Safety in Mines Research Advisory Committee

Final Project Report

Safe mining face advance and support installation practice in mechanical miner workings under different geotechnical conditions

I Canbulat and JN van der Merwe

Research agency	**:**	**CSIR : Division of Mining Technology**
Project Number	**:**	**COL 609**
Date	**:**	**March 2000**
Report No	**:**	**2000-0154**

Executive summary

SIMRAC has initiated a research programme in order to determine safe cut out distances in mechanical miner workings. This research programme includes a detailed literature review on acts and procedures in other major coal producing countries, an underground monitoring programme in various mining districts and numerical modelling.

The literature review showed that in major coal producing countries it is stated that no one may go under an unsupported roof, and therefore the cut out distances are limited to 6.0 m. In South Africa the standard cut out distance is 12 m. However, permission may be obtained to exceed this distance from the district inspectorate in all these countries, including South Africa. Various procedures have been adopted by the district inspectorates to obtain extended cut out distances, including extensive underground monitoring and roof hazard plans together with ventilation plans and risk assessment investigations.

The references relevant to cut out distances cover 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. However, there are relatively few published references on determining effective cut out distances as compared to other aspects of coal mining. The majority of 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 almost as safe as the mining of nonextended cuts from a roof fall accident and fatality perspective.

Underground experiments were conducted in 13 sites at six collieries in four seams. These sites were selected by conducting a detailed investigation into the identification of different geotechnical areas. A comprehensive database was used which was collected over 350 panels in South African collieries, and roof and discontinuity ratings were used to identify the geotechnical areas in South African collieries. The results showed that South African collieries may be divided into five groups with the majority of South African coal production originating from one in particular. Therefore, the test sites were concentrated in this area in order to obtain representative results.

A sonic probe extensometer was used to monitor the roof and support performances in the experiment sites. Two holes were drilled and instrumented with sonic probe anchors in each site. The first hole was drilled and instrumented at the face before any mining took place, and the second hole drilled in the middle of the cut out distance. In order to determine the effect of time on roof deformation, the sites were left for 48 hours unsupported, where appropriate. The underground results showed that the most critical parameter with respect to ground control aspects of extended cut out distance is the bord width, which determines the amount of deformation that will take place in the roof. This was also confirmed by the numerical modelling.

The other significant finding of the underground monitoring programme was the effect of cut out distance. Most of the maximum deformation in the roof takes place before the cut out distance to bord width ratio exceeds two. This indicates that once the face is extended more than twice the bord width, the roof will stabilise, assuming that no geological discontinuity is present and that there will be no further stress changes in the area as a result of for example the development of an intersection. It is known that any site can experience instability at some time due to stress changes, unseen geological discontinuities, changes in the roof lithology and weathering. Therefore, it is recommended that the limitation to cut out distance for a particular geotechnical condition should be based on detailed observations, roof hazard plans, geological information, roof lithology and support performance. Also, an underground monitoring programme should be conducted for a given site for a range of conditions, in order to determine the effective cut out distances in mechanical miner workings.

Acknowledgements

The authors gratefully acknowledge co-operation given by many mines and funding from SIMRAC. Mr. D. Minney, Mr D. Fenn and Mr. L. Munsamy of AMCOAL Rock Engineering Department, are gratefully acknowledged for assisting the project personnel during the project.

Prof. A. Vervoort is also acknowledged for his contribution in the analysis of underground results.

Lastly, Mr. G. Prohaska who spent many days and nights underground to conduct the in situ experiments is acknowledged.

Table of contents

List of Figures

List of Tables

List of contracted Enabling Outputs

NO.	ENABLING OUTPUT	PAGES
1	Survey of international best practice for support installations,	11 - 23
	methods and procedures.	
2	Critical stable roof deflections for various strata.	28 - 80
3	Effect of unsupported face advance on roof stability for different roof	28 - 80
	strata.	
4	Time effects on the stability of various unsupported bord lengths.	28 - 80
5	The effect of time before installation of support on support effectiveness.	28 - 80
6	Collation and interpretation of data and compilation of report.	1 - 100
7	Transfer of technology.	83

1.0 Literature review

1.1 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. Recent studies have shown that, under certain circumstances, 42 per cent of the total roof deflection takes place before the support is installed (Canbulat and Jack, 1999), possibly compromising the support effectiveness. It is thus important to determine, for different roof strata, the critical deformations that can be tolerated before instability sets in.

With the introduction of more sophisticated equipment, such as continuous miners (CM), remote controlled CM and powerful scrubbers for ventilation, there has been a trend to increase cut out distances worldwide. In order to investigate the best cut out distance practice, a comprehensive literature review was conducted.

