Chapter 4.0

Effect of cut-out distance on roof performance

4.1 Introduction

One of the critical parameters in mechanical miner sections is the unsupported face advance, which determines not only the stability of the initial unsupported roadway but also that of the final supported roof. Therefore, underground monitoring programme is continued in 13 monitoring sites using 26 stations with the aim of establishing the effects of unsupported cut-out distance on roof and roof bolt performances.

While increased cut-out distances can increase production significantly, the extended cutting may endanger workers by exposing them to greater risks of injury due to roof falls. The major concern regarding extended cutting is that the unsupported roof area is larger, and that the time before permanent support installation is longer.

The standard cut-out distance of 6.0 m is regulated in other major coal producing countries and it is currently set at 12 m in South Africa. However, extended cuts (longer than 12 m) have been approved by the Department of Minerals and Energy (DME) and the maximum unsupported distance is as long as 24 m in some of South African collieries.

However, it was found that there are relatively few published references on determining effective cut-out distances as compared to other aspects of coal mining. References covering various aspects of the problem were selected and are summarized together with regulations for extended cut-out distances in major coal producing countries.

4.2 Research conducted

The references relevant to cut-out distances include remote-control operation of continuous mining machines, the control of dust and methane, elimination of frictional ignitions, effective ventilation methods, and human factors (worker/machine interaction). Cut-out distance ground control aspects are also mentioned in relatively few references. This thesis investigated the ground control problems associated with extended cut mining. Therefore, only the literature which deals with ground control aspects of extended cut mining was reviewed.

Remote control, ventilation and human factors aspects of extended cut mining can be found in the following references:

Remote control mining: Warner (1973a), Lindsay (1973), Davis (1977).
Ventilation: Divers et al. (1982), Taylor et al. (1992), Volkwein et al. (1985), Campbell (1979), Jayaraman (1987).
Human factors: King and Frants (1977), Sanders and Kelly (1981), Love and Randolph (1991 and 1992), Randolph (1992a).

The majority of research into the effects of extended cut-out distances on ground control have been conducted in the USA by the National Institute for Occupational Safety and Health (NIOSH) during the period 1993 to 1998.

Bauer et al. (1993) conducted a preliminary examination of coal mine roof-fall fatalities from 1988 through 1992. They reported that extended cutting was a contributing factor in approximately 23 per cent of the fatal roof falls, and that geology was an influence in over 80 per cent of the roof fall fatalities in both extended- and non-extended-cut mining. They also reported that nearly 65 per cent of the extended-cut roof fall fatalities were the result of non-approved extended cutting (mining of cuts deeper than 6.0 m without an extended-cut permit or mining deeper than the approved extended-cut depth). Overall, the fatality rate was found to be 37 per cent lower for extended-cut mines. They concluded that in nearly 40 per cent of all roof fall fatalities, the victim was behind the last row of permanent support (Bauer, 1998).

Grau and Bauer (1997) reported on an underground study that addressed the long-term stability of extended-cut areas (over a 10 month period) as compared to non-extended-cut areas. They used a rating system modified from one developed by Mucho and Mark, 1994. The long-term stability was analysed by comparing how cuts in each rating category changed and how the extended cuts changed with respect to the nonextended-cuts. They concluded that a high percentage of extended cuts experienced roof damage over time even though these areas initially had stable roof conditions. In non-extended cut areas where changes occurred, the damage was more severe.

Bauer (1998) investigated site specific stability associated with the mining of extended cuts. He concluded that there was no significant increase in roof fall incidence rates after the mines were granted approval to mine extended cuts. The underground investigations revealed a relationship between depth-of-cut and roof conditions; i.e. that extended cuts were generally mined where the roof was stable and non-extended cuts were mined where the roof showed signs of instability. Also, the study indicated that extended cuts were twice as likely to experience changing roof conditions over time than non-extended cuts. He stated that this occurred because it was easier to detect changing roof conditions in areas originally found to have no visible stability problems (the areas where extended cuts are mined), than it was to detect changes in areas already experiencing stability problems.

Bauer (1998) also found that there was an increase in worker injuries during the remote-control mining of extended cuts. Accident and fatality information suggested that the mining of 90 deg. crosscuts presented additional worker-safety concerns. An alternative shown to minimise these concerns was the mining of angled crosscuts instead of right- and left-hand 90^0 crosscuts.

Two dimensional (2D) numerical modelling was also conducted to understand roof and pillar reactions during extended-cut mining. The numerical modelling successfully predicted where roof displacements would be expected to occur, and delineated the roof-stability concerns caused by geological discontinuities.

Bauer (1998) established the following formula to estimate the safe cut-out distances:

$$CutDepth = 8.1 + 0.564\ (CMRR) - 0.152\ (B) - 0.0029\ (H) \qquad [4\text{-}1]$$

where $CMRR$ = Coal Mine Roof Rating

B = Bord width (ft)

H = Depth below surface (ft)

Bauer (1998) also investigated the applicability of analytical solutions to determine the safe cut-out distances. He concluded that the strength of the rock is not as important in determining the maximum safe cut-out distances as is the type, number and/or frequency of discontinuities in the immediate roof. Finally, he suggested that until another method is proposed, tested, and verified, the decision as to the safe depth of each individual cut must be left to the CM operator.

Bauer (1998) stated that one of the major concerns of extended cuts is the time delay for support installation. In general, it is expected that support should be installed as soon as the mining takes place to prevent bed separation. The stand-up time is dependent upon the geotechnical parameters in the roof of the excavation. Where mobilisation of low friction parting planes occur, the beds delaminate inducing tensile and shear forces which can cause the beam to fail.

The tensile strength of rock is only 1/10 of the compressive strength, and strata failure is often initiated by tensile cracks at the edges of the unsupported span. With time these cracks grow. Also, the material is affected by oxygen and moisture (ventilation), which decrease its inherent strength with time, van der Merwe, 1995.

Currently, the effect of time on roof behaviour cannot be quantified mathematically, although, there have been studies to identify the effect of time on support and roof performance.

Buddery (1989) suggested that, in order to gain maximum benefit, roof bolts should be installed as soon as possible after the roof has been exposed as this will limit the amount of roof deflection and bed separation. Small cut-out distances are therefore implied.

Radcliffe and Stateham (1980) investigated the interval between exposure and support of the roof, in a mine in the USA, using both instrumentation and statistical methods. Instrumentation studies were designed to equate displacement and rates of displacement with areas of roof left unsupported from 15 minutes to more than four days. A statistical study was completed to compare roof fall occurrence with time-lapse intervals encountered during normal bord and pillar mining. Results from this investigation showed the time lapse, in this specific mine, to be insignificant with respect to roof stability.

The following basic relationships that govern stand-up time were originally formulated by Austrian tunnelling engineers (Mark, 1999).

- for a given rock mass, a tunnel's stand-up time decreases as the roof span becomes wider, and
- for a given roof span, a tunnel's stand-up time decreases as the quality of the rock mass reduces.

Using data collected from numerous tunnels and mines, Bieniawski (1989) was able to quantify this relationship. Bieniawski used Rock Mass Rating (RMR) as the measure of rock quality. His data indicated that an unsupported 4.3 m wide tunnel would be expected to collapse immediately if the RMR of the roof was less than 33. If the tunnel was 6.0 m wide, immediate collapse would be expected if the RMR was less than 41. The following equation expresses the relationship for this range of tunnel spans (approximately the range encountered in underground coal mining), (Mark, 1999).

$$RMR = 13 + 1.4\,B \qquad [4\text{-}2]$$

where B is the bord width, in feet.

Mark (1999) stated that because roof bolting normally takes place within several hours of mining, the collapse of an extended cut may be considered "immediate".

In order to identify the lithological factors that influence the structural competence of a mine roof, Molinda and Mark proposed a Coal Mine Roof Rating (CMRR) in 1994.

In developing the CMRR, field data were collected from nearly 100 mines in every major coalfield in the USA.

Mark (1999) used the CMRR to determine stand-up times at 36 mines with a questionnaire being used to identify the stand-up times. The results were divided into three classes: Class 1 "always stable", Class 2 "sometimes stable" and Class 3 "never stable". Mark concluded that the CMRR/depth of cover and CMRR / bord width are statistically significant to determine the stability in the extended cut sections using the CMRR.

The following relationships for CMRR-depth of cover and CMRR-bord width are given respectively by Mark (1999):

$$CMRR_{crit} = 40.9 + (H/100) \quad [4\text{-}3]$$

$$CMRR_{crit} = 19.2 + 1.64\ B \quad [4\text{-}4]$$

where $CMRR_{crit}$ = CMRR value below which instability may start to occur.

H = depth below surface (ft)

B = the road width (ft)

4.2.1 Summary of current knowledge

In South Africa the standard cut-out distance is 12 m. However, if the roof is defined as self-supporting then systematic support is not required and work under unsupported roof is permitted. Under these conditions defining a maximum cut-out distance becomes irrelevant with respect to ground control, and the issue becomes one of sufficient ventilation at the face and dust control.

The issues which are given most consideration in determining cut-out distances include remote-control operation of continuous mining machines, the control of dust and methane, elimination of frictional ignitions, effective ventilation methods, and human factors (worker/machine interaction). Given the general requirement that no person should be allowed under unsupported roof, roof stability is seldom considered a major issue in determining cut-out distance and few references covering this topic could be found. Nevertheless a number of detailed studies relating stand-up time to rock mass quality and mining dimensions have been carried out and could be used for determining maximum cut-out distances. A serious limitation is the possibility of unexpected changes in roof stability, and also verification of the empirical relationships would have to be carried out for local conditions.

However, further research into the effects of extended cut-out distances on ground control was conducted in the USA by the National Institute for Occupational Safety and Health (NIOSH) during the period 1993 to 1998 by Bauer (1998). He concluded that extended cut-mining is about as safe as the mining of non-extended cuts from a roof fall accident and fatality perspective, mainly because extended cut mining was only allowed in good quality roof conditions.

4.3 Underground monitoring

4.3.1 Introduction

In order to determine the displacements in the roof for various cut-out distances, in different geotechnical areas, an underground monitoring programme was carried out. A total of 13 sites at six collieries in four seams were monitored as part of this task. Approximately 80 per cent of coal production comes from Witbank and Highveld Coalfields, and therefore the monitoring sites were concentrated in these two coalfields.

Table 4-1 Distribution of test sites

Number of sites	Colliery	Sites	Seam
1	S	1	Vereeniging 2B
10	A	4	Witbank No 2
	B	2	Witbank No 2
	G	1	Witbank No 2
	K	3	Highveld No 2
2	T	1	Highveld No 2
	B	1	Witbank No 5

Two sonic probe extensometers were again used to monitor the roof and support behaviour in the sites. In the initial tests it was observed that drilling a 7.0 m hole into the roof was difficult and in many sites there was not a proper drilling machine available. Therefore, a 4.0 m sonic probe extensometer was used as drilling of this length hole was more readily accomplished.

4.3.2 Underground monitoring procedure

In order to monitor the roof behaviour in continuous miner and road-header sections, two different monitoring programmes were established, which suited the mining cycles in both mechanical miner sections.

Two different cutting sequences were used in the experiments to cater for different mining equipment.

CM sequence: The full cut length was completed in four steps, Figure 4-1. The first sonic probe hole was drilled and instrumented next to the last row of the support approximately 1.0 m from the face. The face was then advanced by half of the standard cut-out distance with a single drum cut and the second hole was drilled and instrumented at the face. Then, the second, third and fourth lifts were cut. A sonic probe reading was taken following the first, second and the fourth mining steps to monitor movements into the roof as mining takes place.

Road-header sequence: The full cut length was completed in two steps, Figure 4-2. The first sonic probe hole was drilled and instrumented just behind the last row of support, approximately 1.0 m from the face. The face was then advanced by half of the standard cut-out distance in full bord width. The second monitoring hole was drilled and instrumented at the face. Then the second half of the full length was cut. A sonic probe reading was taken after each mining step.

To record all the information relevant to roof strata deformation prior to the installation, the first monitoring holes were drilled and instrumented close to the last row of support, and the second hole approximately 1.0 m away from the face in the unsupported ground.

Drill bit sizes, resin quantities and support types and lengths were recorded in each monitoring site. This information is presented in each graph from each monitoring site. In addition, the support installation and drilling were also monitored in each section. However, the performance of the drilling crew tends to improve when the crew is being observed. Therefore, it was decided that the support installation should be monitored in the sections by visual observations, and van der Merwe's (1998) support installation and roof damage checklists were adopted in each site, Figure 4-3.

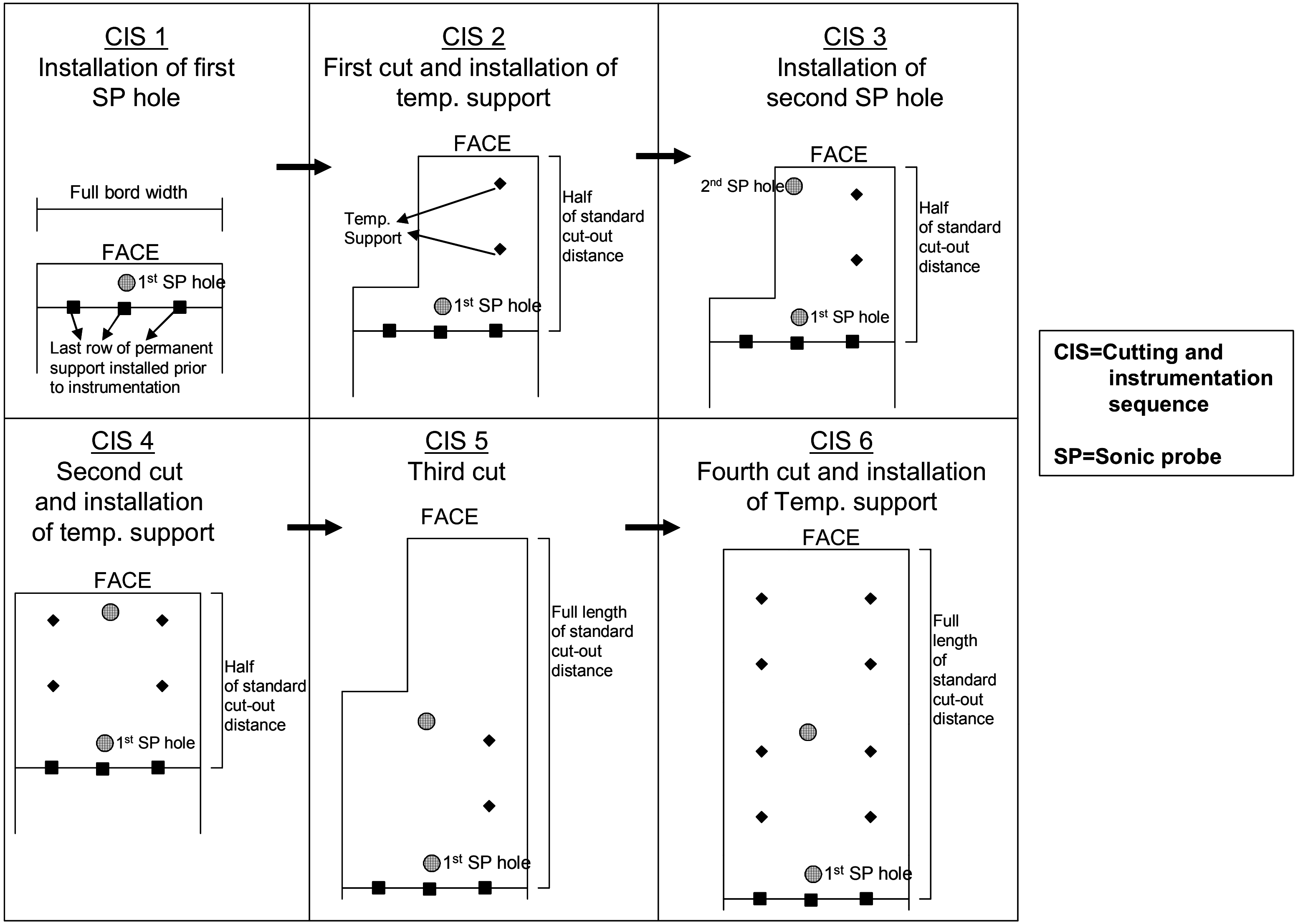
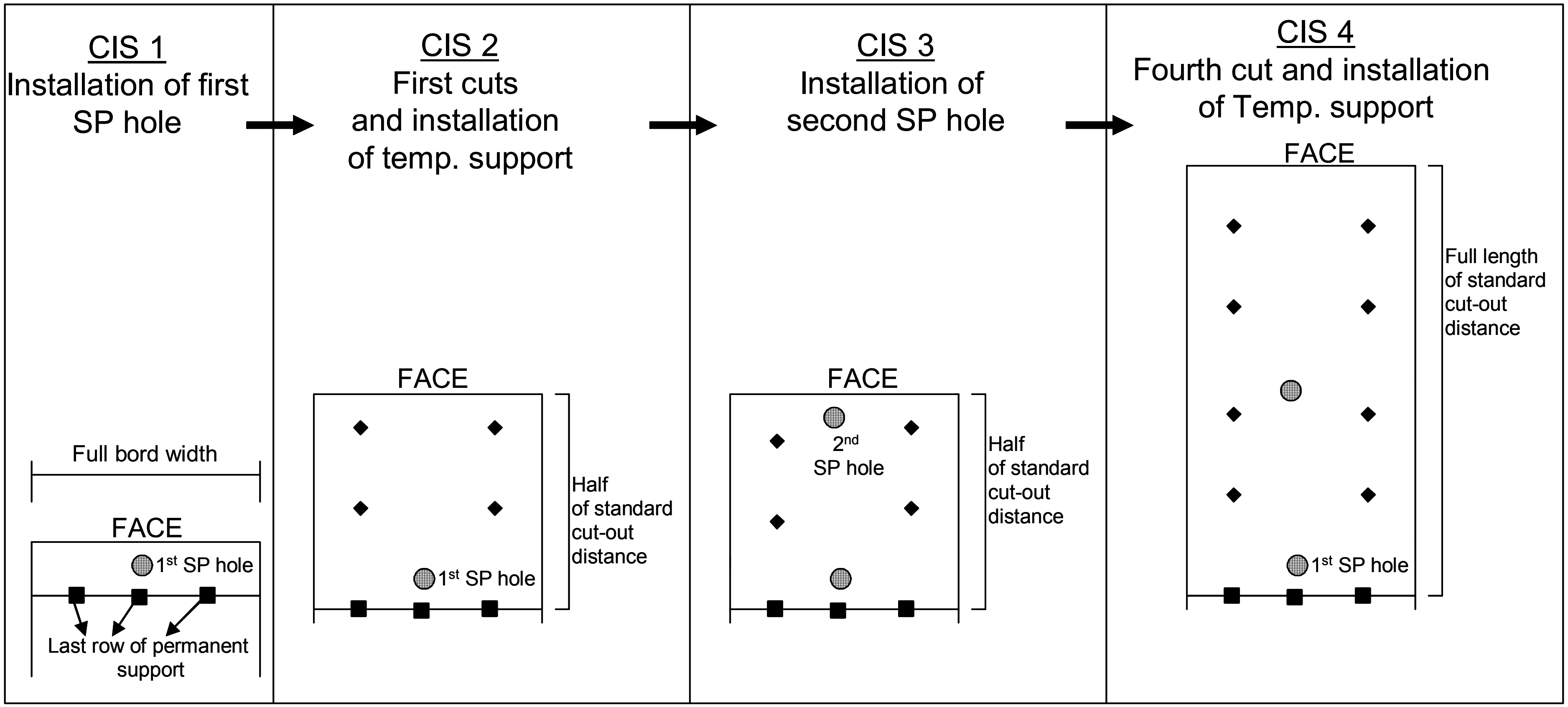


