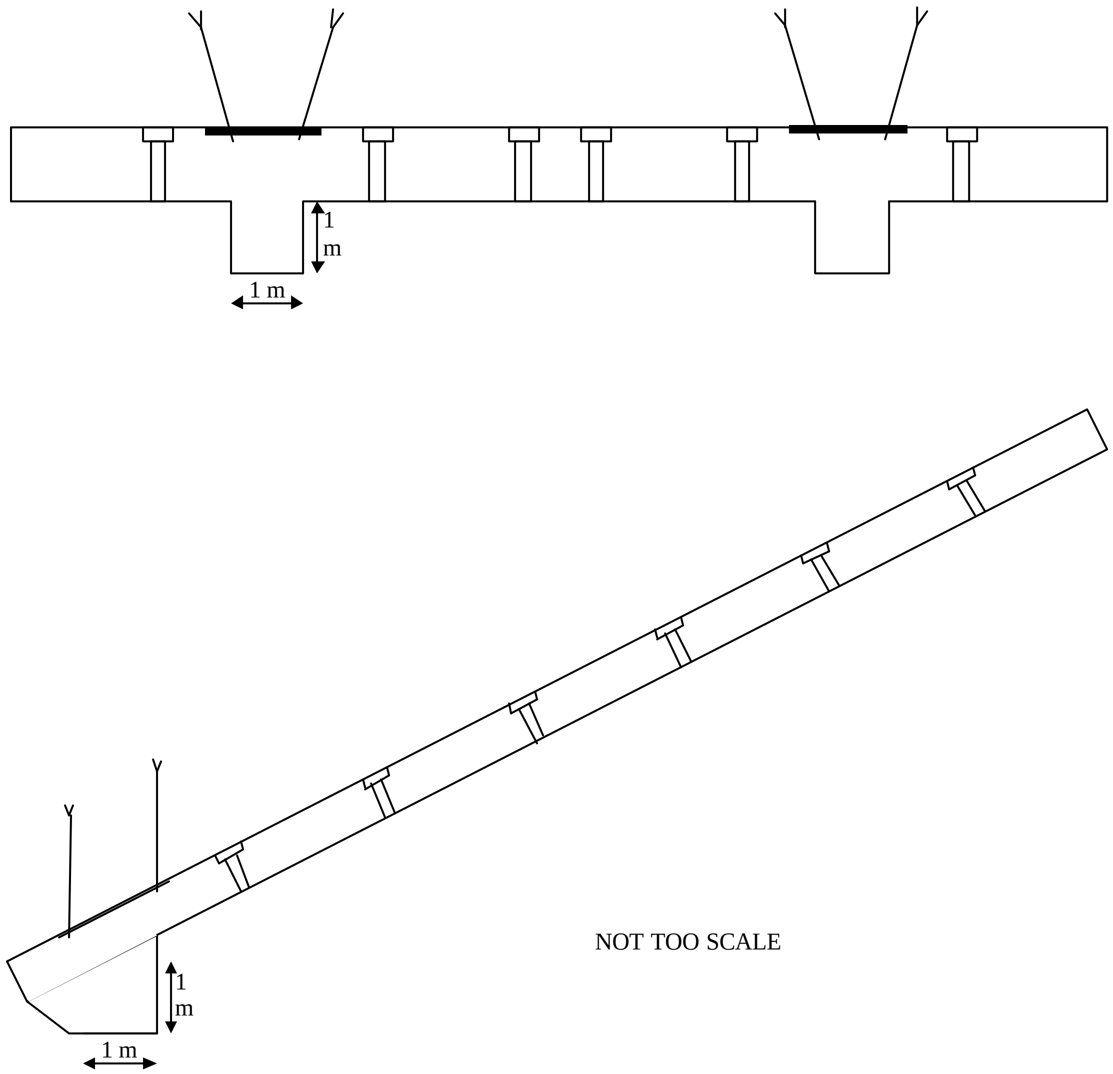


Figure 2.4.3**:** Gully and service way support.

Sweeping will be kept up to date by sweeping strike panels to within one line of pillars of the up-dip advancing face. Two winch drivers per winch will be used.

2.4.5 Stope support

Figures 2.4.2 and 2.4.3 illustrate the placing of props or elongates with headboards. Depending on conditions skeleton packs should be adequate along gullies or service ways. Directly above the gullies and service ways rock-bolts with straps across gullies will support the hanging.

2.4.6 Stope workers allocation

The workers allocated per 15m panel are the following:

Team leader	1
Scraper winch operators	2
Drilling machine operator	2
Drilling machine operator assistant	2
Miner-assistant	2
Support installers	2
Multi-skilled (drill, winch, pipes, sweeping)	9
Total	20

The 6 dip panels will have 120 in stope employees. One stope will produce 2520m^2 per month/120 employees=21m^2 per employee per month for a 0,625m face advance per blast. Although the figures might not be 100% comparable it is worth noting that, should 20m advance per month be achieved the efficiency will improve to 28m^2 per month per person working in the stope. For 1m face advance per blast the efficiency improves to 33.6m^2 per person for a face advance of 24m per month.

3 Rock engineering

At workshops held at Miningtek for the DEEPMINE project, it has become obvious that the rock engineering criteria dictates the evaluation of the other criteria involved in the deep mine project. At these workshops the following criteria were investigated.

3.1 Safety

The present poor safety record due to seismic events and falls of hanging wall is no longer acceptable in the mining industry. Numerical modelling indicate that the low-energy-release-rates (ERR) and excess-shear-stresses (ESS) found when using stable squat pillars with small spans between pillars should result in less seismic activity with the possibility of improved hanging wall conditions (de Frey, 1993, Esterhuizen, 1997, Ryder, 1993, Leach, 1998). Less injuries or fatal accidents can thus be expected.

3.2 Regional support

Preliminary reports indicate that regional support consisting of an array of squat pillars not smaller than 10mx10m for a stope width of 1m at 3000m below surface present the following advantages:

- Minimal regional disturbance to rock mass. Esterhuizen, found negative values of excess shear stresses for bord and pillar layouts approaching geological structures. Leach had similar results (Esterhuizen, 1997,Leach, 1998).
- Less seismicity is also expected as Ryder found the average ERR to be 2,1MJ/m^2 which is low, and the average overall regional convergence to be 52mm (Ryder, 1993).
- Tolerable pillar foundation stress levels of 86MPa found by Esterhuizen make it possible to develop within 20 m of the footwall of the reef horizon (Esterhuizen, 1997).
- Blocks of ground, previously not possible to be mined, due to the high safety risk, could now be mined safely.
- Footwall (f/w) predevelopment within 30m of reef horizon.
- Footwall haulages (FW/HLGE) and return airways (RAW) could now be developed as close as 30m to the reef horizon.
- Extraction of 75% of ore body with the possibility of maintaining it with increased depth, while percentage extraction decreases with depth for the other known stope layouts.
- Concentrated and scattered mining layouts are possible.
- Due to the small spans between pillars, horizontal as well as vertical clamp assistance is improved, which should result in better h/w conditions.
- Less movement on geological structures with less water inflow.
- Backfilling not essential.
- Necessity of leaving high-grade bracket pillars against geological structures is no longer necessary.
- Enhances the possibility of early shaft pillar removal.
- Diminished squeeze of excavations in the vicinity of stoping operations.

- Stiffness of the pillar support allows better control of tensile stresses in the hangingwall (h/w) and footwall (f/w) of the excavations.
- Because of the smaller risk of major seismic events as a result of low ESS found near geological structures, defining geo-technical disturbances is less critical.
- The stresses are distributed evenly onto the pillars keeping excess stresses away from leads or lags of advancing stope faces.
- Removing previously unpay blocks at a later date, when the gold price increases, from a seismic point of view, will be relatively easy
- Low ESS will, from a seismic point of view, have minor effects on geological structures.
- Low levels of energy release rates will have little if any destabilizing effects on geological structures
- Placing of backfill if required can be done under better control circumstances.

From a rock-engineering point of view the major disadvantage is the lack of knowledge of what to expect regarding h/w conditions around the pillars. The possibility of a ‘pillar run’ is also put forward as a possible risk. This argument can only be valid if the pillars are designed to sub standard dimensions for the overburden they have to support.

3.3 Local support

The array of smaller pillars in situ, with small spans between pillars, provide higher stiffness resulting in greater h/w and f/w deformation control.

Less sophisticated and expensive h/w support is therefore required. Ordinary timber props with headboards should suffice in friable h/w conditions. Where h/w conditions require it, skeleton packs will be adequate. Because of low closure deep gullies are not required. Support of dip and strike gullies should be less critical due to the horizontal and vertical clamping effect of the in situ pillars and small spans in the immediate vicinity of gullies.

3.4 Literature survey

Below follows the literature study of Handley into the rock engineering criteria when examining the effect of SSSP on the rockburst hazard.

Bord and pillar mining at depth appears to be a favourable approach because mining spans, and therefore theoretical elastic convergences, remain small. This results in good control of the Energy Release Rate (ERR) and mining induced stresses in the footwall and hangingwall. According to current theory, the outcome of such a mining strategy should be a consistent extraction rate of about 75% and a safe, accessible mine, even when mining near geological structures. Because of the good control of mining induced stresses in the hangingwall and footwall, access development need not be placed far from the reef horizon, which will result in shorter crosscuts to reef, and generally less off-reef development.

This literature survey addresses two issues. First, it reviews bord and pillar layout designs that have been assessed for deep level hard rock mines, and second, it reviews pillar foundation stability in deep level mines together with laboratory experiments to quantify pillar foundation strength. The review continues with an evaluation of the risk of introducing a bord and pillar layout in a deep level hard rock mining environment using the layouts that have been analysed. The review concludes that the current bord and pillar layouts have a high risk of pillar punching (foundation failure) using current design criteria, and a new pillar design method is proposed to eliminate this risk.

3.4.1 Review of bord and pillar designs for deep level mines

As far as is known, three studies have been carried out to evaluate bord and pillar mining for deep level hard rock tabular ore bodies. The first was speculative and preliminary, and arose because of a discussion between Dr. FSA de Frey and Dr. JA Ryder on hard rock pillar design published by Ozbay et al. (1995a). In response to this discussion, Dr JA Ryder used BEPIL (Ryder and Ozbay, 1990), a simple 2-dimensional boundary element program that accurately models regular arrays of rectangular pillars in planar seams, to evaluate the potential of a regular bord and pillar layout in hard rock at great depth. The results were intriguing. Ozbay et al. (1995b) report an average pillar stress of 320 MPa for a 10m bord and 10m square pillar layout 3000 m below surface i.e. 75% extraction, and an average ERR of 2.1MJ/m^2. The low ERR arises mainly because of the low elastic convergence, which was calculated to be 52mm in the bords, 104mm in the rooms, with an overall regional average convergence of about 52mm. The average pillar stress falls well within the accepted criterion of $2.5\sigma_c$ where σ_c is the uniaxial compressive strength (UCS) of an intact cylindrical rock sample, typically about 200 MPa for quartzites from the gold mines of the Witwatersrand Basin.

Ozbay et al. (1995b) went on to report that the nature of fracture of the pillars is difficult to assess beforehand, but that it would occur, even as the pillars were being cut. Hangingwall conditions may also prove to be blocky and difficult to support. They did not assess seismicity in the mined out areas in detail, stating that there was a possibility of seismic damage because of geological weaknesses such as joints, bedding, faults, and dykes in the rockmass. Ozbay et al. (1995b) stated that their discussion of seismicity was speculative and that back area seismicity may be rare in practice because the *ubiquitous presence of strong pillars could provide a haven of safety to mine workers at all times*. Second-phase mining of the pillars was cursorily examined with BEPIL, and they concluded that conditions would most likely be untenable, even with stiff backfill in place.

This preliminary study was followed up by Esterhuizen (1997), who undertook a more detailed study of a theoretical bord and pillar layout assuming a depth of 3000m below surface. He used the MINSIM-D stress analyser for tabular deposits (Chamber of Mines Research Organization, 1985). The mining layout consisted of a 10m bord and a 10m square pillar, and assumed a 1m mining height, giving the pillars a 10:1 width:height ratio. The layout would result in 75% extraction, leaving 25% of the reef behind in the pillars. The stresses in the surrounding rock were determined, and the effect of these stresses on geological structures was evaluated. Esterhuizen found the following:

- The proposed bord and pillar layout will have a minimal effect on the surrounding rockmass;
- For all practical purposes, the rock stress 30m below the mined out reef horizon will be effectively equal to the virgin stress state;
- Tunnels 25m or more below the reef will be unaffected by stresses induced by unmined geological features or by the abutment;
- According to the model, geological structures are unaffected by Excess Shear Stress (ESS) caused by the bord and pillar mining.

These findings suggest good mining conditions with low Energy Release Rates and a low potential for mining induced seismicity when mining close to geological features. Esterhuizen (1997) specifically states at the end of his synopsis that (sic): “The study did not investigate the stability of the rock in the immediate vicinity of the pillars”. This includes the stability of the immediate hangingwall, the potential for footwall heave, and pillar sidewall stability. Although Esterhuizen (1997) does not report the Energy Release Rates calculated, he states in the conclusions that the layout will result in “extremely low levels of energy release rate”. This is concluded to have no effect on the stability of geological features, barring local instability.

Esterhuizen's (1997) results are impressive, but not complete because the stability of the pillars and the surrounding rockmass cannot be objectively assessed with an elastic model such as the MINSIM-D model he analysed.

Leach (1998) undertook a more sophisticated study in which he evaluated different pillar sizes and layouts for different depths. Having determined an acceptably stable layout, he then determined the *stability of the pillar itself*, and not the hangingwall and footwall above and below the pillar. Leach's (1998) model shows damage to hangingwall and footwall rocks around the pillar edges (Figure 7, p.11). This is to be expected, and will occur at all depths considered in the model. He notes (p. 15) that "When the FLAC3D average quartzite case is considered, the general pillar strength has a near constant minimum level of 450MPa. If correct, it would appear feasible to use pillars which are fully stable and remain intact up to extraction limits of 73% at 5000 m depth, 80% at 4000 m and 85% at 3000 m." Leach (1998) emphasizes again on page 15 that pillar width to height ratios should exceed 10:1 to ensure strain hardening behaviour, i.e. a stable failure mode in which seismicity associated with the pillars would be minimised. These figures exclude the use of backfill, which will reduce the pillar stress levels to below 450MPa if it is effectively placed.

Leach's (1998) conclusion begins with the statement that "The analysis is based solely on numerical modelling as practical experience does not exist of using small pillars at these depths". The conclusions continue that a room and pillar method is viable at depth, and that it may provide improved levels of regional support, even without backfill. The author is unsure of the meaning of this statement, but presumes that the "improved levels of regional support" mean lower levels of stope closure and seismicity together with improved pillar stability than has been observed with strike stabilising pillar systems.

3.4.2 Review of pillar foundation stability

As far as is known, no room and pillar mining has yet been undertaken in deep level hard rock tabular mines. It is therefore difficult to objectively evaluate the expected actual performance of small pillars in such an environment. Rockmass failure around the pillars themselves is specifically omitted in all three room and pillar studies mentioned above. The reasons for this are as follows:

- The first two studies employed elastic models (Ryder 1995, and Esterhuizen 1997), therefore the potential for failure cannot be accounted for;
- The third analysis by Leach (1998) accounts for failure in the pillars themselves, but deliberately does not consider failure in the rockmass immediately surrounding the pillar;
- Numerical models still do not reliably predict rock failure (especially violent failure), its extent, or nature.

Leach (1998) mentions concerns about the indestructibility of the pillars, with particular reference to Lenhardt and Hagan (1990), who describe the failure of 20m wide strike stabilising pillars on the Carbon Leader Reef at Western Deep Levels Limited. He therefore recommends pillars with width:height ratios exceeding 15:1, to avoid any strain softening behaviour in the pillars. This does not address the failure of the foundation itself, which Hagan (1988a), and Lenhardt and Hagan (1990) describe in their discussion of strike stabilising pillar failures at Western Deep Levels Limited. Although the pillars are seemingly indestructible because of width:height ratios of 20:1, they still fail violently and without warning despite satisfying all pillar design criteria then in use. Jantzon et al. (1990) also describe pillar foundation failure on the Ventersdorp Contact Reef at depths between 2000 and 2300m below surface.

