Assessment of current mining strategy for steep reefs at Mine A

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Synopsis

The purpose of this investigation was to assess the mining strategy for steep reefs at Mine A mine in 2005 and 2006. Reefs steeper than 35° were discovered when production commenced at the mine. At that time reefs steeper than 35° were not popular in South African tabular gold mines, therefore new techniques of mining them had to be developed. The underhand mining method was chosen for this purpose.

The project was about assessing ventilation, support costs, local support cycles, adherence to the mine's support standards, mine planning and efficiencies of the underhand mining method. The project work was covered by doing a literature research for underhand mining methods and performing a literature review of the mining methods, support methods, ventilation standards at Mine A.

Underground visits were conducted at Mine A mine for data collection, and Mine B was visited to obtain information on support costs and efficiencies. During the investigation it was realized that the underhand mining method was an appropriate method for mining steep reefs; however, there were some challenges related to ventilation and support installation. Efficiencies could be realized with the underhand mining method and planning and scheduling were not different from breast mining.

Introduction

Mine background and general information

Mine A mine is found next to the towns of Orkney and Klerksdorp about 160 km south-west of Johannesburg[1]. The decision to develop Mine A was taken in 1989 and the construction of the main surface shaft was completed in 2000[1]. In 2005, 19 343 m[2] of ground weres stoped and 17 350.4 m were developed[2]. These indicate that Mine A was still an infant mine compared to other gold mines in 2005. At that time the mine had an estimated life of at least 10 years.

Mine A was a tabular gold mine whose reef had a complicated structure, and was found within a complex geological ground. The mine had large-scale and abundant small-scale faults with shale layers, which could cause stability problems[1]. The reef had been tilted by faults and had dips ranging from about 11° to 70°.[2] At the time of the investigation the mine was using a blend of breast mining and underhand mining. Breast mining was used for exploiting both shallow and steep dipping ground and underhand was adopted for steep dipping areas.

Project background

Initially when planning of Mine A was done no steep dipping reef was expected, therefore the planning and design of mining layouts were based on the information that the reef was flat dipping[8]. When production commenced and more drilling was done, it was discovered that some parts of the reef were steep dipping[8]. The geology of the mine was also complex due to abundant small-scale faulting. At that time the mining method in use was breast mining and it was considered inappropriate for mining steep reef[8]. To come up with a mining method best for Mine A a workshop was conducted on steep reef mining on 1 August 2005[8].

Three mining methods were discussed[9]. These are breast, overhand and underhand mining methods. A decision had to be made as to which of the three methods would be the best option for mining steep reef in geologically complex ground. Advantages and disadvantages of each method were considered. These three methods with their advantages and disadvantages are discussed below.

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Breast mining

In this layout the reef is mined in the strike direction and the face is parallel to the raise or the true dip[10], therefore workers are directly exposed to the reef dip. If the reef is dipping by 60° stope employees must work directly on 60°. This method was tested for panels dipping at 32° and was found to be viable[9]. It had the advantage of being the most flexible method for a lead/lag configuration between adjacent panels. This method, however, was not safe and effective for mining panels that were very steep. Gate stulls were required to stop falling objects from hurting people down dip[9]. For panels of about 60° drilling was difficult and installation of supports was not easy[11]. Another problem with this method was difficulty in controlling broken rock after blasting. During drilling operations rocks could fall and easily hurt drill operators and other stope workers[11]. People could not easily move around the stopes and this could affect stope productivity. The layout is indicated in the Figure 1.

Overhand stoping

'In this mining method the stope face is on the up-dip side from workers. Mining progresses in the strike direction but the face is parallel to the apparent dip'[10]. This method had the advantage of reducing the dip to which workers were directly exposed. It also had a best backfill position as workers would always be positioned up-dip from the backfill[9]. Even if a backfill bag loosened, workers would still be safe as it would roll down-dip. Should any spillage occur, people who are on the face would not be exposed to danger as the slurry would flow into the gully down-dip. However, backfill bags would be damaged by flying rock after blasting and water flowing from face would also damage the bags through erosion[9]. See Figure 2.

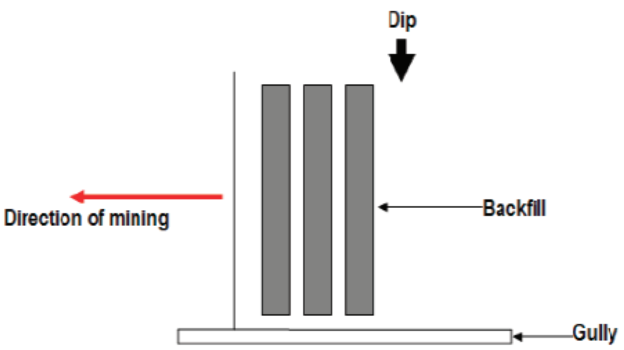


Figure 1—Layout for breast mining method

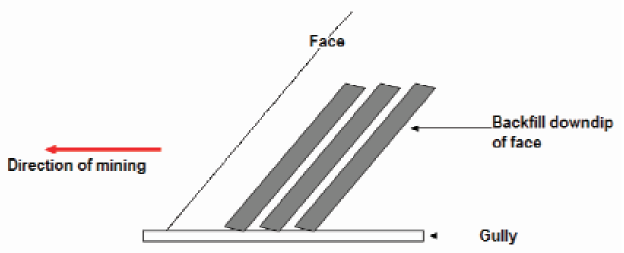


Figure 2—Layout for overhand stoping method

Underhand stoping

In this method the face is down-dip from the workers and the face is parallel to the apparent dip[10]. This method is currently in use because of the advantages it has over other methods. These advantages are:

- 'Drilling is easier
- Broken ore lies on the face
- Good gully control
- Sweepings easy
- Allows any apparent dip less than 30°
- Flexible approach to structures
- Best without backfill, both safety and production
- Easily created from the ledge'[9]

One disadvantage with underhand stoping is that workers are down-dip from the backfill bags, therefore the bags are unsafe[9]. See Figure 3.

A decision was made after the steep reef workshop to use the underhand mining method. After deciding on which method to use, a safe and effective face configuration had to be chosen. Two options, which are indicated in the Figures 4 and 5, were considered.

The face configuration for option A was creating sharp corners where stress would be highly concentrated. The stable gully was also excavated in the fractured zone as the gully followed stoping[9]. This configuration also created a ventilation problem as indicated in Figure 4. Option A indicates the direction of ventilation air. Air would not ventilate the area encircled as it always follows the shortest route (indicated by red arrows in the figure).

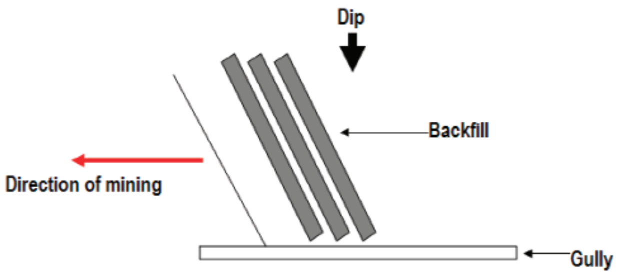


Figure 3—Layout for underhand stoping method

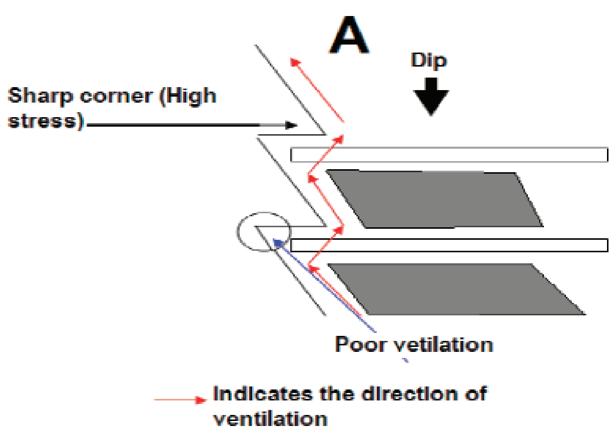


Figure 4—Face configuration A

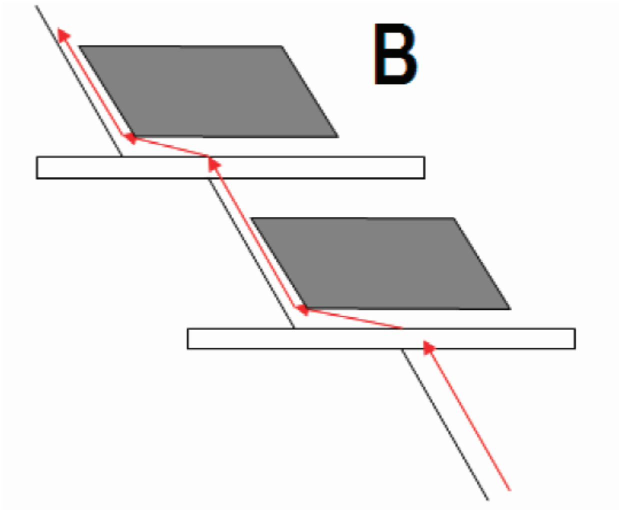


Figure 5—Face configuration B

Option B, which was taken to be the best configuration, prevents the creation of sharp corners and had a good sweeping control[9]. The underhand mining method with the face configuration indicated in Figure 5 started to be implemented in August 2005[8]. A f ew concerns arose with the application of this mining method. Some of those concerns were:

- Whether the planned efficiencies as agreed on by the corporate office and Mine A would be met with this mining method
- Whether support standards could be adhered to with this mining method
- Whether the ventilation standards set by the mine policy and procedures could be met with the underhand mining method and whether the mining method created any ventilation issues
- Whether the mining method could be handled by planning and scheduling techniques used with breast mining
- How the support costs were affected by the use of the underhand mining method[8].

An investigation, which had already been done at the time of the project study, was how the underhand mining method affected mine planning and scheduling. So the study of other four issues of the concern mentioned above was necessary; hence the importance of this project to Mine A.

Problem statement

Assessment of the appropriateness of the underhand mining strategy for steep reefs at Mine A.

Objectives

The following objectives were set in order to complete the project:

- Investigate whether the mine planned efficiencies could be realized with the use of the underhand mining method
- Investigate whether ventilation standards as set by the mine policy and procedures could be adhered to and whether the underhand mining method created ventilation issues or not
- Investigate support methodologies in terms of:
  - Support costs (comparison between breast and underhand mining methods)
  - Whether support standards could be adhered to or not
- Do m^2/m comparison between steep dipping and flat dipping reef.
- Compare planning and scheduling between the breast and underhand mining methods and see if the underhand layout could be handled in terms of planning and scheduling.
- Compare flat dipping reef efficiencies at Mine B with steep dipping reef (where the underhand mining method is employed) efficiencies at Mine A.
- Compare steep dipping reef's support costs at Mine A with flat dipping reef's support costs at Mine B.

Scope of the study

All the objectives were covered except that for mine planning and efficiencies. The available information was not sufficient for a detailed analysis to be made. This also affected on a comparison of efficiencies between Mine B and Mine A. A more detailed study on the efficiencies and mine planning can be done when more information about underhand working places becomes available.