Figure 4-1 Cutting and instrumentation sequence in CM sections



CIS=Cutting and instrumentation sequence

SP=Sonic probe

Figure 4-2 Cutting and instrumentation sequence in road header sections

Observation \ Cause	Installation problems											System problems	
	Hole too short	Hole too long	Weak washer	Spin/wait times wrong	Temperature	Insufficient tension	Worn adaptor	Crimp too strong	Crimp too weak	Torque too high	torque too low	Spacing too wide	High stress
Too much thread protruding	●			◑ ⊕	◑ ⊕				○	○			
Too little thread protruding		●						◑ ⊕			○		
Loose washer	◑			○ ⊕	○	● ⊕		◑ ⊕			○		
Rounded nut							● ⊕	○		◑ ⊕			
Deformed washer			◑							○		○	● ⊕

● Most likely cause ◑ Possible cause ○ Potential cause ⊕ Most serious

(a)

Observation \ Cause	Roadway too wide	Spacings too wide	Bolts too short	Insufficient tension	High horizontal stress	Weak anchor	Weak steel	Support installed too late	Alarm rating
Tension crack	●		◑	○		○		◑	1
Guttering	◑				●			◑	1
Falls, bolts broken	○	◑			○		●		2
Falls, bolts pulled out	◑		○		○	●			2
Falls higher than bolt length	◑		●		○			○	2
Falls, bolts hanging free		◑		●		○		○	3
Falls, rock columns on top of bolts	○	●						◑	3
Seriousness rating	1	3	4	6	4	5	7	2	

●Most likely cause ◑ Possible cause ○Potential cause

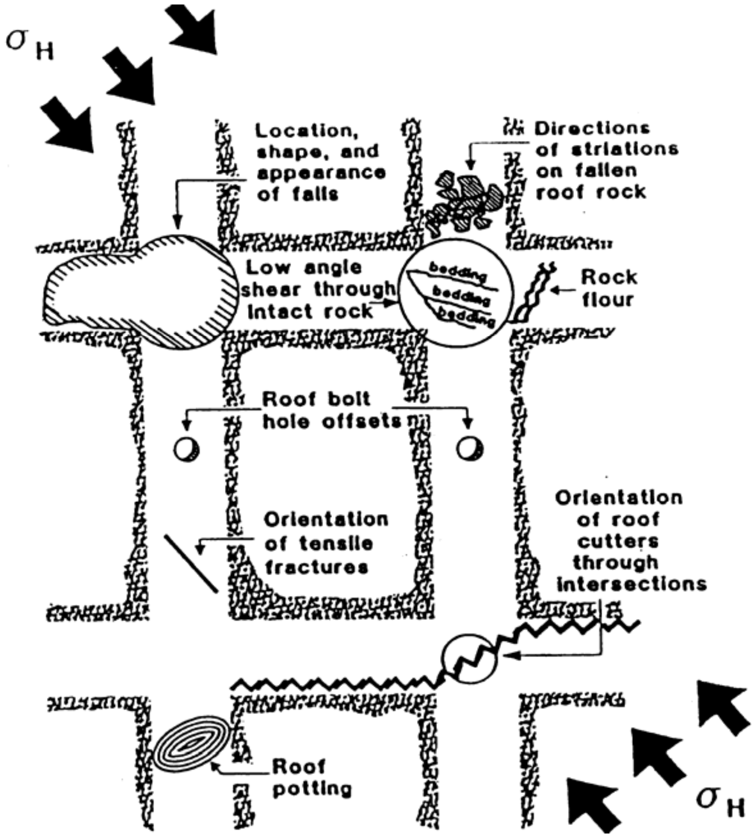
Seriousness rating: 1 is the most serious non conformance

Alarm rating: 1 is the most dangerous situation

(b)

Figure 4-3 a) Probable cause of observed roof damage. b) Probable cause of observed roof bolt defects (after van der Merwe, 1998)

Horizontal stress manifests itself in a variety of features that can be observed underground. Therefore, indicators of horizontal stress in each section and site were also carefully monitored using the technique developed by Mark and Mucho (1994), Figure 4-4.



Feature	Observation Noted	Relationship to σ_H
CUTTER - GUTTERING OR KINK ROOF	Location in entry, especially tendency passing through intersections	Entry location gives indication of angle of mining to stress field. Through intersections tries to align with σ_H
TENSILE FRACTURES	Direction	Gives direction of σ_H
ROOF POTTING	Direction of major and minor axes	Major axis gives direction of σ_H
ROOF BOLT HOLE OFFSETS	Direction of roof movement	Roof layers move in direction of σ_H
SHEAR PLANES AND ROCK FLOUR	Direction	Planes and rock flour lines are in the direction of σ_H
STRIATIONS ON ROOF ROCK	Direction	Striations are parallel to σ_H
ROOF FALLS	Location , shape and appearance	Location gives clues as to the general directionality of the stress field. High angular shape usually indicates high horizontal stress with stepped shear failures usually predominating on one side

Figure 4-4 Summary of underground stress mapping techniques (after Mark and Mucho, 1994)

Depending on the mining method and rate of face advance, the time lapse between further sets of readings varied from hours to days. In a typical development section underground, the centre roadway in the section (belt road) was usually monitored. When monitoring in the belt road was not possible, the holes were drilled in the closest roadway to the belt road. The reason for this was for the tensile stress and deformations to be at their maximum in the middle of the panels.

Initially, it was planned that in each site petroscope holes should be drilled next to each sonic probe hole in order to gain maximum information. However, it was found that, because these experiments interfered significantly with production, it was not possible to drill extra holes, which would further delay the production and support installation in the sections. Also, the possibility of underground core drilling to obtain the stratigraphy of the first 2.0 m into the roof was investigated. Difficulties were experienced in drilling with the available roofbolters and problems also arose because of delays caused to production. Therefore, it was decided not to core drill in the sections, where experiments took place. However, detailed borehole logs from the vicinity of the experiment sites were obtained from the geology departments at each colliery. The detailed logging of the immediate roof strata from those boreholes is also presented in each graph from each monitoring sites.

4.3.3 Processing of information

The information obtained from each site was processed similar to previous chapter. However, modifications are made in the presenting the results; the results from both sonic probe extensometer holes are presented in one figure in which the individual graphs from each hole have been cropped at the 3.0 m elevation.

After the installations were completed, the initial readings were taken from both holes. These comprise a minimum of three sets from each hole, which were screened for any obvious anomalies or booking errors. They were then entered into the program where they were averaged, and the calculations carried out to produce the graphic results necessary for interpretation. All the subsequent sets of readings were treated in a similar manner with the program comparing them to the first (datum) set of readings from which the displacements were calculated.

4.4 Colliery ‘A’

Four sites in three different sections were monitored at Colliery ‘A’. The colliery is situated in the Witbank Coalfield and mining No 2B Seam at a depth of 32 to 59 m using a continuous miner. While the CM experiment sequence was used in the first two sites, the road header sequence was used in the remaining two sites.

4.4.1 Colliery 'A' Site 1, Test 1

In Site 1 two experiments were conducted approximately 150 m apart from each other. Site 1 was an eight-roadway, primary bord and pillar production section, and in both experiments the sonic probe monitoring holes were drilled in one-left roadway (one left-hand from the centre, belt road). Because of a major water aquifer 5.0 to 6.0 m into the roof, some degree of damage in the workings was observed. Initially, it was aimed to drill 8.0 m sonic probe holes into the roof, however, because of the aquifer, the holes were limited to 5.0 m into the roof. Also, scaling in the pillar - roof contacts indicated some degree of horizontal stress. This was confirmed by the stress mapping technique. Installation and performance of support were found to be excellent in the section.

After the installation of the first hole was completed, the initial reading was taken from this hole. Then the face was advanced by 8.0 m in full bord width of 5.8 m, and the second hole drilled and instrumented. Readings were taken from both holes. Then, the face was advanced a further 8.0 m and readings were taken from both holes. The face was left unsupported for 48-hours in order to determine the effect of time on deformation. After 48-hours readings were taken and entered into the program. Further readings were taken 5 and 11 days after the support installation. The face advance was approximately 30 m, when the last reading was taken.

The results obtained from both holes during the first experiment in Site 1 are presented in Figure 4-5. The summary of site performance in this site is given in Table 4-2.

Table 4-2 Site performance Colliery 'A' Site 1, Test 1

Coalfield:	Witbank	Seam:	No 2
Site:	One – Test 1	Positions:	Roadway
Road widths:	5.8 m	Pillar widths:	9.0 m
Mining height:	3.0 m	Depth:	32 m
Mining method:	CM and shuttle cars		
Cut-out distance	16 m		
Roof strata:	0.13 m grit, 0.65 m grit/coal, and 0.08 m coal overlain by 0.74 m thick sandstone		
Support:	1.5 m x 20 mm OZ-Bar, full column resin in 25 mm hole. Support density 0.57 bolt/m^2.		

Performance:	Although there were indications of the presence of high horizontal stress within the section, there was no visual evidence to indicate its development at either monitoring site.
	Roof separation was only measured at one site up to a maximum of 1.1 m into the shale allowing the roof skin a total relaxation of 2.5 mm after a face advance of 66 m over a 66 day period.

4.4.2 Colliery 'A' Site 1, Test 2

Because the experiment sites in Site 1 were very close to each other, descriptive information obtained in the first experiment was used for the second experiment.

The same cutting and instrumentation sequence as Test 1 was used during the second experiment. The results obtained from both holes during the second experiment in Site 1 are presented in Figure 4-6. In this experiment the final readings were taken after 542 m face advance, which indicated that even after this face advance, the displacement in the roof was not significant (± 0.5 mm). The summary of the site performance in this site is given in Table 4-3.

Table 4-3 Site performance Colliery 'A' Site 1, Test 2

Coalfield:	Witbank	Seam:	No 2
Site:	One – Test 2	Positions:	Roadway
Road widths:	5.8 m	Pillar widths:	9.0 m
Mining height:	3.0 m	Depth:	32 m
Mining method:	CM and shuttle cars		
Cut-out distance	16 m		
Roof strata:	0.13 m grit, 0.65 m grit/coal, and 0.08 m coal overlain by 0.74 m thick sandstone		
Support:	1.5 m x 20 mm OZ-Bar, full column resin in 25 mm hole. Support density 0.57 bolt/m^2.		
Performance:	Approximately 1.5 mm dilation, 1.5 m into the roof, was observed in No 1 hole. Initial dilation of 1.0 mm was recorded after the completion of 16 m		

unsupported face advance. No further dilation was recorded after 48-hours stand-up time. Further 0.5 mm dilation took place after the face advanced by 542 m.

The total dilation in the second hole, which took place 1.0 m into the roof at the same interface of grit/coal

4.4.3 Colliery 'A' Site 2

Site 2 was an 11-roadway primary bord and pillar production section, and the sonic probe monitoring holes were drilled in the two-left roadway. Similarly, in some localised areas in the section, floor heave and scaling of roof-pillar contact indicated horizontal stress driven damage. The underground dimension control, installation and performance of support were excellent in the section.

After the installation of the first hole was completed, the initial reading was taken from this hole. The face was then advanced by 8.7 m at full bord width and the second hole drilled and instrumented with sonic probe anchors, and readings were taken from both holes. Advancing the face by 8.0 m, the full cut-out length of 16.7 m was completed. The readings were again taken from both holes. The face was left for 48-hours to monitor the effect of stand-up time. Further readings from both holes were taken 96 hours after the support installation, 15 days after the support installation (approximately 50 m face advance), and after 364 m face advance.

The results obtained from both holes during the experiment in Site 2 are presented in Figure 4-7. The results showed that after 364 m face advance, No 2 hole indicated some degree of displacement. However, it is known that after the initial phase of the experiment (up to 15 days after support installation, as indicated in Figure 4-7), an intersection was developed between the two holes during the mining cycle. Therefore, it was thought this movement was due to stress changes in the area. The summary of the site performance in this site is given in Table 4-4.

Table 4-4 Site performance Colliery 'A' Site 2

Coalfield:	Witbank	Seam:	No 2
Site:	Two	Positions:	Roadway
Road widths:	6.85 m	Pillar widths:	7.5 m
Mining height:	3.4 m	Depth:	59 m

Mining method: Road header and shuttle cars

Cut-out distance 16.7 m

Roof strata: 0.54 m sandstone overlain by 1.8 m shale

Support: 0.9 m x 20 mm OZ-Bar, full column resin in 25 mm hole.
Support density 0.41 bolt/m^2.

Performance: No 1 hole was stable throughout the experiment period.

The total dilation recorded in the No 2 hole was 1.5 mm, which took place 0.5 m into the roof at the sandstone shale contact. This displacement took place between 60 m to 364 m face advance. Initially, this movement was thought to be due to the effect of face advance. However, detailed investigation showed that an intersection was developed during this period, and this movement was due to stress changes during the development of the intersection.

4.4.4 Colliery 'A' Site 3

Site 3 was a three-roadway shortwall development section, and the sonic probe monitoring holes were drilled in the centre roadway. A road header together with the shuttle cars were used to mine No 2B Seam in the Witbank Coalfield. The stress mapping technique showed that there was no apparent horizontal stress driven damage in this section. In general, the pillar and roof conditions were excellent. The installation and performance of support were also found to be excellent in the section.

The road-header experiment sequence was used at this site. After the installation of the first hole was completed, the initial reading was taken from this hole. The face was then advanced by 8.0 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. The further readings from both holes were taken after the completion of 16 m face advance and 96-hours stand-up time. Before the experiment was completed, two more readings were taken, after approximately 20 m (7 days after the support installation) and 100 m face advance.

The results obtained from both holes during the experiment in Site 2 are presented in Figure 4-8. The results indicated that while No 1 hole was stable during the experiment, No 2 hole showed a 3.0 mm displacement after 8.0 m face advance (16 m full length completed) and 48-hours stand-up time. No further displacement was recorded in No 2 hole. Similar to Site 2, an intersection was developed between the two sonic probe monitoring holes after the second last reading (seven days after support installation as indicated in Figure 4-8); however, no further displacement took place after this stress change.

The summary of the site performance in this site is given in Table 4-5.

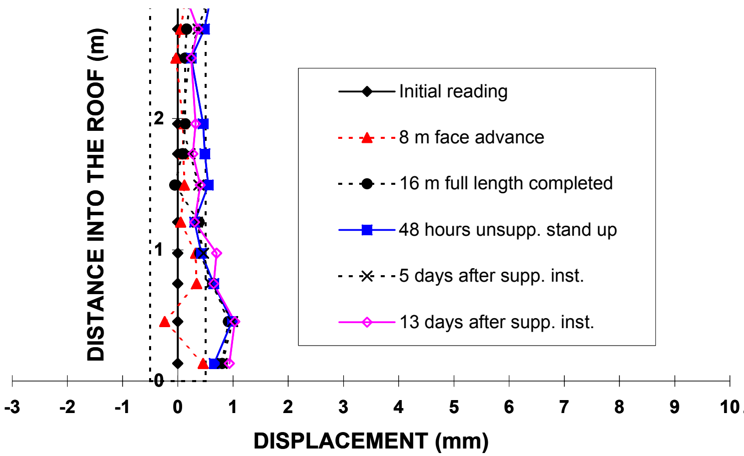
Table 4-5 Site performance Colliery 'A' Site 3

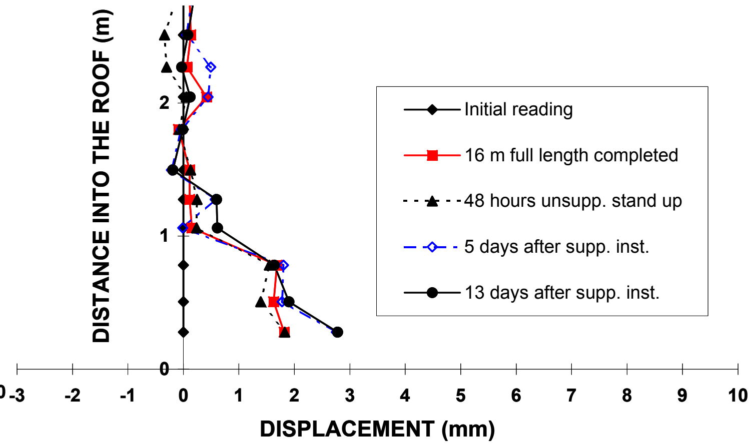
Coalfield:	Witbank	Seam:	No 2
Site:	Three	Positions:	Roadway
Road widths:	6.1 m	Pillar widths:	30 x 10 m
Mining height:	3.0 m	Depth:	53 m

Mining method:	Road header and shuttle cars
Cut-out distance	16.7 m
Roof strata:	0.28 m laminated sandstone overlain by 0.85 m grit which overlain by thick sandstone (>2.0 m)
Support:	1.8 m x 16 mm Re-bar, full column resin in 22 mm hole. Support density 0.44 bolt/m^2.
Performance:	No 1 hole was stable during the experiment. No 2 hole showed approximately 3.0 mm dilation after 8.0 m face advance (16 m full length completed) and 48-hours stand-up time which took place at the grit/sandstone interface, 1.0 m into the roof. No further displacement was recorded in No 2 hole, even after 100 m face advance.