The significance of these papers is that pillar failure occurred on both the Carbon Leader Reef and the Ventersdorp Contact Reef at Western Deep Levels. This suggests that the pillar failure might be inevitable, since it has been observed to occur in two completely different geotechnical environments. Although other deep level mines have employed similar pillars, they have not reported any failures because their seismic data is either inferior to that at Western Deep

Levels, or non-existent (Vieira et al., 1998b). They report further that the pillar failures recorded at Western Deep Levels are independent of depth and independent of pillar width. This represents a failure of the average pillar stress criterion, set at 2.5 times the uniaxial compressive strength of the rock, since all the strike stabilising pillar designs at Western Deep Levels satisfy this criterion, yet many fail. Other processes at work must therefore lead to pillar failure.

Ozbay and Ryder (1989) undertook laboratory tests on squat rib pillars to determine both the force required for, and the mechanism of pillar failure, in Witwatersrand quartzite, and in Bushveld Complex norite. The intact specimens of both rock types were strong, with laboratory-measured uniaxial compressive strengths (UCS) of 267 and 395MPa respectively. Details of how they fashioned the pillars and open stopes in the test specimens are given in the paper. They found that foundation failure in the laboratory took place when at an APS/UCS ratio of up to 2.8, for a k-ratio of 0.5, with the APS/UCS ratio increasing to 4.8 for a k-ratio of 1.

The experiments suggest the pillars themselves remained intact while fractures propagated into the rock above and below the stopes. Ozbay and Ryder (1989) show that these fractures do not meet vertically above and below the pillar centreline, as would be expected for a pillar foundation failure mechanism. However, the authors recognise fracture convergence above and below the pillar centreline as a valid foundation failure mechanism. Microseismic and borehole evidence put forward by Hagan (1988a), and Lenhardt and Hagan (1990) suggest that failure does take place in the rockmass directly below the pillar centreline, which supports the pillar foundation failure model. Wagner and Schumann (1971) also verify this mechanism in early load bearing tests of rock foundations.

These failures are significant since it appears, from Vieira et al. (1998b:pp. 4-27 to 4-28) that the average pillar stress at which foundation failure takes place is of the order of the laboratory uniaxial compressive strength of the rock. They also report a parametric study of pillar foundation failure, which shows that the only factor to significantly influence the potential for failure is the ratio of horizontal to vertical stress above or below the pillar, which agrees with the findings of Ozbay and Ryder (1989). All the parametric models investigated showed fracture development towards the centreline of the pillar above and below it, i.e. the formation of a wedge above and below the pillar.

All other parameters excepting the ratio of horizontal to vertical stress had a minimal effect on the failure potential. For example, in a 20m wide strike stabilising pillar layout and 85% extraction there is a 95% potential for failure for the depth range considered (Vieira et al., 1998b, p 10-6). This result is all the more significant when considering that the parametric modelling was specifically designed to assess the potential for pillar foundation failure using the wedge formation mechanism.

3.5 Risk evaluation and means of ameliorating the risk

The risk of pillar failure and means of ameliorating the risk is discussed.

3.5.1 Risk evaluation

If the average pillar stress criterion has failed, it is not because it is intrinsically wrong, but because the uniaxial compressive strength of a rockmass is typically much smaller than the uniaxial compressive strength of an intact laboratory sample. This suggests that using the criterion as it has been in the past introduces a high potential for pillar foundation failure at depth because:

1. The average pillar stress for the above extraction ratio will be of the order of 320MPa 3000m below surface, which may exceed the uniaxial compressive strength of the foundation rocks;

2. Leach (1998) proposes pillar width:height ratios of at least 10:1 to avoid strain softening behaviour in the pillars themselves;
3. Pillars with width:height ratios even greater than 20:1 have been observed to be unstable both in strike stabilising pillar layouts in deep hard rock mines and in parametric models, which suggests a foundation stability problem rather than a pillar stability problem;
4. Vieira et al. (1998, p. 3-23) observed that total closure between strike stabilising pillars may have reduced the average pillar stress by about 6%, whereas in the bord and pillar layout, total closure will not be possible in the small-span rooms, therefore the pillars will not be able to shed load by any other mechanism other than by punching into the footwall or hangingwall;
5. The high stresses in the pillars will lead in the long term to deterioration of the pillar rock as well as the rock in the immediate hangingwall and footwall;
6. Strata control problems similar to those noted for the 20m wide strike stabilising pillars in Hagan (1988b) will be likely in the bords;
7. a “pillar run”, i.e. failure of many pillars in quick succession, similar to those reported in Ozbay et al. (1995), cannot be ruled out as a possibility;
8. There is no modelling package currently available that can objectively evaluate the potential for time-dependent deterioration of the rockmass and pillars, or can estimate the potential for regional failures because of the punching of many pillars.

Because pillars can fail with such violence - seismic event magnitudes associated with the failures of strike stabilising pillars are typically 3.0 or larger – the bord and pillar layout applied at depth may have catastrophic consequences, especially if failures of many pillars over a large area occur in quick succession. Ozbay and Ryder’s (1989) experiments were carried out with intact rock; hence, their results indicate higher pillar foundation load-bearing capabilities than has been observed in mining environments. Pillars with width:height ratios of 20:1 or more are essentially indestructible, but they still fail because the foundation rocks have limited bearing capacity. It is therefore concluded that the risk of catastrophic failure of the foundations above or below the pillars will be high if conventional pillar design criteria are applied to deep bord and pillar layouts.

3.5.2 Means of ameliorating the risk of foundation failure

A new pillar design approach based on foundation strength is proposed to remove the risk of catastrophic pillar foundation failure. We can safely assume that at a width to height ratio of 20:1 the pillar is indestructible, but from observations of pillar instability that the foundation strata are not. Conventional pillar strength calculations are therefore irrelevant at great depth, and another approach based on the foundation strength is necessary. Ryder and Ozbay (1989) found that for intact rock in the laboratory, pillar punching loads were 2.8 times the intact rock UCS for k=0.5. Note from their experiments and from the observations of Hagan (1988a), and Hagan and Lenhardt (1990) that the pillars themselves did not fail, but that wedging and foundation punching occurred.

The in-situ strength of the rockmass is considerably lower than that measured for intact samples, hence a rockmass UCS should be used. The Hoek-Brown Failure Criterion (Hoek and Brown, 1980) can be used to obtain an idea of the footwall uniaxial rock strength. For a very good quality rockmass with an intact rock UCS of 200MPa, assume m = 10, s = 0.1, then:

$$
\begin{aligned}
s_1 &= s_3 + \sqrt{m s_c s_3 + s s_c^2} \\
&= 0 + \sqrt{10 \times 200 \times 0 + 0.1 \times 200^2} \\
&= 63\,MPa
\end{aligned}
$$

This suggests that if there is no confinement, the rockmass UCS is about a third of the intact rock UCS. Such a low strength is not expected because the foundations at depth will be confined. If we assume a k-ratio = 0.5 and a depth of 3000m, the horizontal stress is approximately 40MPa. For a reef dipping 23° the confining stress in the foundations parallel to the reef on dip is 46MPa assuming the above virgin stress tensor. However, the confining stress in the foundations along strike is still 40MPa, and this value should be used for the confinement in a regular bord and pillar layout if the pillars are square. Substituting this value into the above equation together with the other rockmass parameters yields the expected footwall strength of 330MPa at 3000m below surface. This is close to the global average pillar stress that can be computed from extraction percentages of deep level mines given in Vieira et al. (1998). Thus, pillars seem to be loading the foundation rocks at levels close to their in-situ strength, and therefore failures are likely to occur from time to time.

The bord and pillar layout with 75% extraction will be applying similar foundation loads above and below the pillars. Ozbay and Ryder (1995b) estimate average pillar stresses at 324 MPa for a layout 3000 m below surface. It is proposed that the pillars be designed on a *foundation strength criterion* assuming a tributary area loading for a regular pillar layout or a model determined average pillar stress and a factor of safety. The foundation strength criterion is obtained using the Hoek-Brown failure criterion as shown above, using either a measured or calculated stress for the horizontal confinement. It is assumed, from the modelling of Esterhuizen (1997) and Leach (1998), that the horizontal virgin confinement will not be significantly affected by the bord and pillar layout even close to the pillars, and that processes of creep or other rock movements will not somehow contrive to reduce it over time. The foundation strength criterion derived for regular pillar layouts in deep level hard rock mines is given by (Handley 2001):

$$e = 1 - \frac{f s_v}{s_f}$$

Where e= the extraction expressed as a fraction, f = a factor of safety, s_v is the stress component perpendicular to the pillar, and s_f is the strength of the foundation calculated as described above. If the above rock and depth parameters specified above are used, and a factor of safety of 1.7 is chosen, an extraction ratio of 62% is obtained for a reef dip of 23°, which translates to an average pillar load on the foundation of 194MPa. The average stress perpendicular to the pillar must be the same. The bord and pillar dimensions should therefore be designed using the following process:

1. The pillars must have a minimum 20:1 width:height ratio, therefore pillar sizes are determined by the mining height, and will be 20m for a 1m stoping width;
2. The bord widths must therefore be 12.5m to satisfy the above pillar size and extraction percentage requirements.

Because of the dip there will be shear stress acting on the pillar. A simple way of assessing the pillar stability with this shear stress is as follows:

$$\begin{aligned} ó_{sp} &= \frac{ó'_{21}}{1-e} \\ &= -37.87\text{MPa} \end{aligned}$$

The shear stress in the pillar is concentrated in the same way as the normal stress is, and is here called $ó_{sp}$. Assuming that the material in the pillar has no cohesive strength and a friction angle of 30º, the Coulomb Stability Criterion is given for a dry pillar by:

$$
\begin{aligned}
C &= \left|\tau_{sp}\right| - \mu\sigma_{np} \\
&= 37.87 - 0.58 \times 194.45\text{MPa} \\
&= -74.91 = -75\text{MPa}
\end{aligned}
$$

The above result suggests that the pillars should be stable in shear.

The extraction percentage is significantly less than that of 85% for 3000 m below surface given by Leach (1998). However, the following factors must be taken into account:

- There will be pillar sidewall scaling, which will reduce the effective size of the pillars at an as yet unknown rate, and to an unknown degree;
- There will be hangingwall and footwall damage around the pillars and therefore a strata control problem that will increase the effective height of the pillars at an as yet unknown rate and to an unknown degree;
- Bord and pillar layouts have never been tried at depth; hence, a conservative approach is indicated until more practical experience is gained.

It is considered viable to introduce bord and pillar layouts at depth in hard rock mines with all the claimed benefits, if foundation stability is taken into account in the pillar design. The above *foundation strength criterion* is still untested, but it may provide the solution for stable bord and pillar layouts at depth in hard rock mines.

3.6 Comparison of bord and pillar method (bpm) with deepmine layouts

Vieira et al. (2001) studied four possible mining layouts to mine tabular reefs at ultra-depth (3000 m to 5000 m below surface). The four layouts are:

1. Breast longwall with strike stabilising pillars (referred to as LSP in Vieira et al., 2001);
2. Sequential grid method with dip stabilising pillars (referred to as SGM in Vieira et al., 2001);
3. The sequential down dip method (referred to as SDD in Vieira et al., 2001);
4. The closely spaced dip pillar method (referred to as CSDP in Vieira et al., 2001);

Vieira et al. (2001) identified the extension of the Ventersdorp Contact Reef to the south of the Carletonville Gold Field as a potential ultra-deep orebody, and named it the Iponeleng orebody. The structure of this orebody has been imaged by subsurface seismic methods, therefore the positions of faults and dykes at reef elevations are reasonably well known.

It is assumed that a regular bord and pillar geometry will eventually result across the entire orebody, since geological structures can be ignored. Using tributary area methods, and taking an average reef dip into account, it is possible to calculate preliminary rock engineering results for the bord and pillar method. Table 3.6.1 below has been drawn up from Vieira et al. (2000), Esterhuizen (1997), Leach (1998), and preliminary bord and pillar results calculated above.

Table 3.6.1: Comparisons between DEEPMINE and bord and pillar layouts.

Layout	Depth[1] (m)	APS[2] (MPa)	ERR[3]	Efficiency[4]	Extraction (%)
LSP	3000	450	0.2	$24m^2/m$	67
	5000	695	0.6		
SGM	3000	320	0.1	$40m^2/m$	59
	5000	520	0.4		
SDD	3000	310	0.0	$33m^2/m$	58
	5000	510	0.0		
CSDP	3000	300	0.0	$32m^2/m$	62
	5000	500	0.04		
BPM	3000	194	0.0	N/a	57
	5000	257	0.0		

Notes:
1: Depth below surface;
2: APS is Average Pillar Stress;
3: ERR is the *probability* that the Energy Release Rate exceeds 30 MJ/m^2 anywhere in the layout at the given depth;
4: Layout Efficiency is here defined as the area of reef mined per metre of access development required.
Extraction at 5000m below surface

The table shows that bord and pillar compares very favourably with the other methods for the same orebody from a rock-engineering point of view. In particular, the average pillar stress and energy release rates are significantly lower than those for the other methods, without compromising percentage extraction. This is a strange result because average pillar stresses for similar extractions should be similar, regardless of the layout geometry.

If the access development is kept 30m below reef in the footwall with bord and pillar mining, it can be used to explore the reef ahead of the mining front without concern for abutment stresses. This is a considerable advantage over all the other methods, which induce significant stresses in footwall development. These results must be confirmed by a more detailed study of the mining layout in the same orebody.

The following advantages are also important:
1. 60% of the area mined will be trouble free.
2. With backfill it will probably be possible to get up to 85% or close to this level of extraction safely in a second pass.
3. Mining will be rate-independent (can mine at a high rate) and geology-independent.
4. Average pillar stress is considerably lower for bord and pillar than for the other methods with comparable extraction ratios (See table 3.6.1)

It is clear from Handley's study that the untested foundation strength is still a risk factor that has to be considered. If however one considers his findings when comparing the SSSP (BPM) method with the other methods in use, one must come to the conclusion that pillar foundations and hangingwall conditions in the vicinity of the pillars and advancing faces, in the case of the SSSP method should be more stable than the other methods. De Frey is of the opinion that the foundations of the pillars are as stable as the spans between pillars allow it to be. This is also the case when one considers the overall equilibrium and stability of the rock masses on a long and short-term basis.

3.7 Dynamic rock mass response to mining

A close study and literature survey was done by de Frey of papers of The Fifth International Symposium on Rockburst and Seismicity in Mines (RaSiM5) as this was seen to cover the latest developments in the rock-engineering field (G van Aswegen, RJ Durrheim and WD Ortlepp: 2001).

Where applicable de Frey comments on the relativity of the findings and how it supports the SSSP concept.

3.7.1 Rock mass equilibrium and stability

Ortlepp discovered that shear rupture had been driven through an intact, undamaged rock material from which it can be inferred that the origin of the rupture was some distance from the plane of the excavation and that the fracture front propagated towards the stope horizon (van Aswegen et al., 2001:50). He concluded that the fracture erupts spontaneously when the.deviatoric stress state at a point some distance out of the plane of the stoping excavations and usually ahead of the face, exceeds some critical value (van Aswegen, Durrheim and Ortlepp, 2001:50). It is therefore important to mine so as not to allow stresses to exceed this critical value. The SSSP will assist to address these excess stresses.

Missich and Lang refer to Durrheim's rockburst classification in terms of the source and damage mechanisms (van Aswegen et al., 2001:59). In one of the cases they investigated the strength of a pillar and found it was significantly reduced when the bearing area was halved and the slenderness ratio reduced by approximately 40% (van Aswegen et al., 2001:67). They claim there is a need to adopt a more systematic approach to the investigation, analysis and documentation of rock failure events in order to better understand the underlying causes and their effects on the mine structure (van Aswegen et al., 2001:61). De Frey is of the opinion that a critical review of the past and present theories and approaches to rock engineering would enable rock mechanics to systematically analyse what has been achieved and set objectives for the present and long term future. This could be a valuable future research project.

CI Trifu concluded that at Kidd mine in Timmins, Ontario analysis of 86 microseismic events (<0) allowed for 58 reliable solutions. Of these 37 events were characterised by a general mechanism with an average 56% tensile and 33% pure-shear failure components. This clearly indicates that tensile failures are far more common than previously reported (van Aswegen et al., 2001:79). We should therefore be concentrating on criteria that would assist in controlling tensile stresses of the rock mass. Smaller gaps between pillars would assist the control of tensile stresses.