Methodology

In order to achieve the objectives set the following steps where followed:

- Literature research on the mining of tabular steep reefs
- Literature research on the underhand mining method
- Literature review of other mining strategies at Mine A
- Literature review of support strategies and available options at Mine A
- Literature review of the ventilation layouts and standards at Mine A
- Collect data on support costs and mine planning at Mine A, and also interview relevant individuals
- Visit underground workings at Mine A to collect data on:
  - adherence to support standards
  - ventilation practices
- Visit Mine B mine to collect data on:
  - support costs
  - efficiencies

Results from literature study

The literature study was conducted on two topics, steep reef mining and underhand mining methods. The objective was to identify any issues on steep reef mining from other operations which could be relevant to Mine A; and to see if there were any mining limitations that were generic to the underhand mining method. Information was gathered to from previous and current steep mining operations about improving efficiency of the operations, overcoming difficulties associated with support installation, handling planning and scheduling that would be relevant to this investigation. Also it was investigated how underhand operations were ventilated, and how some of the ventilation difficulties associated with underhand mining were overcome. Below are the results of the literature study:

Mining of tabular steep reefs

Durban Roodepoort Deep (DRD) is the only operation that was identified that had mined steep reefs in the past. The operation had an average dip of 60°. It employed walled shrinkage for the extraction of the orebody[3]. The mining method was named walled shrinkage because it was semi-shrinkage stoping, and the broken ore was kept within erected walls. The mining of ore was done by overhand face shape and afterwards ore would be stored in the erected walls. Walls were built on the down-dip side of the face, and they were such that they form a Christmas tree shape[3]. A detailed description of the mining method can be found in Duggan[3].

Before a firm decision was made to mine the ore by overhand face shape, an underhand shape was put into trial. The reasons for putting an underhand face shape into trial were:

- To reduce the danger to personnel from face-bursting or slabbing. It was felt that slabs would lie on the underhand face where they had been removed.
- To reduce the number of accidents caused by slipping or falling[3].

There are other rock engineering reasons why an underhand face shape was put into trial but they are not mentioned in this report as they are not relevant to the investigation.

After 18 months of trial the face shape was changed from underhand to overhand because the underhand face shape did not satisfy some of the performances required. The reasons for abandoning the underhand face shape were:

- Accidents from face bursting caused by underhand were more than those caused by overhand. This is because slabs falling from the face rolled down and hence hurt workers[3].
- Poor breaking efficiency because of gravity. The force of gravity reduced the volume of rock broken[3].

The facts mentioned above are important because:

- Accidents resulting from slabs rolling down the face would still be relevant to steep reef mining presently.
- The breaking efficiency affects the metres advanced per blast and hence the area mined per month. This has a direct effect on the m^2/stope employee per month, which is one of the objectives to be investigated in this project.
- Slipping and falling affect the efficiency of workers. Installation of supports is also difficult in slippery conditions and this might affect the ability of workers to adhere to support standards.

It is worth mentioning at this stage that the major reasons why the underhand face shape was abandoned were related to the support mechanisms used by DRD. Just because the method was not viable in terms of safety does not mean it cannot be used by other operations. Because Mine A is using different support mechanisms, major issues which lead to the failure of underhand are irrelevant to it.

Another important observation made was that the delay in the entire cycle of operation was increasing with the steepness of the reef[4].

Underhand mining method

Around 1964 DRD was still mining steep reefs at a dip of 60° or more[4]. The challenge at that time was that the reef had a width of 20 ft, which is approximately 6 m, in some places. That meant that the reef could not be mined in one go. Because of rock engineering constraints only 1.5 m width could be stoped by semi-shrinkage stoping with an overhand face shape. The other part of the reef had to be recovered by footwall lifting or hangingwall stripping.

The lead time between the actual mining of the reef and reclamation of the hanging and the footwall had to be short. This was done to prevent deterioration of ground conditions before the recovery process was started. The total time for mining an area also had to be short. From these requirements it can be seen that the technique used for the recovery process had to be fast.

Among the factors considered for developing a technique that would meet the above requirements was the shape of the face, which is what makes this part of the report important. There were three face shapes from which to choose: underhand, overhand and breast face shape. An underhand face shape was chosen because of the advantages it had over other face shapes[4]. The advantages are not mentioned in this paper. Important facts worth noting from implementing the underhand face shape for reclamation are:

- When underhand is utilized, the apparent dip should be chosen such that there is a balance between the difficulty in installing supports and the rate at which cleaning is done. This is because the rate of cleaning improves with increasing apparent dip, and the installation of supports become increasingly difficult with increasing apparent dip. Both cleaning and support installation have a direct impact on the overall stoping cycle[4]. So the apparent dip should be chosen such that the stoping cycle is optimized.
- The underhand face shape fulfilled the production requirements of the reclamation process. Reclamation could be completed in the required time before the ground conditions deteriorated and other problems arose[4].
- Productivity was below the levels expected because of the difficulty of support transportation and installation[4].

From the above discussion it is evident that the implementation of underhand for the recovery of the hanging and footwall at 60° was successful. However, installation of supports was a difficulty.

Observations, measurements and data collection

This section of the report gives data collected from Mine A and Mine B. Methods of data collection and reasons why data was collected are discussed.

Mine efficiencies

Data had to be collected for the calculations of mine efficiencies in m^2/direct stope employee. The data had to consist of the number of direct stope employees, that is, stope rock drillers, those workers who install supports and the cleaning crew. Direct stope employees include everyone involved in direct stope activities from the position of the miner downwards. Panels worked by different teams of workers every month and the square metres broken also had to be collected.

For the purpose of doing a comparison between breast mining and underhand mining it would be appropriate to classify the data in terms of data for areas using underhand and data for areas employing breast mining. This was not possible because at the time of data collection the mine had been using underhand mining for less than three months. The data available at the time of study was for areas using the breast mining method only.

The efficiencies then were assessed by comparing areas of various reef dips. Table I gives the true dips of panels mined by specific teams each month and Table I in Appendix I shows the number of direct stope employees, area mined and the efficiencies. The data collected would be used to evaluate whether planned efficiencies were reached or not.

Ventilation

The main objective of having effective ventilation was to create an environment that was safe and conducive to work in; that is, to ensure that there were no harmful gasses or fumes where people were working and that the level of heat employees were exposed to could not cause heat illnesses[5]. To ensure the accomplishment of this ventilation air had to be properly directed towards the working face at a correct velocity and cooling power. Figure 6 shows the stope ventilation layout that was used at Mine A mine. The layout is for breast mining; however, it is relevant to this project as it illustrates principles that were employed to control ventilation.

A face temperature of 27.5°C wet bulb had to be maintained at all times[5]. When the temperature reached 32.5°C wet bulb or 37°C dry bulb workers were removed from the working place to a safer area. The air had to have a face air velocity of 1.5 m/s.[5]

For the specifications mentioned above to be met the following standards were put in place:

- Ventilation brattices were to be installed on the centre gully as shown in Figure 6 because the quality of air at the face is determined by the effectiveness of the ventilation seals on the centre gully
- Gas testing had to be conducted in accordance with the procedures specified by the management to improve detection of harmful gases before they reached dangerous levels
- Dust had to be kept down at all times by watering down
- Double ventilation brattices had to be installed at the top of the gullies
- Effective ventilation seals had to be created to restrict the ventilation air towards the working face[5].

Two working places, which were using the underhand mining method, were visited to investigate some of the points mentioned above. These two panels had ventilation issues at the time of the investigation. The entire study on ventilation was conducted in them. They were:

- 92H-102 X Cut-Raise 1-Panel 2, which had a reef dip of 45°.
- 95H-101A-Panel 4, which had a dip of 60°.

On the 10/01/2005 these panels were stopped from operating because of poor ventilation at working faces. Poor ventilation could be caused by air velocity less than 1.5 m/s hence resulting in an unacceptable temperature (more than 32.5°C wet bulb).

In both panels the ventilation issue was solved by installing a Venturi next to the face so that it could blow cool, fresh air straight on the working face. The venturi is indicated in Figure 7.

The solution was effective at 95H-101A-P4 because the face length was short, about 8 m. In 92H-102 X Cut-RSE 1-P2, the solution was not effective because the working face was about 35 m. At the top of the face, where the venturi was situated, the working conditions were acceptable; however, the temperature increased towards the bottom of

Table I

True dips of panels mined by teams each month

Team Name	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct
Golden supervisors	21	21	21	40	40	40	40	40	40	-
Sky high	21	21	21	21	21	21	30	30	30	-
Golden eagles	-	-	-	-	-	-	-	11	11	11
Dragon team	11	11	11	-	-	-	-	21	21	21
Thaitha	-	-	-	-	-	-	-	-	-	-
Simunye	38	38	38	38	38	38	21	21	21	38

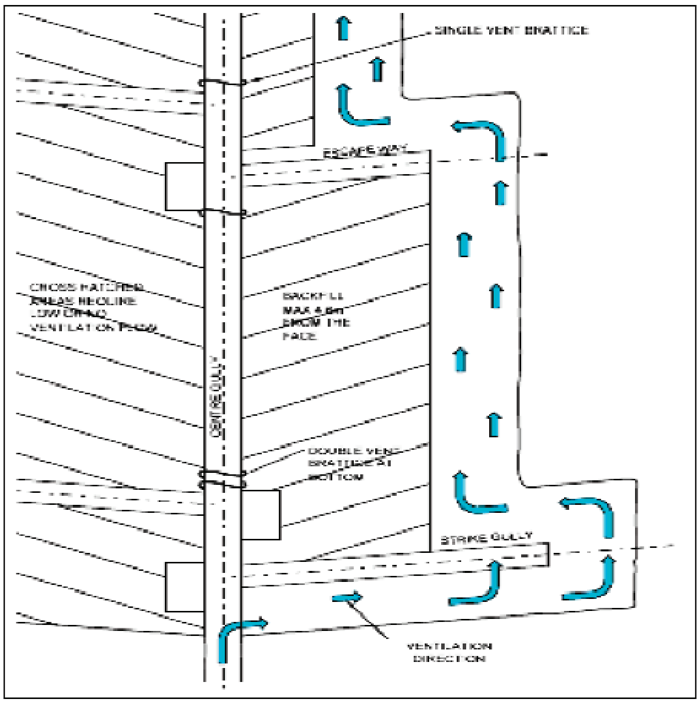


Figure 6—Ventilation standards for breast mining at Mine A

Figure 7—Venturi

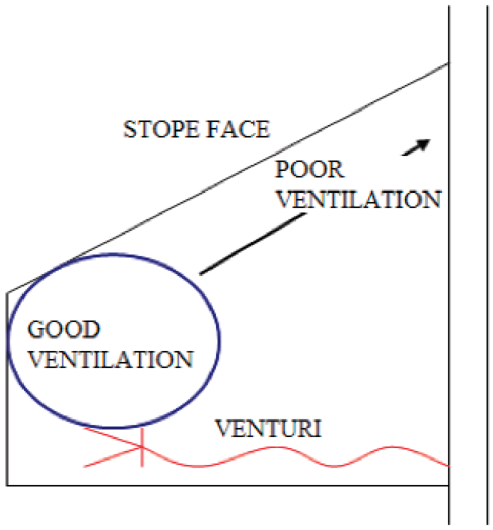


Figure 8—Venturi unable to solve ventilation problem

the stope, with very high temperatures at the bottom. The temperature at the bottom was about 32°C wet bulb. The situation is illustrated in Figure 8. Note that when the wet bulb temperature reaches 32.5°C employees are withdrawn from the working place.

Supports/rock engineering aspects

Two issues were investigated under supports. These are the support costs and whether the support standards can be adhered to with underhand mining being used. The support costs were investigated so that a comparison could be made between the cost of supporting underhand stopes and breast stopes. This would help decide whether the cost of supporting underhand is reasonable or not.

Adherence to support standards was investigated to see if it was possible to adhere to the standards with the underhand mining strategy, and whether there were any issues in terms of keeping standards that needed special attention.

Support costs

Three sets of data would be necessary for calculating the costs in R/m^2 supported. These are the data on the area to be supported for both underhand and breast mining, the number of support units to be used, and the cost for each type of support unit. Support prices were obtained from the costing department. The area and the number of units were obtained from the support standards. These sets of data are given below.

Support units' prices

The support units' prices as obtained from the costing department are given in Table II.