WITBANK COALFIELD
No 2B Seam

COLLIERY "A" - SITE 1 (TEST1)
(CUT OUT DISTANCE 16 m)





Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.6		>2	SHALE, black, fine grained, fissile
0.86		0.74	SANDSTONE, white, coarse to medium
0.78		0.08	COAL, dull lustrous, 10 - 40% bright
0.13		0.65	GRIT/COAL LAMINAE
0		0.13	GRIT, white, medium grained, coaly

Site performance

BOLT TYPE:	OZ-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	25
BOLT LENGTH (mm):	1500
HOLE LENGTH (mm):	1500
NUMBER OF BOLTS IN A ROW:	5
DISTANCE BETWEEN THE ROWS (m):	1.5
BOLT/m^2	0.57
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	5.8
PILLAR WIDTH (m):	9
DEPTH (m):	32
MINING HEIGHT (m):	3
SAFETY FACTOR:	5.19

Cutting sequence

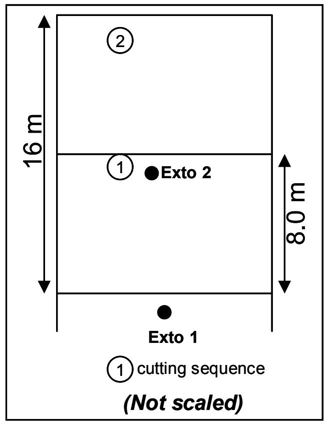
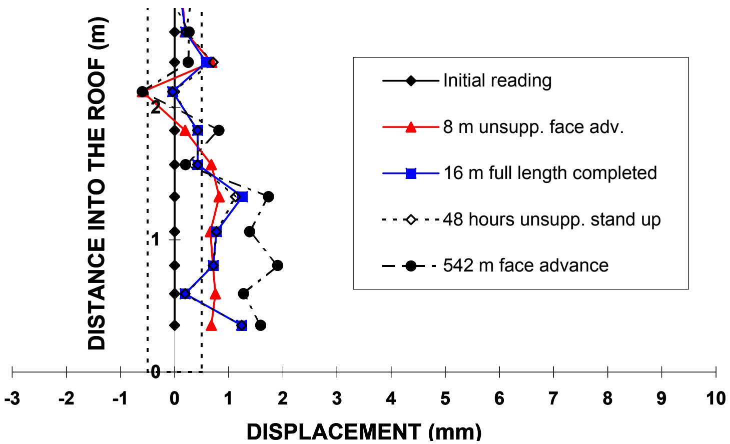


Figure 4-5 Colliery 'A' Site 1, Test 1

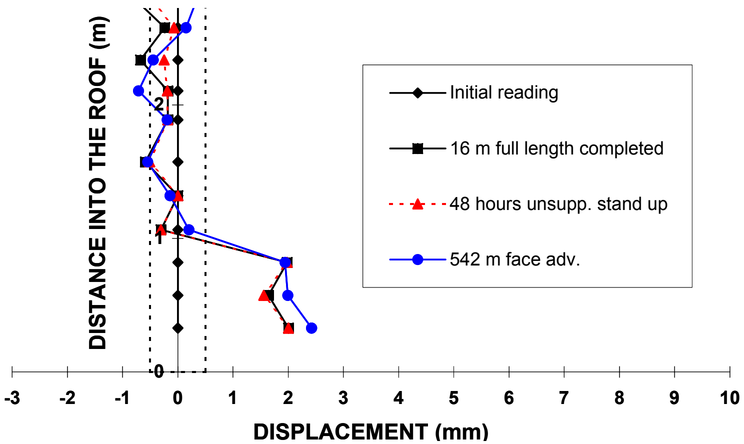
WITBANK COALFIELD
No 2B Seam

COLLIERY "A" - SITE 1 (TEST2)
(CUT OUT DISTANCE 16 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.6		>2	SHALE, black, fine grained, fissile
0.86		0.74	SANDSTONE, white, coarse to medium
0.78		0.08	COAL, dull lustrous, 10 - 40% bright
0.13		0.65	GRIT/COAL LAMINAE
0		0.13	GRIT, white, medium grained, coaly

Site performance

BOLT TYPE:	OZ-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	25
BOLT LENGTH (mm):	1500
HOLE LENGTH (mm):	1500
NUMBER OF BOLTS IN A ROW:	5
DISTANCE BETWEEN THE ROWS (m):	1.5
BOLT/m^2	0.57
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	5.8
PILLAR WIDTH (m):	9
DEPTH (m):	32
MINING HEIGHT (m):	3
SAFETY FACTOR:	5.19

Cutting sequence

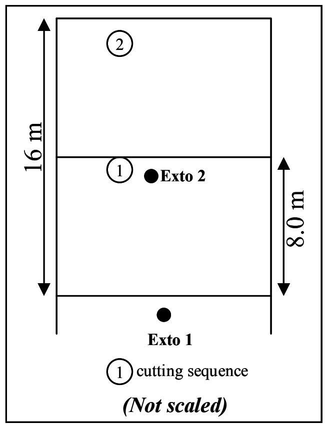
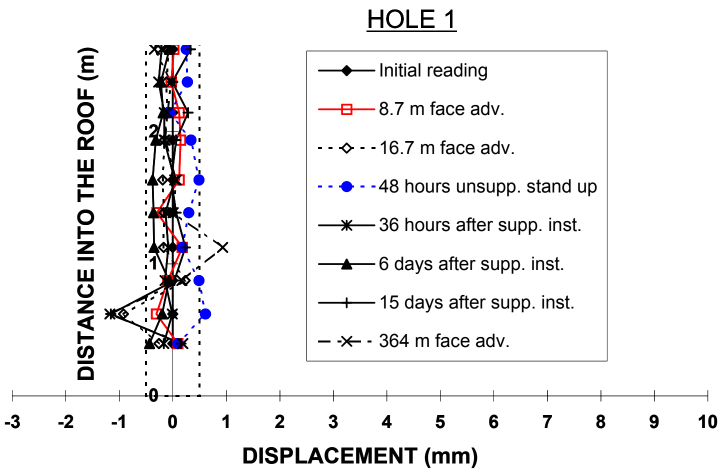
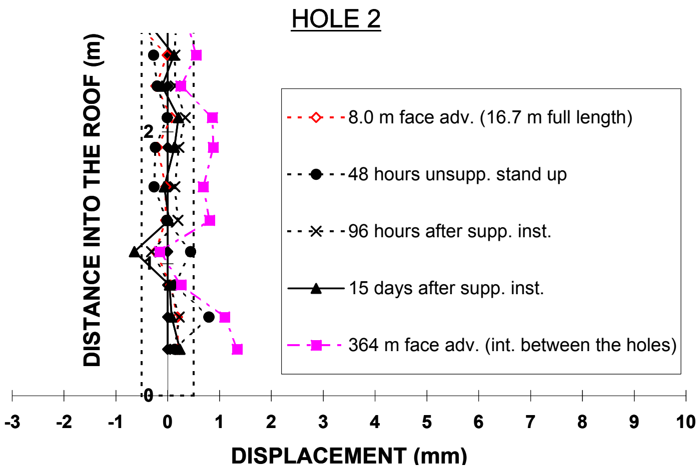


Figure 4-6 Colliery 'A' Site 1, Test 2

WITBANK COALFIELD
No 2B Seam

COLLIERY "A" - SITE 2
(CUT OUT DISTANCE 16.7 m)





Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
0.54		1.8	SHALE, black
0		0.54	SANDSTONE, gritty, silty

Site performance

BOLT TYPE:	OZ-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	25
BOLT LENGTH (mm):	900
HOLE LENGTH (mm):	900
NUMBER OF BOLTS IN A ROW:	4
DISTANCE BETWEEN THE ROWS (m):	1.5
BOLT/m^2	0.41
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	2
BORD WIDTH (m):	6.58
PILLAR WIDTH (m):	7.5
DEPTH (m):	59
MINING HEIGHT (m):	3.4
SAFETY FACTOR:	1.89

Cutting sequence

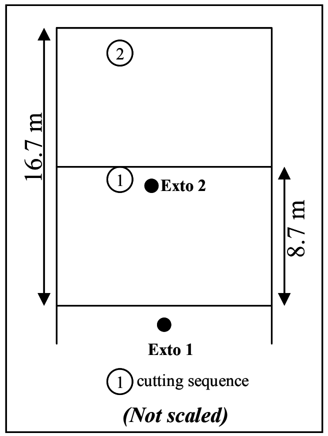
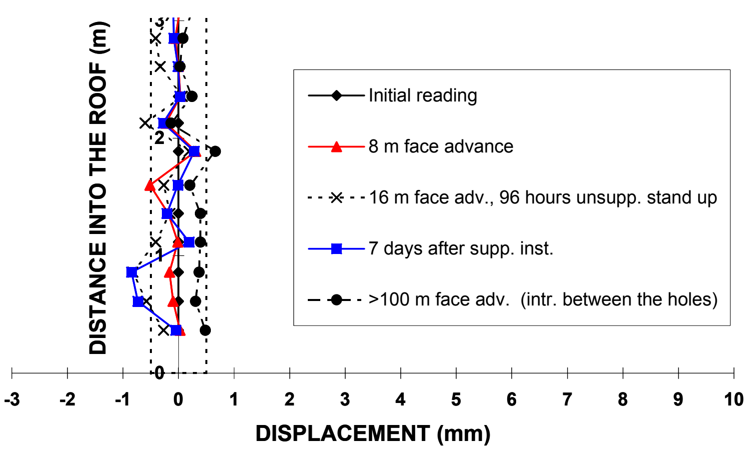


Figure 4-7 Colliery 'A' Site 2

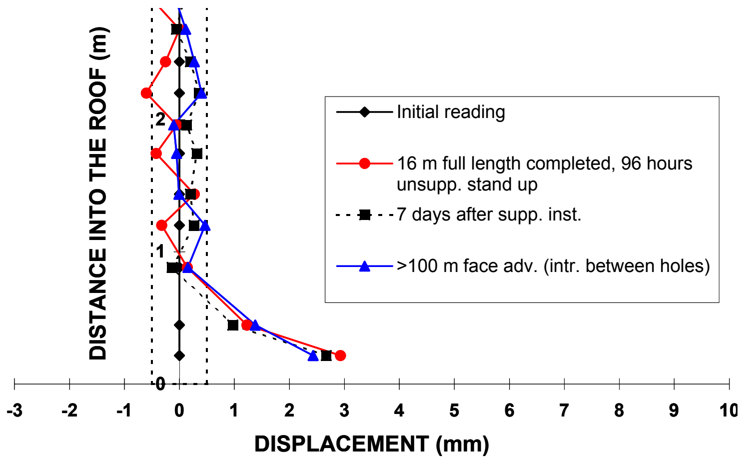
WITBANK COALFIELD
No 2B Seam

COLLIERY "A" - SITE 3
(CUT OUT DISTANCE 16 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.13		>2	Sandstone, light grey, thinly lam.
0.28		0.85	GRIT, white, very coarse grained
0		0.28	SANDSTONE, dark gray, thinly laminated

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	22
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1800
NUMBER OF BOLTS IN A ROW:	4
DISTANCE BETWEEN THE ROWS (m):	1.5
BOLT/m^2	0.44
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	6.1
PILLAR WIDTH (m):	15
DEPTH (m):	53
MINING HEIGHT (m):	3
SAFETY FACTOR:	5.09

Cutting sequence

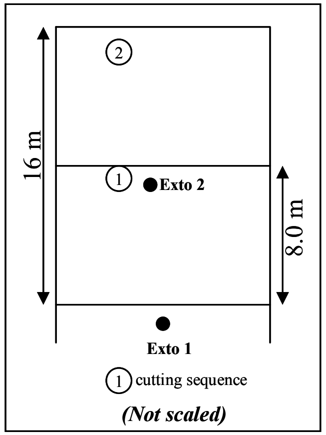


Figure 4-8 Colliery 'A' Site 3

4.5 Colliery 'B'

Three sites in three different sections were monitored at Colliery ‘B’. The colliery is situated in the Witbank Coalfield and mining is currently being conducted in the No 2 and No 5 Seams.

4.5.1 Colliery 'B' Site 1

Site 1 was a five-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the belt road (centre roadway). A CM together with shuttle cars was used in the section to mine No 2 Seam in the Witbank Coalfield. There were no excessive stress indications in the section. While the underground dimension control was satisfactory, the roof and the pillar conditions were good.

The road header experiment sequence was used at this site. After the instrumentation of the first hole, completed at the face, the initial reading was taken. The face was then advanced by 10 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. The cut-out length was then completed by advancing the face by 14 m. The readings from both holes were taken again and the face was left for 48-hours unsupported in order to determine the effect of time. Further readings were taken 20 days after support installation, when the face advance was approximately 50 m.

The results showed that during the experiment both holes were stable and no displacement was recorded in either hole. The results obtained from both holes during the first experiment in Colliery 'B', Site 1 are presented in Figure 4-9. The summary of site performance in this site is given in Table 4-6.

Table 4-6 Site performance Colliery ‘B’ Site 1

Coalfield:	Witbank	Seam:	No 2
Site:	One	Positions:	Roadway
Road widths:	6.5 m	Pillar widths:	10.5 m
Mining height:	4.4 m	Depth:	75 m
Mining method:	CM and shuttle cars		
Cut-out distance	24 m		
Roof strata:	1.0 m coal overlain by 1.05 m thick sandstone		

Support: 1.8 m x 16 mm Re-bar, full column resin in 22 mm hole.
Support density 0.15 bolt/m^2.

Performance: Both holes showed no dilation during the experiment. This is thought to be due to the 1.0 m thick coal left in the coal.

4.5.2 Colliery 'B' Site 2

While this section was a 12-roadway primary bord and pillar production section, the experiment took place in an area where the number of roadways was reduced to five. The sonic probe holes were drilled in the one-left roadway. A road header together with shuttle cars was used in the section to mine the No 2 Seam in the Witbank Coalfield. Stress mapping techniques showed that there was no excessive horizontal stress in the section. The roof and the pillar conditions were good.

The road header cutting sequence was applied in a 31 m cut-out distance in the section. After the instrumentation of the first hole was completed at the face, the initial reading was taken. The face was then advanced by 16 m at full bord width and the second hole drilled and instrumented with sonic probe anchors, and readings were taken from both holes. Advancing the face by 15 m then completed the cut-out length and readings from both holes were taken. Because of the long cut-out distance in the experiment, the face was not left unsupported for 48-hours, and the area was supported as soon as the readings were taken. The face was then advanced by a further 21 m and readings were taken from both holes.

Installation and performance of support were found to be good in the section. During the experiment both holes were stable and no movement was recorded in either hole. One reason for this can be that displacement in the roof had occurred before instrumentation of the second hole, as the length of the first face advance was 16 m. This will be investigated further in the following section.

The results obtained from both holes during the first experiment in Site 1 are presented in Figure 4-10. The summary of site performance in this site is given in Table 4-7.

Table 4-7 Site performance Colliery 'B' Site 2

Coalfield:	Witbank	Seam:	No 2
Site:	Two	Positions:	Roadway
Road widths:	6.7 m	Pillar widths:	15.7 m
Mining height:	4.2 m	Depth:	44 m

Mining method:	Road header and shuttle cars
Cut-out distance	31 m
Roof strata:	0.856 m coal overlain by 0.65 m thick shale/siltstone
Support:	1.8 m x 16 mm Re-bar, full column resin in 22 mm hole. Support density 0.15 bolt/m^2.
Performance:	Both holes showed no dilation during the experiment.

4.5.3 Colliery 'B' Site 3

The section was a 17-roadway, primary bord and pillar production section, and the monitoring holes were drilled in the belt road (centre roadway in the section). A CM with shuttle cars was used in the section to mine No 5 Seam in the Witbank Coalfield. Localized bord and intersection failures and pillar-roof contact scaling in various parts of the section raised possibility of horizontal stress damage. The detailed stress mapping technique also showed that the horizontal stress was higher than the strength of the immediate rock layer. However, while excessive horizontal stress caused damage in the roof in some areas, there was no movement in the roof in the experiment site, again indicating the variable nature of the conditions.

The road header experiment sequence was used in the section. After the installation of the first hole was completed, the initial reading was taken. Then the face was advanced by 6.0 m at full width, the second hole drilled and instrumented and initial readings were taken from the second hole. The face was advanced a further 6.0 m and readings were taken from both holes. Without leaving the face for 48-hours unsupported, the installation of the support was started and readings were taken after the installation of each row of support, in order to determine the effect of bolting in the roof. The readings were taken up to a point where the support passed the second hole in the experiment site. However, because no movement took place during the 12 m cut-out distance, the effect of roof bolting could not be monitored.

In general, the roof and the pillar conditions were good as well as the quality of support installation.

The results obtained from both holes during the first experiment in Site 1 are presented in Figure 4-11. The summary of site performance in this site is given in Table 4-8.

Table 4-8 Site performance Colliery 'B' Site 3

Coalfield:	Witbank	Seam:	No 5
Site:	Three	Positions:	Roadway
Road widths:	6.2 m	Pillar widths:	6.4 m
Mining height:	1.9 m	Depth:	43.7 m

Mining method: CM and shuttle cars

Cut-out distance 12 m

Roof strata: 0.27 m thick interlaminated sandstone overlain by 1.27 m massive sandstone.

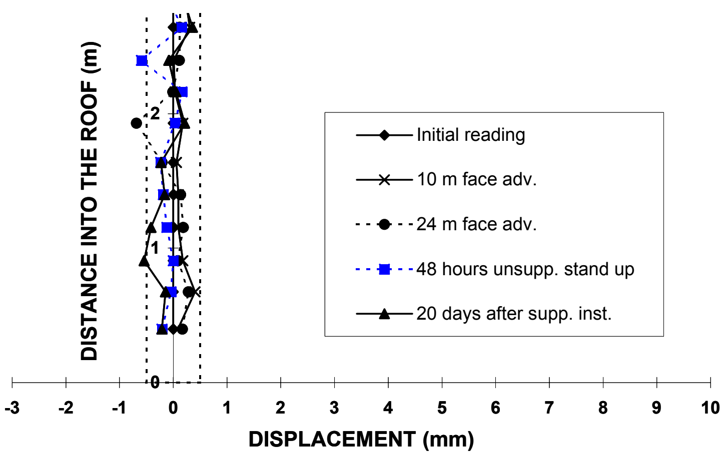
Support: 0.9 m x 20 mm OZ-Bar, full column resin in 25 mm hole.
Support density 0.24 bolt/m^2.

Performance: Both holes showed no dilation during the experiment.