3.7.2 Simrac projects

Adams and vd Heever review interesting research projects that have contributed to a better understanding of the seismic hazard in deep mines (van Aswegen et al.'2001: 205-212).

The projects GAP 034 and 223 suggest that although seismicity is inevitable with deep level mining, there are techniques for managing both seismic activity and resulting rockburst damage.

Research in GAP 223 found that stabilizing pillars yield seismically with an average pillar stress of as little as 250MPa, well before the suggested tolerable levels of 600MPa. Surely this warrants further investigation!

Where 28 rockbursts were investigated in GAP 201, it was found that mining layout and geological structure usually control the rockburst source mechanism. Esterhuizen found that SSSP mining endorsed these findings (Esterhuizen, 1997:3).

It is commendable when SIMRAC comes to the conclusion that it is important to seek the early involvement of end users of the technology that is being developed.

Durrheim concludes that with foreknowledge of potentially hazardous structure, it is possible to adapt rock-engineering technologies (mine layout, stope and tunnel support systems etc.) to manage the levels of seismicity expected at ultra-depth (van Aswegen et al.2001: 213). The idea is to reduce the hazard by seeking to engineer as stiff a system as possible through the choice of pillar spacing, placement of backfill, the sequencing of mining and the rate of mining.

Murphy found during his investigation of backfill in AngloGold's TauTona deep mine that as long as backfilling is placed to support 70% of the mined area and kept at close proximity to faces being mined, a significant reduction in production losses occurred as a result of seismic related activities. Re-establishing of seismically damaged panels was improved from 3 weeks to 3 days. There were also major reductions in seismically related accidents and fall of ground accidents. Backfilling also resulted in greater beam stiffness resonating at high frequencies thus dissipating seismic energy more efficiently reducing the potential seismic-induced damage in the stope face area (van Aswegen et al. 2001: 231). Van Aswegen and Mendecki state that a stiffer system will limit both the frequency and magnitude of intermediate and large events, considering that all mining induced seismicity is essentially driven by stope closure.

Vieira and Durrheim state that the mine designer could opt to stiffen the overall regional system by changing pillar dimensions and mining spans, or else introduce backfill (van Aswegen et al. 2001: 261).

Konecny concluded that methods of active prevention should avert, if possible the accumulation of deformation energy in stress exposed parts of the rock massif, by selecting a temporal and spatial arrangement of mine workings that would entirely prevent the formation of critical stress exposed zones (van Aswegen et al., 2001: 335)

Butra, Debkowski and Pytel found that where overburden strata have no weak inclusions, the magnitude of energy dissipation associated with failure is in proportion to roof strata stiffness. In roof strata consisting of many thin layers of weak material, one may expect frequent but small seismic events. Thick and stiff roof layers favour the postponement of failure that leads to a high level of strain energy accumulation that could cause large damaging seismic events (van Aswegen et al., 2001: 477).

Considering the above findings de Frey can but come to the conclusion that the SSSP mining method will address most of the above reasons for siesmicity. It will provide a stiff environment with minimal stope closure and resultant low tensile stresses. This will prevent the formation of unstable conditions as a result of disturbing the equilibrium in the rock-mass. Spreading the stresses and strains equally across the area being mined can keep it within the required limits, so as to prevent failure due to excess tensile strains and/or compressive stresses.

3.8 Rock engineering practice

De Frey also reviewed the book 'A handbook on Rock engineering practice for tabular hard rock mines' edited by AJ Jager and JA Ryder (Jager and Ryder 1999).

The objective was to investigate the criteria that would support the SSSP concept and where applicable point out possible problem areas that would be encountered.

Jager and Ryder found that the severity of the consequence of rockbursts in the goldfields is currently three times that of rockfalls. It is necessary to devise and implement significantly improved layout, support and mining methodologies aimed at reducing and containing deep level rockburst hazards (p4).

The dominant risk area for both rockfall and rockburst accidents, was the stope face area (p7).

Abnormal local values of the k-ratio for example k<0,3are likely to affect rock pillar strength and foundation stability to some extent, and the incidence and severity of seismicity in deep mines to a considerable extent (p12).

Large seismic events are accompanied by significant stope closure (p24).

Low convergence areas require stiffer systems than those for higher convergence areas (p35).

Reduced extraction ratios and smaller open spans could reduce the ERR of ultra-deep mines to as little as 10-30MJ/m2, maintain a largely elastic regional environment, and contain stope closures to 200mm or less (p37).

Studies carried out in deep longwall-type mines have shown a convincing correlation between average ERR level and seismicity or incidence of damaging rockbursting (p50).

Although Jager and Ryder do not suggest the SSSP concept as a system that provides stiff support, so as to reduce volumetric closure and hence stress concentration in front of the mining faces, it can be seen as a definite alternative (p53).

The criterion that APS<2.5σ where σ is the uniaxial compressive strength (UCS) of the foundation rock material is suspect. Current ongoing research is aimed at refining this criterion, which probably should take into account the friction and dilation properties of the foundation rocks, as well as possibly the prevailing k-ratio (p56).

Jager and Ryder emphasise that, with regular stabilizing pillars and with no total closure between pillars, the stope width no longer features in the ERR equation; ERR values increase rapidly with extraction ratios; ERR values increase directly with half centre spacing between pillars. Thus use of small spans between pillars is also favoured at great depths of mining (p62).

Laboratory simulations, together with numerical modelling studies, suggest that the pillars' primary design objective (to reduce face ERRs) should not be significantly compromised if the APS <2.5σ criterion is met (p65).

Jager and Ryder found that with room and pillar mining at depth, field stresses did not rise significantly above virgin levels, and on-reef or deep off-reef development were not subjected to high abutment or face stresses. The main consideration of the stability of the pillars and their reaction to transient seismic waves seems to be the area requiring further investigation (p93).

With extraction, remnants become smaller and pass through critical minimum dimensions, stress and ERR levels, stress-induced fracturing and seismicity all tend to increase (P101).

Once the problem of significant seismicity and rockbursting is recognised, the influence of dynamic loading on excavation stability must be considered as the dominant factor in support design (P115).

The height of the tensile zone is a function of the stope span (P117).

The bending of beams or plates result in tensile stresses developing on the convex surfaces sufficient to fail the rock in tension (119).

The depth of fracturing ahead of the face is directly related to the ERR (P120).

The main criteria to be considered in assessing the potential for seismicity include the levels of ERR, ESS, the volume of rock that may be subjected to significant levels of ESS and the presence and orientation of geological structures (P124).

The functions of stope support systems are to provide, with a factor of safety, sufficient support resistance or energy absorbing capacity and areal coverage or support unit interaction to maintain the integrity of a discontinuous hangingwall that may be subjected to seismically generated ground velocities. Design factors that are important are strength, stiffness, yieldability, energy absorption ability and whether it is an active or passive support (P137).

Under seismic deformation and subsequent differential decelerations across the spans between support units, the stability situation is much more complex. To improve safety, and reap the immense economic benefits of reducing fallout dilution and stoping widths, improved support interaction and areal coverage specifications designed for specific geotechnical conditions are required (P144).

Mining wide reefs in excess of 2m width at depth poses a problem in that the conventional timber type of support become increasingly inefficient with increasing height, particularly in terms of stiffness which decreases inversely with the height of the support unit (P184).

Where pillars or remnants are of critical dimensions they can become overstressed and fail violently (P250).

Jager and Ryder point out that the synthesis of rockburst investigations for the most part endorse existing guidelines, but in some instances reflect a slow evolving change in knowledge and approach (P276).

When Jager and Ryder consider increasing pillar size and reducing spacing between pillars to ensure that APS levels are kept below locally established critical levels (P277). They also point out the importance of determining the stable 'dynamic span' for the support system and geotechnical area (P280).

Jager and Ryder describe the routine seismic monitoring in mines (P287-325).

At present numerical models are used to test proposed design specifications against well-established predefined design criteria such as APS, ERR, ESS and Rockwall Condition Factor (RCF) (P339).

When one considers the above literature review the following thoughts and conclusions come to mind:
It is not clear how the ERR for the different layouts, such as where in both cases the percentage extraction is 75%, the ERR can vary between 20MJ\m^2 and 5MJ/m^2 while the APS at the same depth remains fairly constant. De Frey is of the opinion that insufficient attention is being given to the influence of the span between pillars. Pillar strength in quartzite is largely determined by the competence of its hangingwall and/or footwall strata. The smaller the span between pillars the bigger the confinement of the pillars with less chance of the weaker strata flowing into the open span.

Further research into the validity of the APS<2.5σ as a criterion is now needed. The emphasis is still on the strength of the pillar without taking into consideration the effect of the span between pillars.

As a mining alternative this report proves that the SSSP method is viable seen from the non-rock engineering criteria point of view, and from the information presently available signifies it to be better than any of the other layouts presently in use.

With the SSSP layout numerical modelling has shown that ERRs remain fairly constant unless the pillars are removed. Where larger unpay blocks (remnants) have to be left one could expect that their effect would improve the ERR around the unpay block.

When considering the influence of dynamic loading on excavation stability as the dominant factor in support design, the SSSP layout, is providing regional support at the same time as addressing the problems of local support, and especially coping with dynamic loading!

Considering the fact that the tensile zone is a function of the stope span special attention must be given to the fact that the span should be limited to the competence of the immediate strata to support the tensile stresses ahead of the advancing stope face as well as between and across pillars. The SSSP concept addresses this criterion more effectively than any of the other mining layouts.

The SSSP layout will address the problem of tensile stresses getting out of control. This should be seen as one of the main reasons why there are foundation failures of stabilizing pillars accompanied by seismicity.

Because the SSSP layout is a method that has the lowest ERR, fracturing of rock ahead of the face will be better controlled than with the other mining layouts. The further one allows the shear fractures to migrate ahead of the face the larger the mass of unstable rock becomes. The aim should therefore be to keep the ERR values as low as possible and that means keeping stope closure as low as possible.

Considering the potential for seismicity, the low levels of ERR, ESS and the effect of the orientation of geological structures, the SSSP layout has been proven to be superior to the other known methods.

One of the greatest advantages of the SSSP concept is the fact that it serves as local as well as regional support. When considering design factors as a local support the pillars will be strong, stiff, have the ability to absorb energy and act as an active and passive support. It will however not be as yieldable as other support designs. The challenge is to maintain stiffness of the support, which is a pre-requisite for absorbing energy!

Mining of wide reefs should be safer using SSSP layouts. SSSP layouts will provide the stiffness that will not be achieved with conventional timber support.

One is inclined to visualise a burst as the disintegration with explosive force of the reef horizon. It is however the hangingwall and/or footwall that usually fails as a result of stresses that exceed the critical strength of the rock in the vicinity. This results in sudden violent energy release, manifesting itself in damage to the excavations in the vicinity. Pillars and spans should therefore not be allowed to reach the critical dimension size that will place it in a state of unstable equilibrium.

Jager and Ryder point out that for the most part research findings endorse existing guidelines but in some instances reflect a slow evolving change in knowledge and approach. Hopefully this Simrac GAP 828 report will stimulate and support the evolution in finding the optimum safe and economic solution for the prevailing conditions and environment.

One realises that it costs money to install, maintain, and run or manage the monitoring systems, as some of them are of a sophisticated nature and need competent people to manage and control them. It would therefore be of considerable advantage if a mining layout can be designed that would require minimal monitoring because of the fact that changing of face advance directions to negotiate geological disturbances or lags and leads will not affect the rock engineering criteria.

Until better design criteria are established and proven the present criteria should be seen as meaningful indications of what to expect in the field of rock engineering practice and also applied when measuring safety risks.

3.9 The Driefontein case study

The methods that were compared were a closely spaced dip pillar method, a double cut mining method, a drift and fill mining method and a bord and pillar method (Riaan Carstens July 2001). This report being of a confidential nature let it suffice to say that it was found that in terms of rock engineering, extraction ratio, flexibility, cost (labour and support), efficiencies, ventilation and ore handling the bord and pillar method was ranked the best option. It was ranked first in all the categories except the extraction ratio where it was ranked second and in the ventilation where it was ranked third. All the rock engineering design criteria are met except for the pillar stress to strength criteria. The pillar strengths can only be calculated once the CSIR laboratory testing and in-elastic numerical modelling has been completed.

3.10 Discussion

Considering all the above findings it is felt that the first and most important consideration is the maintenance of the equilibrium in the rock masses surrounding the excavations so as to have stable conditions in the workings of a mine. To establish this state of stability, minimum movement in the rock mass is essential. This can be achieved if closure of stopes is kept to a minimum. It is therefore important to have stiff support that will prevent or at the least resist closure to its maximum extent. The stiff support must also be able to act as a shock absorber with as little as possible deformation or slip and ride. Large spans make available large areas for the rock masses to move into. Spans must therefore be kept to a minimum. It is probably also time for the rock engineers to investigate the influence of the size of spans between pillars on the strength and stresses of the pillars. Special attention should be given to the amount of horizontal confinement of the h/w and f/w of the pillars brought about by the spans between pillars. Pillars should therefore be as small as their inherent strength allows them to be for the given stope widths, while the size of spans will be dictated by the strength of the pillars as well as the hanging and footwall competence and strengths. It seems that ERR is still a good criterion to measure the propensity for seismic activity in non-geological complex areas. SSSP at the moment is the best mining layout that addresses the above criteria.

4 Ventilation

4.1 Introduction

At a workshop on the 6th January 1999 at Miningtek it was concluded, that although it would not be impossible to ventilate the bord and pillar system it would probably be done at greater cost with more rigorous controls. The probability of ventilation ducting or curtains interfering with production was given as a serious disadvantage.

The possibility of doing predevelopment of twin haulages as close as 30m normal or at right angles to the reef horizon and the small rate of closure of working places, from an environmental point of view, support the bord and pillar mining system. The following issues are therefore relevant:

- Concentrated and/or scattered mining
- Better control of air usage

- Less heat radiation for the 75% extraction ratio
- Shorter x-cuts mean less waste development with less heat loading
- Improved environmental management - u/g fires
  - stopping of air fans
- Improved intake and return airway maintenance
- Better management of ventilation districts
- Better possibility for recycling or reconditioning of air
- Less heat transfer from underground (u/g) water fissures
- Less pumping means less heat load added
- Better condition for placing of refrigeration equipment and excavations.

4.1.1 Background and general information

The literature study and analyses done by Webber follows.

Ultra-deep mining (to depths of 5 000m and greater) would be a world first and, accordingly, no previous experience in the determination of acceptable heat stress limits, criteria or indices is fully applicable. However, some South African gold mines are already operating at depths beyond 3 500m and much of the knowledge gained in reaching and working at such depths will be helpful in making adequate provision for acceptable environmental control at the greater depths being contemplated.

The findings detailed in previous research indicated that it is likely that an appropriate combination of heat stress indices will be required in planning for and ultimately controlling thermal conditions in ultra-deep mining (Webber, 2000). The depths being contemplated and the concomitant potential heat hazard, present too great a risk for reliance on a single heat stress index, such as the wet-bulb temperature index at present in common use locally.

The need to quantify the costs associated with various levels of wet-bulb temperature and air velocity (the two most important determinants and means of controlling heat stress) and a direct indication of the Air Cooling Power (ACP) was addressed through an analysis of a model mine operating at a depth of approximately 5 000m (Webber, 2000:71). From this investigation it was found that it is possible that the provision of a given level of cooling power at a working depth near 5 000 m would be more cost-effective through increasing refrigeration and reducing the amount of ventilation air. This implies in effect that it would be "cheaper" to increase the cooling of air and reduce the air velocity in the mining environment.

To further contain the cost of providing a given air velocity at great depth, recommendations were provided on the implementation of controlled recirculation strategies. From the various recirculation models for a longwall follow-behind mining layout that were investigated, it was found that global recirculation of air seemed to be the most cost-effective system in planning ventilation requirements at ultra depth (Webber, 2000:100). It appears that the application of recirculation strategies will be imperative in future. The inclusion of re-circulation strategies are not relevant to this investigation and is not done, but will in future, if the bord and pillar concept is adapted, have to be investigated in detail. This is the case, no matter what mining layout is being considered.