Table II

Support units' prices

Unit name	Unit cost (R)
Camlock Prop	758
High yield Elongate	169.7
Xpanda bolt	49.9
Pack pre-stressing plate	63.5
Mine pole	49.6
Durapack	15.4

Table III

Material for constructing one backfill bag

Item	Price (R)
1 x bag	1735.2
2 x blue wire	80
2 x other wires	960
Slurry/bag	1706.66

Table IV

Data used for estimating the cost of slurry/bag

	Total backfill plant, eng operations, labour and stores	
Month	Cost (R)	Tons produced
Jan	240 000	666
Feb	295 000	2066.98
Mar	300 000	3025.28
Apr	375 000	2251.72
May	435 000	3019.13
Jun	336 000	3210.08
Jul	355 000	3330.22
Aug	365 000	3662.16
Sep	390 000	4267.20

The price for backfill will be put separately because it is made up of the price of the material for building one backfill bag and the price of filling it with slurry. See Table III.

The cost of filling one backfill bag with slurry was estimated using the data in Table IV. The calculations for both slurry/bag and the total backfill price are covered in the next section.

Area supported and number of support units

The area to be supported and the number of support units were estimated from the support standards given in Figures 1 and 2 of Appendix II

Adherence to support standards

Adherence to support standards was investigated by visiting various underground working places. Four panels were investigated. Three of these panels were steeply dipping and one of them was shallow dipping. A shallow dipping stope was investigated so that a comparison could be made between panels using breast mining and those using the underhand mining method.

Figures 1 and 2 in Appendix II indicate the support standards used at Mine A. The standard distances between the support units and the distance of temporary and permanent supports from the panel face are indicated in them. The measurements taken underground were compared with these distances.

A 5 m steel measuring tape was used for taking the measurements. These measurements are given in the form of tables. The tables indicate the name of the place investigated; the date of investigation; and the true dip of the place visited. Only Table V is given, the other tables giving details of information collected are given in Tables I–V in Appendix II.

Measurements in all the tables indicate sub-standard measurements, i.e. supports that were not properly spaced.

Mine planning

Under this section two issues will be discussed. These are the effects of changes in true dip on the m^2/m ratio, and the comparison of planning and scheduling between underhand and breast mining.

m^2/m comparison could not be done between underhand and breast mining. However, the behaviour of m^2/m ratio with changes in reef dip was investigated as was proposed by the management of Mine A. The data collected to carry out the study is given below:

m^2/m ratio

A picture of the development layout for Mine A is used for drawing development cross-sections for different reef dips. From the development cross-sections lengths of various components of the tertiary development and the cross-cut is determined. Therefore the total development metres are determined. The area mined is determined from the raise length. The development layout is given in Figure 9.

Additional data used:

- Spacing of 150 m between levels
- Spacing of 25 m between the ore passes
- Stoping of 90 m on either sides of the raise (180 m spacing between the raises)
- The angle between the horizontal and the ore passes was varied from 50° to 80°
- Vertical portion of the ore pass was varied from 10 m–50 m (see section view in Figure 9).

Planning and scheduling

Table VI was used for evaluating the rate at which an area of approximately 20 500 m^2 would be mined when employing underhand or breast mining. The table indicates the areas scheduled for mining in different months. Areas under 'normal' are for using breast mining and those under steep are for underhand. The table was obtained from the planning department.

Tables I and II in Appendix III indicate the data collected for analysing differences in planning between underhand and breast mining. The data consist of the true dips with corresponding face lengths and angle of inclination from the horizontal. These parameters are used to indicate differences

Table V

Measurements collected to check adherence to support standards

Place visited			92H, 102 X cut, RSE1, P2		
Date			11/01/06		
Dip			45°		
Issues to be investigated	Whether support is normal to the hangingwall	Spacing parallel to face	Spacing perpendicular to face (skin-skin)	Distance of first line from face	Distance from gully centre
Temporary	correct	1.7-2.2, avg 1.9	Only 1 line	1.6	Not measured
Elongates	correct	0.9-1.1, avg 1	1.3-1.6, avg 1.4	3.3	Not measured
Durapak	correct	none	1.4	5.3	1
Xpanda bolts	-	-	-	-	-

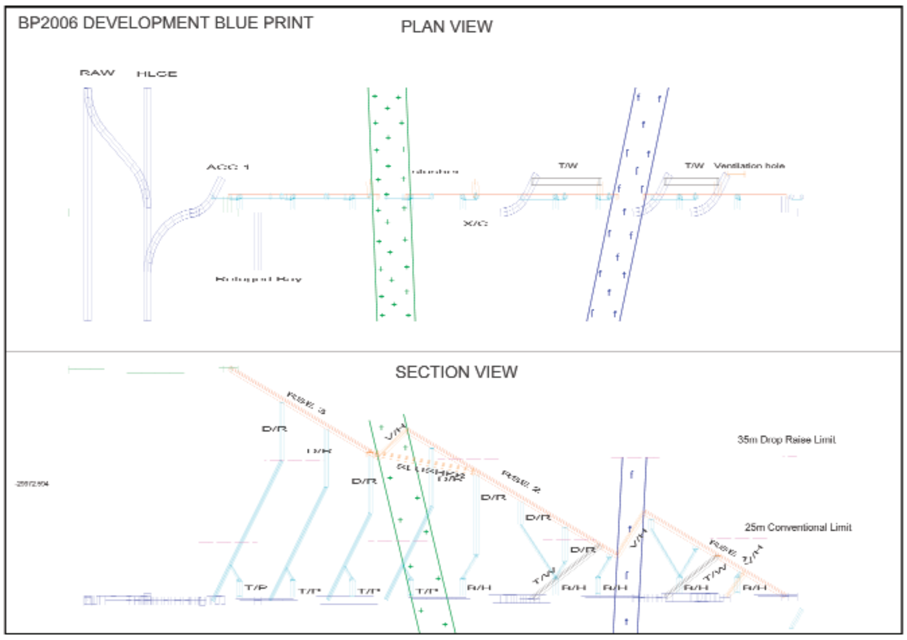


Figure 9—Development layout for Mine A

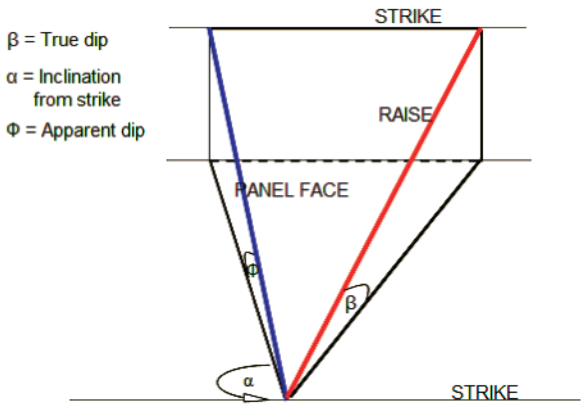


Figure 10—Figure used for the analysis of face length behaviour

in planning between underhand and breast mining. Figure 10 was used for the analysis of the behaviour of the face length and the angle of inclination from the strike, which are important for planning underhand stoping.

Additional information on planning and scheduling was collected through a personal interview with Mr. K. Steyn, senior mine planner at Mine A.

Data from Mine B

Two sets of data had to be collected from Mine B. These are data on support costs and mine efficiencies, i.e. (m^2 broken)/(direct in stope worker). These sets of data were used to compare support costs and efficiency between Mine B and Mine A.

The data was collected on a one-day visit to Mine B. The support costs were obtained from the costing department, and mine efficiencies were obtained from the human resources department. The data are given in Tables VII–IX. Support standards were also collected as given support costs were checked against the calculated ones.

Analysis and evaluation of research results

This section of the report is on the analysis and evaluation of research results. All the problems mentioned under the objectives are investigated.

Table VI

Table used for comparing the rate of mining of mining methods

m2		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total	
2005	Normal										270	280	270	820	
	Steep										98	329	352	779	
2006	Normal	287	318	280	280	277	294	283	280	280	280	275	166	3 299	
	Steep	372	509	605	681	925	1 035	1 080	1 092	1 020	1 142	1 249	1 321	11 031	
2007	Normal	53				133	262	403	628	902	1 142	1 220	1 318	6 061	
	Steep	1 372	1 433	1 179	941	933	730	481	306	199	182	165	148	8 069	Total
2008	Normal	1 443	1 120	1 161	1 257	1 481	975	861	745	590	425	128		10 185	20 366
	Steep	139	143	144	129	95	6							654	20 533

Table VII

Efficiencies at Mine B in 2005

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Area	29 628	34 307	35 242	34 618	36 025	37 116	37 736	37 581	38 049	37 737	37 035	36 880
ISW	1 971	2 006	1 952	1 966	1 968	1 974	1 932	1 928	1 967	1 991	2 001	2 024
m2/ISW	15.0	17.1	18.0	17.6	18.3	18.8	19.5	19.4	19.3	18.9	18.5	18.2

Table VIII

Support costs at Mine B in 2005

Month	Jan	Feb	Mar	Apr	May	June	Jul	Aug	Sep	Oct	Nov	Dec
R/m2	170	133	144	146	120	131	133	127	133	134	131	139 centre

Table IX

Support units' costs at Mine B in 2005

Product	Camlock props	Elongates	Composite pack	Roof bolt
Unit price (R)	758	169.67	9.64	49.89

Mine efficiencies

A direct comparison could not be made between the efficiencies for areas using underhand mining method and those employing breast mining. This was due to the insufficiency of data at the time of study. The mine started employing underhand mining method only in August 2005. At the time of study, is December 2005, there was insufficient data from underhand stopes to study the efficiencies. The data were then analysed by looking at areas of different dips and analysing how working of steep areas affected the efficiencies.

Since the underhand mining method would be used for mining steep areas, this analysis might give an indication as to whether the efficiencies would be satisfied in steep areas. The analysis is as follows:

Figure 11 indicates graphs of planned and actual efficiencies from January to October 2005. Efficiencies were met in April, May and June. If a relationship can be established between the behaviour of the actual efficiencies and the reef dip, then a conclusion can be drawn about the effects of mining steep areas on the efficiencies. Figure 12 was drawn in attempt to establish such a relationship.

The figure indicates the number of steep panels mined every month in 2005. It can be seen that there were few steep panels being mined in 2005. However, steep panels made a significant contribution to the efficiencies since there were between 6 and 13 panels being mined every month in 2005. The contribution made by steep panels in terms of the percentage of the total area stoped every month is shown in Table X.

The reef dips of some panels were not recorded. These panels could be steep dipping or flat dipping. This means that there are some inaccuracies concerning the ratio of flat dipping panels to steep dipping panels in Figure 12 and the percentages given in Table X. The areas of unknown dip

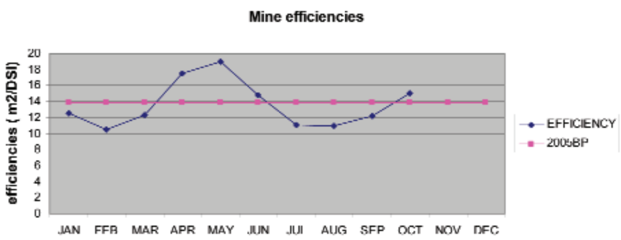


Figure 11—m2/ direct stope employee for Mine A in 2005

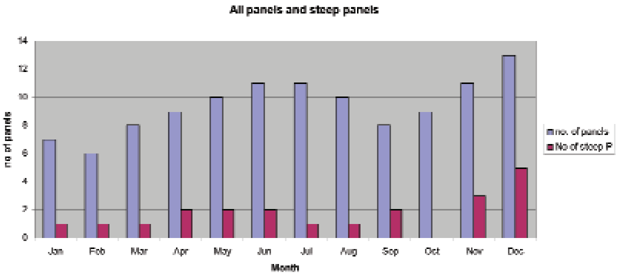


Figure 12—Number of steep and flat dipping panels worked per month in 2005

Table X

% area mined from steep panels

Month	No. of panels	No. of steep panels	% steep panels
Jan	7	1	14
Feb	6	1	17
Mar	8	1	13
Apr	9	2	22
May	10	2	20
Jun	11	2	18
Jul	11	1	9
Aug	10	1	10
Sep	8	2	25
Oct	9		0
Nov	11	3	27
Dec	13	5	38

were counted as flat dipping so that the contribution made by steep panels as shown in Table X would be conservative values. A more accurate analysis could be done in the future when more accurate data become available.