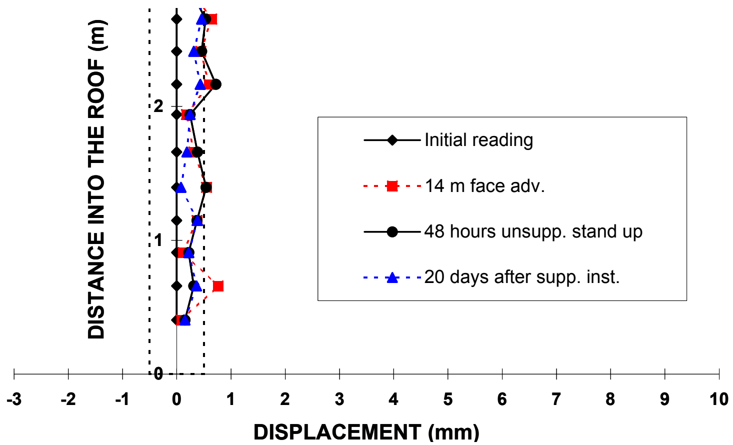
WITBANK COALFIELD
No 2 Seam

COLLIERY "B" - SITE 1
(CUT OUT DISTANCE 24 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.46		0.59	SANDSTONE, grey, fine grained, micaceous bedding planes
1		0.46	SANDSTONE, shaly, with bioturbation
0		1	COAL

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1800
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.15
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	6.5
PILLAR WIDTH (m):	10.5
DEPTH (m):	75
MINING HEIGHT (m):	4.4
SAFETY FACTOR:	1.86

Cutting sequence

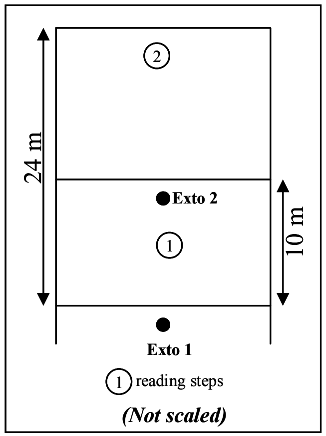
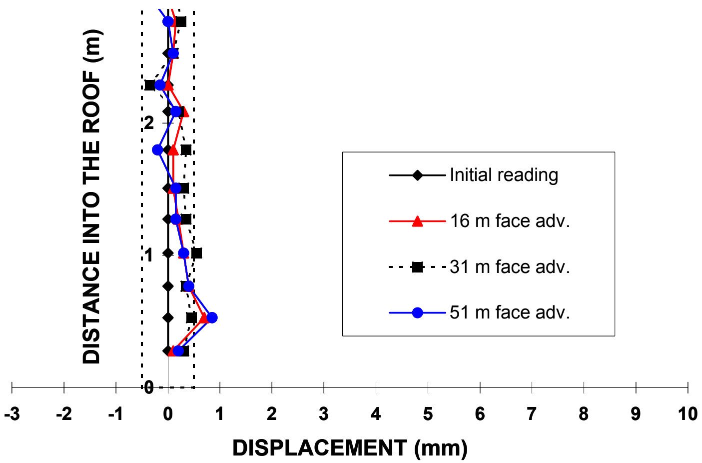


Figure 4-9 Colliery 'B' Site 1

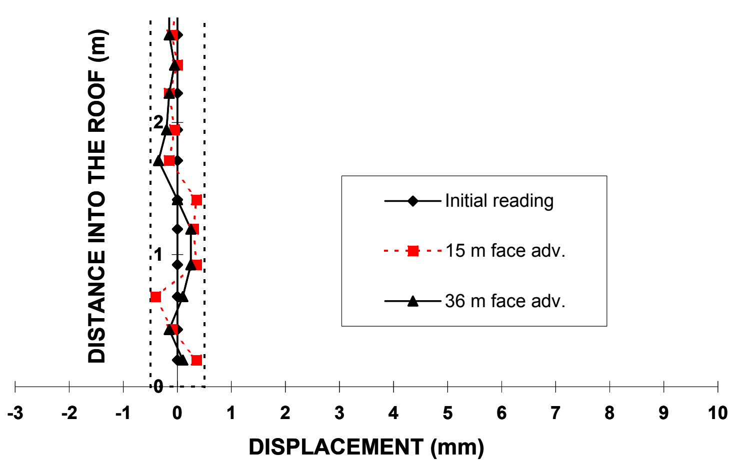
WITBANK COALFIELD
No 2 Seam

COLLIERY "B" - SITE 2
(CUT OUT DISTANCE 31 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1		0.65	SHALE/SILTSTONE, dark greyish-black car. Mic.
0		0.85	COAL

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1800
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.15
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	6.7
PILLAR WIDTH (m):	15.7
DEPTH (m):	44
MINING HEIGHT (m):	4.2
SAFETY FACTOR:	4.85

Cutting sequence

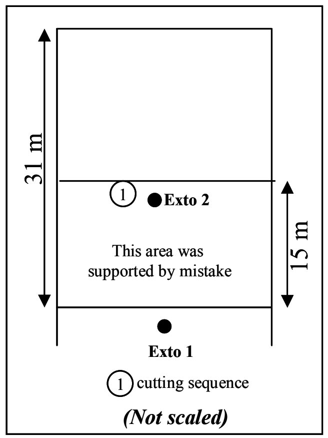
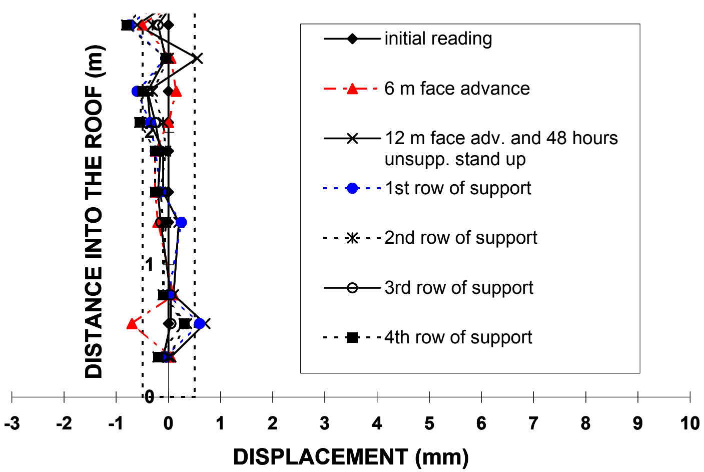
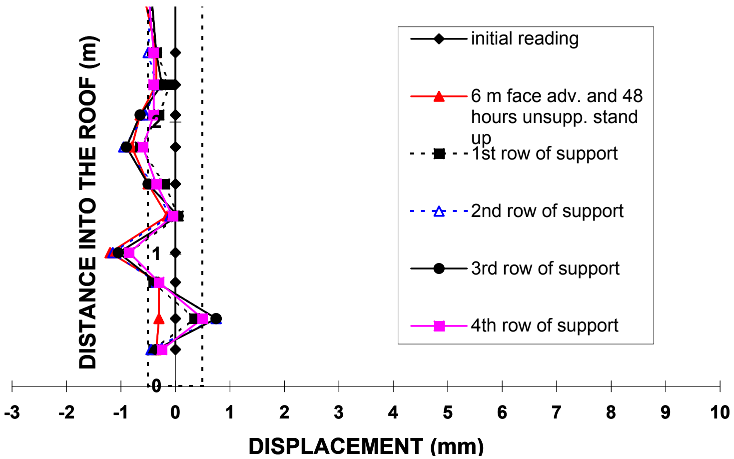


Figure 4-10 Colliery 'B' Site 2

WITBANK COALFIELD
No 5 Seam

COLLIERY "B" - SITE 3
(CUT OUT DISTANCE 12 m)





Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.54		> 9	SHALE / SILTDTONE
0.27		1.27	SANDSTONE, greenish-grey, coarse grained, massive
0		0.27	SANDSTONE, light grey, medium grained, interlaminated

Site performance

BOLT TYPE:	OZ-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	26
BOLT LENGTH (mm):	900
HOLE LENGTH (mm):	900
NUMBER OF BOLTS IN A ROW:	3
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.24
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	2
BORD WIDTH (m):	6.2
PILLAR WIDTH (m):	6.4
DEPTH (m):	43.74
MINING HEIGHT (m):	1.66
SAFETY FACTOR:	3.56

Cutting sequence

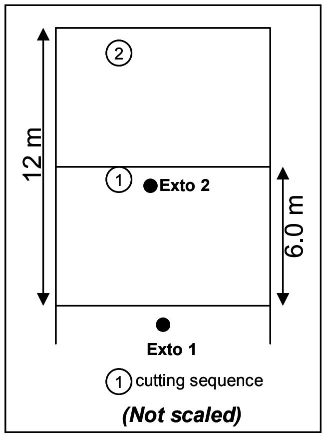


Figure 4-11 Colliery 'B' Site 3

4.6 Colliery 'C'

Three sites in three different sections were monitored at Colliery ‘C’. The colliery is situated in the Highveld Coalfield and mining is currently underway in the No 4 Seam.

4.6.1 Colliery 'C' Site 1

Site 1 was a 17-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the two-right roadway. A road header together with shuttle cars was used in the section. There were no excessive stress indications in the section. The roof and pillar conditions were good as well as the quality of support installation.

The road header experiment sequence was used at this site. After the instrumentation of the first hole was completed at the face, the initial reading was taken. The face was then advanced by 6.0 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. Advancing the face by 6.0 m then completed the cut-out length, and the face was left for 48-hours unsupported. The readings from both holes were taken. A further reading was taken four days after the support installation, when the face was not advanced. The experiment was completed by taking one last reading when the face advance was 64 m.

The results showed that while hole No 1 was stable throughout the experiment, No 2 hole showed 1.5 mm dilation, which took place approximately 1.0 m into the roof at the coal/mudstone laminae and sandstone contact. The results obtained from both holes during the first experiment in Colliery 'C', Site 1, are presented in Figure 4-12. The summary of site performance in this site is given in Table 4-9.

Table 4-9 Site performance Colliery 'C' Site 1

Coalfield:	Highveld	Seam:	No 4
Site:	One	Positions:	Roadway
Road widths:	7.0 m	Pillar widths:	10 m
Mining height:	4.3 m	Depth:	54.2 m

Mining method: Road header and shuttle cars
Cut-out distance 12 m

Roof strata: 0.94 m thick coal/mudstone laminae overlain by 2.25 m thick shale/sandstone

Support: 1.8 m x 16 mm Re-bar, partial column resin in 22 mm hole. Support density 0.14 bolt/m^2.

Performance: While No 1 hole was stable throughout the experiment. No 2 hole showed 1.5 mm dilation, at 1.0 m into the roof at the coal/mudstone laminae and sandstone contact. This movement took place after 6 m face advance (12 m full length completed) and 48-hours stand-up time. No further dilation was recorded even after 64 m face advance.

4.6.2 Colliery 'C' Site 2

Site 1 was an 11-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the centre roadway. A road header together with shuttle cars was used in the section. There were no excessive stress indications in the section. A major problem observed in support installation was the overdrilling of boltholes.

The road header experiment sequence was used at this site. After the instrumentation of the first hole was completed, the initial reading was taken. The face was then advanced by 6.0 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. Following readings were taken once the face was advanced by 6.0 m and after a 48-hour unsupported stand-up time period. Further readings were then taken after the area was supported at 60 m and the face was advanced by 200 m.

The results obtained from both holes during the first experiment in Colliery 'C', Site 2, are presented in Figure 4-13. The summary of site performance in this site is given in Table 4-10. Figure 4-13 shows that the total dilation in No 1 hole was 1.0 mm at the skin anchor. Initial 0.9 mm dilation, 0.5 m into the roof, took place after the face was advanced by 6.0 m. A further 0.1 mm movement, 1.0 m into the roof, was recorded after the completion of 12 m face advance and 48-hours stand-up time. After the support installation was completed, the results from No 1 hole indicated that there had been an upwards movement into the roof. Initially, this behaviour was thought to be due to roof bolting, which took place after the unsupported stand-up time. However, the roof bolting should affect the roof skin first before influencing movement further into the roof. As can be seen from the figure, the roof skin deflected 1.0 mm during the experiment. Therefore, it was decided that this movement was an anomaly and the reading may be discarded.

The total dilation in No 2 hole was 1.0 mm, at 1.0 m into the roof, at the coal/mudstone and shale/sandstone interface. This movement took place after 6.0 m face advance (12 m full length completed) and 48-hour stand-up time. No further dilation was recorded in hole No 2.

Table 4-10 Site performance Colliery 'C' Site 2

Coalfield:	Highveld	Seam:	No 4
Site:	Two	Positions:	Roadway
Road widths:	6.7 m	Pillar widths:	12 m
Mining height:	4.7 m	Depth:	70 m
Mining method:	Road header and shuttle cars		
Cut-out distance	12 m		
Roof strata:	0.8 m mudstone/coal/sandstone laminae overlain by 2.3 m thick shale/sandstone		
Support:	1.8 m x 16 mm Re-bar, partial column resin in 22 mm hole. Support density 0.15 bolt/m^2.		
Performance:	Both holes showed 1.0 mm dilation after the 12 m cut-out length and 48-hours unsupported stand-up time completed. This movement took place 0.5 m and 1.0 m into the roof respectively in No 1 and No 2 holes. Both holes stabilised and no further dilation was recorded even after 200 m face advance.		

4.6.3 Colliery 'C' Site 3

Site 1 was an 11-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the centre roadway. A road header together with shuttle cars was used in the section to mine the No 4 Seam in the Highveld Coalfield. There were no excessive stress indicators in the section. Under drilling of bolt holes was observed, however, in general support installation and performance were good.

The road header experiment sequence was used in the experiment to monitor the 18 m cut-out distance. After the installation of the first hole was completed at the face, the initial reading was taken. The face was then advanced by 6.0 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. Further readings were taken once the face was advanced by 12 m and after the 48-hours unsupported stand-up time period. Further readings were then taken after the area was supported and while the face was at 34 m and at 143 m.

The results obtained from both holes during the first experiment in Colliery 'C', Site 3 are presented in Figure 4-14. The summary of site performance in this site is given in Table 4-11. Figure 4-14 indicates that both holes, No 1 and No 2 hole, were stable during the experiment. Although approximately 0.5 mm dilation was recorded in No 2 hole, it was within the accuracy of the system.

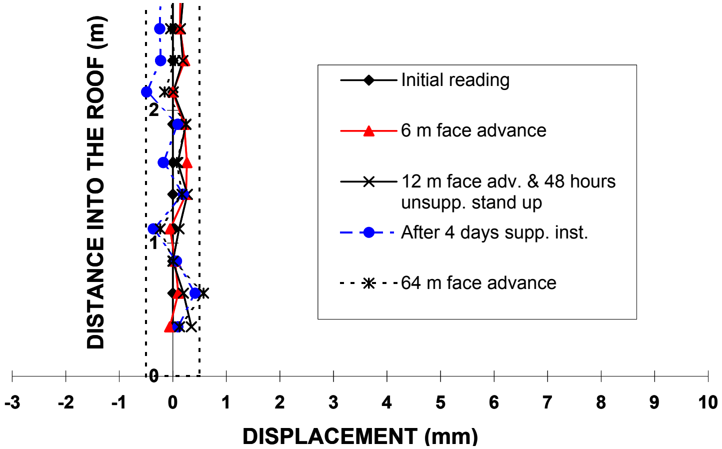
Table 4-11 Site performance Colliery 'C' Site 3

Coalfield:	Highveld	Seam:	No 4
Site:	Three	Positions:	Roadway
Road widths:	6.1 m	Pillar widths:	9.0 m
Mining height:	4.3 m	Depth:	61 m
Mining method:	Road header and shuttle cars		
Cut-out distance	18 m		
Roof strata:	0.83 m shale/coal/sandstone laminae overlain by 2 m thick shale/sandstone		
Support:	1.8 m x 16 mm Re-bar, partial column resin in 24 mm hole. Support density 0.16 bolt/m^2.		
Performance:	While No 1 hole showed no dilation throughout the experiment, No 2 hole showed 0.5 mm dilation, approximately 1.1 m into the roof, which was the within the accuracy of the system.		

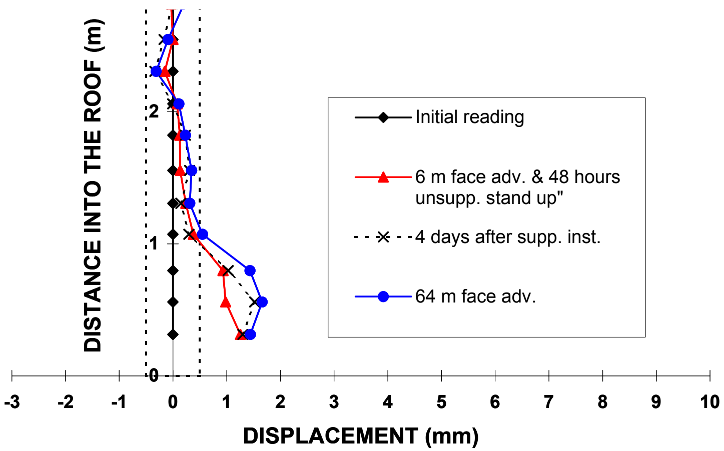
HIGHVELD COALFIELD
No 4 Seam

COLLIERY "C" - SITE 1
(CUT OUT DISTANCE 12 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
0.94		>2.25	SHALE/SANDSTONE, light grey, interbedded
0.71		0.23	SHALE, dark grey, silty
0.56		0.15	COAL
0.39		0.17	MUDSTONE
0.24		0.15	COAL
0		0.24	MUDSTONE

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1750
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.14
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	2
BORD WIDTH (m):	7.03
PILLAR WIDTH (m):	10
DEPTH (m):	54.19
MINING HEIGHT (m):	4.3
SAFETY FACTOR:	2.33

Cutting sequence

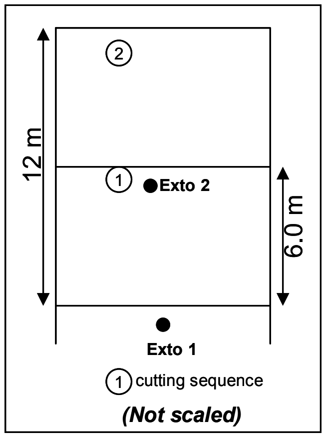
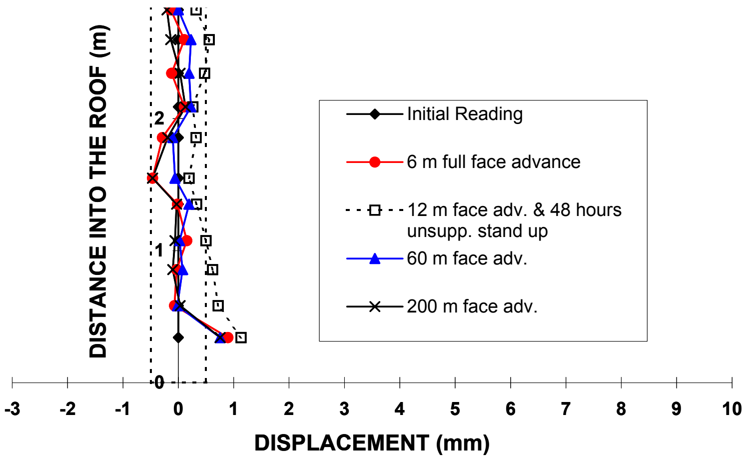