In an investigation of this nature it is important to identify all relevant work that was done in the past and to compare all results with those already achieved in other investigations. Bluhm and Biffi examined the heat, ventilation and cooling effects of four DEEPMINE project stoping layouts for narrow reef mining in ultra-deep operations in very hot rock (Bluhm,S.J. and Biffi, M. 2001). The various layouts that were considered had a broad division between those using strike pillars and those using dip pillars. For strike pillars, breast mining was investigated and for a layout including dip pillars, down dip mining and breast mining (overhand and underhand) were considered.

In his paper Bluhm was mainly concerned with the micro-layout (in stope) aspects of this work and the effect that the different stoping methodologies would have on ventilation and cooling requirements. In his paper Bluhm gives detailed versions of the various stope layouts mentioned, as well as their advantages and disadvantages.

Bluhm in his paper concluded that the in-stope heat load generated per ton mined varied for the different layouts. Qualitatively, the dip-pillar down-dip mining layout fared better than the others. The dip-pillar breast layout was the best in terms of escape and rescue considerations, while the strike pillar layout offered the best advantages in terms of development and multi-blast needs. However, the four layouts were grouped fairly closely. From this work done various strengths and weaknesses for each mining method have been identified (Vieira et al., 2001).

From this paper by Bluhm it was also concluded that although the investigation mainly dealt with the micro level (in stope), there are two important observations to note on macro level. First, the contribution of in-stope heat to the total mine heat load is less than that of the intake tunneling (for all 4 layouts). This means that for ultra-deep mines, the air conditioning balances of stopes are less than that of the intake system. Second, the total mine-wide costs of owning and operating the ventilation and cooling systems vary by about 20% for the different layouts, with the strike-pillar breast mining the highest cost and dip pillar overhand breast mining the lowest.

The expected high cost for ventilation, cooling and pumping associated with ultra deep mining, therefore warrants further research to investigate the possibilities of maximising profits by optimising ventilation cooling and pumping costs by also including other layouts that might be feasible in the ultra-deep mining context.

The purpose of this research was therefore to investigate the various issues pertaining to the ventilation and cooling requirements for a bord and pillar mining layout (obviously for the "flat reef" mining environment dipping at approximately 26°). De Frey, in work done to identify mine design alternatives, evaluated the bord and pillar mine layout as a mine design alternative (de Frey, 1999). De Frey concluded his report by saying that it is necessary to introduce bord and pillar as a mining method and that early indications are that it is economically viable with major safety advantages. In addressing the problem it was very important to investigate the feasibility of a bord and pillar layout, in terms of the requirements to ventilate and cool the said layout.

4.1.2 Problem statement

Investigate the feasibility of a bord and pillar mining layout, in terms of the requirements for ventilation and cooling.

4.1.3 Objectives

- To evaluate all previous work done pertaining to bord and pillar layouts for ultra-deep mines
- Simulate the ventilation and cooling requirements for a bord and pillar layout on a micro scale
- To identify and highlight all relevant environmental design issues pertaining to bord and pillar mining and identify all other mining related issues that will influence the ventilation design for a bord and pillar mining layout.
- Make recommendations to adequately and safely ventilate the bord and pillar mining layout

4.1.4 Methodology

- A literature search
- Draw up a computer simulation program

- Evaluation of current bord and pillar design and the feasibility thereof in terms of ventilation requirements
- Documentation of results
- Evaluation of results
- Recommendations
- Conclusions and identification future research issues

4.2 Research / Work Done

Research work done is discussed below.

4.2.1 Background investigation

Bluhm and Biffi in their paper identified various factors to be considered in comparing various mine layouts and specifically when considering the ventilation and cooling requirements for different mining layouts. Layouts can be examined on a macro and micro-level and a macro analysis should consider the following:

- ❖ The scheduling of the ventilation requirements in relation to other infrastructure needs.
- ❖ The production build-up and the related capital and running cost of the ventilation and cooling systems.
- ❖ Large scale ventilation tactics including re-circulation and the use of primary and secondary bulk air coolers as well as in-stope coolers.

Bluhm and Biffi in their paper only considered the micro-layout aspects and the effect that different stoping methodologies would have on ventilation and cooling requirements. In the context of comparing a bord and pillar layout with other previous designs, the comparison on a macro design was not done in the same detail and only relative factors that can be compared on a qualitative basis are later discussed. The objective therefore will be to investigate the feasibility of ventilating a bord and pillar layout for ultra deep mines on a micro scale.

4.2.2 Bord and pillar mining layout

In his evaluation of the bord and pillar method for ultra-deep mines, De Frey gave detailed descriptions of the various aspects pertaining to the design (de Frey, 1999). The essence thereof was discussed in chapter 2 on mine layout.

4.3 Ventilation and cooling requirements

Global air requirements and in stope air and cooling requirements are looked at and compared in the following paragraphs.

4.3.1 Global air requirement parameters

From the production requirements mentioned above the following parameters and assumptions need to be made (with the aim to be used in the determination of the air requirement calculations).

- ❖ A stope will consist of 6 up dip panels (a panel being 15m long and the pillar width 15m as indicated before)
- ❖ Total advance per panel per month will be 15m
- ❖ The total production per month will be 45000m^2 (+- 156 000 tons)
- ❖ Development tons +- 10% of the above figure
- ❖ Total tons mined per month therefore +- 170 000 tons

- ❖ Take an average of 4.5kg/s of air needed per kilo ton of rock mined
- ❖ As per de Frey's recommendation, 6 stopes (production units per level)

From the above parameters given, the total amount of air needed for production can be determined.

To simplify the calculations we assume that the total amount of air needed per level and per stope will be the same.

The amount of air needed to sustain full production will be 765kg/s. This means that for the proposed 6 levels there will be an amount of 128kg/s of air available for each level. Considering the amount of air needed for other layouts it can therefore be assumed that one will arrive at the same figure of the total amount of air needed, if the same air factor per kilo ton of rock is used. Development will be done on each level (footwall haulage(s), RAW(s), X-cut(s) and the boxholes). In terms of the amount of air needed to ventilate the stopes, it will be important to establish what cooling affect the available air will have and what effect the additional fans as well as the change in mining layout will have on the heat load in the stope. This comparison will be done with the help of a simulation and will be shown in detail.

In terms of the establishment of ventilation districts, it is important that wide raises should be holed over at least three levels. In terms of ensuring availability of stopes sooner, raise bore ventilation passes can be established from the RAWs on specific levels to the raise lines above, and in this way ensure that a return air way can be established before holing with the next upper level has been affected.

4.3.2 In stope air and cooling requirement (micro)

In the design for ventilating the proposed layout it is assumed that the air is already cooled when it reaches the stope. The aim is therefore to prove that the ventilation of the bord and pillar will be possible/not possible on a micro scale and then to evaluate it later on a global basis in terms of total costs (capital and working). Various issues, like in-stope cooling will have to be investigated in future, but it must be remembered that in terms of the current layout that the moving of cooler fans and piping of chilled water to these fans in the limited spaces available, would be difficult and not easy to carry out (although not impossible). It is therefore most likely that the cooling of the air will be done either through bulk cooling at pre-determined optimum positions (through macro analysis) in the total mining layout, or through cooling cars in the x-cuts below.

The ability of the air to remove heat is a combination of the mass flow of the air and the temperature of the intake air. A higher mass flow will ensure a larger amount of heat that can be removed. If the intake air temperature is lower, it means that the air has the ability to absorb more heat for a specific reject temperature and that the mass flow could then be less. The ability of the air to remove heat from the work environment therefore has to be optimised between the amount of air supplied and the temperature thereof.

Figure 4.3.1 shows the basic layout of a 6 up-dip panel stope and the possible face configuration that can be expected, as well as the proposed ventilation flow through such a stope. From Figure 4.3.2 it is obvious that the faces that need to be stoped to meet the required production can basically be from any of the up-dip raises/panels/headings A, B, C, D, E, or F and that production faces could be mined up-dip or on strike. The other aspect that also has to be taken into account, is that an up dip panel will have to be stoped more than 30m, to ensure that the panels on strike (east and west) can be broken away and the next line of pillars and subsequent strike airway, be established.

From the proposed layout it needs to be noted that coursing of the air through the stope has to be done and that dip ventilation controls between pillars on dip have to be established.

Installation and maintenance of ventilation brattices on dip should not create problems to as they will not be in the direct line of scraping. At this point in time it is important to note that dip ventilation control will become a critical aspect to prevent air short circuiting through the back areas of the stope, making less air available for the up dip panels. The air from two or more intake up dip panels from the back areas will join and increase the volume flow in the last "strike ventilation" gully, making more air available for heat removal. The air from the last strike airway will then join with return air from the up dip panels that have not holed yet. The return air from the stope will therefore be leaving the stope through the pre-developed wide raise at the top of the stope. This means that the wide raise becomes critical in terms of maintaining the airflow through the stope.

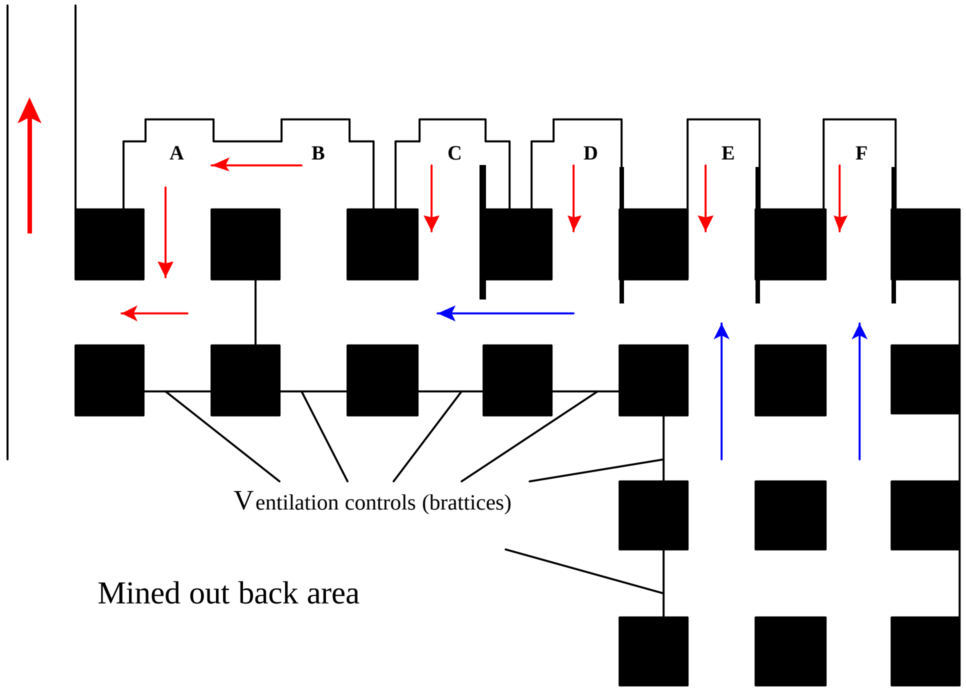


Figure 4.3.1: Various stages of stoping in bord and pillar mining.

From Figure 4.3.1 it is also obvious that each up dip panel could be in a different stoping phase and the production "timing" of these different phases will be critical in establishing through ventilation and a better working environment. Considering the stoping and how it will be done, it is important to note that due to the fact that up-dip mining will be done, there will be situations, where, before the holing between pillars and panels have been created, that a dead end/back stope scenario will be created, and therefore special air supply requirements will have to be met. We are therefore here in a position where we have a possibility of six back stopes, which will have to be ventilated mechanically, individually, until through ventilation has been established between two lines of pillars. The stoped area of each back stope is approximately 500m^2. Figure 4.3.2 below shows a close up view of the expected flow of air after a holing has been affected between up dip panels A and B. This will ensure that until such time that the new up dip panel has advanced "high" enough, that fan ventilation for that specific panel will not be necessary. This holing will also ensure better working conditions in the headings A and B (in terms of available air on the face).

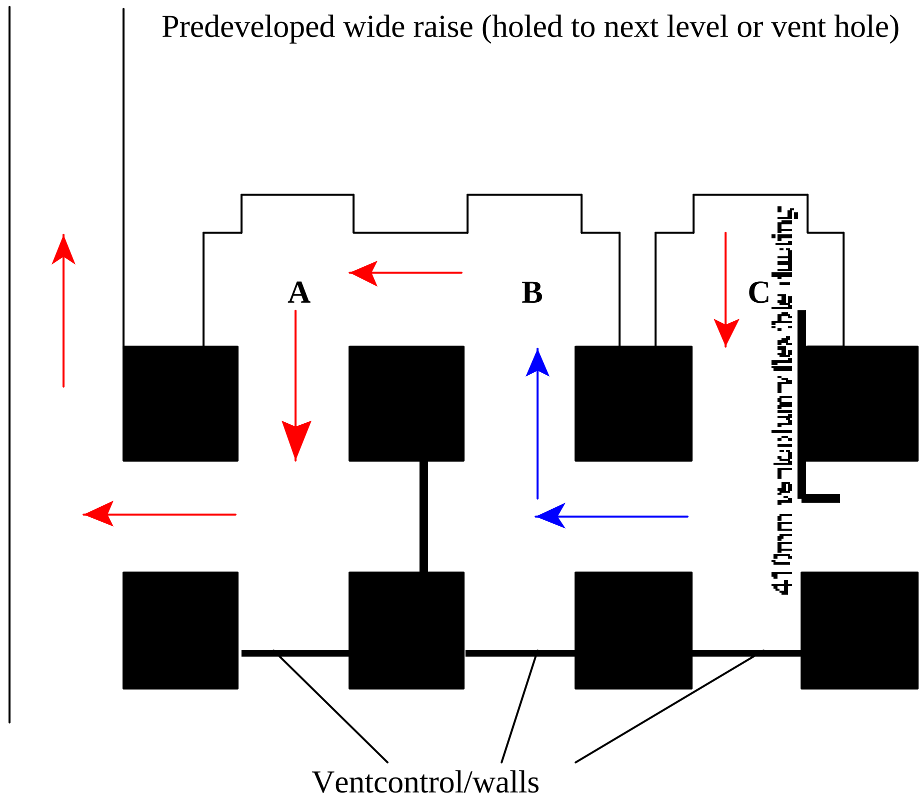


Figure 4.3.2: Airflow arrangements in a bord and pillar layout.

With a bord and pillar mining layout, where the working height is 3m or more, it is easier to course the air with the help of ventilation curtaining that can be hung from the hanging wall, and in this way creating a resistance to re-route the flow of air to an area where it is needed. This is being done on almost all the bord and pillar layouts for coal mines and is fairly easy to install and maintain due to the available height in which the ventilation can be hung. In the case of the low stoping width bord and pillar method as proposed, this will be an area of concern, as the quality and amount of air on the face will become very critical to control. A method will have to be devised which will have to be easy to install and easy to remove in the event of scraping that will take place on dip. These curtains can be hung, say for instance in the up-dip heading (coursing of air through the up-dip back stope), to possibly replace the fan (which if present, will force the air mechanically onto the face). This will mean that this curtaining will have to be removed at the end of each shift, as scraper cleaning will take place in that heading during the next shift, and if not removed, will damage the curtains. This will create an unsafe working environment for the cleaning crew on the next shift, as the heading will be unventilated.

From the layout as shown in Figure 4.3.2 it is also obvious that strike ventilation control will also be a critical aspect to consider. The problems associated with the strike ventilation control will be the same as mentioned above, as it will have to be removed before scraping can take place and be reinstalled before stoping or any other work can commence. This cycle will repeat itself continuously through the life of the stope and might cause problems to keep in place. This means that strike ventilation control will always have to be a “temporary” arrangement, where dip ventilation control measure could be permanent walls between pillars. If this strike control is not done properly, it will mean that the available air along the last through ventilation strike “gully” will also become less, affecting the ability to remove heat, quite drastically. The inclusion of a strike ventilation pillar between each set of up-dip stopes between levels might also become something to consider when the detailed ventilation planning is done.

4.3.3 Comparing bord and pillar with conventional breast panel

The VUMA heat and airflow simulation program provides a handy tool to compare various layouts with one another and what would happen to the heat flow capacity of the air, if certain parameters such as mass flow and production rates are changed. The purpose of this section is therefore to show what effect the mining method would have on, for argument sake, the mass flow of air required for a specific reject temperature.