As mentioned earlier in this section, the objective of giving all the information is to establish a relationship between reef dip and the efficiencies.

From January to March the actual efficiencies were below the planned efficiencies. In these months there was only one steep panel operating. This panel contributed 14, 13, and 17% to the total area stoped. From April to June the number of steep panels doubled to two; during the same period the steep panels contributed 22, 20, and 18% to the total area stoped, and the planned efficiencies were met. In July and August the number of steep panels dropped to one, with an overall decrease in the percentage of area contributed. In this period the efficiencies were not met. The efficiencies for the remaining months were not available. From the above analysis it can be seen that the increase in the number of steep panels and the percentage contributed to the total area mined almost coincided with the improvement in the efficiencies. A conclusion that can be drawn at this moment is that the mining of steep area has not contributed negatively to the efficiencies. With the use of the underhand rather than breast mining method, the author of this report anticipates an improvement in the efficiencies because of improved working conditions.

Another investigation was carried out to add some certainty to the conclusion drawn from the above analysis. The performances of two stoping teams under steep and flat dipping conditions were investigated. The two teams were chosen because the information on where they were working, the dips of the areas, and their efficiencies from January to September was available (see Table I). The two teams are the Golden Supervisors and Simunye. The efficiencies of these teams and the true dips of the panels they were working are shown in Figures 13 and 14.

From January to March Golden Supervisors were working in an area of 21°. In these months they could not meet the planned efficiencies. From April to September they were working in an area of 40°, and from April to June they met the planned efficiencies. From July to September they missed the planned efficiencies. The fact that efficiencies could be met for three months in an area of 40° indicates that it is possible for mine efficiencies to be met under steep mining conditions.

Simunye was working a panel of 38° from January to June. In all these months the efficiency was slightly missed only once, in February. In September they were also working a panel of 38° and they did not meet the efficiency. The fact that efficiencies could be met for five months indicate that planned efficiencies can be met under steep conditions. Failure to meet the planned efficiencies cannot be directly attributed to the working of steep reef.

Ventilation

One can be tempted to conclude that the underhand mining method induces ventilation problems at the working faces, since the panels that were used as areas of study were both having ventilation issues. (In a earlier section of thise report explains how specific problems that were identified were related to the panels, and also some of the ventilation issues caused by the use of underhand.)

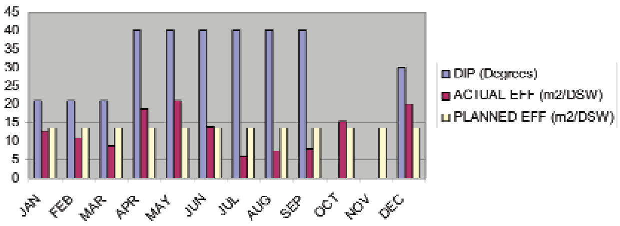


Figure 13—m^2/In-Stope worker for Golden Supervisors

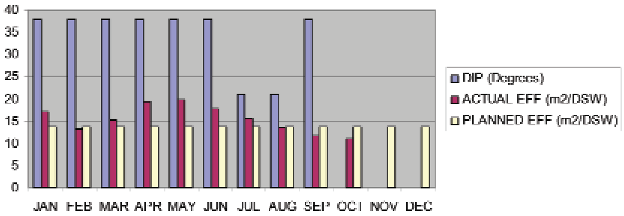


Figure 14—m^2/In-Stope worker for Simunye

95H, 101A, P4

Figure 15 is used to describe the problem encountered in 95H-101A-P4.

Figure 15 indicates the panels in level 95, 101A raise. Panel 4 was a steep panel, dipping at an angle of 60°. At the time of study the panel was using theunderhand mining method. As stated earlier, the panel had to be stopped at some stage because of poor ventilation on the face of the panel. The main cause of the problem was high air resistance on the escape gully leading to panel 4. Mine ventilation standards require that 4 m^2 of cross-sectional area be available for the passage of sufficient ventilation air[12]. To remedy the situation air going down the centre gully had to be forced through the escape gully by providing higher resistance at the bottom part of the centre gully. This could be accomplished by putting sealing brattices on the centre gully and ventilation curtains on the western gullies. Also the backfill marked R had to be removed to reduce the resistance. These solutions were not implemented and another solution, which was installing the Venturi near the face, was used. Although the ventilation problem was identified in a panel using the underhand mining method, the problem was not due to the use of the mining strategy but to high air resistance in the gully.

92H-102 Xcut-Raise 1-P2

Figure 16 describes the problem encountered in 92H-102 Xcut-Raise 1-P2.

The air comes in through the cross-cut and it has to go up the raise, and then enter between the wall and the ventilation curtains at the top of the panel (position C). As a fluid the air will naturally turn to move downwards, especially when it is cold like ventilation air. Because of this the ventilation air does not follow the path indicated in the figure. This results in poor ventilation at the panel face. Another problem was lack of ventilation curtains on the first line of elongates to restrict the air from taking the shortest route after leaving the last pack, as indicated on the sketch.

According to Johan Mass, manager of the department of environmental engineering, a solution to this problem would be to install a force fan at position B and force air through a column to position C. Once the air is forced to position C, ventilation curtains can be installed properly to direct the air to the face. At the time of study this solution had not been implemented.

It is important to note again at this stage that this problem was not caused by the fact that an underhand mining layout was used. The problem was specific to the above panel.

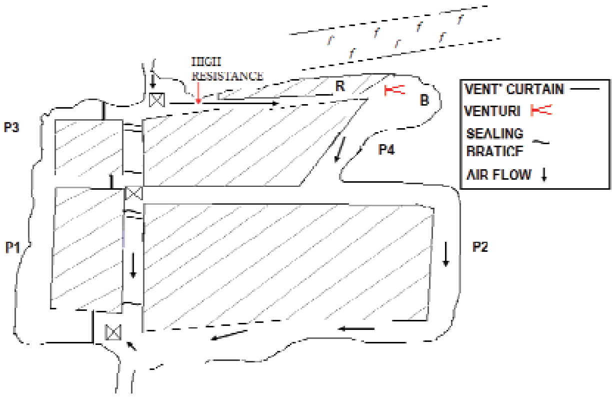
Problems with underhand

Two problems that result from the use of the underhand mining method were identified. One was the formation of a gas accumulation zone on the up-dip side of the panels. Gases that were likely to be trapped were those that are lighter than air, like methane. This problem is illustrated in Figure 17.

The other problem was the use of ventilation curtains. Ventilation curtains were an issue because workers easily neglect them. They were poorly installed and at times not installed at all. An example of this is 92H, 102 X Cut RSE 1, P2. The reason why ventilation was fine next to the Venturi and poor away from the Venturi is because there were no ventilation curtains forcing air to the panel face (see section 3.2). In Figure 17 it can be seen that if ventilation curtains are not installed along elongates air will not follow the direction indicated by the purple arrows, but it will follow the shortest route, which is indicated by a blue arrow.

Failure to install ventilation brattices on the centre gullies was also a problem. In the two places that were investigated ventilation brattices were not installed. There were no dust problems identified.

According to Johan Mass, ventilation issues in underhand panels were mainly due to failure to adhere to ventilation standards as given by the environmental department. This, however, does not mean standards cannot be adhered to. It is possible to adhere to ventilation standards with the underhand mining method.



Picture adopted from Mr C Somers' suggested environmental layout, 14/12/05

Figure 15—Ventilation problem at panel 95H,101A,P4

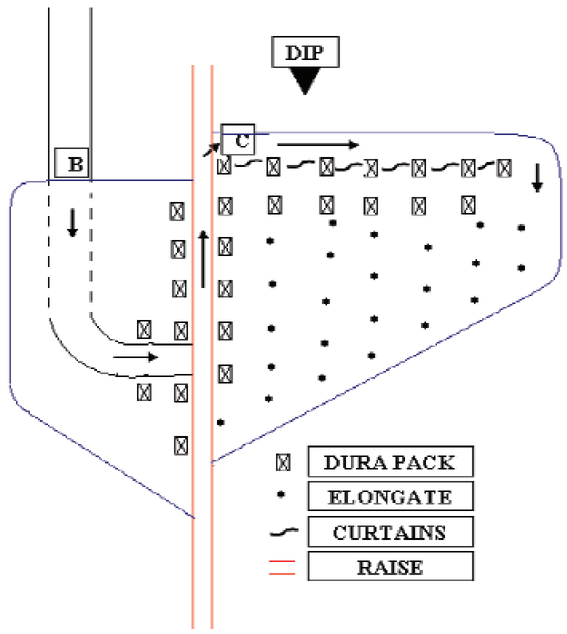


Figure 16—Ventilation problem at panel 92H-102 Xcut-Raise 1-P2

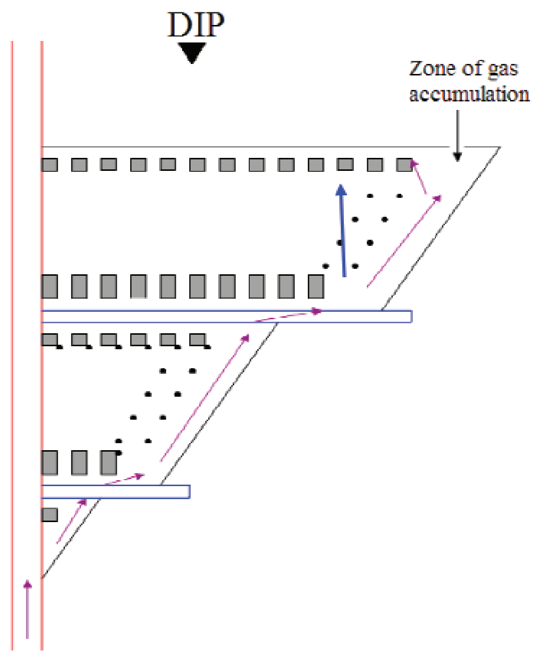


Figure 17—Formation of gas accumulation zone on the area of poor ventilation

Underhand panels need the installation of ventilation curtains on the first line of elongates to improve ventilation on the panel face. Also, double ventilation brattices must be installed on the raise to restrict the movement of ventilation air to the face. Workers should be educated on the importance of adherence to these standards so that they can see the need to adhere to them.

Supports/rock engineering aspects

This part of the report focuses on the analysis of support costs for both underhand and breast mining methods, and also the analysis of the results obtained from investigating adherence to support standards.

Support costs

A comparison had to be made between the cost for supporting breast and underhand mining methods. The cost for the breast mining method includes backfill. For underhand mining, two calculations were done, one including backfill and the other excluding backfill.

Before the overall cost calculations were done the cost for constructing one backfill was calculated. Two calculations were done, one including slurry per bag and the other one excluding it. This was done because the management of Mine A wanted the cost of backfill excluding slurry per bag; but the author of this paper thought that a more detailed analysis, which includes the cost of slurry per bag, would be necessary. The calculations are given in Tables XI and XII.

The cost of slurry per backfill bag was estimated from the data given in Table III. This data was analysed using an Excel spreadsheet and the results indicated in Table XI were obtained.

From Table XII it can be seen that the cost of backfill in R/m³ was not constant from January 2005 to September 2005. Since the cost was not constant a convenient value had to be calculated. This was done by looking at the distribution of cost for every month from January to September. Figure 18 illustrates this behaviour.