Figure 4-12 Colliery 'C' Site 1

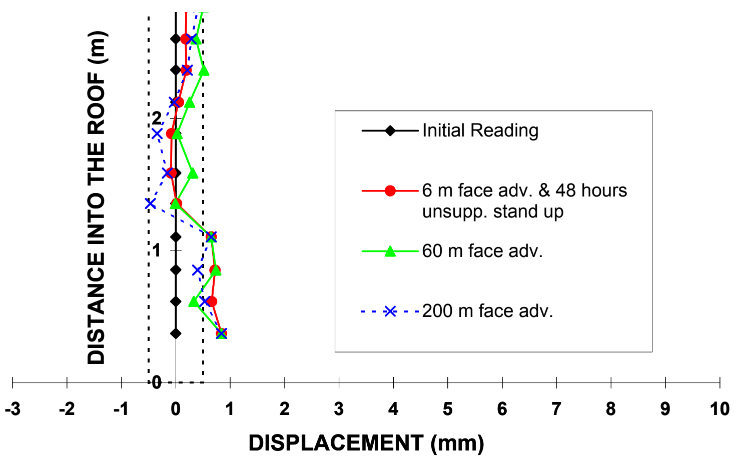
HIGHVELD COALFIELD
No 4 Seam

COLLIERY "C" - SITE 2
(CUT OUT DISTANCE 12 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
0.83		2.3	SHALE/SANDSTONE, laminated, grey
0.79		0.04	COAL
0.59		0.2	SANDSTONE, light green fine grained
0.32		0.27	COAL
0		0.32	MUDSTONE

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	21
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1800
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.15
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	FAST
NUMBER OF RESIN CAPSULES:	1
BORD WIDTH (m):	6.7
PILLAR WIDTH (m):	12
DEPTH (m):	70
MINING HEIGHT (m):	4.7
SAFETY FACTOR:	2.16

Cutting sequence

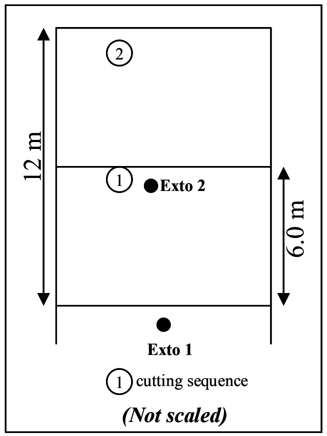
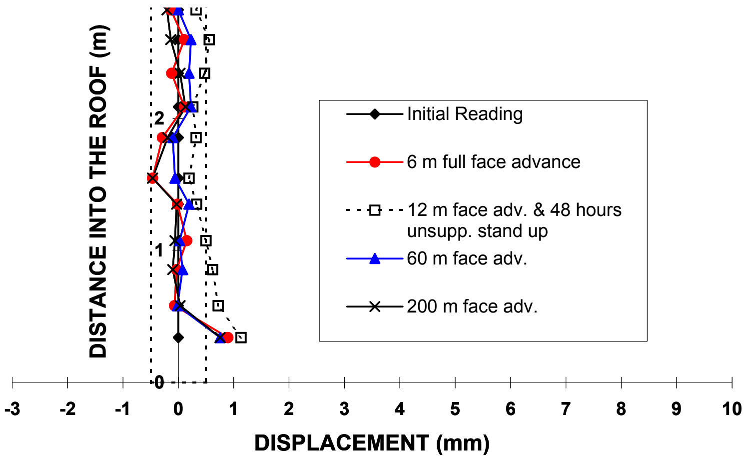


Figure 4-13 Colliery 'C' Site 2

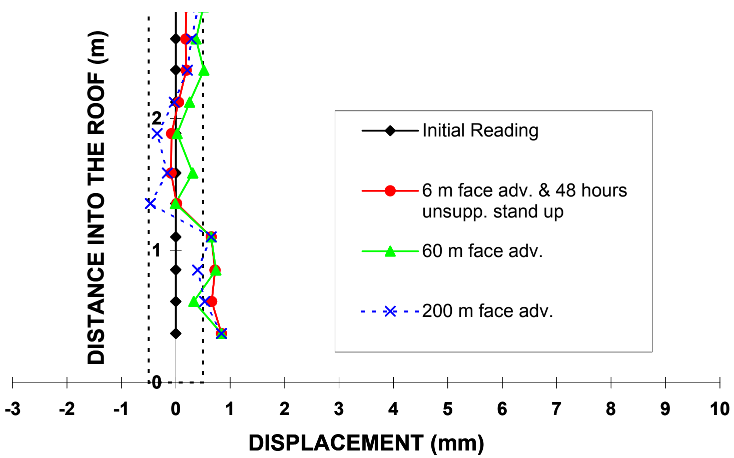
HIGHVELD COALFIELD
No 4 Seam

COLLIERY "C" - SITE 2
(CUT OUT DISTANCE 12 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
0.83		2.3	SHALE/SANDSTONE, laminated, grey
0.79		0.04	COAL
0.59		0.2	SANDSTONE, light green fine grained
0.32		0.27	COAL
0		0.32	MUDSTONE

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	21
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1800
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.15
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	FAST
NUMBER OF RESIN CAPSULES:	1
BORD WIDTH (m):	6.7
PILLAR WIDTH (m):	12
DEPTH (m):	70
MINING HEIGHT (m):	4.7
SAFETY FACTOR:	2.16

Cutting sequence

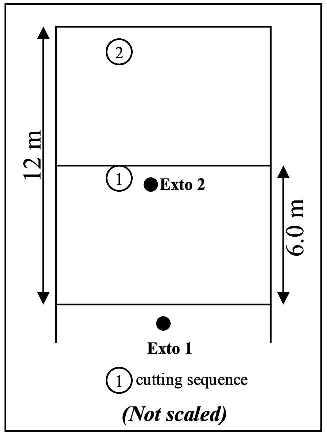


Figure 4-14 Colliery 'C' Site 3

4.7 Colliery 'D'

One site was monitored at Colliery 'D'. The colliery is situated in the Witbank Coalfield and mining the No 2A Seam.

4.7.1 Colliery 'D' Site 1

Site 1 was a seven-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the centre roadway. A remote controlled road header together with shuttle cars was used in the section. While there were no excessive stress indicators in the section, many geological discontinuities were present in the roof and pillars. However, the pillar, roof and underground dimension control were good.

The road header experiment sequence was used in the experiment. In order to monitor the roof displacement profile, three sonic probe-monitoring holes were used. These holes were situated at the face, 6.0 m and 8.0 m into the advancing section. After the installation of the first hole was completed, the initial reading was taken. The face was then advanced by 6.0 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. The third hole was then drilled after the face was advanced by 2.0 m, and readings were taken from all three holes. The final cut-out distance was reached after the face was advanced by a further 8.0 m. Because of the amount of time potentially spent under an unsupported roof during the drilling and instrumentation of holes, it was decided to support the roof immediately after the 16 m cut-out distance was completed. An attempt was made to monitor the effect of roof bolting. Therefore, readings were taken after installation of each row of support up to a point where the last row of support passed the third monitoring hole. The last readings were taken from all three holes when the face was advanced by 60 m. The results obtained from both holes during the first experiment in Colliery 'D', Site 1 are presented in Figure 4-15. The summary of site performance in this site is given in Table 4-12.

Figure 4-15 shows that while hole No 1 showed the least dilation of 2.0 mm, 2.5 mm dilation was recorded in both No 2 and No 3 holes. In all three holes the dilations took place approximately 1.5 m into the roof, and before the support was installed. Also, holes No 1 and No 3 showed a further 0.5 mm dilation after the face advanced by 60 m. It was noted that this deformation could be due to stress changes caused by development of an intersection 0.5 m away from the third hole.

As mentioned earlier, one of the aims of this experiment was to monitor the effect of roof bolting and/or tensioning in the roof. While it was not possible to observe the effect of roof bolting in the No 1 or No 3 holes, because the initial displacements took place at the bolted horizon as a whole beam, Hole No 2 presented an ideal site to attempt to determine the effect of the installation of the roof bolts. The installation of the bolts appeared to have little if any effect on the bed separation already evident within the bolted interval as can be seen in Figure 4-15. However, as this information came from only one monitoring hole, it cannot be concluded that this is typical and that the installation of pre-tensioned roof bolts has no remedial effects on pre-existing openings within the bolt horizon.

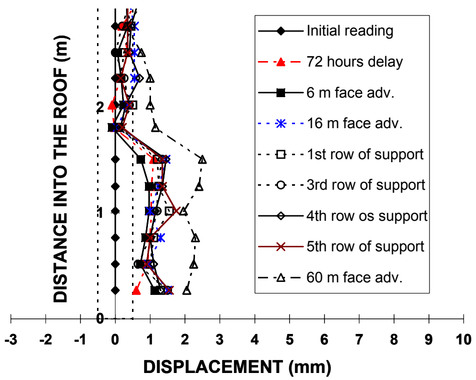
Table 4-12 Site performance Colliery 'D' Site 1

Coalfield:	Witbank	Seam:	No 2A
Site:	One	Positions:	Roadway
Road widths:	6.6 m	Pillar widths:	8.2 m
Mining height:	4.2 m	Depth:	53 m
Mining method:	Road header and shuttle cars		
Cut-out distance	16 m		
Roof strata:	1.83 thick laminated coal/shale		
Support:	1.5 m x 16 mm Re-bar, partial column resin in 22 mm hole. Support density 0.23 bolt/m^2.		
Performance:	While No 1 and No 3 holes showed 2.5 mm, No 2 hole showed 2.0 mm displacement.		

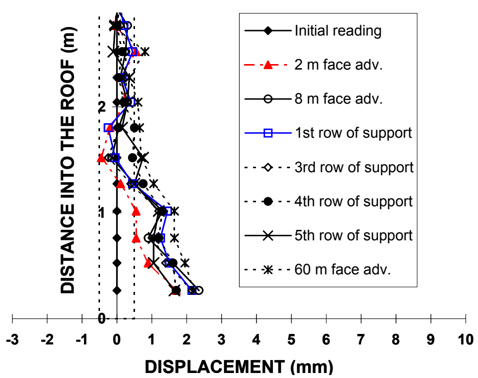
WITBANK COALFIELD
No 2A Seam

COLLIERY "D" – SITE 1
(CUT OUT DISTANCE 16 m)

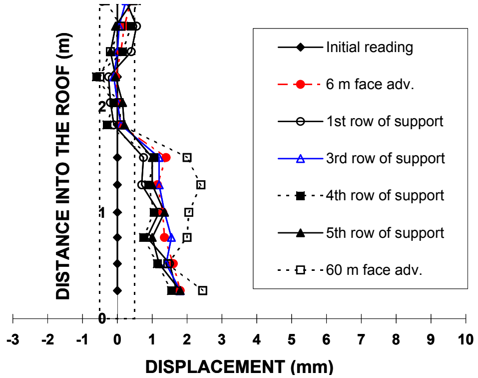
HOLE 1



HOLE 2



HOLE 3



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.83		0.65	SHALE, carbonaceous
0		1.83	COAL, SHALY COAL, bright bands

Site performance

BOLT TYPE:	Resin Point Anchor
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	22
BOLT LENGTH (mm):	1500
HOLE LENGTH (mm):	1500
NUMBER OF BOLTS IN A ROW:	3
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.23
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	FAST
NUMBER OF RESIN CAPSULES:	2
BORD WIDTH (m):	6.6
PILLAR WIDTH (m):	8.2
DEPTH (m):	53
MINING HEIGHT (m):	4.2
SAFETY FACTOR:	2.03

Cutting sequence

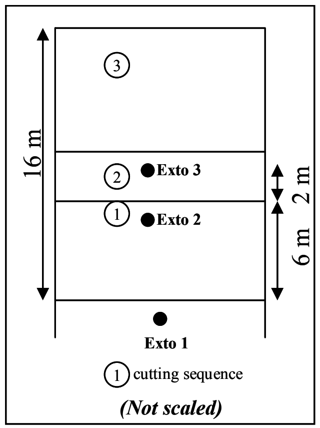


Figure 4-15   Colliery 'D' Site 1

4.8 Colliery 'E'

One site was monitored at Colliery 'E'. The colliery is situated in the Highveld Coalfield and mining is being carried out on No 4 Lower Seam.

4.8.1 Colliery 'E' Site 1

Site 1 was a seven-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the centre-roadway. A remote controlled CM together with shuttle cars was used in the section. There were no excessive horizontal stress indicators in the section. The installation and performance of support was excellent.

The road header experiment sequence was used at this site. After the installation of the first hole was completed at the face, the initial reading was taken. The face was then advanced by 12 m in full bord width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. The final cut-out distance was reached after the face was advanced by a further 12 m, and the face was left unsupported for 40 hours. Further readings were taken from both holes. The last reading was taken after the face was advanced by a further 13 m. The results obtained from both holes during the experiment in Colliery 'E' are presented in Figure 4-16. The summary of site performance in this site is given in Table 4-13.

No 1 and No 2 holes showed dilation of 1.0 mm and 5.0 mm respectively, in both cases extending 0.3 m into the roof. In both monitoring holes, the displacements took place at the coal/sandstone/siltstone contact after the completion of 24 m cut-out length and 48-hours stand-up time.

Table 4-13 Site performance Colliery 'E' Site 1

Coalfield:	Highveld	Seam:	4 Lower
Site:	One	Positions:	Roadway
Road widths:	7.0 m	Pillar widths:	25 m
Mining height:	3.8 m	Depth:	178 m
Mining method:	CM and shuttle cars		
Cut-out distance	24 m		
Roof strata:	0.35 m thick coal/sandstone laminae overlain by 0.65 m thick siltstone, and thick gritstone		

Support: 1.5 m x 20 mm Re-bar, partial column resin in 25 mm hole.
Support density 0.24 bolt/m^2.

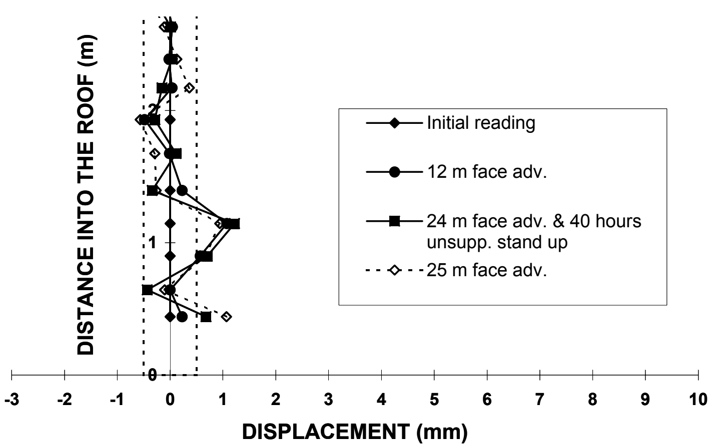
Performance: A maximum of 1.0 mm was recorded in No 1 hole.

No 2 showed 5.0 mm displacement, which was the largest of all the monitoring sites. Displacements in both holes took place 0.3 m into the roof at the coal/sandstone and siltstone interface.

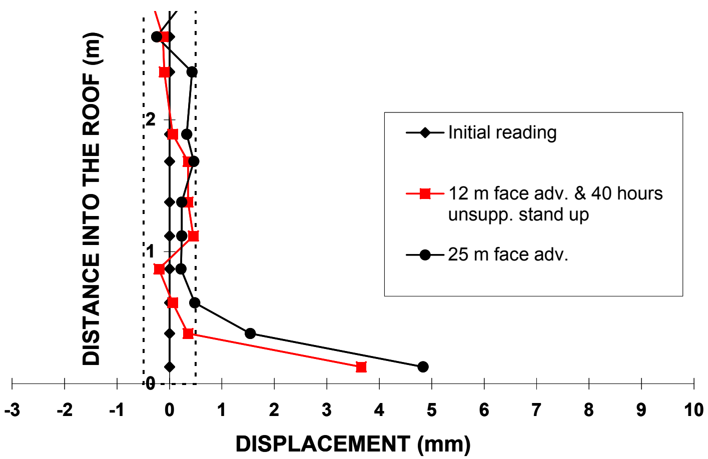
HIGHVELD COALFIELD
No 4 Lower

COLLIERY "E" - SITE 1
(CUT OUT DISTANCE 24 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1		>3.9	GRITSTONE
0.35		0.65	SILTSTONE
0.27		0.08	COAL
0.15		0.12	SANDSTONE
0		0.15	COAL, shale

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1500
HOLE LENGTH (mm):	1500
NUMBER OF BOLTS IN A ROW:	5
DISTANCE BETWEEN THE ROWS (m):	3
BOLT/m^2	0.24
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	2
BORD WIDTH (m):	7
PILLAR WIDTH (m):	25
DEPTH (m):	178
MINING HEIGHT (m):	3.8
SAFETY FACTOR:	1.91

Cutting sequence

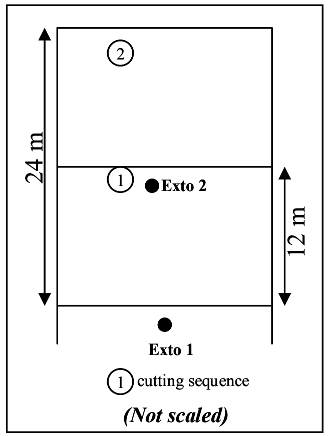


Figure 4-16 Colliery 'E' Site 1

4.9 Colliery 'F'

One site was monitored at Colliery 'F'. The colliery is situated in the Vereeniging Coalfield and mining was conducted on the No 2B Seam.

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4.9.1 Colliery 'F' Site 1

Site 1 was a four-roadway, primary bord and pillar production section. The sonic probe monitoring holes were drilled in the one-right roadway. An onboard CM together with shuttle cars was used in the section. The installation and performance of support was also excellent.

The CM experiment sequence was used at this site. After the installation of the first hole was completed at the face, the initial reading was taken. The face was then advanced by 5.0 m with a 3.5 m drum width and the second hole drilled and instrumented with sonic probe anchors. Readings were taken from both holes. The second lift was then mined up to 5.2 m bord width, and readings were taken again from both holes. The experiment was completed by cutting a further 5.0 m. After this stage of the experiment, the area was supported due to a slip running across the roadway next to the second monitoring hole. A further reading was taken when the face advance was 60 m. The results obtained from both holes during the experiment in Colliery 'F' are presented in Figure 4-17. The summary of site performance is given in Table 4-14.

Similar to Colliery 'A' Site 1, Test 1 and 2, the water aquifer limited the sonic probe hole lengths. Therefore, approximately 3.0 m long sonic probe holes were drilled and monitored. The results showed that both holes were stable throughout the experiment.