In drawing a simulation model for the bord and pillar and breast panel layout, certain assumptions have to be made, of which the following are the most important:

- Each up dip panel has to be ventilated by it's own fan until such time a holing has been established (therefore looking at worst case scenario where 6 panels are mechanically ventilated and mined)
- Stopes are wet, as indicated per code used in VUMA, in other words, not dry mining (therefore influencing heat transfer negatively)
- The air supplied by the fans remain constant
- Air leakages and losses taken as nil
- In the simulation program, the bord and pillar layout is shown as development ends and development tunnels following on one another, and for this layout therefore 6 development end headings and 6 development tunnels. The average distance of mining an up dip panel with the help of a fan is taken as 28m. This allows for the break aways of the east and west strike panels in the heading.
- The air is therefore coursed into one development end by means of a 15kW fan, where the return air from this development end would join the through ventilation, and then to the next development end, and so forth.
- The development end headings are taken as 15m wide and 1.25m high and the advance per month is assumed as 15m. The average distance between development end headings is 30m, which means that the length of the development end tunnels along which the air must flow on its way to the next development end, is also 30m.
- For this simulation all the other parameters such as depth, virgin rock temperatures (VRT), rock properties, production rates and so forth, is kept exactly the same as for use in the breast panel layout.
- Three breast panels are used in the simulation with the breast panel method with 70% back filling and a 6m ventilation channel along the face.
- In both cases a heat load of 450kW for winches is assumed.
- It is assumed for both cases that the air would be cooled to 20/20ºC, before it is sent into the stope (bord and pillar and breast conventional stoping). The air mass flow rate is changed to see what effect it will have on the reject temperature out of the stope (breast panels and bord and pillar). This is an important aspect, to remember, as one of the objectives of this investigation, is to find out on a micro scale, what the conditions of the air leaving the stope will be, if the bord and pillar method is used. The total amount of air and cooling needed, can only be established once a total mine layout for bord and pillar mining have been done.
- For the sake of the comparison, air losses are not taken into account, but it is obvious that the control of the airflow in the bord and pillar method will be an area of concern.
- The reject temperature for all the simulations is 27.5ºC.

Tables 4.3.2 and 4.3.3 below show the results from the simulations done for conventional breast panels and the bord and pillar stope (upcast airflow).

Table 4.3.2: Mass flow of air versus reject temperature (breast stope panels).

Mass flow of air (kg/s)	Quantity of air (m^3/s)	Density of air (kg/m^3)	Reject temperature (ºC)
5.0	3.3	1.5	30.0/32.0
7.5	4.9	1.5	27.8/29.8
10.0	6.5	1.5	26.0/28.0

Table 4.3.3: Mass flow of air versus reject temperature (bord and pillar).

Mass flow of air (kg/s)	Quantity of air (m^3/s)	Density of air (kg/m^3)	Reject temperature (ºC)
5.0	3.4	1.5	31.3/35.5
7.5	5.1	1.5	29.1/33.3
10.0	6.7	1.5	27.6/31.7

It must be noted from the above results, that the amounts indicated are the minimum airflow required, but that in practice it would be much more, due to ineffective use and back area losses occurring (losses could be anything between 20 and 30%). For the purpose of the comparison, the results as indicated, will be accepted.

From the simulation results mentioned above, the following comments can be made:

- ❖ The amount of air required to ventilate the bord and pillar stope (six panels on strike) will be approximately 30% more for the same reject temperature of 27.5ºC (7.5kg/s to 10kg/s). The mining and break away of the east and west strike panels while the heading is still ventilated by the fan, have not been included in the design and the amount of air needed can and will be even more than the indicated extra 2.5kg/s, as was calculated in the simulation.
- ❖ The higher amount of mass flow of air required for the bord and pillar layout, will mean that more air will have to be cooled and that it will have a major influence on the total amount of air needed per stope (it was assumed that the air quantity was already cooled at 20/20ºC, when it reaches the first stope panel and an increase in mass flow therefore indicates more air that will have to be cooled). The only scenario where less air will be needed for total mine production, is when it is proved, on a macro scale, that less stopes will be needed to give the same amount of production, if compared with other layouts and therefore decreasing the amount of air needed per kiloton of rock mined.
- ❖ The additional fans in the bord and pillar layout will definitely have a detrimental effect on the heat load in the stope.

4.4 Critical evaluation of results

In ventilating a bord and pillar layout for narrow reef mines, various issues have been discussed and dealt with so far. There are however, key issues that need to be highlighted again and other issues need to be mentioned as well. These issues will be mentioned briefly, but in some cases can and will have to be dealt with in detail later.

- Until such time that a holing between two pillar lines on strike has been established, all up dip panels will have to be ventilated mechanically with fans. The duty was assumed as 15kW and a possible air delivery of 3kg/s.
- The potential for air losses are much greater for the bord and pillar method as dip and strike control will become very important to have in place. It is thought that strike control especially, is going to be very difficult to be maintained in the bord and pillar layout as proposed.
- Coursing of air by means of ventilation brattices from one end to the other is possible, but is most likely not going to be practical and will be hard to do.
- From previous research it was also found that an increase in air quantity in the ultra-deep environment would have a detrimental effect in terms of costs, and should be avoided.
- The supply of electricity to each of the fans ventilating the up-dip headings, will have to go through the up-dip panels in which scraping is continuously done, creating a possibility of constant fan breakdowns due to cable damage and subsequent fan failure. The problem can be avoided by having fans with their own dedicated power line in the pre-developed wide raise, but you still have the problem of the cable entering the stope on strike and in the path of the scraper.
- The fans can and will have a detrimental effect on the heat load in the stope.
- The 4 up-dip winches and possible strike winches in the proposed layout, will also have a detrimental effect in terms of the heat added to the air in the stope, as they are all situated in the main intake air stream, affecting the quality of the air going to the up-dip panels immediately.
- Although each fan will supply $3m^3/s$, it might be possible that the amount of through ventilation needed past the 6 up-dip panels can and will be a significant amount, and as was shown by the simulation, at least 10kg/s.
- The minimum velocity of air on the face as required by currently used recommendation is 0.25m/s, but that we know that the industry averages approximately 1m/s. This means that a high air velocity for the bord and pillar layout would be possible, but that a constant heat load will be added continuously due to the presence of the fans. The through ventilation velocity on the face for a bord and pillar layout is only possible, once a holing on strike between two pillars has been affected.
- Installation and moving of fans in a limited space such as a stope, can and will be cumbersome.
- The installation and removal of ventilation brattices and fans during the blasting and cleaning cycle can and will be an area of concern and can have a serious effect on the work cycle and the eventual advance per panel per month.
- The handling of fans and ducting in the initial stages of establishing the various pillars will be easier, because of the fact that there will be a pre-developed strike gully. It may be necessary that footwall lifting or taking down hangingwall to make enough space available for the fans and ducting will have to be done in the other strike gullies higher up in the stope. This can create an additional burden to mining activities
- The utilisation of the available air will become crucial, as losses can occur through back area pillar holings not properly sealed off.
- In terms of lack of space and the continuous scraping that will take place in the dip gullies, the creating and establishing of positions for in-stope coolers will have to be done in detail. In-stope cooling will be feasible, but in terms of the infra structure in the stope needs to be critically evaluated. The additional piping to coolers in stope will also create an additional work burden.
- Constant supply of electricity to the fans will be very important and critical to the feasibility of mining with a bord and pillar layout.
- Other environmental issues such as methane accumulations in the up-dip panel and the danger of “used” air being drawn over the next set of fans, makes one believe that flameproof fans will become an option to consider, which will make this type of layout even more expensive.
- Another important aspect to consider is the water handling in the working environment. The handling and control of wastewater is an area of concern, as the strike gully in which the

boxholes from the footwall haulage hole into will have to collect all the water from the panels above. The main intake of air according to the proposed layout will be through this strike gully and the excess “hot” return water will have a negative impact on the cooling ability of the air.

- ❖ A total mine heat flow and air requirement analysis (macro analysis) needs to be done to quantify the total costs of the final airflow and cooling requirements for a bord and pillar layout.

4.5 A recommended alternative method

From the layout that was given, the ventilation and cooling requirements were done on a micro scale and various areas of concern were highlighted. In terms of a bord and pillar layout for an ultra deep mine, it is important to keep in mind that two critical parameters will determine the feasibility of mining a specific area, and that is the rock mechanic and ventilation and cooling constraints.

In terms of ventilation and cooling requirements for such a layout, it is important to ensure that the layout must be simple and easy to ventilate. The layout as it stands now, is not conducive to safe and easy mining (in terms of the supply of good and healthy air) and too many factors do exist (as mentioned in this report) that will have a detrimental effect on the mining process. The best way of solving the problem to ventilate a bord and pillar layout, is to eliminate the method of using fans for ventilating production stoping panels, and to make the coursing of the air as easy and convenient as possible.

From experience, it was found, that although it might take a little bit longer to establish, that pre-developed raises and panels gave higher production potential, but that airflow control and maintenance were better manageable. The layout that was proposed is workable, but there is an indication based on physical comparison, that it would be quite difficult to sustain. Figure 4.5.1 shows a slight modification of the design and will be discussed shortly.

The main difference between the layout as proposed in Figure 4.5.1 and the previous layout discussed, is that all the up-dip panels will “disappear “as they would have been carried as strike or breast sidings in the T-raise as shown in Figure 4.5.2. In this design, through ventilation is established by developing the reef drive on both levels and between the two adjacent x-cuts, until holing is affected. This will ensure that fresh air will be available for the 6 up dip T-raises to be developed. If we assume that the bord width will be 15m as per the original design, it will mean that with a 2.2m wide gully (to cater for rock storing capacity), that 6.4m shoulders (edge of gully to end of siding) will be carried. This represents an approximate 16128m^2 (210m gully length and 12.8m shoulders) of production in the pre-development phase of establishing a stope comprising of 6 pre-developed raises.

The T-raise concept has been widely used in the industry, and is nothing new. The advantage of doing the T-raise is that the ventilation column and services can be carried in the raise without being damaged by the scraper during cleaning operations, which in the previous case would hamper ventilation control. The width and depth of the raise is dependent on the ground holding capacity of the boxes, as well as the production rate required from the panels, once through ventilation has been established. Figure 4.5.3 shows a plan view of the T-raise, with the shoulders lagging behind within a design parameter distance.

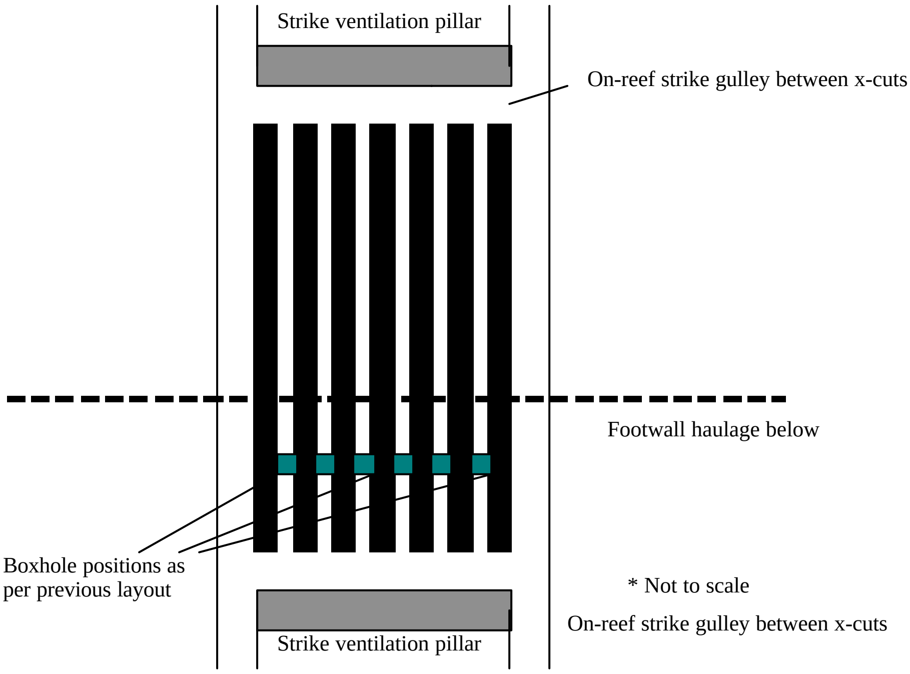


Figure 4.5.1: Airflow arrangement for bord and pillar layout (pre-developed).

The ventilation column in the T-raise could be a standard 570mm force column and a 20-30kW powered fan (with possible airflow quantities delivered between 3.5 and 7m^3/s), as the distance to develop this T-raise would be substantial (in the order of 200m). It is important to note that once through ventilation has been established, the fans can be removed and will not be used in the stopes again. In mining the shoulders in the T-raise, the air can be coursed onto the siding by making use of ventilation brattices. This will make this design very attractive, as it takes away all the problems with the in-stope fans and pillar holings, as was shown in the previous design. The strike ventilation pillar will play a very important role in the coursing of the available air and it will also enable the easier design and identification of ventilation districts.

The other advantage of developing the 6 T-raises together is that it is in effect defining the mining area exactly, as faults and dykes can be picked up as you develop and in this way establish off-reef pillars. In the other design it will not be possible. The establishment of the dip pillar line can therefore be controlled much more easily. The other advantage in terms of the ventilation of the T-raises is that once the T-raise holes in the reef drive above, the raise will immediately be available for production. This means that a stope can be in the development and stoping phase without any changes to the amount of air available. Once all pre-developed T-raises have holed, all the dip pillar lines would have been established and the coursing of the air to the strike mining sections as indicated in Figure 4.5.4 can be done quite easily by hanging ventilation brattices in the T-raise at positions where the air has to be coursed onto the strike faces. In Figure 4.5.4, the ventilation brattices and the possible positions thereof, is shown with the dotted lines (only shown in T-raise A).

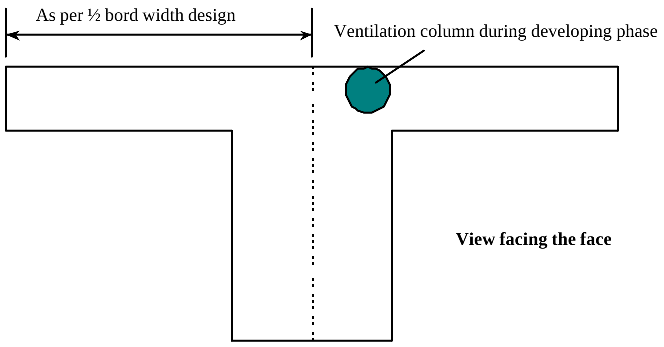


Figure 4.5.2:Section of basic T-raise development (view facing the face).

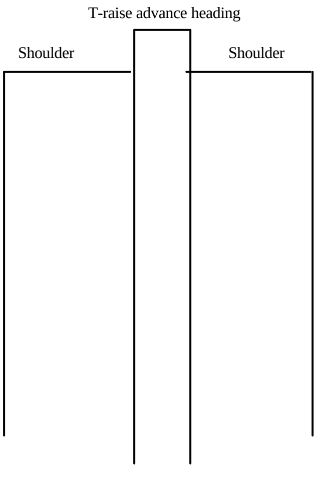


Figure 4.5.3: Plan view of T-raise with shoulders.