The graph indicates that as the year progressed the cost of backfill decreased. This behaviour can be described in terms of the tonnage of slurry pumped underground. Mine A was still an infant mine at the time of the investigation. The mine had not yet reached full production. As the production increased more area had to be supported. For this reason more slurry was required for backfill. The tonnage of backfill per month increased while the cost of pumping increased only slightly (fixed costs). This resulted in an overall decrease in the costs. From these results it can be concluded that the cost of backfill in R/m³ will continue to decrease with increasing production. Since the production of Mine A was still increasing at the time of the study a decision was made to use the lowest available cost. This would be conservative because the cost is expected to decrease with increasing production and hence with time until peak production is reached. The lowest, cost as can be seen in Table XII, is R160/m³.

Table XI

Backfill price excluding slurry bag

Item	Price (R)
1 x bag	1 735.2
2 x blue wire	80
2 x other wires	960
Total	2 775.2

Table XII

Calculation of cost of slurry in R/m³

Month	R/m³
Jan	630.63
Feb	249.76
Mar	173.53
Apr	291.44
May	252.14
Jun	183.17
Jul	186.54
Aug	174.41
Sep	159.94

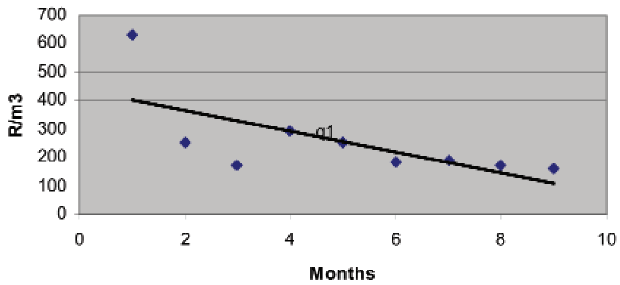


Figure 18—The distribution of backfill costs in 2005

From the cost of R160/m^3, a cost in rand per slurry/bag was calculated. The cost was determined as follows:

A stoping width of 1.5m would be used for all the analysis. Using this stoping width the cost in R/m^2 can be calculated. This cost is R240/m^2. This cost was multiplied by the area/slurry/bag to give the cost in R/slurry/bag. Area/slurry/bag was estimated from the support standards. This area was 16 m^2 for backfill covering a length of 10 m. The cost of backfill is then R3 840/slurry/bag.

From this value the cost of the entire backfill unit including slurry can be calculated, as indicated in Table XIII.

Support costs for underhand and breast mining methods

As mentioned previously, the number of support units and the total area supported were estimated from the support standards. Three calculations were done. The first calculation excludes backfill, the second one includes backfill material only, and the last one includes backfill material and slurry. The number of support units estimated from the standards was multiplied by the unit price to give the total price for the support type. Prices for all support types were added and divided by the total area in the appendices. The results of the calculations are given in Table XIV.

The cost of supporting underhand panels was low when backfill was not included (R101/m^2). The cost increased significantly when backfill was included. It exceeds the cost of supporting breast mining panels by R13/m^2 when the cost of material only was excluded and R25/m^2 when including the cost of slurry bag.

Adherence to support standards

As mentioned previously, four working places were investigated. About four problems on adherence to support standards were identified. Some of the problems were found in more than one stope. Problems that were found to occur in more than one working place were taken to be the ones that needed attention. Table XV summarizes the problems identified.

In 95H, 101A, P4 measurements could not be taken due to the steepness of the stope and lack of safety gate stulls. However, the spacing between elongates looked acceptable as observed from a distance of less than six metres.

In 95H, 102 X cut, RSE1, P1, both the temporary and permanent supports were correctly spaced. One factor that seemed to be a problem was the angle of the supports relative to the reef dip. The supports were not perpendicular to the reef dip as required. An observation made was that this problem was caused by the steepness of the reef. In 92H, 102 X cut, RSE1, P2, where the underhand method was already in use, this problem was not as serious. A conclusion was therefore drawn that the use of underhand enhanced correct installation of supports.

Another observation made was that some elongates which were not perpendicular to the reef dip split at the bottom. See Figure 19.

When the elongate is not perpendicular to the reef dip it will wend to slide during stope closure. Elongate number one on Figure 19 indicates the sliding action. The red arrows indicate the direction of sliding if there was no friction between the hangingwall and the elongate. Since there is friction between the hangingwall and the elongate the latter cannot slide freely. If there are defects at the base of the elongate it might split, as indicated by elongate number two.

Table XIII

Cost of backfill including slurry bag

Item	Price (R)
1 x bag	1 735.2
2 x blue wire	80
2 x other wires	960
Slurry/bag	3840
Total	6 615.2

Table XIV

Results of cost calculations

Description	Cost (R/m^2)
Underhand without backfill	101.05
Underhand with backfill material only	139.69
Underhand with backfill material and slurry bag	193.15
Breast mining with backfill material only	126.76
Breast mining with backfill material and slurry bag	167.85

Table XV

Summary of identified support sub-standards

Work place	Dip	Problem 1	Problem 2	Problem 3	Problem 4
92H, 102 X cut, RSE1, P2	45°	Spacing of camlocks parallel to face (1.9)	Distance of camlocks from face (1.6)	First line of elongates from face (3.3)	First line of Durapak from face (5.3)
95H, 101A, P4	60°	Temporary supports not installed	-	-	-
95H,101A, P2	28°	Spacing of camlocks parallel to face (1.8)	First line of elongates from face (3.3)	-	-
92H, 102 X cut, RSE1, P2	45°	Spacing of camlocks parallel to face (2)	Distance of camlocks from face (1.6)	Distance of elongates-from face (4.8)	
95H, 102 X cut, RSE1, P1	65°	-	-	-	-

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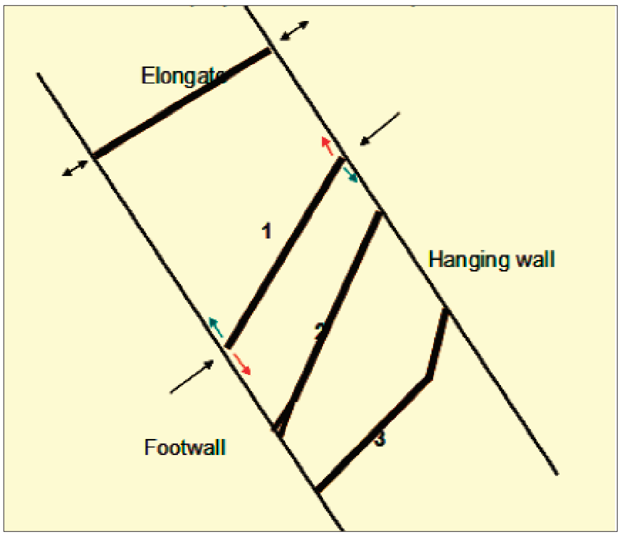

Figure 19—Elongates not perpendicular to reef dip

Elongates used at Mine A are designed to have 100% yield. Damage on elongates due to incorrect installation precludes them from giving the required yield.

Four issues would need to be addressed to improve adherence to support standards. These are:

- Spacing of the camlock props in the direction parallel to the face
- Distance of the camlock props from the face
- Distance of the first line of elongates from the face
- Installation of elongates perpendicular to the reef dip.

Table XV indicates that 95H,101A, P2, which was dipping at 28°, also had sub-standard supports. Spacing of the camlock props parallel to the face and the distance of the first line of elongates from the face were incorrect. This indicates that sub-standard support installation is not caused by the working of steep reefs. It is a result of negligence from stope workers.

Failure to install supports perpendicular to the reef dip could be caused by slippery conditions in steep stopes. This observation was made on a visit to investigate how stope workers install supports.

Mine planning

This section is about the analysis of data collected for mine planning. The behaviour of the m^2/m ratio with increasing true dip will be discussed first, followed by mine planning and scheduling.

m^2/m ratio

m^2/m ratio was analysed by calculating the total development metres required for mining a specific area and the total area to be mined. The development considered includes the tertiary development and the cross-cut (see Figure 9). The area mined was calculated using the length of the raise and 180 m on strike. 180 m was used based on the assumption that 90 m would be mined on either side of the raise.

Two techniques were used for the analysis. One technique employed sketches drawn to scale and the other one calculation. In the drawing technique the angle of the rock passes from the horizontal was taken to be 70°, the straight section (dog leg) of the ore pass to be 40 m. m^2/m ratios were then calculated for reef dips ranging from 25° to 60°. The results obtained are given in Table XVI.

These results were used to plot the graph in Figure 20.

Figure 20 indicates that development metres decrease with increasing reef dip. The rate at which linear metres are decreasing is low for the raise and the cross-cut, but the total metres decrease significantly from 1 917 m at 25° to 813 m at 60°. The graph indicating the behaviour of the m^2/m ratio is given in Figure 21 so that the results obtained from the drawings can be compared with those obtained from the calculations.

In order to check the results obtained from the drawings, calculations were also done. Calculations not only serve to check the drawings but they also provide an easy way of making detailed analysis. This is achieved by varying the angle of the rock passes, the straight section of the ore passes, and the reef dip. By varying them the behaviour of m^2/m ratio for different situations can be evaluated.

The angle of the rock passes from the horizontal was varied from 50° to 80°. The straight section of the ore pass was varied from 20 to 40 m, and the reef dip was varied from 25° to 60°. 25° represent panels that are shallow dipping and 35° and more panels that are steep dipping.

Calculations where done by running a small program on Microsoft Excel. These enabled different inputs for rock pass angle, reef dip and straight section of the rock pass to be made.

A graph of reef dip against m^2/m ratio was drawn from the results of the calculations. This graph and the graph from the drawing results are presented in Figure 21.

There is a small difference between the corresponding m^2/m values from the drawings and the calculations. The difference ranges between 0.9 and 2.5. However, both graphs

Table XVI

m^2/m ratio results from drawings

Dip	25°	35°	40°	50°	60°
Raise (m)	360	265	232	193	172
X/cut (m)	338	238	192	138	106
Rock passes (m)	1 219	866	736	604	535
Total development	1 917	1 369	1 160	935	813
m^2 stoping	64 800	47 700	41 760	34 740	30 960
m^2/m ratio	33.8	34.8	36	37.1	38.1

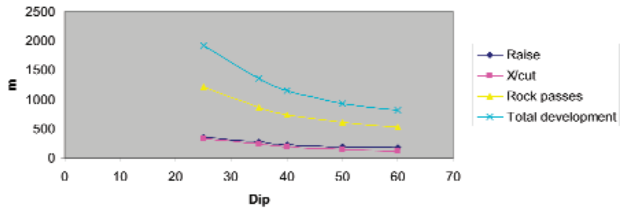

Figure 20—Variation of tunnel meters with increasing dip

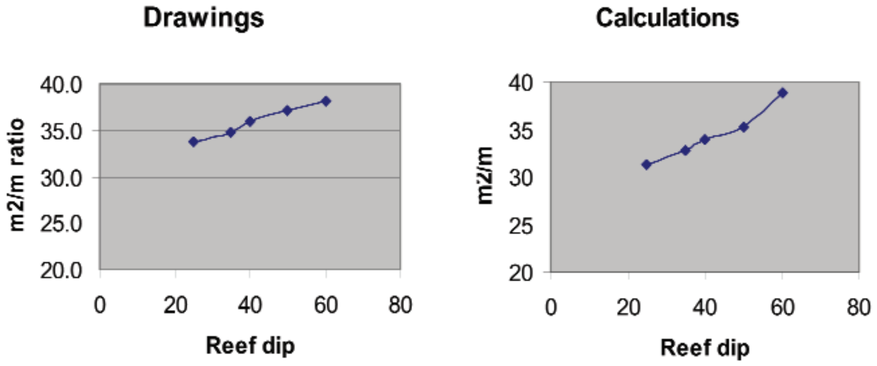

Figure 21—Efficiencies estimated from drawings and calculations

indicate a similar trend, the ratio increases with increasing reef dip. It can be seen from Table XVI that the area broken decreases with increasing reef dip because of shorter raises. This reduction in the area is offset by a significant reduction in the total linear metres, (see Figure 20), resulting in an overall increase in m^2/m ratio. The same behaviour was observed for different rock pass angles and straight sections of the ore pass, emphasizing the results give above. An increase in m^2/m ratio means that there is an advantage for the mining of steep reefs.