Table 4-14 Site performance Colliery 'F' Site 1

Coalfield:	Vereeniging	Seam:	No 2 B
Site:	One	Positions:	Roadway
Road widths:	5.2 m	Pillar widths:	44.1 m
Mining height:	2.6 m	Depth:	44.1 m
Mining method:	CM and shuttle cars		
Cut-out distance	10 m		
Roof strata:	1.41 m thick coal/shale layers overlain by 1.1 m thick coal/shale		

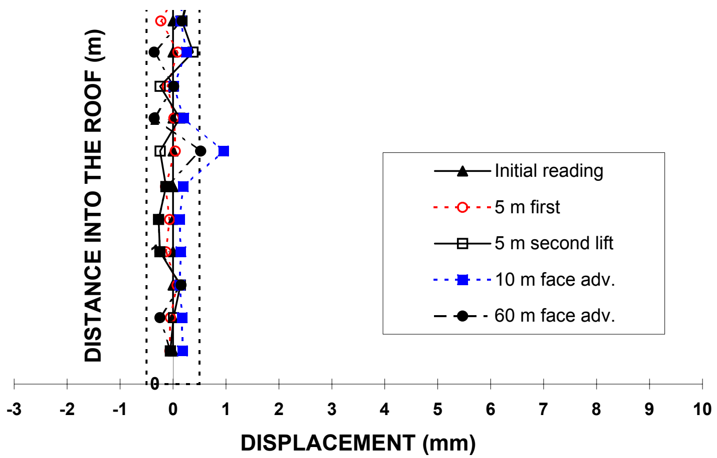
Support: 1.8 m x 20 mm Re-bar, partial column resin in 25 mm hole.
Support density 0.77 bolt/m^2.

Performance: Both holes were stable throughout the experiment, and no displacements were recorded in either hole.

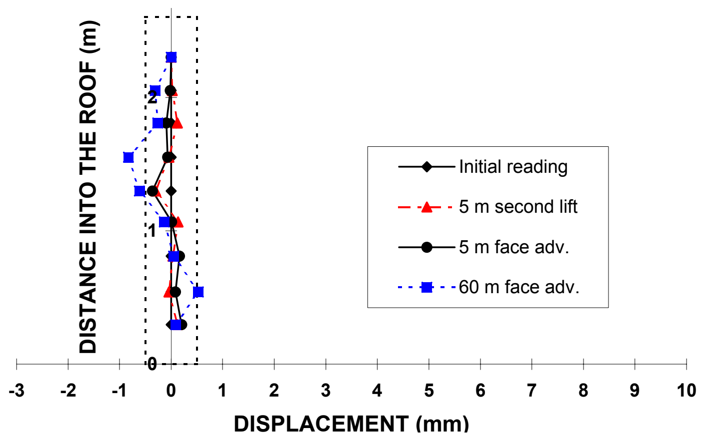
VEREENIGING COALFIELD
No 2B Seam

COLLIERY “F” - SITE 1
(CUT OUT DISTANCE 10 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
1.41		1.1	COAL - SHALE
1.28		0.13	DULL COAL
0.89		0.39	COAL - SHALE
0.54		0.35	DULL COAL
0		0.54	COAL - SHALE

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	20
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1830
NUMBER OF BOLTS IN A ROW:	6
DISTANCE BETWEEN THE ROWS (m):	1.5
BOLT/m^2	0.77
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	SLOW & FAST
NUMBER OF RESIN CAPSULES:	3
BORD WIDTH (m):	5.2
PILLAR WIDTH (m):	28.8
DEPTH (m):	44.1
MINING HEIGHT (m):	2.6
SAFETY FACTOR:	12.31

Cutting sequence

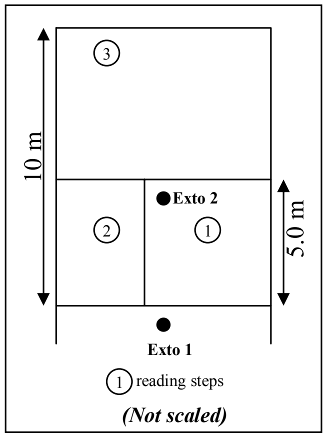


Figure 4-17 Colliery 'F' Site 1

4.10 Analysis of underground monitoring results

In the initial phase of the underground experiments, it was intended that in all the sites the cut-out distance would be extended to a point where the roof would fail and the critical deformations could be determined. However, discussions with mining personnel highlighted the fact that this was not possible, because all the experiments were to take place in normal production sections where roof failure could not be tolerated. Therefore, in 9 of 13 sites, only the standard cut-out distances of individual sections were monitored. In the remaining four sites (Colliery 'B' Sites 1 and 2, Colliery 'C' Site 3, and Colliery 'E' Site 1) extensive cut-out distances of 18 m to 31 m were monitored. The results, however, showed that the roof deformations were as small as for the significantly shorter cut-out distances. For example, in Colliery 'B' Site 2, the 31 m cut-out distance showed no dilation in the roof. This of course could be due to installing the sonic probe holes late or due to the very competent coal roof.

The results obtained from both the No 1 and No 2 sonic probe monitoring holes from all the monitoring sites are given in Table 4-15 and Table 4-16. The results did not show any obvious correlation between the maximum dilation and other variables. In fact this could be expected as there are many parameters, which can affect the roof performance. These include: the support density, roof lithology, bord width, stress changes in the roof as well as cut-out distance. While these parameters may affect the roof on an individual basis, it is shown that generally more than one of the above factors will have an influence on roof behaviour. Therefore, in order to obtain repeatability of experiment results, more than one experiment is required from each site, which was impossible to achieve in this study.

The relationships between the maximum dilation and the support density, roof lithology, bord width and cut-out distance are shown from Figure 4-18 to Figure 4-21. It can be seen from these figures that there is no obvious correlation between roof dilation and the other variables. Consequently, the data does not show meaningful relationships between these parameters. There is thus no indication of the dependence of roof stability on cut-out distance in the range of cut-out distances that was investigated.

There was, however, one significant correlation observed, and this was the relationship between the position where separation occurred and the thicknesses of lithological units. This is shown in Figure 4-22. The immediate roof thicknesses were obtained from the borehole logs and the position of the separation from the sonic probe monitoring observations. This figure confirms that the position of dilation or separation in the roof agreed with the position of likely partings or change in rock type in the roof. This finding indicates that as a fundamental analytical tool, beam theory may be used to estimate the expected roof behaviour.

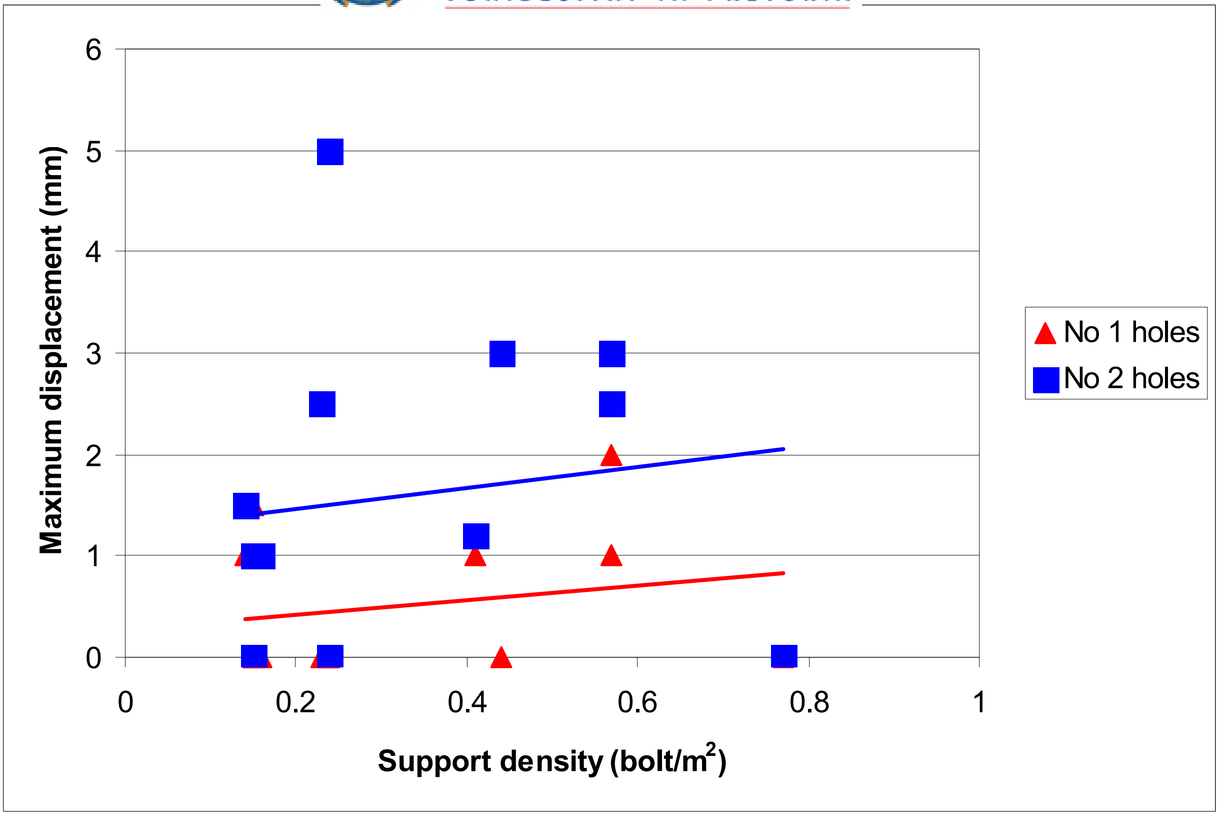


Figure 4-18 The relationship between the support density and total displacement

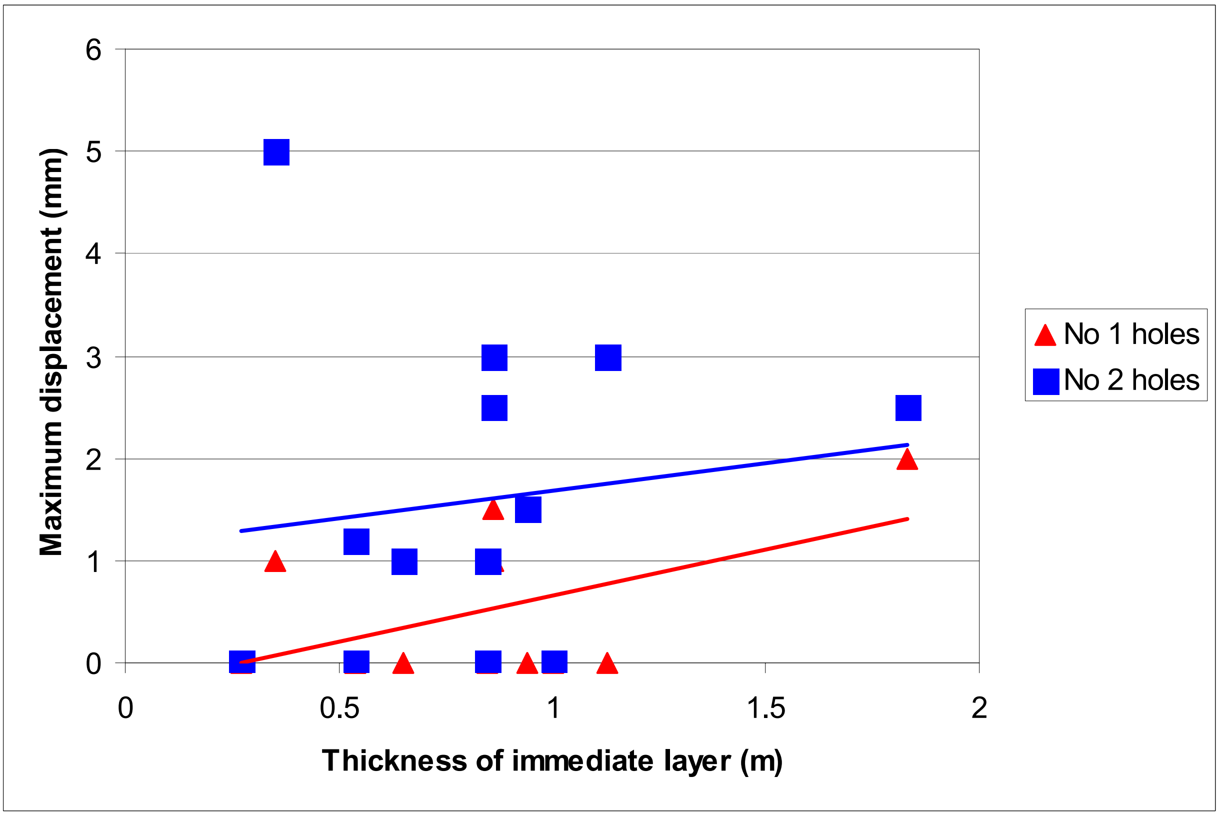


Figure 4-19 The relationship between the thickness of the immediate layer and total displacement