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.

1.2 Regulations and exemptions with respect to cut out distances in major coal producing countries

South Africa

In South Africa the effective cut out distance has not been investigated and no criteria to establish this distance in different geotechnical conditions have been established.

The introduction of new technology into existing mining systems very often has both positive and negative impacts on worker safety; extended cutting is no exception (Bauer, 1998). 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.

While the standard cut out distance is 6.0 m in other major coal producing countries, 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.

In South Africa, the Mine Health and Safety Act was published in 1996 to provide for the assurance of health and safety of employees and other persons at mines. In this act the mine manager is responsible for the health and safety on the mines, and the responsibilities of managers are outlined as follows:

- Supply all necessary health and safety facilities and equipment to each employee
- To the extent that is reasonably practicable, maintain those facilities and that equipment in a serviceable and hygienic condition (Mine Health and Safety Act, 1996, Section 6).
- Ensure that every employee complies with the requirements of the Act
- Institute the measures necessary to secure, maintain and enhance health and safety

- Appoint persons and provide them with the means to comply with the requirements of this Act and with any instruction given by an inspector
- Ensure that work is performed under the general supervision of a person trained to understand the hazards associated with the work and who has the authority to ensure that the precautionary measures laid down by the manager are implemented (Mine Health and Safety Act, 1996, Section 7).
- Every manager must establish a health and safety policy (Mine Health and Safety Act, 1996, Section 8)
- Any manager may prepare and implement a code of practice on any matter affecting the health or safety of employees and other persons who may be directly affected by activities at the mine.
- A manager must prepare and implement a code of practice on any matter affecting the health or safety of employees and other persons who may be directly affected by activities at the mine if the Chief Inspector requires it.
- A code of practice required by the Chief Inspector must comply with guidelines issued by the Chief Inspector
- The manager must consult with the health and safety committee on the preparation, implementation or revision of any code of practice.
- The manager must deliver a copy of every code of practice prepared in terms of subsection (2) to the Chief Inspector
- The Chief Inspector must review the code of practice of a mine if requested to do so by a registered trade union with members at the mine, or a health and safety committee or a health and safety representative at the mine.
- At any time, an inspector may instruct a manager to review any code of practice within a specified period if that code of practice -
    (a) does not comply with a guideline of the Chief Inspector; or
    (b) is inadequate to protect the health or safety of employees.
- The manager must also assess and respond to risk (Mine Health and Safety Act, 1996, Section 11)
    1) Every manager must -
        (a) identify the hazards to health or safety to which employees may be exposed while they are at work;
        (b) assess the risks to health or safety to which employees may be exposed while they are at work;
        (c) record the significant hazards identified and risks assessed; and
        (d) make those records available for inspection by employees.
    2) Every manager, after consulting the health and safety committee at the mine, must determine all measures, including changing the organisation of work and the design of safe systems of work, necessary to
        (a) eliminate any recorded risk;
        (b) control the risk at source;
        (c) minimise the risk; and
        (d) in so far as the risk remains
            (i) provide for personal protective equipment; and
            (ii) institute a programme to monitor the risk to which employees may be exposed.
    3) Every manager must -
        (a) periodically review the hazards identified and risks assessed, including the results of occupational hygiene measurements and medical surveillance, to determine whether further elimination, control and minimisation of risk is possible; and
        (b) consult with the health and safety committee on the review.

As can be seen from the above the mine manager can provide for specific areas or mining practices in his code of practice, and, the code of practice to combat the roof fall accidents must be approved by the Inspectorate. These previsions can relate to the unsupported cut out distance and the procedures required to maintain these areas in a stable and safe condition.

In order to compile codes of practice, the DME has published a guideline for the compilation of mandatory codes of practice to combat rockfall accidents in collieries.

In these guidelines, roadway support strategies were outlined, and coal mining roofs were broadly classified into three main types:

- **No systematic support required (i.e. massive sandstone roof):** Strong roof requiring no systematic support shall be investigated for changes in lithology at monthly intervals and the strategy adapted where necessary.

- **Suspension type (i.e. relatively thin layer of weak material overlain by stronger layer):** Suspension type roofs shall be supported by means of a suitable support system designed by a suitably qualified rock engineering practitioner. The design shall be based on the weight of the weak material, multiplied by a suitable safety factor, balanced by the load bearing capacity of the support system. The load bearing capacity of the system shall be determined by tests.