No conclusion about the appropriateness of the underhand mining method can be derived from this analysis. However, it serves to indicate that in steep areas, where the underhand mining method will be employed, the ratio of area mined to the development done to access that area should be high. Moreover it satisfies one of the objectives of this investigation, which is to find out for the management of Mine A how the reef dip affects the m^2/m ratio.

Planning and scheduling

K. Steyn from the mineral resource management department at Mine A was interviewed about planning and scheduling for underhand and breast mining methods. According to him, there were no significant differences between planning and scheduling done for underhand and that which was done for the breast mining method.

Planning

What had to be considered under planning for underhand mining was the fact that the face lengths of the panels always changed as mining progressed. This is illustrated in Figure 22.

The figure indicates three panels: A, B and C. On each panel the current stoping face (1) and the future one (2) are indicated. Panel A and B have already reached the maximum face length so they are mining at constant face length. Panel C has just started mining so the face length will continue to increase until it reaches the maximum length. As can be seen the face length of 1 and 2 on panel C are not equal. This has to be considered when planning is done.

Also, the maximum face length for a given panel depended on the reef dip of that panel. This meant that panels with different reef dips would have different maximum face lengths. This becomes apparent when two adjacent panels have different reef dips. e.g. panel A and B above can have dips with a difference of 30°, resulting in a face length difference of more than 16 m.

The two points mentioned above never had to be considered with breast mining as the face length was always constant and did not depend on the steepness of the reef.

The angle between the panel face and strike direction was also important as it always changed to ensure an apparent dip of 32° all the time. From Figure 10 it can be seen that as the true dip increases the panel face must be swung towards strike so as to maintain a constant apparent dip.

Calculations of maximum face length and the angle between strike and panel face were done to investigate how they change with reef dip. Calculations were done using the relationships indicated in Figure 10 and the following formulas:

$\sin\Phi = \sin\alpha \times \sin\beta$

Maximum face length = $25/\sin\alpha$

where:

Φ = Apparent dip

α = Inclination from strike

β = True dip

The results of the calculations are given in Table II of Appendix III. From these results the graphs of the maximum face length against true dip, and angle of inclination from strike against true dip were drawn. The graphs are given in Figure 23.

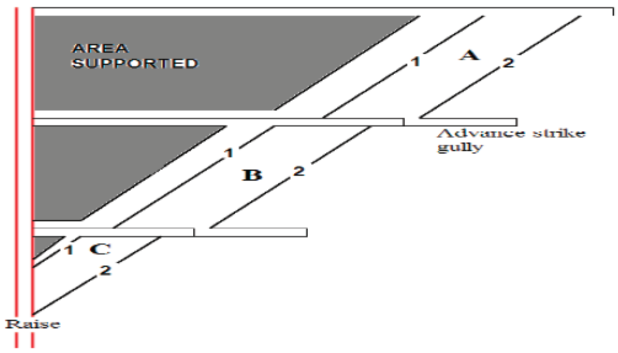

Figure 22—Change in face length as underhand panels are mined

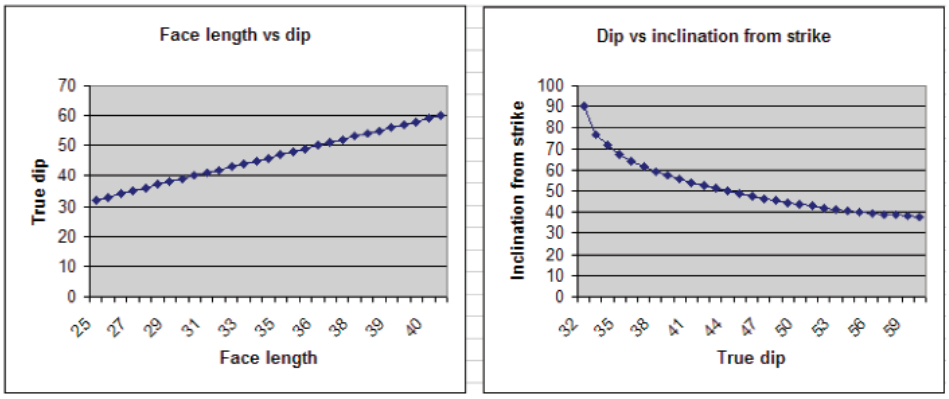

Figure 23—Face length against true dip, and inclination from strike against dip

From Figure 23 it can be seen that the maximum panel face length increases with increasing reef dip, and the inclination from strike decreases with reef dip. These two factors had to be accounted for in planning. According to Mr K. steyn, apart from these two factors, no complicated issues arose in planning as a result of using the underhand mining method.

Scheduling

Scheduling did not depend on the mining method used at all. Two factors that were said to be important for scheduling were the local geological conditions and rock engineering considerations. These factors where used to schedule panels such that mining progressed away from dangerous areas. For example, in an area that had a dangerous fault, mining would be done away from the fault for safety reasons. This meant that panels that were close to the fault would be scheduled for mining at an earlier stage than those that were far from it.

The other consideration was the maintenance of lead/lags of not more than 5 m for panels that were on opposite sides of the raise. This was done to prevent the creation of a potentially hazardous area in terms of rock fall. (See Figure 24). According to Mr K. Steyn, these two issues did not create complications in handling scheduling for the underhand mining method.

Another important issue under scheduling is that an exercise was done by Mr Steyn to determine which of the mining methods was quicker. An area of approximately 20 500 m^2 was considered. The outcome of the exercise was that, beginning from October 2005, with underhand mining the area would be completed by June 2008, and with breast mining the area would be completed in November 2008. There is a difference of five months, indicating that underhand mining is quicker. This agrees with one of the findings in the literature research, where underhand face shape was quicker to reclaim access reef in shrinkage stoping.

Comparison between Mine A and Mine B

Compared to Mine A, Mine B was already a mature mine at the time of investigation. The mine lies on the northern side of Mine A[6]. The two mines are adjacent to each other; their lease areas share a boundary. Although they had significant differences in their production rates at the time of the investigation, they were compard in an attempt to check if the support costs and the efficiencies at Mine A were reasonable or not. The data given earlier on support costs and efficiencies from Mine B ares analysed in this section. Support costs are analysed first, followed by efficiencies.

Comparison of support costs

An average support cost of R137/m^2 was calculated from the support costs of 2005 for Mine B. An average costs was obtained by adding the cost for every month from January to December and dividing them by the number of months.

Another method was used to calculate the support costs for Mine B. The method made use of the support standards and the supports' unit prices. There were two support layouts used at Mine B. One layout made use of elongates and the other one used pack supports. Prices in R/m^2 were calculated for both layouts. All the calculations were done through an Excel spreadsheet.

A cost of R150/m^2 was calculated for the elongate layout and R120/m^2 for the pack support layout. A simple mean was taken between the two values as the percentage covered by each layout was not available for taking a weighted mean. A value of R135/m^2 was obtained. This value agrees with the R137/m^2 average obtained from the costs of 2005.

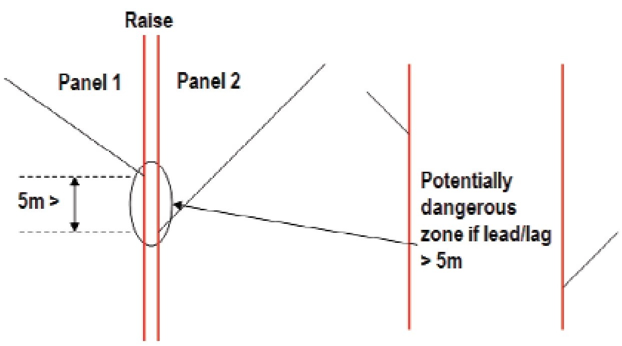

Figure 24—Lead/lags for adjacent panels

Assessment of current mining strategy for steep reefs at Mine A

This value is higher than the value of supporting underhand panels excluding backfill at Mine A by R36/m^2. When including backfill with material costs only for underhand mining at Mine A the cost at Mine B becomes less by R3/m^2. The cost of R101/m^2 for Mine B, which gives the difference of R36/m^2, is an important cost since at the time of investigation Mine A was not using backfill in underhand stopes. If no changes are introduced, underhand mining would save costs in terms of supports.

Comparison of efficiencies

The graph in Figure 25 was drawn to compare efficiencies. Data available at Mine A was only up to October. At that time not many panels used the underhand mining method. From January until August steep areas were mined using the breast mining method, so the graphs do not give a clear indication as to whether efficiencies from areas employing underhand were reasonable. However, based on the assumption that the underhand mining method provides better working conditions compared to the breast mining method, we can say the results might indicate if the efficiencies will be satisfactory.

Efficiencies at Mine B were fairly constant for the rest of the year because in 2005 the mine was already a mature operation. With Mine A the efficiencies went up and down because it was an infant operation. In April and May the efficiencies at Mine A were nearly equal to the efficiencies at Mine B. This might reflect a potential for Mine A to operate at reasonable efficiencies in the future. Note a value of 19m^2/ISW for Mine A in May. This value is higher than 18m^2/ISW for Mine B, which is positive for Mine A.

A clear conclusion cannot be drawn from these results because of insufficient data. As enough data become available, a similar analysis can be made. However, April and May indicate that Mine A has potential to meet efficiencies equal to those at Mine B.

Conclusions

Two teams were used to investigate how working of steep areas affects the efficiency of workers. Both teams indicated a positive impact in the efficiencies as reef dip increases. Efficiencies increased in some months when the teams were working in steep places. The working of steep areas does not decrease the efficiency of workers. Analysis of 2005 data indicates a similar trend: an improvement in m^2/ISW with increasing numbers of steep panels.

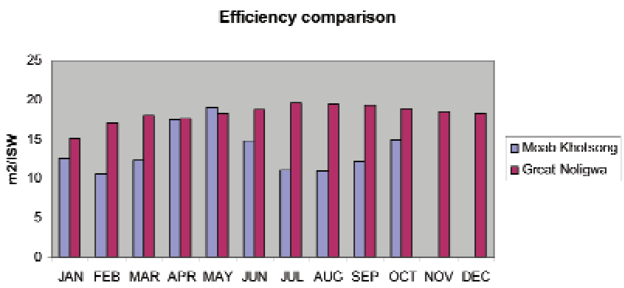

Figure 25—m^2/m comparison between Mine A and Mine B

Ventilation issues were found to exist in underhand panels because of failure to adhere to ventilation standards. Underhand panels need the installation of ventilation curtains on the first line of elongates from face to improve ventilation on the panel face. Also, double ventilation brattices must be installed on the raise to restrict the movement of ventilation air to the face. These two standards were not practised. Workers should be educated on the importance of adherence to these standards so that they can see the need to adhere to them. Underhand mining creates a zone of gas accumulation on the up-dip side of the panels. This area should be ventilated by a Venturi to prevent the accumulation of hazardous gases.

The cost of supporting underhand panels is R101/m^2 when backfill is not included. This cost is lower than the cost of supporting breast mining panels, which is R126.7/m^2. The cost increases significantly when backfill is included. It exceeds the cost of supporting breast mining panels by R13/m^2 when the cost of material only is excluded and R25/m^2 when including the cost of slurry bag.

Sub-standard support installation is not caused by the working of steep reefs or the use of the underhand mining method. It is a result of negligence by stope workers.

Four issues need to be addressed to improve adherence to support standards. These are:

- Spacing of the camlock props in the direction parallel to the face
- Distance of the camlock props from the face
- Distance of the first line of elongates from the face
- Installation of elongates perpendicular to the reef dip.