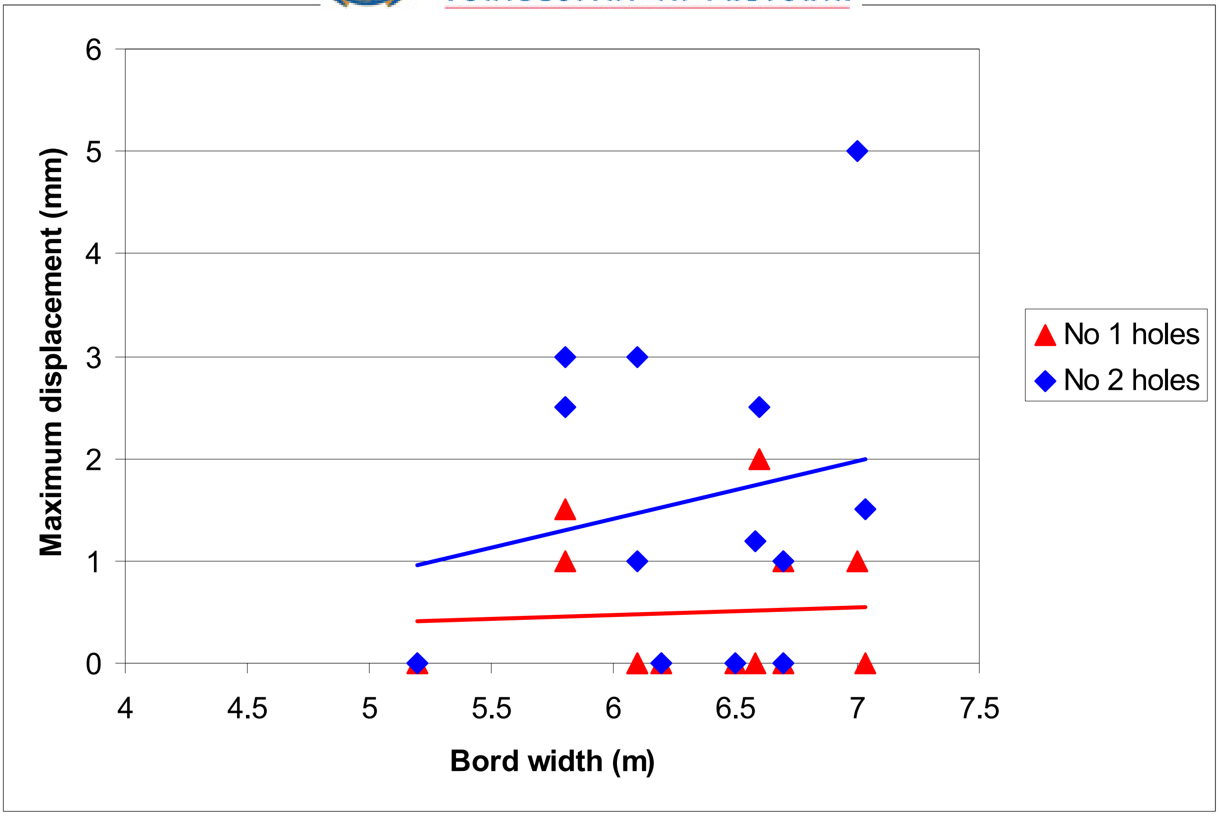


Figure 4-20 The relationship between the bord width and total displacement

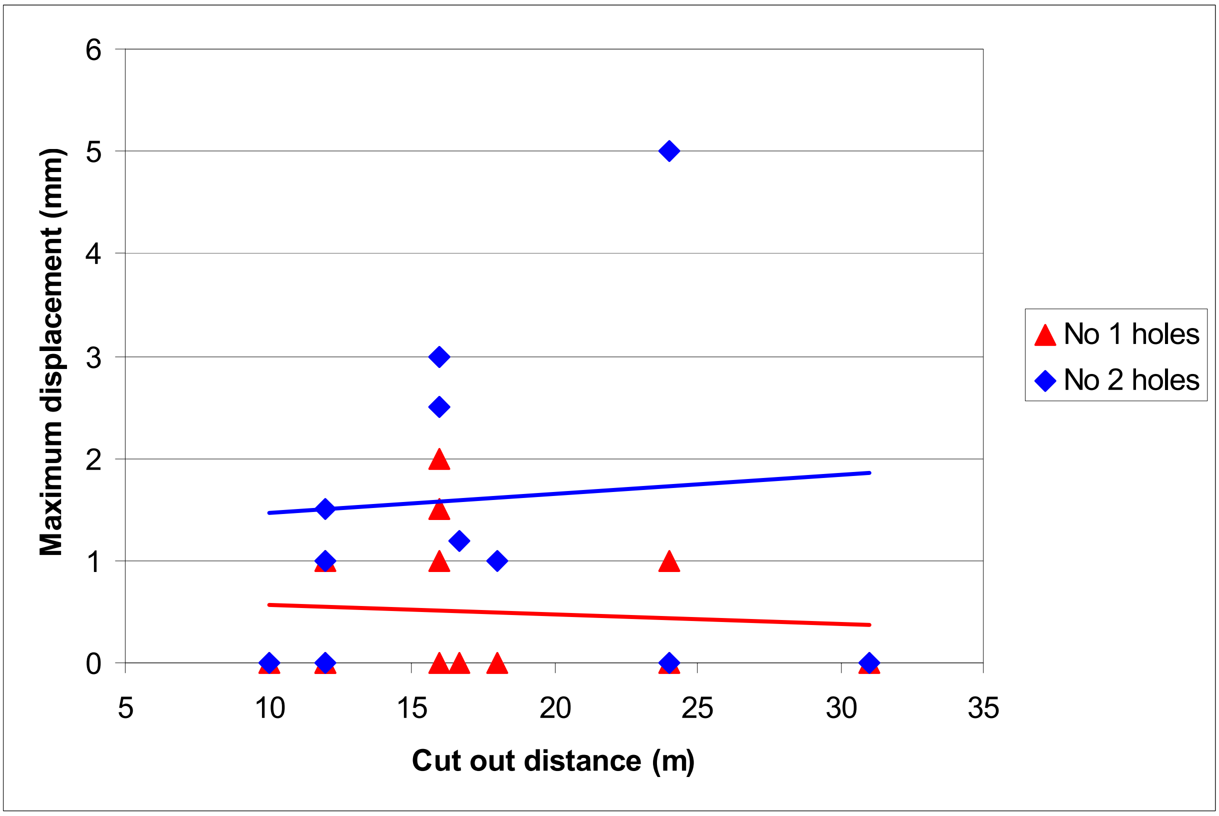


Figure 4-21 The relationship between the cut-out distance and total displacement

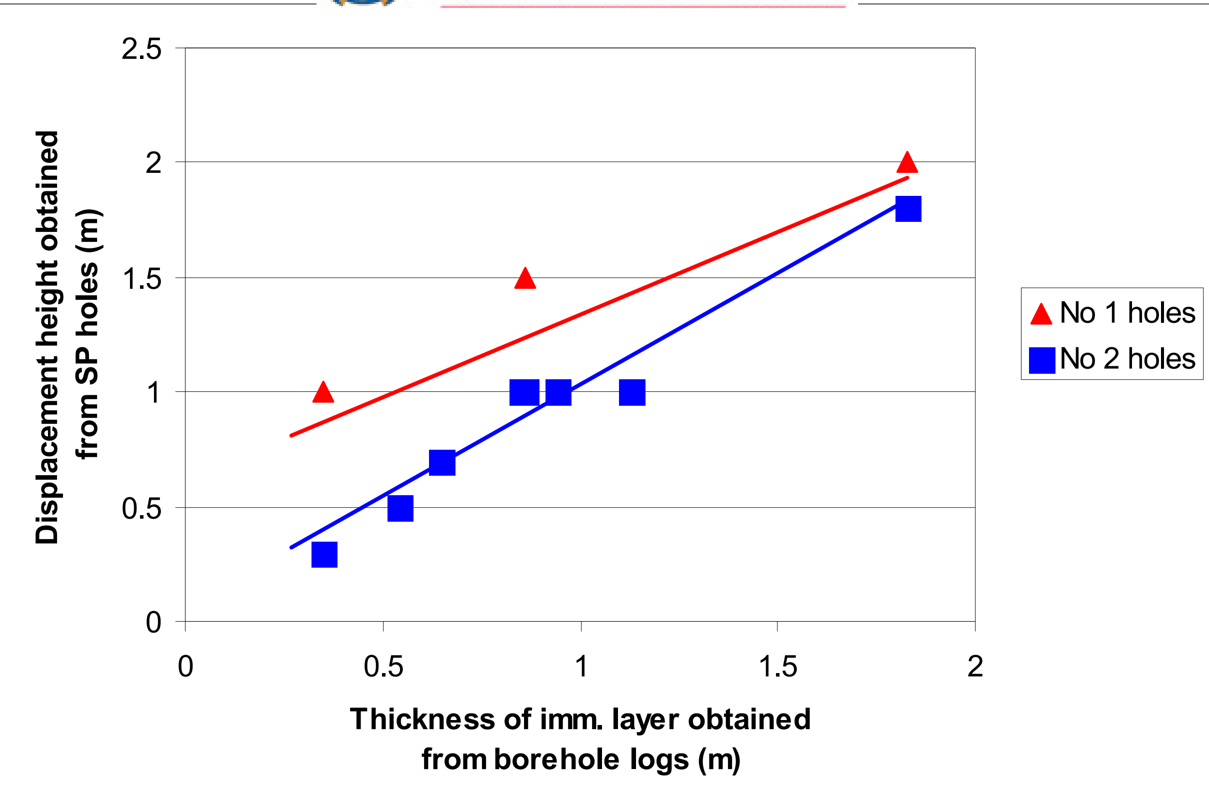


Figure 4-22 The relationship between the thickness of the immediate layer obtained from the borehole logs and height of the displacement obtained from underground sites where some degree of dilation was recorded

The time it took for deformation to occur was similar in all the sections. In all the sections the greater portion of the displacements took place during and immediately after mining took place at the face. On average 69 per cent of the maximum dilation measured took place during the face advance, and a further 11 per cent during the following 48-hours unsupported stand-up time. This figure is based on data from both No 1 and No 2 sonic probe monitoring holes where some degree of dilation was recorded in the roof.

This indicates that on average, 69 per cent of the deformation takes place immediately after the mining takes place at the face, before the support has been installed. If the support installation is delayed by 48-hours, this percentage rises to 80 per cent. Although the percentage increase after a 48-hour unsupported stand-up time is not significant, it may change the roof behaviour from elastic to plastic, due to the weathering and development of micro fractures.

Two attempts were made to identify the effect of roof bolting and tensioning of roof bolts. This was done in Colliery 'B', Site 3 and Colliery 'D', Site 1. However since no roof movement was recorded in holes No 1 and 2, in Colliery 'B', and although some initial displacement took place above the bolt horizon in holes No 1 and No 3 in Colliery 'D', no displacement was recorded within the bolt horizon. Hole No 2 in Colliery 'D' presented an ideal site to attempt to determine the effect of the installation of the roof bolts. The installation of the bolts appeared to have little if any effect on the bed separation already evident within the bolted interval as can be seen in Figure 4-15. However, as this information came from only one monitoring hole, it cannot be concluded that this is typical and that the installation of pre-tensioned roof bolts has no remedial effects on pre-existing openings within the bolted interval.

Although the effect of the installation of pre-tensioned roof bolts was specifically monitored at only one site, in general in the monitoring holes, where the displacements occurred within the roof bolt horizon, there was no evidence that the installation of the bolts partially closed pre-existing openings within the roof strata. Whether the roof stability would be improved by reversing some of the relaxation that takes place prior to the installation of the roof bolts is open to debate. This requires a further investigation.

Based on the good correlation that was found between the thicknesses of the nether roof units and the positions where dilation (or separation) occurred, it was concluded that the roof displayed characteristics of discrete plate behaviour. This behaviour was investigated further by comparing the measured roof deflections with that which is predicted by a conventional gravity loaded beam theory. As the length of the roadways exceeded twice their width in all cases, the valid simplification of using beam formulae was introduced.

The deflection of the roof was predicted using the following equation:

$$\eta = \frac{\gamma B^4}{32Et^2} \qquad [4\text{-}5]$$

where η = Maximum deflection (m),
γ = unit load *(ρg)*,
E = Modulus of Elasticity,
t = thickness of layer and
B = bord width.

Where appropriate, allowance was made for additional load resulting from softer layers overlying stiffer ones. Where laminated layers consisted of materials of different stiffnesses, a weighted average stiffness was used in the calculations.

The comparison between calculated and measured deflections is shown in Figure 4-23. Although the correlation coefficient of the linear line is low (53 per cent), the good correlation is immediately apparent, as is the fact that the magnitudes of the displacements are in the same range. It can therefore be concluded that the gravity loaded beam analogy is valid for predicting roof behaviour in the study area. The correlation coefficient is affected by mainly two cases where very little deflection was predicted. This may have been caused by incorrect loading assumptions.

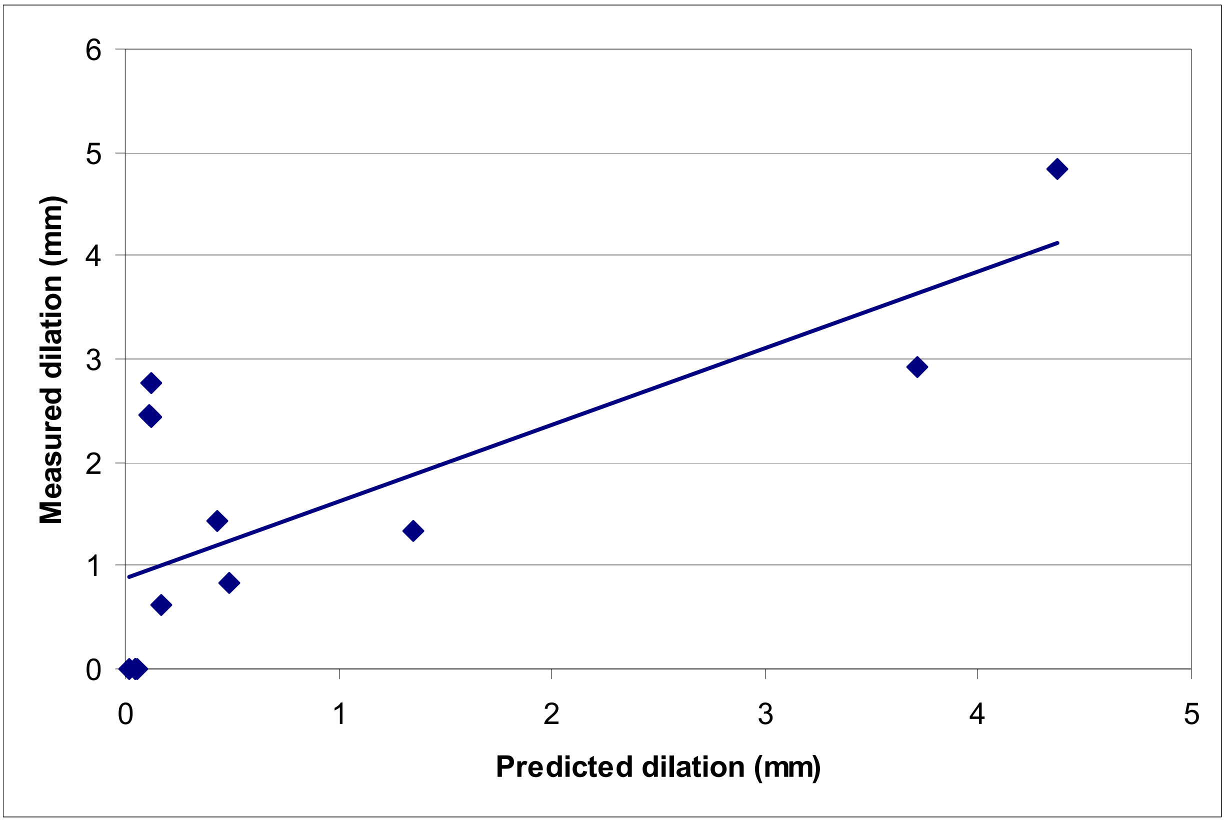


Figure 4-23 Relationship between measured and predicted dilation

The simplification of using beam theory rests on the assumption that once the length of a plate exceeds twice its width, further increases in length will not result in meaningful additional deflection. The good correlation between the predicted beam deflections and the measured plate deflections confirm that further increases in length (i.e. cut-out distance) will not result in meaningful additional roof deflection. This agreed with the underground observations, where it was indicated that stability was reached very soon, i.e. predominantly during mining of the limited cut-out distance.

Table 4-15 Summary results obtained from No 1 sonic probe monitoring holes

Colliery	Site	Max. height of dilation (m)	Support density (bolt/m^2)	Bord width (m)	Cut-out distance (m)	Total displ. (mm)	Immediate roof lithology	Thickness of imm. roof (m)
A	1, Test 1	0.8	0.57	5.8	16	1.0	Grit/grit coal laminae	0.86
A	1, Test 2	1.5	0.57	5.8	16	1.5	Grit/grit coal laminae	0.86
A	2	0	0.41	6.58	16.7	0.2	Sandstone	0.54
A	3	0	0.44	6.1	16	0.2	Sandstone and grit	1.13
B	1	0	0.15	6.5	24	0.2	Coal	1
B	2	0	0.15	6.7	31	0.2	Coal	0.85
B	3	0	0.24	6.2	12	0.2	Sandstone	0.27
C	1	0	0.14	7.03	12	0.2	Mudstone/coal	0.94
C	2	0.5	0.15	6.7	12	1.0	Mudstone/coal/sandstone	0.85
C	3	0	0.16	6.1	18	0.2	Shale/coal	0.65
D	1	1.8	0.23	6.6	16	2.0	Coal/shaly coal	1.83
E	1	0.5	0.24	7	24	1.0	Coal/sandstone	0.35
F	1	0	0.77	5.2	10	0.2	Coal/shale	0.54

Table 4-16 Summary results obtained from No 2 sonic probe monitoring holes

Colliery	Site	Max. height of dilation (m)	Support density (bolt/m^2)	Bord width (m)	Cut-out distance (m)	Total displ. (mm)	Immediate roof lithology	Thickness of imm. roof (m)
A	1, Test 1	1	0.57	5.8	16	3.0	Grit/grit coal laminae	0.86
A	1, Test 2	1	0.57	5.8	16	2.5	Grit/grit coal laminae	0.86
A	2	0.5	0.41	6.58	16.7	1.2	Sandstone	0.54
A	3	1	0.44	6.1	16	3.0	Sandstone and grit	1.13
B	1	0	0.15	6.5	24	0.2	Coal	1
B	2	0	0.15	6.7	31	0.2	Coal	0.85
B	3	0	0.24	6.2	12	0.2	Sandstone	0.27
C	1	1	0.14	7.03	12	1.5	Mudstone/coal	0.94
C	2	1	0.15	6.7	12	1.0	Mudstone/coal/sandstone	0.85
C	3	0.7	0.16	6.1	18	1.0	Shale/coal	0.65
D	1	1.8	0.23	6.6	16	2.5	Coal/shaly coal	1.83
E	1	0.3	0.24	7	24	5.0	Coal/sandstone	0.35
F	1	0	0.77	5.2	10	0.2	Coal/shale	0.54

4.11 Investigation of trends using numerical modelling

As mentioned earlier, the behaviour of the roof is a function of many variables. These include the stress environment, roof lithology and strength of roof materials, bord width, etc. A further complication is that the variables govern the roof behaviour according to their combination with the others. There are a great number of different combinations, and although great care was taken to include the widest possible range of parameters in the study, it was clearly not possible to include all or even a sufficient number to derive all the answers experimentally.

However, important trends were identified. To investigate these trends further, a numerical modelling code was added to the study to conduct a comparative analysis. The three dimensional (3D) boundary element code Map3D was used in the analysis. The basic 3D model that was used is shown in Figure 4-24.

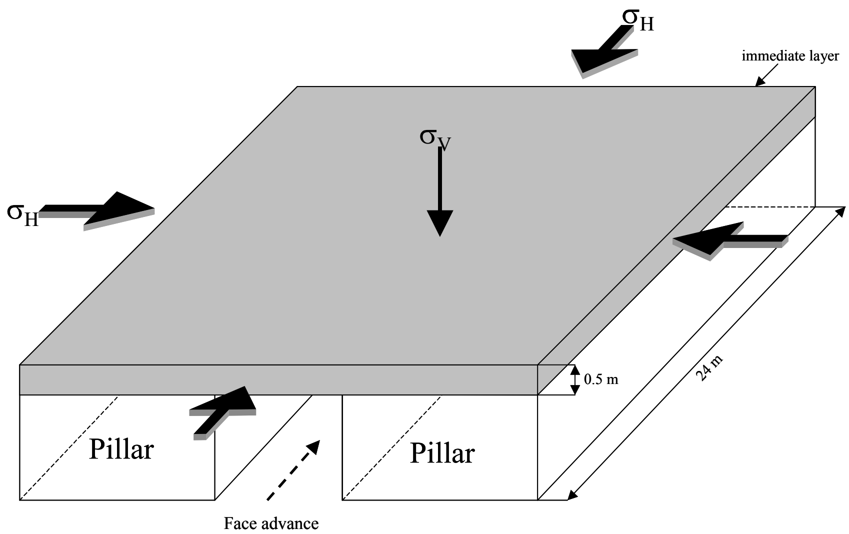


Figure 4-24 MAP3D model that was used in the numerical modelling analysis

The following material properties were used in all models:

Table 4-17 Input parameters used in numerical modelling

Material	Elastic modulus (GPa)	Passion's Ratio	Density (kg/m^3)
Host material	15.0	0.20	2500
Immediate layer	8.0	0.25	2500
Coal	4.0	0.28	1600

The first variable that was investigated was bord width, Figure 4-25. The trend is clear – the greater the bord width, the greater the roof deflection, as would be expected.

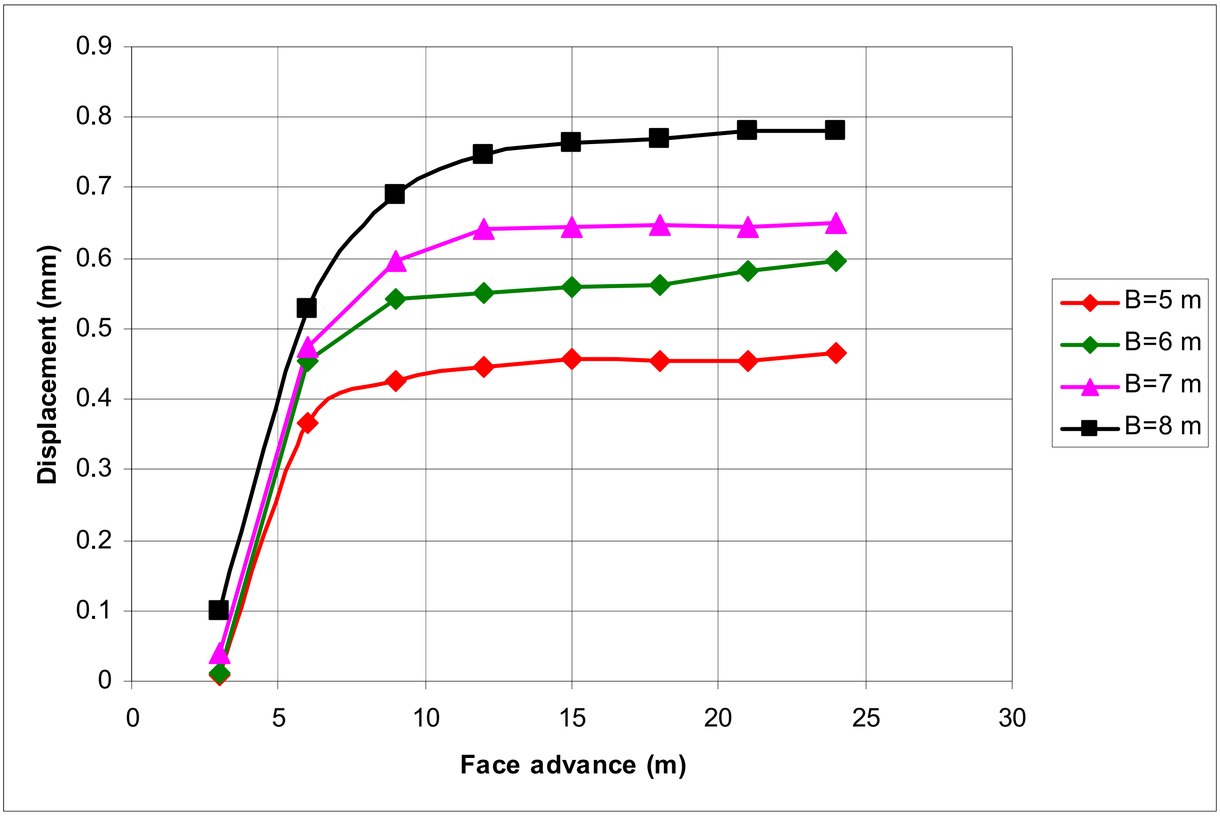


Figure 4-25 Effect of bord with on dilation

The effect of horizontal stress was investigated next. A 0.5 m thick immediate layer overlain by sandstone overburden was modelled, and the maximum displacements at 6.0 m face advances were measured in the centre of the bord. While all the parameters were kept constant, the model was run with two k-ratios (ratio of horizontal stress to vertical stress) of 3.0 and 10. The results are shown in Figure 4-26. This figure indicates that increasing horizontal stress by a factor of 3.3 increased the deformations in the roof by a factor of 1.3 at 6.0 m advance and by smaller proportions at greater advances. That increasing the horizontal stress did not accelerate the roof deflection is confirmed by observations at the three sites where horizontal stress was regarded as a problem. At those sites, the roof deflection did not differ significantly from that at any other site. However, the magnitude of the stresses at the sites was not known.