- **Beam creation type (i.e. thick layer of weak material like shale, mudstone or other laminated material):** Where a thick, weak roof occurs, an artificial beam shall be created by placing roof support in such a way that the shear stresses in the roof are countered. Alternatively, prop support or long anchors to provide sufficient suspension resistance shall be provided. This support shall be designed by a qualified rock engineering practitioner/consultant.

It is evident for the first type of roof that in the code of practice the mine manager can define mining under unsupported strong roof conditions as a safe mining practice. Thus the effective cut out distance becomes meaningless.

In order to control the levels of dust and combustible gases, and to improve roof stability, the Chief Inspector issued a directorate limiting the cut out distance to 12 m, provided at all times the dust level is not more then 5 mg/m^3 at the face.

The regional inspectorate can approve cut out distances greater than 12 m. In order to obtain permission to cut further than 12 m, the mine should submit a risk assessment investigation report. This report includes design parameters and the safety factors of the area, a detailed geological mapping and ventilation plans, as well as a history of falls of ground and influence of depth below surface. In the ventilation plans the type of scrubbers, number of sprayers and effectiveness of ventilation should be detailed. Considering the ventilation and geotechnical plans, if the conditions are described as good, a 24 m cut out distance is permitted. If the conditions are medium a 12 m cut out distance is allowed, and if the conditions are poor, a maximum of 6 m cut out distance is granted to the mine. Currently, six collieries in South Africa have permission to cut further than 12 m.

U.S.A.

The USA Coal Mine Health and Safety Act (1969) states that "no person shall work or travel under unsupported roof…." (30 CFR 75.202b) (U.S. Code of Federal Regulations, 1994a). The accepted practice in the USA underground coal industry has been to advance the face until the inby edge of the CM operator's compartment is just outby of the last row of roofbolts, and then to proceed to support the exposed mine roof (Bauer, 1998).

The Mine Safety and Health Administration (MSHA) defines extended cut mining as "any cut in which the onboard manual controls of the continuous mining machine are advanced by the last row of permanent supports or any cut in which the mining machine advances more than 6.0 m beyond the last row of permanent roof support". Integral to extended cutting is the use of remotely controlled continuous mining machines equipped with flooded-bed scrubbers or fan-spray systems, to suppress dust.

In the USA, cut out distances or extended cut mining must comply with all existing support and ventilation regulations. Additional restrictions on extended cut mining are included in each mine's roof control and ventilation plans. These plans can vary from one MSHA district to another. However, the Federal Coal Mine Health and Safety Act of 1969 limits the amount of respirable coal mine dust to a permissible exposure limit (PEL) of 2.0 mg/m^3 for a working shift. If the dust sample contains more than 5 per cent silica by weight, the dust standard is reduced.

The maximum extended cut out distance approved by MSHA for bituminous coal mines is 19.5 m with a remote controlled CM and a flooded-bed scrubber. The anthracite coal mines are not allowed to cut longer than 6.0 m (standard maximum cut out distance) under any circumstances.

The guidelines for evaluating requests for approval to extract extended cuts were summarized by Bauer, 1998. These guidelines were developed by many of the state regulatory agencies as well as the MSHA in the USA. For instance, Pennsylvania was one of the first states to develop guidelines, having them in place in 1989. West Virginia has had written guidelines since 1990, while MSHA implemented their suggested guidelines in 1993. Because these are merely guidelines and not regulations, regulatory agencies can only use them in plan evaluation, leaving some leeway for mine operators to fine-tune their approval requests.

All of the guidelines were written to address and eliminate the major safety concerns of extended cutting in the areas of ventilation, frictional ignitions, roof control, and human factors. This is accomplished on a mine-by-mine basis by evaluating specific factors pertinent to extended cutting such as the mining system utilised (scrubber vs. fan spray, and provision to indicate cut depth), roof drills and Automated Temporary Support (ATRS) system, and location of personnel and equipment during the mining process. The MSHA guidelines specifically state that the following factors should be considered when reviewing the mine history:

- Coal seam characteristics;
- Unintentional roof falls;
- Remote-control machinery;
- Frictional ignitions;
- Face methane liberation; and
- Ventilation and respirable dust compliance (MSHA, 1993)

Although the guidelines may differ, the areas concerning ground control are similar. The ground-control aspects common to nearly all guidelines include:

1. The CM operator shall not be positioned inby of the second outby row of permanent supports while the CM is in operation;
2. The cut depth shall be reduced when subnormal (adverse) roof conditions are encountered or in areas where ATRS will not set firmly against the roof. To resume extended cutting, at least two normal cuts in good roof must be taken and the roof evaluated by a certified foremen;
3. No more than two or three extended cuts shall be unsupported at any time during normal production shifts on any given section. Unsupported areas that will be left standing over extended periods (holidays and weekends) will be limited to normal cut depth (6.0 m)
4. Prior to mining a side cut, permanent supports shall be installed inby rib of the proposed side cut for a distance of at least one extended cut, or totally supported if less than one extended cut;
5. Side cuts will be permanently supported prior to working inby the intersection; and
6. An ATRS-equipped roof-bolting machine shall be used to install permanent supports in unsupported areas of an extended cut.