A direct m^2/m ratio comparison could not be made between underhand and breast mining method. However, analysis indicates that the m^2/m ratio increases with increasing reef dip. This is an advantage for mining steep reef and also raises anticipation for underhand workings to operate at a higher m^2/m ratio.

In planning how panels are mined three issues make underhand different from breast mining. These are the change in face length as panel mining progresses, increase in maximum face length with increasing true dip, and decrease of an angle between the panel face and strike direction. However, these issues do not complicate the planning of underhand workings.

Factors that must be considered when scheduling are not dependent on the reef dip, therefore scheduling of underhand workings is done similarly to that of the breast mining method.

Although the data on efficiencies from Mine A weres insufficient for making a detailed analysis, it was realized that efficiencies at Mine A exceeded efficiencies at Mine B in May 2005. In April efficiencies were slightly different. This indicates the potential for Mine A to operate at reasonable efficiencies in the future, even under steep conditions.

A value of R137/m^2 was obtained for support costs at Mine B; this value is higher than the value of supporting underhand panels excluding backfill at Mine A by R36/m^2. When including backfill with material costs only for underhand at Mine A the cost at Mine B becomes less by

R3/m2. The cost of R101/m2 for Mine B, which gives the difference of R36/m2, is an important cost since at the time of investigation Mine A was not using backfill in underhand stopes. If no changes are introduced, underhand mining would save costs in terms of supports.

Based on the issues discussed above, the underhand mining method is appropriate mining steep reefs at Mine A.

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- AngloGoldAshanti for financial support
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- Paul Brenchley, my supervisor at the mine
- Mr H. Yilmaz, my supervisor at the University of the Witwatersrand.

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Appendix I: Efficiencies

Table I

Number of in stope employees, area mined and the efficiencies

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct
2005BP	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87
Golden surpervisors										
m2	281	224	169	300	419	264	120	133	167	320
Labour	21.7	20.8	19.3	16.1	19.9	19	20	18	21	21
Efficiency	12.949	10.769	8.7565	18.63	21.06	13.89	6	7.389	7.9524	15.24
Sky high										
m2	371	163	357	397	405	353	219	241	267	308
Labour	23.7	19.6	21.6	20.6	21.1	21	19	16	21	17.9
Efficiency	15.654	8.3163	16.528	19.27	19.19	16.81	11.53	15.06	12.714	17.21
Golden eagles										
m2	143	193	172	227	409	107	75	126	280	119
Labour	15.1	16.8	15.2	16.4	16.5	16	16	13	16	15
Efficiency	9.4702	11.488	11.316	13.84	24.79	6.688	4.688	9.692	17.5	7.933
Dragon team										
m2	174	160	192	257	149	281	271	92	194	219
Labour	21.9	18.3	21	16.4	16.2	16	14	11	14	13
Efficiency	7.9452	8.7432	9.1429	15.67	9.198	17.56	19.36	8.364	13.857	16.85
Simunye										
m2	257	263	319	373	419	373	280	232	236	212
Labour	15	19.7	20.7	19.3	21	21	18	17	20	19
Efficiency	17.133	13.35	15.411	19.33	19.95	17.76	15.56	13.65	11.8	11.16
THAITHA										
m2									144	349
Labour									14	16
Efficiency									10.286	21.81
Total stope Efficiencies										
m2	1226	1003	1209	1554	1801	1378	965	824	1288	1527
Labour	97.4	95.2	97.8	88.8	94.7	93	87	75	106	101.9
Efficiency	12.587	10.536	12.362	17.5	19.02	14.82	11.09	10.99	12.151	14.99
2005BP	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87	13.87

Appendix II: Supports

Special Support Standard Stoping	
For 11-7	For Stoping
Valid from 14/12/2005	

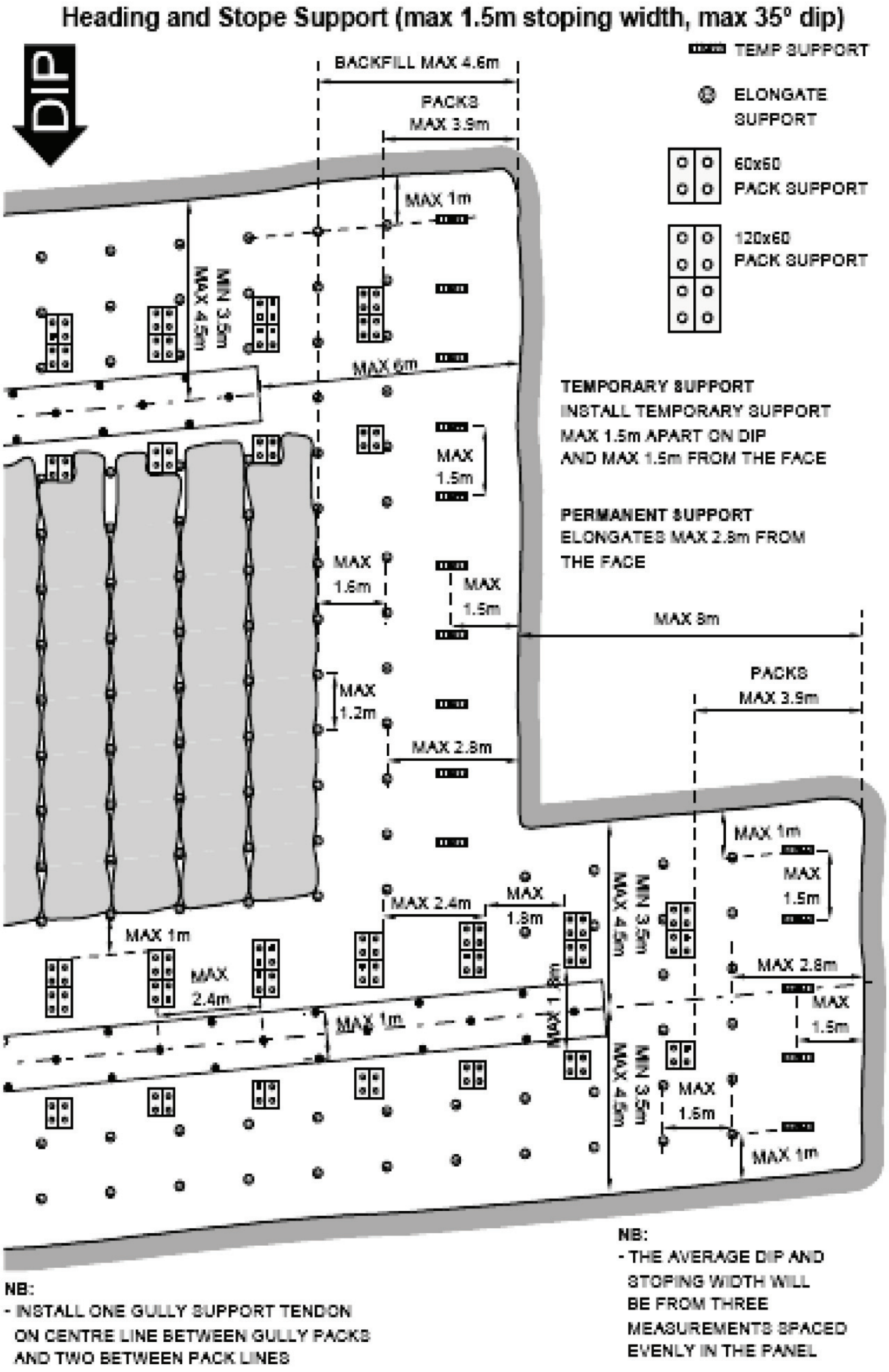


Figure 1—Support standards for breast mining method

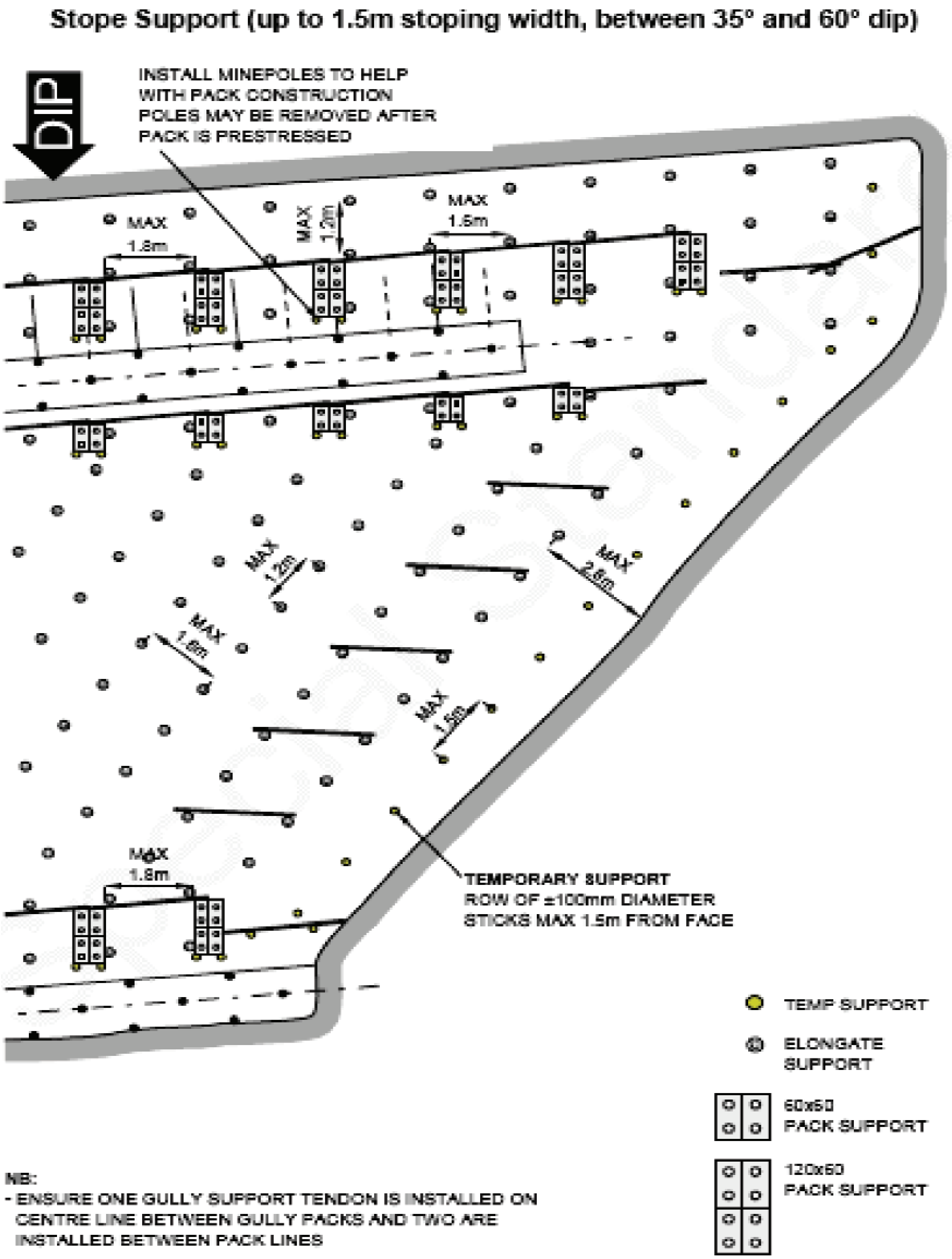


Figure 2—Support standards for underhand mining method

Table I
Measurements taken to check adherence to support standards

Place visited			95H, 101A, P4		
Date			11/01/06		
Dip			60°		
Issues to be investigated	Perpendicularity	Spacing parallel to face	Spacing perpendicular to face	Distance of first line from face	Distance from gully centre
Temporary					
Elongates	Correct	0.9	1.4	2	Not measured
Durapak	-	-	-	-	-
Xpanda bolts	-	-	-	-	-

There were no temporary supports on the day of thevisit. Stope workers could not install them the previous day since they were stopped from working because of ventilation problems. The Durapak lines had been installed while the stope was still using breast mining; however, they were up to standard. The Xpanda bolts were not measured since the steep stope was lagging behind the adjacent flat dipping stope.