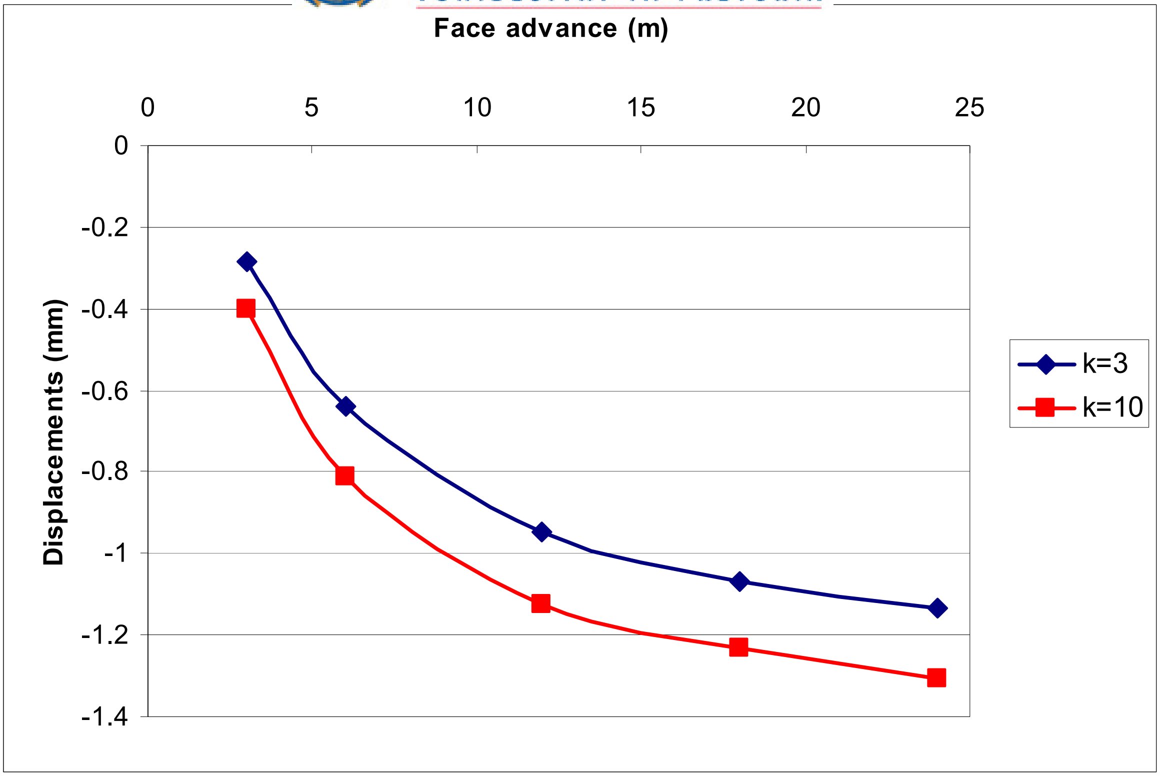


Figure 4-26   Effect of k-ratio on roof deformations

Note that this observation should not be read as implying that elevated levels of horizontal stress are irrelevant. It merely means that if the stress is not high enough to result in failure of the roof material, it will not dramatically increase the deflection of the roof (until of course it reaches stress levels that are sufficient to induce buckling of the roof beam, when sudden failure can be expected).

The effect of the thickness of the immediate layer was also investigated using the same model. In this model the k-ratio was kept constant at 3.0, and 0.25 m, 0.5 m and 1.0 m thick, immediate layers were modelled. The results indicated that decreasing the thickness of the immediate layer increases the displacements in the roof, Figure 4-27. This trend was also apparent in the monitoring, indicated by relatively good correlation between measured deflections and the beam predictions.

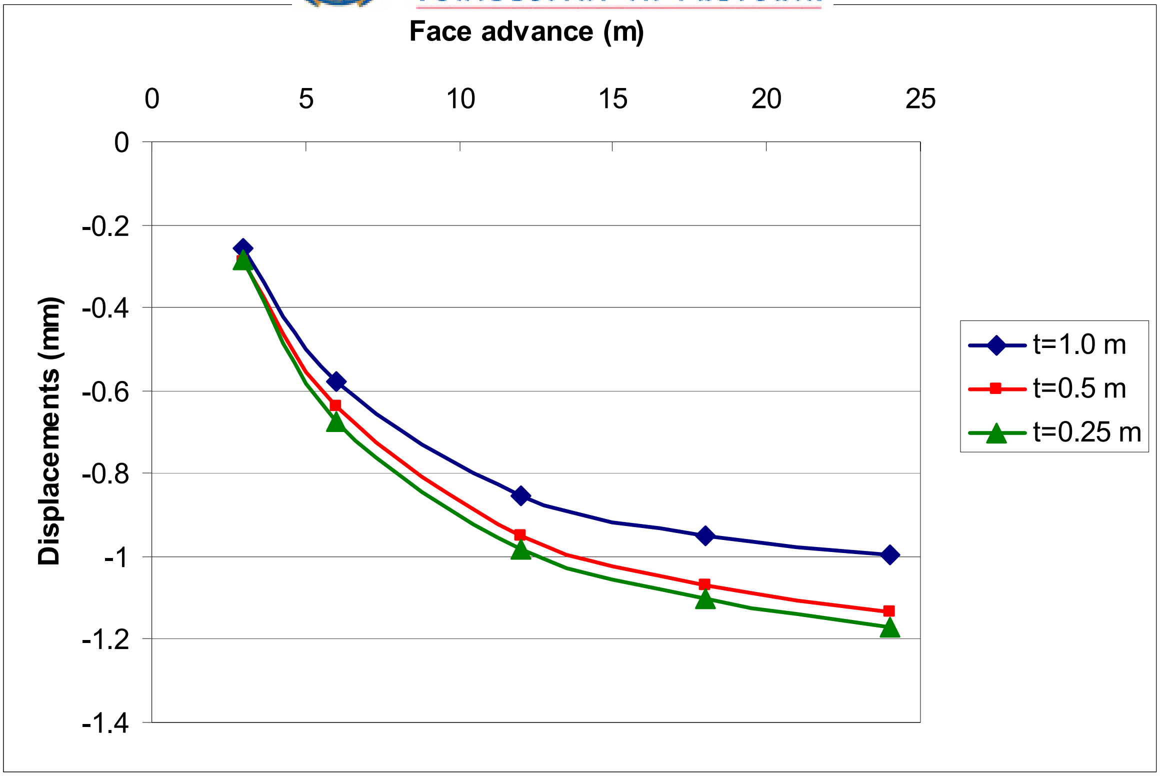


Figure 4-27 Effect of the thickness of the immediate layer on roof deformations

Other critical parameters in determining the deformability of the roof are the elastic properties of the immediate layer. The same model as given in Figure 4-24 was used in the analysis. While the k-ratio was kept constant (k = 3), the material properties of the immediate layer were changed. In the first model, the Elastic Modulus and Poisson's ratio of the immediate layer were taken as 10 GPa and 0.18 respectively. In the second model less stiff material was used. The Elastic Modulus and Poisson's ratio in the second model were 2.5 GPa and 0.22, respectively. The results obtained from the modelling are shown in Figure 4-28. As can be seen from this figure, the properties of the immediate layer have a major effect on the deformations in the roof.

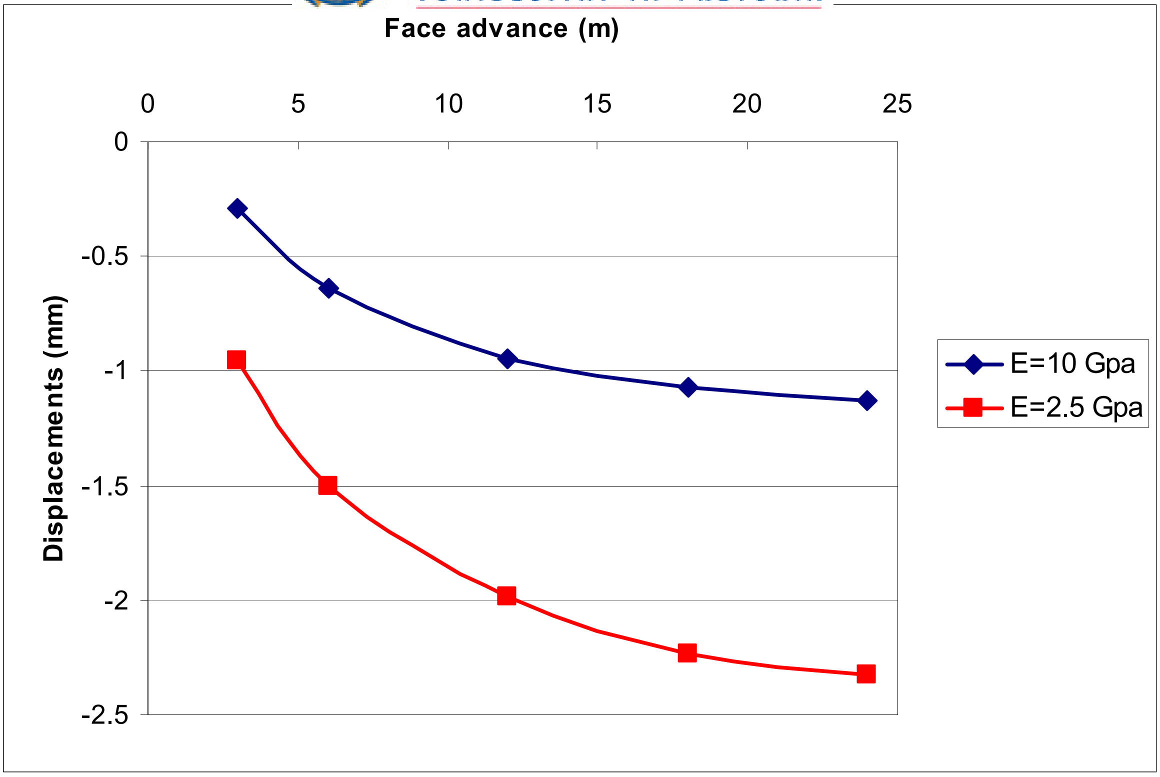


Figure 4-28 Effect of the strength of the immediate layer on roof deformations

The conclusion drawn from the numerical modelling is that the trends observed by underground monitoring are confirmed. It is clear that roof behaviour is the result of a complex combination of several variables. The benefit of modelling is also illustrated. It is the only practical way in which selected parameters can be varied while the others are kept constant in order to isolate the contribution of the different parameters to the overall observed behaviour.

The most important conclusion is that face advance only plays an important role during the initial advance, until the advance equals twice the bord width. Thereafter the additional roof deflection is not significant. However, bord width is important from the beginning and the effect never diminishes. The same can be concluded for the thicknesses of the lithological units and their properties.

The underground observations represent points on the trend curves of overall behaviour. There were insufficient data points to determine trends for each of the variables in isolation, but with the aid of the models it was seen that the observations fitted the patterns exhibited by the models.

4.12 Conclusions

The literature survey yielded little in the form of directly applicable research. It appeared that little work on determining roof failure per se as a function of cut-out distances has been done elsewhere. The limitations on cut-out distances were mainly due to other issues, like preventing underground workers being under unsupported roof and methane and dust control. Recent work done by researchers in the USA seems to indicate that extending the cut-out distance in the USA had little effect on roof stability, mainly because operators tended to reduce the cut-out distance under adverse roof conditions and only extend it if roof conditions were good.

The ideal research methodology from a scientific viewpoint would have been to advance unsupported faces until failure occurred. If this could be done under a sufficient number of different situations, it would have been possible to provide direct answers for different situations. However, it was not possible to do that without exposing people to considerable risk. The next best was to monitor the universally accepted precursor to roof failure, which is roof deflection, under a range of different situations. This was done under the widest possible range within the constraints of time and funds, but it was still found that there were too many combinations of the variables that determine the roof deflection to derive complete answers.

The measurements were then complemented by numerical monitoring, which affords the possibility to vary only certain parameters and keep the rest constant. It was then found that the underground observations fitted the patterns derived from the models and consequently there is a high level of confidence in the final conclusions.

The most important conclusion from this investigation was that once the face had advanced to a distance equal to twice the bord width, there was insignificant additional roof deflection with further face advance. This conclusion was confirmed by numerical modelling and is in line with the analytical beam solutions. For typical South African conditions, with bord widths in the range of 5.5 m to 7.2 m, the implication is that roof stability would not be adversely affected by advancing further than 11 m to 14 m. Majority of all of the total roof deflection that would take place, would occur during the first 11 m to 14 m of development. Therefore, if it is intended to limit roof deflection by restricting the cut-out distance, the cut-out distance would have to be limited to less than the bord width. During the investigation, it was observed that where adverse roof conditions existed, this was in fact done by underground personnel.

With regard to the effects of time on roof deflection, it could only be studied for the initial period of 48-hours following roof exposure. The reason for this was operational, as leaving faces for longer periods would have had an adverse effect on production and the sequence of mining.

The instrumentation was usually done on Friday afternoons, preceding weekends during which faces would not be mined. It was found that the roof continued to deflect during that period, but that the amounts of deflection were not significant. However, it is still deemed necessary to support a roof as soon as possible, as even minute fractures resulting from the additional deflection may change the roof behaviour and eventually result in failure.

Results from one sonic probe monitoring hole showed that roof bolting had minimal remedial effect on roof deformations. Although the effect of roof bolting was specifically monitored by only one sonic probe monitoring extensometer, in general, the results showed that in none of the monitoring holes where roof displacements were recorded, was there any evidence of the roof being lifted due to installation of pre-tensioned roof bolts. This indicates that the roof bolt tensioning was not sufficient to close the pre-existing openings within the roof strata, where roof displacements were recorded. However, as indicated by the differences in the maximum displacements between the No 1 and No 2 holes, it may be concluded that roof bolting prevented further deterioration from taking place. In all the cases the displacements recorded by the No 1 holes (drilled next to the previously installed bolts) were less than those recorded by the No 2 holes (drilled in the centres of the unsupported areas) during the same monitoring period.

It was found that the lithological composition of the roof strata played a major role in the amounts of deflection that were recorded. Bedding separation was seen to occur at the positions where different strata types joined. This implies that the roof behaved like a set of composite beams of different characteristics. It was then also found that the amounts of deflection corresponded with the deflection that would be expected from gravity loaded beams.

Within the limits of horizontal stress that were present in the study areas (three of the sites exhibited obvious signs of elevated horizontal stress), the stress appeared not to have had a noticeable effect on roof deflection. This was confirmed by the numerical modelling. It was concluded that as long as the magnitude of the stress is insufficient to result in failure of the roof, it does not contribute meaningfully to deflection.

The implication of this is that the dilation in the roof is determined by bord width and roof lithology rather than cut-out distance, once the cut-out distance exceeds twice the bord width.

This last conclusion is significant, as it offers the first possibility to predict roof deflection and consequently roof failure. The recommended process is as follows:

- Determine the thicknesses of the roof plates (or beams) by careful scrutiny of borehole logs.
- Calculate the maximum deflection for the desired road width using standard beam solutions.
- Calculate the induced beam stresses using the standard beam solutions.
- If failure is not predicted, the road width is confirmed.
- The cut-out distance should be determined by other considerations (ventilation requirements, etc), but at least it is known that there is little to be gained in terms of roof stability by restricting it to any distance that is greater than twice the bord width.

Roof deflection should then be monitored underground and the first warning sign should be where the amount of deflection exceeds the calculated amount, as that would indicate a change in conditions. Where that occurs, it would be prudent to reduce the cut-out distance, but even more so to reduce the road width.

Exemption from the 12 m restriction on cut-out distance may be obtained from the Principal Inspector provided that the mine can show that the risk to underground workers will not be adversely affected. This implies that a comprehensive risk assessment is required to obtain the exemption. The results of this investigation show that in general, the increased risk to roof instability due to extended cut-out distances is not a major factor and that the emphasis in the risk assessment should be on the other factors, namely the control of dust and methane and the probability of workers being under unsupported roof.

As with any matter relating to roof stability, it is recommended to base this type of exemption on a comprehensive hazard analysis. It is important to obtain a broad view, based on a general roof hazard plan that is required for other purposes as well.

The following steps are recommended for determining the effective cut-out distances for a given site:

1. General roof hazard plans should be drawn up for each section based on the borehole logs,
2. A detailed geotechnical analysis should be conducted. This analysis should include mapping of geological discontinuities, stress regime and roof lithology,
3. The characteristic behaviour of the roof should be determined for the range of conditions, such as change in the thickness of the immediate roof layer, stress regime and bord widths,
4. Once the bord width and support method are established from the above, the cut-out distance can be determined as well. The most important control parameter is

the bord width. If the bord width is chosen such as to result in deflection that is less than that resulting in failure using beam theory, there is little to be gained by restricting the cut-out distance.

5. With the previous steps in place, it remains to also stipulate a procedure that will prevent any person being under unsupported roof.
6. The support system that will be used in the section should also be monitored by continuing the monitoring after the installation of support. The critical factors in determining the support performance are the height of the instability into the roof, which determines the length of support, and the separations within the bolt horizon, which determine the stiffness of the support.
7. Once the cut-out distance is determined with regard to ground control, it should be checked against the ventilation and risk assessment plans.

The study area included one site where there was a high incidence of jointing, but in that area the effects of the jointing did not materialise in the measurements, most probably due to "experimental gremlins." The irony is that the roadways next to the one where the instrumentation was done suffered severe damage and the cut-out distances in those were reduced substantially by the operational crews. However, in the instrumented roadway, no damage occurred and the roof deflection was minimal.

Finally, logic dictates that the longer the cut-out distance, the higher the probability of encountering unexpected jointing with its accompanying negative effects on roof stability. This may be countered by instituting measures that will prevent personnel being under unsupported roof.