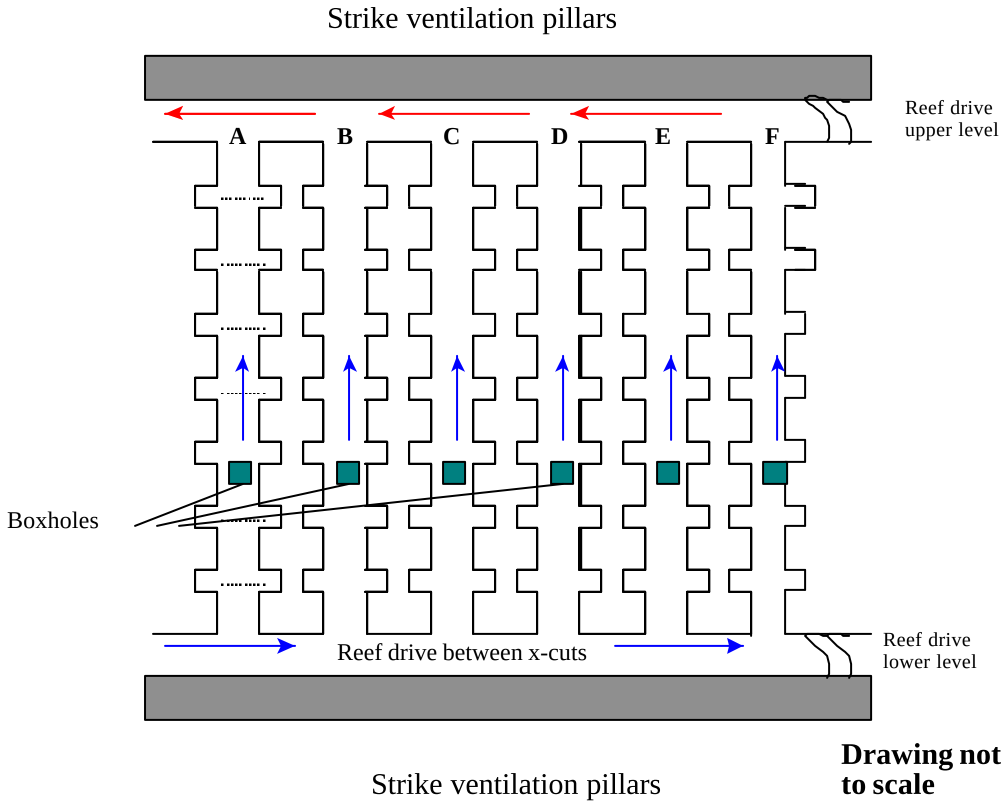


Figure 4.5.4: Detailed airflow arrangements in pre-developed wide raises.

The boxholes indicated in Fig 4.5.4 were also in the previous design, and height allowing, could easily be one box hole and a y-leg to two T-raises and this aspect should be investigated. The part of the raise below the boxes can also be mined with the help of the winches that were installed at the bottom of the raise in the reef drive, immediately making additional mining ground available after the T-raise has holed.

After holing of the T-raise, a dip distance of 210m (as with the previous layout) will be available for mining east and west. In the layout shown in Figure 4.5.4 it will mean that 7 panels of 15m each, will be available for mining on either side of the T-raise. Half of the 210m of face will be actual faces being mined. The other half of the 210m raise will consist of pillars that were already established during the development phase. If we assume that the airflow will be coursed through the actual faces and we assume a distance of 5m from the face of ventilation available, it will mean that the minimum mass flow on the face would have to be 2.34kg/s (density taken as 1.5kg/m^3 and the air velocity 0.25m/s minimum), say 2.5kg/s. This might be the minimum air mass flow needed, but might not be the amount of air mass flow needed to ensure a reject temperature of 27.5ºC per stope, as was achieved with the previous design. In the other layouts, the mass flow of air needed was 7.5kg/s and 10kg/s for the conventional breast and bord and pillar layouts respectively. Keeping the assumptions for the simulation the same, it is important to establish what the amount of air needed on the face will be will be for this design. The results of the simulation for the new design are shown in Table 4.5.1.

The layout is simulated by having a raise and 6 breast panels, 15m long, on either side of the raise being mined from it. From one breast panel the air is coursed and then sent to the next breast panels east and west. In this way the pillar position and effect on heat flow is also included. It was assumed in this case that only one winch per raise line would be needed. The air was also introduced at the bottom of the raise at 20/20ºC and it was important to establish what the reject temperature for various mass flows of air would be. The results are shown in Table 4.5.1.

Table 4.5.1: Mass-flow of air versus reject temperature (recommended design).

Mass flow of air (kg/s)	Quantity of air (m^3/s)	Density of air (kg/m^3)	Reject temperature (^oC)
5.0	3.4	1.5	30.1/32.8
7.5	5.1	1.5	27.5/30.1
10.0	6.7	1.5	25.9/28.4

From the results for the pre-developed design it is obvious that it compares favourably with the results as indicated for the micro analysis for breast panels as shown in Table 4.3.2 which is very promising, in the sense that there is no additional new heat load in the stope, due to the fans in the stope, being present in the previous design. The reject temperature for the above is also almost the same for various mass flows of air, as for the breast panels.

From the proposed new layout mentioned above, the following abbreviated findings are:

- It is imperative, whatever mining design that will be used to do bord and pillar mining, that the pre-development of raises and raise lines, will be a very important part of the design.
- The supply of air by means of fans in the stoping environment should not be part of a global planning strategy for ultra-deep mines. The risks associated with fan ventilation in stopes are too big.
- Fan ventilation in stopes, will increase the risk of methane related problems and will enforce the inclusion of flameproof fans.
- The repetitive nature of installing fans and ventilation brattices and removing them daily can and will create a feeling of resistance to the initial proposed system.
- Once through ventilation has been affected in the stope, no fans are needed in the stope,
- Once through ventilation has been affected, the flexibility of having different panels available for mining becomes quite obvious. In this design 7 possible panels on either side of the T-raise will be available. In a stope consisting of 6 holed T-raises, this means 84 of 15m each, possible panels to be mined. In our initial design it was calculated that 6 panels of 15m each, with 15m face advance per month ($1350m^2$) was needed per stope to deliver the total required production rate of $45000m^2$ per month.
- In the proposed layout the advantage of having through ventilation and the subsequent ease of coursing the air onto the faces where it is required, is evident.
- Cleaning of dip gullies becomes easier as no fans and electrical cabling is in the path of the scraper. Dip and strike ventilation controls are also much easier to install and maintain.
- Once a pillar holing has been affected between two T-raises, the pillar holing can be cleaned and the dip ventilation controls be installed immediately.
- The positioning of in-stope coolers is much easier in this design, as they can be easily be put at the bottom of any T-raise.
- An approximate $16000m^2$ is possible per stope during the pre-development phase. The same principal of fan ventilation for doing stoping and development applies, with the main difference being, that in the case of the pre-developed T-raises, it will be much easier to be controlled.

4.6 Comparison criteria used as for other layouts

The following list of criteria was used by Bluhm to compare the layouts mentioned in his research. The original design of continuously using fans to ventilate the up-dip headings in the bord and pillar layout, have been discussed and evaluated in terms of various set design parameters. Various shortfalls have been identified and a new design (slight deviation from the original layout), which is based on the pre-development of T-raises to establish through ventilation from one level to the next level, was proposed as an alternative and various advantages for this new design were identified. How the new design of a pre-developed T-raise layout for a bord and pillar layout will fare in terms of the various criteria set will briefly be discussed.

❖ Planning and general layout criteria such as: degree of concentration of ventilation areas and potential for creating ventilation districts, controlled re-use and re-circulation of air, minimising secondary ventilation leakage, multi shift blasting, reducing re-entry periods, vamping and closure.

  - In the design as proposed, the establishment of ventilation districts will be much easier to do and the introduction of a re-circulation strategy can also be introduced quite readily. The control of ventilation leakages will be very much easier as there is no real open stope back areas as pillar holings on strike will be sealed off immediately once the holing has been cleaned and swept. The permanent ventilation control in terms of strike walls will also not be damaged by scraper cleaning, as no cleaning on strike will be necessary after the two breast faces between two raises and holings have been affected and cleaned.
  - Multi shift blasting will be possible in the sense that each raise line has it's own dedicated return air-way. This however needs to be investigated further.
  - Re-entry time periods for stope raises can be shorter as there is a dedicated ventilation route per raise.
  - Vamping and sealing off of each raise line that has been mined and completed, within a stope becomes very easy to do and control.

❖ In stope vent control criteria such as: potential for in-stope ventilation control, minimising in-stope ventilation leakage, reducing uncontrolled re-circulation and avoiding short -circuiting.

  - The issue of in-stope control has been dealt with, but it is obvious that the in-stope control with this method will be very much easier to do and maintain. The reduction in in-stope control can also be minimised in this stope. Short-circuiting might be a problem once pillar holings have been affected and not sealed off properly.

❖ Cooling arrangement criteria such as: potential for using in-stope coolers and for water handling/management (in-stope and crosscuts)

  - As mentioned before the use of in-stope coolers become very attractive, as each raise or groups of raises can be serviced by an in-stope cooler and the installation will only have to be made once and not moved around in the stope, which will be a major advantage. Water management can be done much easier, as water will be collected from the six individual raise lines by means of pump columns fitted to the hangingwall in the raises and portable pumps in the breast panels. Water pumped in pipe columns and immediately removed from the stope environment, will cause an improvement in the stoping conditions.

❖ Development requirements criteria such as: potential for minimising the need for multi blasting.

  - Multi blasting as such is not always detrimental in terms of the work environment, but in this case it will be possible to do if properly planned and controlled.

❖ Contaminants criteria such as: potential for minimising build-up in air contaminants.

- From the layout it is obvious that contaminants will immediately be removed from the stope area and can be sent to the RAW which each level does have, therefore minimising the risk to workers. The establishment of real ventilation districts for this type of layout still needs to be investigated in the context of a macro layout.

❖ Escape and rescue criteria such as: ease of escape and evacuation, ease of fire fighting within layout, potential for minimising risk and it's impact on safety and production.

  - In terms of a rescue strategy it is obvious that each stope will have 6 possible main escape routes to through ventilation. The fire risk is reduced dramatically as no timber other than local support sticks will be used between the pillars. In the event of a fire in the area the affected raise line or stope can be easily sealed off. The impact on production, depending on the ventilation district and the size thereof should be minimal.

The idea with the above comparison was to show that a bord and pillar layout and with specific reference to the ventilation issues, is feasible, but that the design thereof needs to be done in great detail without sacrificing any of the real issues of concern as mentioned before.

4.7 Conclusions and future research needs

From the above mining layouts discussed and the various issues pertaining to the ventilation requirements for a bord and pillar layout the following conclusions are drawn:

❖ Ventilating and cooling of a bord and pillar layout as suggested by de Frey is possible, but has a lot of complicating design aspects as highlighted in this report.
❖ The initial design proposal can be changed slightly and present very favourable design features, of which the pre-developed T-raises to establish through ventilation in the stopes, are the most important.
❖ In order to create a safe and healthy environment, it is important not to complicate the design with unnecessary mining procedures, such as mechanical ventilating stopes with auxiliary fans if another design is possible.
❖ The airflow quantities in ultra deep mines should be minimised and unnecessary additional heat loads in stope will defy the objective.
❖ The results from the micro design investigation must be used in a macro-analysis with other layouts as was done in previous research.
❖ In terms of a macro layout, once it is available, the following aspects will have to be investigated:
  - The amount of air per level and cooling required based on the actual global bord and pillar layout (specified production as per level).
  - The various heat loads as per level and the temperature differences and reject temperatures per level and the optimisation of cooling needs.
  - The comparison of heat loads, air supply and cooling requirements with other mining layouts such as longwall mining, breast etc.
  - The impact of the different amounts of development needed for bord and pillar versus those of other layouts.
❖ The effect of re-circulation of air on the total air quantity from surface, for a bord and pillar layout.
❖ A macro analysis and comparison with other designs at this stage was not possible, as deep mine project 3.2.1 and deep mine 3.2.1 phase two are deemed intellectual property of several mining companies and Miningtek. It is therefore proposed that a macro analysis of a bord and pillar method be done and that the results thereof be compared with industry air supply and cooling requirement averages.

De Frey felt that the cost of doing the above macro analysis for the SSSP or bord and pillar method at this stage was not warranted and budgeted for. It could however at this stage be reasoned that the following facts and figures derived from the analysis done by Graphic Mining Solution International (GMSI) are the essence of the issue (see chapter 5):

The time to reach full production of 45000m^2 compares favourably with any of the other known mining methods.

The actual m^2 /m ratio is higher and therefore requires less development to produce the target and therefore less heat load.

Should insulation of haulages for inlet air be considered, the cost of insulation should be less.

The number of levels required to produce the 45000m^2 are less than for the other methods.

Concentrated mining is possible.

5 Mine economics

5.1 Mine economic criteria

The criteria that make a substantial contribution to the economic performance of a hard rock mine at depth are: access, transport, productivity, time to get to full production, ore-body extraction, and costs.

5.1.1 Access

The following issues prevail in regards to access ways in the bord and pillar layout:

- Shorter travelling time and handling distances due to shorter X-cuts from the RAW to the reef horizon
- Less maintenance on access and escape ways
- Storage bays for timber and material between pillars where x-cut intersects reef horizon
- Access to isolated pay blocks or problem areas only restricted by the required time to develop or equip
- Less sophisticated and expensive support of access ways due to less squeeze
- Due to smalll closure, easy access to old stoped areas in the event of a fire u/g
- Bigger vertical distances between twin haulage levels, for example 240 m intervals with intermediate f/w drives 50m in the f/w of the ore body, halfway between main levels, will improve the square metres stoped to metres developed ratio
- Having intake and return air facilities readily available on strike makes faster opening up and stoping possible
- Pre- or follow-on development is optional

A disadvantage is that access to stopes will be from the lower level until holings with the upper level are affected. Travelling up and down up-dip panels for a distance of 38 m until holings into strike panels are affected, could also be seen as a disadvantage.

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5.1.2 Transport

The issue of transport can be read in conjunction with the access issues. The possibility of handling less men and material to the stopes and less waste rock out of the mine should have a major influence on the costs of the mine.

High speed tramming and time restrictions for cleaning are limited by the re-entry period only.

Getting the required production over fewer levels should help in keeping transport costs down.

The transport of material and equipment from the lower level until a holing is effected with the upper level is a distinct disadvantage

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5.1.3 Productivity

The following issues are relevant:

- Risk of exposing sophisticated machinery and equipment to seismic events is diminished as low ERR and ESS values indicate
- Better stope width (s/w) control due to less seismicity and better h/w control
- Less dilution of ore as a result of fewer scraper gully excavations and less seismic activities
- Less loss of blasts due to faces being out of production as a result of seismic events and falls of h/w
- Possibility of not having night shift workers in stopes, as cleaning can take place on day shift. Cast blasting will leave faces clear for face preparation and drilling
- Volume of area mined (stoped) kept open as a result of less closure due to low ERR. Area thus made available for waste stowing or backfilling, preferably from waste development in the immediate vicinity
- Less gold loss – more time available for cleaner sweeping and vamping
- Backfilling from surface optional
- Possibility of blasting same or different faces more than once over the 24 hours
- Improved face advance leads to overall productivity improvements
- Improved mining and stoping flexibility
- Ledge and cutting of gullies optional
- Ease of moving and reclaiming old and new equipment (mechanisation and automation)
- Reduced propagation of fire in stopes due to less timber packs having to be used as support
- Grade control is flexible
- Control of waste stoped and developed is excellent
- Up-dip overhand stoping assists negotiation of geological features
- Availability of extra faces ahead of working faces
- Follow behind development irrelevant
- Predevelopment – no problem. Allows timely discovery of geological disturbances
- Double handling of ore – centre gullies not used
- No interference of pillars when positioning service, travelling and airways
- Minimal face scraping due to use of cast blasting – shorter panels (length of face per scraper)
- Slotting of pillars for ventilation is part of the face advance process
- Pre-development without the problem of squeezing due to excessive stresses makes expensive support of tunnels unnecessary
- Flexibility of pillar position good before stoping commences i.e. in the planning stage. Poor after stoping operations have started.
- Bracketing pillars of geological features – not required
- Gullies' stability very dependent on pillars
- Re-establishment of gullies for drainage – probably not required as a result of overhand or up dip mining
- Dependency of f/w development on face advance – nil
- Rolling stock requirements- normal due to flexibility of the mine layout probably less due to getting the required production from fewer levels
- Concentrated supervision – dependent on whether concentrated or scattered mining is being practiced
- Electrical reticulation requirements – normal
- Adaptability to team and multi-skill requirements – excellent

- Adaptability to mechanization and innovative changes -well suited for narrow as well as wide reefs as access ways remain open for longer periods
- Less waste rock transported as a result of less seismic events, better stope width control and shorter x-cuts to reef

Cleaning of strike panels was seen as a disadvantage coupled with poor hangingwall control expected because of tight end blasting. Discussion with authorities on this matter (paragraph.2.4.3) proved it to be of little consequence.