Table II
Measurements taken to check adherence to support standards

Place visited			95H,101A, P2		
Date			11/01/06		
Dip			28°		
Issues to be investigated	Perpendicularity	Spacing parallel to face	Spacing perpendicular to face	Distance of first line from face	Distance from gully centr
Temporary	Not	1.8	Only 1 line	1.1-1.2	Not measured’
Elongates	Correct				
Durapak	Correct	Only 1 line	Not possible to measure due to collapse	3.3	0.8
Xpanda bolts	-	-	-	-	-

The first line of elongates had not been properly installed towards the bottom of the panel but at the top elongates were up to standard.

Table III
Measurements taken to check adherence to support standards

Place visited			92H, 102 X cut, RSE1, P2		
Date			11/01/06		
Dip			45°		
Issues to be investigated	Perpendicularity	Spacing parallel to face	Spacing perpendicular to face (skin-skin)	Distance of first line from face	Distance from gully centre
Temporary	Correct	2.0	Only 1 line	1.6	Not measured
Elongates	Correct	0.9–1.5, avg 1.2	0.6–1.5	4.8	Not measured
Durapak	Correct	Nnone	1.4	-	1
Xpanda bolts	-	-	-	-	-

Bows were properly supported by temporary supports and the supports were installed perpendicular to the reef dip. There was one MX prop, which was broken. The prop was not installed perpendicular to the dip and this could be the reason why it would not yield as expected. The spacing of Durapak parallel to the face could not be measured since there was only one line of packs along the centre gully. The pack on the first line of supports was not yet installed; however, there was an MX prop marking the position were the pack would be installed. People struggled to install temporary supports because of slippery conditions. This is the reason why some supports are not installed perpendicular to the face.

Table IV
Measurements taken to check adherence to support standards

Place visited			92H, 102 X cut, RSE1, P2		
Date			17/01/06		
Dip			45°		
Issues to be investigated	Perpendicularity	Spacing parallel to face	Spacing perpendicular to face (skin-skin)	Distance of first line from face	Distance from gully centre
Temporary	Correct	1.7–2.2, avg 1.9	Only 1 line	1.6	Not measured
Elongates	Correct	0.9–1.1, avg 1	1.3-1.6, avg 1.4	3.3	Not measured
Durapak	Correct	None	1.4	2.4	0.9
Xpanda bolts	-	-	-	-	-

Table V

Measurements taken to check adherence to support standards

Place visited			95H, 102 X cut, RSE1, P1		
Date			09/01/06		
Dip			65° at the bottom		
Issues to be investigated	Perpendicularity	Spacing parallel to face	Spacing perpendicular to face (skin-skin)	Distance of first line from face	Distance from gully centre
Temporary	Not correct	1.3	1 line	0.8	-
Elongates	Correct	1.2–1.4	1	1.1	-
Durapak	Correct	None			-
Xpanda bolts	-	-	-	2.4	-
Backfill	Correct	One back parallel to face	BF against each other	2.6–3.7	-

The panel was very steep at the bottom and it flattened towards the top. Measurements were not only taken on the steep part only but they were also taken on the flat part. Both the temporary and permanent supports were perpendicular to the reef dip at the top, and most supports were not perpendicular to the dip on the steep side.

Table VI

Cost of supporting underhand stopes without including backfill

Steep reef			
Product	**No. of units**	**Unit price**	**Total price**
Prestressing plate	16	63.5	1 016
Elongates	101	169.67	17 136.67
Durapak	630	15.37	9 683.1
Gully support tendons	24	49.89	1 197.36
Backfill	0	2 775.2	0
		Overall price	29 033.13
		Total area	287.3
		R/m^2	101.0550992

Table VII

Cost of supporting underhand stopes including backfill material only

Steep reef			
Product	**No. of units**	**Unit price**	**Total price**
Prestressing plate	16	63.5	1016
Elongates	101	169.67	17136.67
Durapak	630	15.37	9683.1
Gully support tendons	24	49.89	1197.36
Backfill	4	2 775.2	11100.8
		Overall price	40133.93
		Total area	287.3
		R/m^2	139.70

Support costs for breast mining stopes—Two calculations were done. The first calculation includes backfill material only and the second one includes material and slurry. These calculations are shown in Tables IX and X.

Table VII

Cost of supporting underhand stopes including backfill material and slurry bag

Steep reef including backfill material and slurry			
Product	**No. of units**	**Unit price**	**Total price**
Prestressing plate	16	63.5	1 016
Elongates	101	169.67	17 136.67
Durapak	630	15.37	9 683.1
Gully support tendons	24	49.89	1 197.36
Backfill	4	6 615.2	26 460.8
		Overall price	55 493.93
		Total area	287.3
		R/m^2	193.16

Table IX

Cost of supporting breast mining stopes including backfill material only

Flat reef			
Product	**No. of units**	**Unit price**	**Total price**
Plate hydro	22	63.5	1 397
Elongates	108	169.67	18 324.36
Durapak	990	15.37	15 216.3
Support tendons	27	49.89	1 347.03
Backfill	4	2 775.2	11 100.8
		Overall price	47 385.49
		Total area	373.8
		R/m2	126.77

Table X

Cost of supporting breast mining stopes including backfill material and slurry bag

Flat reef including backfill material and slurry			
Product	**No. of units**	**Unit price**	**Total price**
Plate hydro	22	63.5	1 397
Elongates	108	169.67	18 324.36
Durapak	990	15.37	15 216.3
Support tendons	27	49.89	1 347.03
Backfill	4	6 615.2	26 460.8
		Overall price	62745.49
		Total area	373.8
		R/m2	167.86

Appendix III

Mine planning

Table I

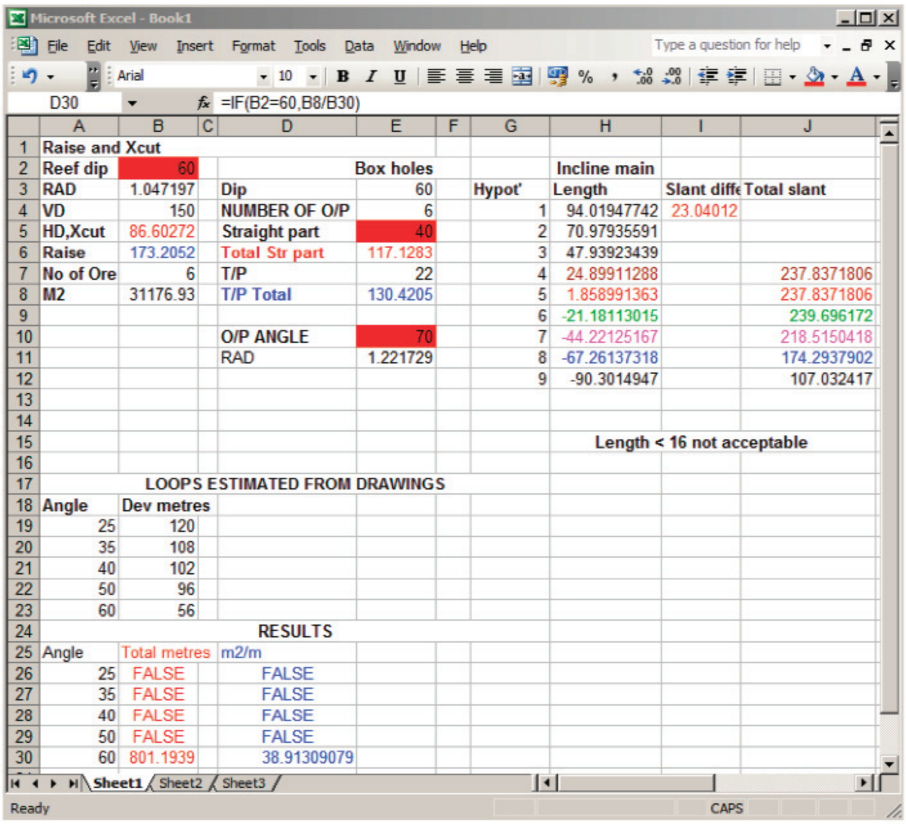
Planning and scheduling, face length and angle from strike for underhand

True dip	Apparent dip	Incline angle from strike	Face length
31	30	76.0713	25.752
32	30	70.3917	26.496
33	30	66.3828	27.232
34	30	63.2356	27.960
35	30	60.3934	28.679
36	30	58.1657	29.389
37	30	56.1059	30.091
38	30	54.1818	30.783
39	30	52.3632	31.466
40	30	51.0355	32.139
41	30	49.3907	32.803
42	30	48.2106	33.457
43	30	47.0901	34.099
44	30	46.0201	34.733
45	30	45	35.355
46	30	44.0201	35.967
47	30	43.075	36.568
48	30	42.1705	37.157
49	30	41.2929	37.735
50	30	40.4445	38.302
51	30	40.024	38.857
52	30	39.2301	39.401
53	30	38.4538	39.932
54	30	38.1022	40.451
55	30	37.3703	40.940
56	30	37.0534	41.452
57	30	36.3549	41.933
58	30	36.074	42.393
59	30	35.4103	42.858
60	30	35.1552	43.301

Table II

Planning and scheduling, face length and angle from strike for underhand mining

True dip	Apparent dip	Incline angle from strike	Face length
32	32	90.0	25.0
33	32	76.6	25.7
34	32	71.4	26.4
35	32	67.5	27.1
36	32	64.4	27.7
37	32	61.7	28.4
38	32	59.4	29.0
39	32	57.4	29.7
40	32	55.5	30.3
41	32	53.9	31.0
42	32	52.4	31.6
43	32	51.0	32.2
44	32	49.7	32.8
45	32	48.5	33.4
46	32	47.4	33.9
47	32	46.4	34.5
48	32	45.5	35.1
49	32	44.6	35.6
50	32	43.8	36.1
51	32	43.0	36.7
52	32	42.3	37.2
53	32	41.6	37.7
54	32	40.9	38.2
55	32	40.3	38.6
56	32	39.7	39.1
57	32	39.2	39.6
58	32	38.7	40.0
59	32	38.2	40.4
60	32	37.7	40.9



Microsoft Excel - Book1

D30 =IF(B2=60,B8/B30)

	A	B	C	D	E	F	G	H	I	J
1	Raise and Xcut									
2	Reef dip	60			Box holes			Incline main		
3	RAD	1.047197		Dip	60		Hypot'	Length	Slant diffe	Total slant
4	VD	150		NUMBER OF O/P	6		1	94.01947742	23.04012	
5	HD,Xcut	86.60272		Straight part	40		2	70.97935591		
6	Raise	173.2052		Total Str part	117.1283		3	47.93923439		
7	No of Ore	6		T/P	22		4	24.89911288		237.8371806
8	M2	31176.93		T/P Total	130.4205		5	1.858991363		237.8371806
9							6	-21.18113015		239.696172
10				O/P ANGLE	70		7	-44.22125167		218.5150418
11				RAD	1.221729		8	-67.26137318		174.2937902
12							9	-90.3014947		107.032417
13										
14										
15								Length < 16 not acceptable		
16										
17		LOOPS ESTIMATED FROM DRAWINGS								
18	Angle	Dev metres								
19	25	120								
20	35	108								
21	40	102								
22	50	96								
23	60	56								
24				RESULTS						
25	Angle	Total metres		m2/m						
26	25	FALSE		FALSE						
27	35	FALSE		FALSE						
28	40	FALSE		FALSE						
29	50	FALSE		FALSE						
30	60	801.1939		38.91309079						

Sheet1 Sheet2 Sheet3

Ready CAPS

Figure 1—Example of calculations of development metres on spreadsheet