5.1.4 Good economic sense

The following reasons make good economic sense and support improved productivity and profit:

Concentrated mining and supervision
Faster production build-up
Constant production rate over a long period
Quick closure at the end of the life of the mine
Less waste development and resultant less handling of waste rock
Better s/w control
Less loss of face blasts
Less loss of gold due to improved conditions for thorough sweeping and vamping
Costly backfilling not a prerequisite for deep mining
High cost of installing sophisticated support in excavations where stoping is taking place is not required.
Lower ventilation costs as a result of the effect of concentrated mining and producing the 45000m^2 from fewer levels.
Better flexibility for grade control.

One could reason that having to leave reef pillars as permanent support is undesirable, but until a better solution is presented one should do the best you can with the present knowledge and technology that is available.

To quantify in detail the different issues would require a more intensive analysis and research. Comparing and weighting different sizes, spans and configurations and their effects on the non-rock engineering considerations does not fall within the scope of this report.

5.2 SSSP and the DEEPMINE project

Graphic Mining Solutions International (GMSI) were subcontracted to analyse the mining of the Iponeleng ore body of the DEEPMINE project using SSSP on the same grounds as those used for analysis of the other methods used for the DEEPMINE.

The primary objective of this project was to produce a life of mine (LoM) design and schedule of the DEEPMINE ore body for the bord and pillar (SSSP) mining method.

In order to expedite the design and schedule, the time constraint of 19 days being fixed, certain details were left out of the design. Firstly, only in the micro design were all ore-passes, travelling-ways, etc designed. Ratios for these were calculated from the micro design and included in the macro design reports only. Secondly, each raise, and therefore the panels, are designed on the average dip of that raise and would not follow the reefs undulations as it was designed in Project DEEPMINE.

The quid pro quo was that the exclusion of this design and schedule allowed an entire "Life of Mine" plan to be produced and analysed. This allowed correct and appropriate comparisons to be made with the original Project DEEPMINE mining methods.

All graphs and data in a report format allowed the SSSP mining method to be compared to the other DEEPMINE project mining methods.

5.2.1 Geological structure

The ore body is divided into three “mining blocks”. The first block is delineated by the geological features on either side of the shaft and the second by a 20 meter throw strike dyke running more or less between the – 4 400 meter and – 4 500 meter contours on the right hand side of the ore body. These three “mining blocks” are mine separately from each other, thus enabling the required production rate of 45 000 m^2 per month to be achieved. The objective is to mine from the centre line (station x/c) laterally and on dip as ore reserves become available.

5.2.2 Production results

Figure 5.2.1 shows the position of the stope faces after 5 years of production as the mine was approaching its target of 45000m^2

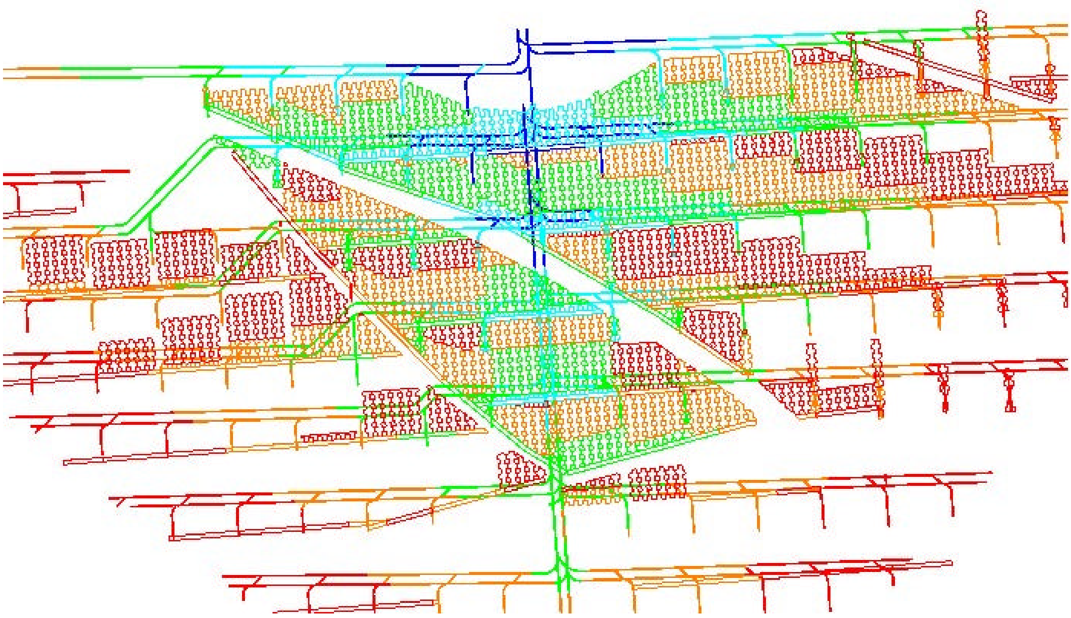

Figure 5.2.1: Plan of the stope faces after 5 years of production.

5.2.3 Scheduling results

Figure 5.2.2 shows the annual production in years and indicates the fast build up at the beginning of the life of the mine as well as the short closing down period.

The production profile is maintained at 45000m2 per month for 17 years (2007 to 2024). The levels of each block are developed out from the station at normal rates of 45 meters per month and the stoping at 15 metres per month. A number of mining blocks may not produce at full capacity where a dyke or fault runs through the block. All optimisation is done on an entire raise line and not on individual panels.

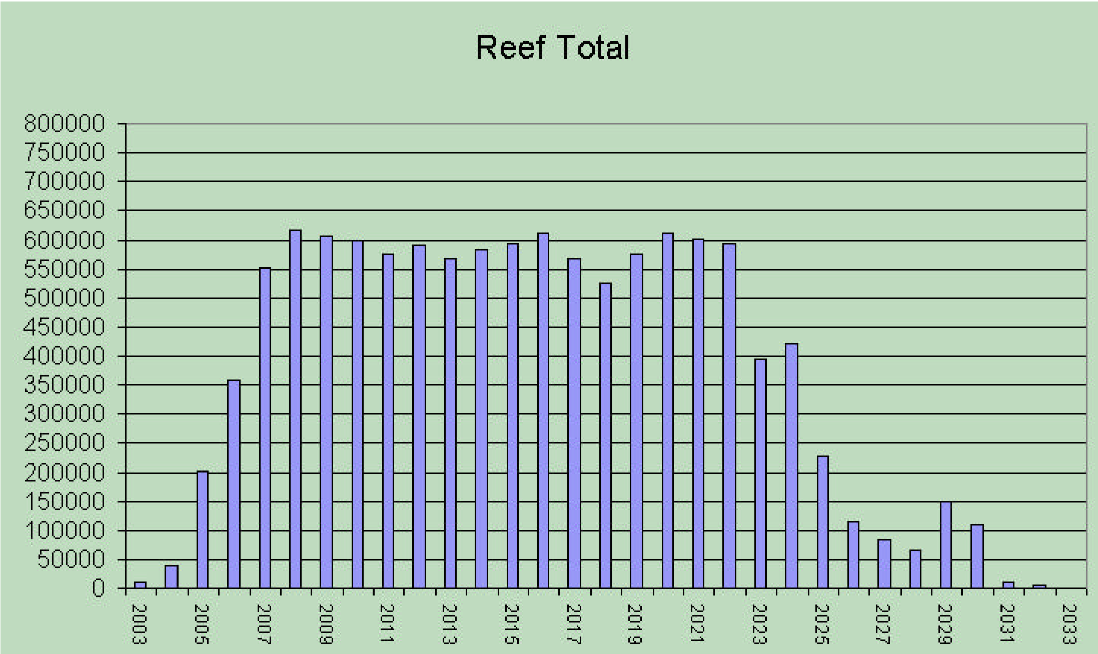

Figure 5.2.2: Production profile.

Figure 5.2.3 is a line diagram showing the annual production from the raise panels, stope panels and bottom strike panels.

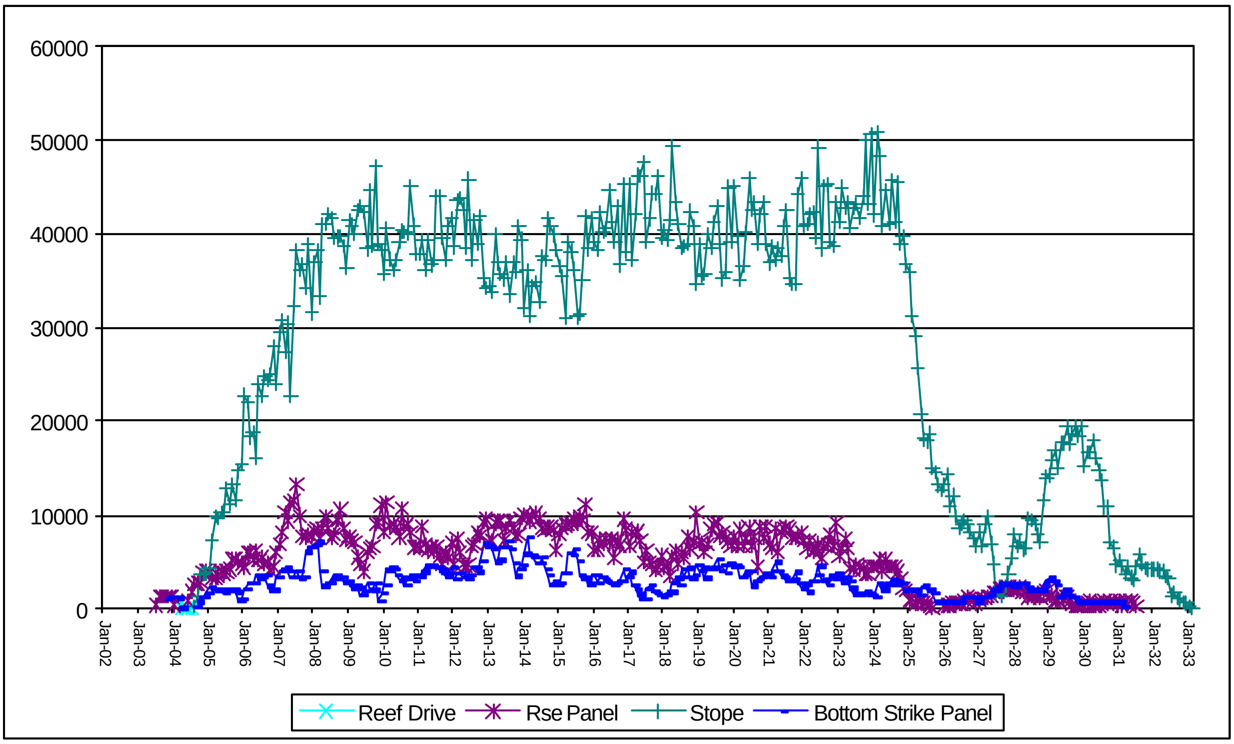

Figure 5.2.3: Production from the in stope reef sources.

5.2.4 Breakdown summary

Table 5.2.1 is a breakdown summary of the performance of the mine over its life.

Table 5.2.1: Breakdown summary of mine performance

Description	
Total Resource Area	18 454 000m^2
Shaft Pillar Mined Out	228 136m^2
"Shaft Pillar" Pillar	12 469m^2
Dip Pillars	0
Clamping Pillars	0
Area Mined over life of mine	12 682 425m^2
Area Mined over life of mine Pay	11 561 509m^2
Area Mined over life of mine Unpay	1 120 916m^2
Off Reef Development	263 123m
Reef Development	2 642m
LOM	31 years
Time to reach 30 000m2	4yr 2m
Time to reach 45 000m2	5yr 5m
Off reef Development to reach 30 000m2/m	576m / month
Off reef Development to reach 45 000m2/m	865m / month
Other Losses (Geological/In-accessible/ etc)	826 000m^2
Actual m2 / m ratio	52m2 / m
Time to reach 1st stoping production	18 months
Max levels mining	5 levels
Extraction Ratio	72%

All mining on the additional levels have been designed according to the same rules and mining rates as set in the DEEPMINE Project. The total production volume is 12 682 425m2, ending in 2033 (start of schedule is 2002), with 243 612 meters of development, ending in 2028. The development ratio over the life of the Mine is 52m^2/m. From the total reef to be mined the overall extraction ratio is 72%. The ameliorating risk formula of Handley was not applied for this exercise.

5.3 Comparing SSSP (bord and pillar) with DEEPMINE methods

Table 5.3.1 compares the outcome of the analysis made by GMSI of the performance of the DEEPMINE project mining methods with the SSSP methods. The DEEPMINE methods were longwall with strike stabilising pillars (LSP), sequential grid method (SGM), sequential down dip method (SDD), closely spaced dip pillar method (CSDP). The comparable figures were compiled from information gathered from the paper by Vieira et al (2000).

Table 5.3.1 Comparing SSSP (bord and pillar) with DEEPMINE methods.

Description	LSP	SGM	SDD	CSDP	SSSP
Cross cut spacing (span) in metres	63	200	100	180	210
Pillar width (metres)	40-50	40-50	25	40	15*15
Length of back (metres)	240	240	240	240	240
Backfill	Yes	No	No	Yes	No
F/w haulage below reef	145	120	50	90	30
Tunnel stresses under dip pillars (MPa)	140-260	190-230	50-91	50-91	Close to virgin stress.
Average pillar stress (MPa)	450-695	320-520	310-510	300-500	290-482
Levels for 45000m^2 /m	11	12	13	12	5
Life of mine (years)	45	31	31	26	31
Area mined in millions (m^2)	13.1	11.7	11.4	12.2	12.7
Resource in millions (m^2)	19.7	19.7	19.7	19.7	18.5
Extraction (%)	67	59	58	62	72
Locked in pillars in millions (m^2)	4.1	7.8	4.7	5.2	5.5
Off reef development (km)	552	290	344	381	243
Span between pillars (m^2)	240	160	75	140	15

From the results shown in tables 5.2.1 and 5.3.1 the SSSP method compares favourably in most categories and shows its superiority when seen from a rock-engineering point of view considering the lower average pillar stresses and field stresses near the footwall haulages. The SSSP method is able to reach the 45000m^2 production target in more or less 5 years, from only 5 levels. It also requires the smallest amount of development to extract the ore body. The extraction of 72% is better than the other methods. One can therefore expect the SSSP method to perform economically better than all the other methods

6 Conclusions

From the results in table 5.3.1 it is clear that the SSSP method, after investigating the different problem areas, on paper proves to be the better mining method.

Handley notes the following important advantages:
1. 60% of the area mined will be trouble free.
2. With backfill it will probably be possible to get up to 85% or close to this level of extraction safely in a second pass.
3. Mining will be rate-independent (can mine at a high rate) and geology-independent.
4. Average pillar stress is considerably lower for bord and pillar than for the other methods with comparable extraction ratios (See table 3.6.1)

Some rock engineers still feel that the strength of the small pillars is suspect and are afraid of a "pillar run". There is however no proof that this would happen. Until this can be proven, the SSSP method, considering the known rock engineering and safety (seismic events) criteria, has the least risk. Considering the poor safety record of the gold mining industry, the stakeholders should insist that this matter receives urgent attention.

Webber in his investigation into ventilation of the bord and pillar "back stopes" illustrated that there are many ways that one can overcome obstacles once one sets your mind to it. When one considers the macro criteria of ventilation for the mining of 45000m^2, the ability to do this from only 5 levels with less development, speaks for itself.

Investigating economic considerations showed that the SSSP method compared favourably with the other DEEPMINE methods. The main advantages being able to produce 45000m^2 from only 5 levels, the possibility of 72% extraction and requiring less off reef development to do it.

7 Recommendations

Because the rock engineering aspect still presents conflicting findings that cause doubt as to its safe implementation and considering the major advantages should it be successfully implemented, it is recommended that stakeholders give the SSSP concept its urgent attention.

De Frey would like to suggest that after the scheduled workshop has been held a panel be elected to define the route to be taken to prove or disprove the doubts and conflicting evidence put forward. Special attention should be given to critically examine the evolution of the theories still being applied. De Frey found that popular formulae are now being queried as to the assumptions being made.

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9 Appendices

9.1 Proposal for project GAP 828.

DEPARTMENT OF MINERALS AND ENERGY
PROPOSAL FOR A PROJECT TO BE FUNDED IN TERMS OF THE MINERALS ACT

- CONFIDENTIAL -

DME REFERENCE NUMBER	TO BE SUBMITTED BY 12:00 NOON ON THE CLOSING DATE
______________________ **(FOR OFFICE USE ONLY)**	

1. PROJECT SUMMARY:

PROJECT TITLE: Examine the criteria for establishing the small span small pillar concept as a safe mining method in deep mines

PROJECT LEADER: F.S.A de Frey

ORGANIZATION: Mine Engineering Consultants CC

ADDRESS: P.O. Box 39005, Faerie Glen, 0043

TELEPHONE:(012) 991 4568 TELEFAX: (012) 991 4568 E-MAIL: miencons@global.co.za

PRIMARY OUTPUT[1]:	Proof that the method is a safer, economic and viable method for mining of deep narrow ore bodies.
HOW USED?[2]:	As a mining method.
BY WHOM?[3]:	Mines that are or will be mining deep tabular ore bodies.
CRITERIA FOR USE[4]:	Successful implementation of the small span small pillar mining (stoping) concept.
POTENTIAL IMPACT[5]:	A quantum improvement in the safety risk and performance of seismic prone mines.

FUNDING REQUIREMENTS (R 000s)	YEAR 1	YEAR 2	YEAR 3
TOTAL PROJECT COST	R212,5-VAT		
TOTAL SUPPORT REQUESTED FROM SIMRAC	R212,5-VAT		

DURATION (**YY/MM**) 2001 June TO 2002 April

SIMRAC SUB-COMMITTEE:	AU/PT	X	COAL		OTHER		GENERIC	

EXPLANATORY NOTES:

This form must be completed in typescript and all required information must be provided for the proposal to be considered.

The following explanatory notes provide definitions of key terms according to the superscript numbering the SIMRAC 1 form.

1.	PRIMARY OUTPUT	Indicate what product, service or information it is intended to deliver as a PRIMARY OUTPUT of the project.
2.	HOW USED?	Indicate how the PRIMARY OUTPUT would be used in practice.
3.	BY WHOM?	Indicates by whom the PRIMARY OUTPUT is expected to be used.
4.	CRITERIA FOR USE	Indicates key characteristics of the PRIMARY OUTPUT which must be satisfied for it to be of value to the user.
5.	POTENTIAL IMPACT	Indicate the extent of changes and benefits which would be achieved by use of the PRIMARY OUTPUT , in quantitative terms where possible.
6.	OTHER OUTPUTS	OTHER OUTPUTS are outputs which would be of relevance for application by industry in the form of secondary or interim results which will be achieved as an integral part of the project and produced as part of the activity designed to produce the PRIMARY OUTPUT.
7.	ENABLING OUTPUTS	ENABLING OUTPUTS are outputs which are necessary to enable production of the PRIMARY OUTPUT. An ENABLING OUTPUT may therefore be seen as an input to the next stage of the project and is not necessarily an output delivered to an end user.
8.	METHODOLOGY	A list of steps outlining the methodology to be adopted to achieve each ENABLING OUTPUT. Where appropriate, comments on special equipment or procedures should be included.

9. Where an output includes a device, mechanism, procedure or system capable of being applied in the mining environment, suggest how the outputs in question might best be applied in practice.

The following practices should be followed when completing the project costing.

10. Organisational overheads and general capital depreciation should be include under the relevant cost category according to the organisation's normal financial policy.

11. Capital depreciation on items of plant or equipment used for the project should be specifically indicated where such provisions need to be included.

2. PROJECT DETAILS

2.1 PRIMARY OUTPUT [1]

Prove to the mining industry that the small span small pillar method will present safer working conditions in the deep mines.

2.2 OTHER OUTPUTS (deliverables)[6]

Due to less delays and losses caused by seismic events, the performance of deep mines will improve.
There will be an overall improvement in the production, efficiencies and the morale of the employees.

2.3 ENABLING OUTPUTS[7]

NO.	ENABLING OUTPUT	MILE -STONE DATE (MM/YYYY)	MAN DAYS
1	A literature study of the work (research) done into small span small pillar mining so as to compare it to other methods.	07/2001	5*2 =10
2	Study of all reports forwarded to or issued by the Deep mine project regarding seismicity, rock engineering, ventilation and, economic viabilities of the other proposed layouts for deep mines.	10/2001	5*3 =15
3	A similar computer simulation of the small span small pillar method as those done of the other methods already completed by Graphic Mining Solutions International (GMSI) and forwarded to the Deep mine project.	12/2001	20*1 =20
4	Prepare and present one or more workshops involving representatives of the Department of Mines and Energy (DME), Simrac, Miningtek, the mining industry and mine labour organisations so as to find a suitable solution.	03/2002	5*3 =15
5	Find a suitable mine and examine the effect of seismic events on its performance. Identify a block of ground that can be used for testing the feasibility of the small span small pillar method of mining	04/2002	10*1 =10
6	Manage the implementation of the method into the suggested or recommended mine.		?

2.4 METHODOLOGY[3]

NO. OF ENABLING OUTPUT	STEP NO.	METHODOLOGY TO BE USED TO ACCOMPLISH THE ENABLING OUTPUT (INDICATE STEPS/ACTIVITIES)
1	1	The project leader and the rock engineer will perform the studies and establish whether the proposed mining method is superior to the other methods from a seismic and rock engineering point of view.
2	1	The project leader, a rock engineer, and a ventilation consultant will study all reports forwarded to or issued by Deep mines regarding seismicity, rock engineering, ventilation, economics and mining methods presently being researched and practised.
3	1	GMSI will use the same computer program as the one used to assess the expected economic results of the other methods tested for the Deep mine project. A similar run will be done for the small span small pillar method .The results will then be compared with the results of the other methods that have been reported on to the Deep mine project.
4	1	Prepare and present workshops for the involvement of the DME, Simrac, Miningtek, the mining industry and the relevant labour organisations.
5	1	Find a suitable mine to test the method.
6	1	Supervise the implementation of the method.

Key Facilities and Procedures to be used in the Project

GMSI facilities.
The analyses of all relevant information available.

2.5 TECHNOLOGY TRANSFER

A Workshop and study of the available relevant information will result in a transfer of technology to the stakeholders.

3. FINANCIAL SUMMARY

3.1 Financial Summary

		R 000s	
	YEAR 1	**YEAR 2**	**YEAR 3**
Project staff costs (from 3.2)	62,5		
Other costs:			
Operating costs (from 3.3)	Nil		
Capital & plant costs (from 3.4)	Nil		
Sub-contracted work (from 3.5)	122,5		
Presentations and Papers (from 3.6)	10,0		
Sub – Total	195,0		
Value added tax*(on R122,5 sub contracts)	17,150		
TOTAL COST OF PROJECT	**212,15**		
Less funding from other sources (from 3.6)	**Nil**		
Support requested from SIMRAC	**212,15-VAT**		

* Only for VAT registered concerns

3.2 Project Staff Costs

Reflect Man Days and Costs separately

NAME AND DESIGNATION	YEAR 1		YEAR 2		YEAR 3	
	MD	COSTS	MD	COSTS	MD	COSTS
FSA de Frey Project leader	25	62500				
TOTAL (R 000s)		62500				

3.3 OPERATING COSTS (Running)

		COST (R 000s)	
ACTIVITY/EQUIPMENT (Items above R10 000)	YEAR 1	YEAR 2	YEAR 3
Nil			
Other miscellaneous items			
TOTAL	Nil		

3.4 CAPITAL AND PLANT COSTS[10]

		COSTS (R000s)	
(i) ITEMS TO BE PURCHASED OR DEPRECIATED FOR MORE THAN R10 000 PER ITEM	**YEAR 1**	**YEAR 2**	**YEAR 3**
Nil			
Other miscellaneous items			
TOTAL	**Nil**		

		COST (R000s)	
ITEMS TO BE MANUFACTURED WITH ASSEMBLED COST OF MORE THAN R10 000 INCLUDING MATERIAL AND LABOUR	**YEAR 1**	**YEAR 2**	**YEAR 3**
Nil			
Other miscellaneous items			
TOTAL **TOTAL (i) and (ii)**	**Nil**		

3.5 SUB-CONTRACTED WORK

SUB-CONTRACTOR	ACTIVITY	COST (R000s) YEAR 1	YEAR 2	YEAR 3
Matthew Handley	Rock engineering	37,5		
Ronny Webber	Ventilation consultant	25,0		
GMSI	Information technology	60,0		
	TOTAL	122,5		

3.6 PRESENTATION AND PAPERS

	COST (R000s)		
ACTIVITY	YEAR 1	YEAR 2	YEAR 3
Work shops etc.	10		
TOTAL	10		

3.7 OTHER FUNDING

ORGANISATION	NATURE OF SUPPORT/ COMMITMENT	AMOUNT (R000s)
Nil		

4. MOTIVATION

(Provide a clear and quantified motivation of justification for the proposal, as well as the main conclusions of a literature survey and the findings of related local and international research. The motivation should include a synthesis of previous work in the project area, both locally and overseas, why the project is proposed what the primary output will achieve and a cost benefit analysis, if applicable. Use continuation pages where necessary but in most cases it should be possible to clearly present the key data and arguments in the space provided.)

The concept of using the small span small pillar mining method as a possible means of addressing the seismic and rock engineering problems of deep mines has been researched by several Rock Engineers. Ozbay, Ryder and Jager "agree wholeheartedly, that further research, together with carefully planned fields trials, is warranted in pursuit of this potentially high rewarding concept." (Addendum 1)

Esterhuizen concluded that "a bord and pillar layout at a depth of 3000m will have minimal effects on the surrounding rock mass…Geological structures, appear to be unaffected by excess shear stresses…There may however be localised effects on geological structures in the immediate vicinity of a bord or pillar. (Addendum 2)

Leach of ITASCA came more or less to the same conclusions as the above and on closer examination of the pillars concluded, "In the worst case, this full 6m of strata would have to be carried by support." (Addendum 3)

De Frey found that "it offered a vast amount of advantages and very few disadvantages regarding rock engineering criteria…. Non-rock engineering criteria put forward as possible showstoppers were: ventilation of dead end panels, cleaning of strike panels, tight end blasting causing poor hangingwall conditions, slow production build up. The main objection remaining would be that it had not been tried at depth." (Addendum 4)

The recent 8 fatalities, as a result of seismic events in 3 different mines, and the comments in the media of the Minister of the Department of Mineral and Energy Affairs and representatives of the labour organisation in the mining industry, again stresses the importance of pro-active control of the causes of seismic events. Resources should be concentrated on resolving the uncertainties presently being put forward by finding a suitable mine where the small span small pillar concept can be put on trial. The parties, who have a vested interest in testing the proposed mining method, could probably negotiate the costs involved.

Over and above the very important safety considerations, the project leader believes that there will be other noteworthy improvements in the performance of the mines as a result of less loss of production due to seismic events. The low ERR will result in keeping access available to do thorough sweeping and effective backfilling, should it be required. Pillar removal at a later date is a possibility. This will improve the percentage extraction. Leaving bracket pillars against geological disturbances in high-grade areas will no longer be necessary. The small pan small pillar method lends itself to mechanisation. The report referred to in Addendum 4 covers the non-rock engineering criteria to be considered in more detail.

5. CURRICULA VITAE OF PROJECT LEADER AND RESEARCH STAFF

5.1 SUMMARY INFORMATION

Project Leader

NAME & INITIALS: de Frey F.S.A. . AGE: 67 years

QUALIFICATIONS (e.g. degree/diploma, issuing institution and date):See addendum 5

SPECIAL AWARDS:

Principal Project Team Members

NAME & INITIALS:. Handley M.F. AGE: 48 years

QUALIFICATIONS (e.g. degree/diploma, issuing institution and date): See addendum 6

SPECIAL AWARDS:

NAME & INITIALS: Webber R.C.W.. AGE: 40 years

QUALIFICATIONS (e.g. degree/diploma, issuing institution and date): See addendum 7

SPECIAL AWARDS:

NAME & INITIALS: AGE:

QUALIFICATIONS (e.g. degree/diploma, issuing institution and date):

SPECIAL AWARDS:

5.2 <u>REVELANT EXPERIENCE AND PUBLICATIONS</u> (one page for each individual listed in 5.1)

NAME: F.S.A. de Frey, M.F. Handley, R.C.W. Webber see addendums 5, 6, and 7

RELEVANT EXPERIENCE: See addendums 5, 6, and7 RELEVANT PUBLICATIONS: See addendums 5, 6, and7

6. DECLARATION BY THE PROPOSING ORGANISATION

I, the undersigned, being duly authorized to sign this proposal, herewith declare that:

- The information given in this proposal is true and correct in every particular.
- This Organization has the basic expertise and facilities required for satisfactory completion of the project and will adhere to the program of activities as set out in this proposal.
- The costs quoted are in accordance with the normal practice of this Organization and can be substantiated by audit.

Signed on this 6th day of October 2000 for and behalf of Mine Engineering Consultants cc

SIGNATURE: ________________________________

NAME: F.S.A. de Frey

DESIGNATION Project leader

9.2 Safety in Mines Research Advisory Committee
Project Summary: GAP828

Project Title	Examine the criteria for establishing the small span small pillar concept as a safe mining method in deep mines

Author:	F.S.A. de Frey	Agency:	Mine Engineering Consultants cc

Report Date:	28th February 2002	Related Projects:	None

Category:	Gold and Platinum	Review of fundamental research	Seismicity and mine layouts

Summary

As a result of investigation by the DEEPMINE project into the criteria required for successful mining of hard rock at ultra-deep depths it was found that bord and pillar (small span small pillar) as a mining method presented very favourable results, from a safety point of view.

The title of the project "Examine the criteria for establishing the small span small pillar concept as a safe mining method in deep mines" speaks for itself.

After comparing the small span small pillar (SSSP) method with other known and practiced deep mine mining methods it was found that the problem areas put forward as possible showstoppers could be overcome. Investigating non-rock engineering as well as the rock engineering criteria proved, in most cases, the SSSP method to be superior.

Some rock engineers are still suspicious of the strength of the pillars and/or competence of the foundations of the small pillars and possible "pillar runs". At present no grounds are available to prove this possibility. Many accepted rock engineering theories, assumptions and practices are now being queried. Considering the possible control of seismicity as well as the poor record of safety regarding rock-bursts and rock falls of deep hard rock mines, stakeholders should insist on resolving this problem as soon as possible.

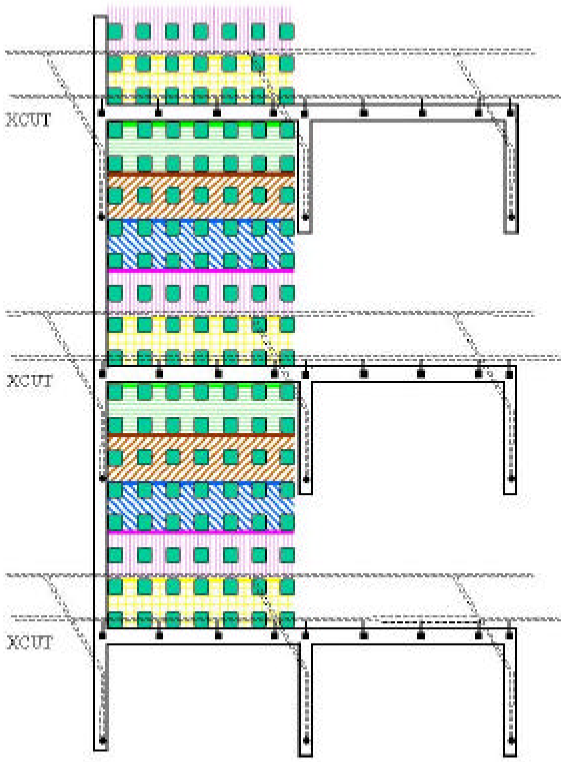
Conclusions

The study has found that the SSSP method on paper is a safer mining method and from an economic point of view compares well with the other known and practiced mining methods.

From a rock engineering point of view Handley claims the following advantages:
That 60% of the area will be mined trouble free.
With backfill it would probably be possible to get up to 85% or close to this level of extraction safely in a second pass.

Mining will be rate-independent (so can mine at a high rate) and geology independent.
Average pillar stress is considerably lower for SSSP than for other methods with comparable extraction ratios.

Webber recommends a way of overcoming the back-stope ventilation problem. Being able to produce the 45000m^2 from only 5 levels, from a macro-ventilation and economic point of view, is a distinct advantage.



General development and stope layout over three levels