When the guidelines are followed, the concerns that are minimised include the necessity of workers to position themselves inby permanent supports, mining of extended cuts when subnormal roof conditions are encountered, leaving large areas of exposed roof unsupported for extended periods, and installing roofbolts without adequate temporary support, Bauer, 1998.

Extended cutting was adopted by over 435 underground coal mines, and, as of 1995, nearly one out of every two underground bituminous coal mines had approval to mine extended cuts in the USA, Bauer, 1998.

U.K.

The legislation in the UK is outdated and is currently being replaced by new legislation, however, in order for the UK mining industry to adopt more modern methods of excavation and support, the mine's inspectorate has, for many years, allowed mines to operate under a series of exemptions from the 1954 Act and 1966 regulations. These exemptions are generally site specific and short term. Regulation of cut out distance, support spacings, roofbolting and other systems has therefore been applied by the local inspectors for each mine.

In practice each mine applies individually to the inspectorate for their required exemptions of the Act and regulations. Usually, a separate application is made for each exemption and the inspectorate normally requires supporting documentation from the mine and independent consultants.

In the case of extended cut out distances, the inspectorate would require notification of a stand up trial, carried out in line with The Deep Mines Coal Industry Advisory Committee (DMCIAC) guidance, followed by monitoring and a short appraisal report by geotechnical consultants, before granting the exemption.

The extended cut out distance needs to be considered a maximum and varied to suit the prevailing geotechnical conditions.

In UK coal mines, cut out distance is not generally as long as the USA and Australian mines. The largest extended cut applied in the UK is 7.8 m in a heading driven by a continuous miner and roofbolted using a Fletcher bolting rig.

Ahead of the forthcoming support legislation, Deep Mines Coal Industry Advisory Committee (DMCIAC), under the auspices of the Health and Safety Commission (HSC), has recently produced "Guidance on the Use of Rockbolts to Support Roadways in Coal Mines". In this guide, a procedure is outlined for assessing 'stand-up time' and determining appropriate cut out distance for UK conditions.

This procedure is given below:

Where a distance is cut greater than that required to set the next roofbolt support, the possible effect of this extended cut out on roof stability needs to be assessed. This investigation ought to take the form of a progressive increase in cut out distance to the planned value with assessment at each of the stages which are given below:

- Assess existing geotechnical information to confirm site suitability for an extended cut trial in terms of strata conditions, total roof movement, rate of movement and position of roof strain zones.
- Excavate the proposed increased cut out distance at the trial site without exposing the driver to unsupported roof.
- Do not install rockbolts.
- Install sufficient temporary standing support to allow a remote reading roof extensometer to be installed halfway along the excavated section.

- Remove the temporary support if appropriate.
- Monitor roof dilation via the remote reading extensometer for the maximum likely delay time before bolts are installed (48 hours is considered to be a minimum).
- The measured pattern of roof deformation should be comparable to previous site results. If there is no significant roof deformation in this period, an extended trial should take place as follows:

(a) assess the overall effect of the cut out distance on roadway stability by monitoring, eg installation of additional telltales and extensometers. The assessment needs to include the possibility of a fall over riding supported roof;
(b) if the results of the assessment are satisfactory, then, in stages, increase the cut out distance to the proposed value. Each stage needs to be subjected to the monitoring and assessment described in (a).

The extended cut out distance needs to be considered a maximum and varied to suit the prevailing geotechnical conditions.

The definition of free-standing supports is supports which are erected between the floor and roof, eg arches, girders, hydraulic powered supports, chocks/cribs, props and bars.

Australia

The standards for underground coal mining in Australia are listed in the Coal Mining Act 1925. In this Act, there is no prescriptive regulation on cut out distances. The cut out distance is determined by ensuring that no one ventures under the last row of support. Hence, in place changing operations under good roof conditions, up to 12 m cut out distances have been granted to mines, provided that the continuous miner is remote controlled and is fitted with a scrubber fan to keep the face ventilated. This places the shuttle car driver under the last row of support. In sections where the operator is on board the CM, the maximum cut out distance is limited to 6.0 m (this places the driver under the last row of support). In very poor roof conditions, the maximum cut out distance can be as small as 0.5 m. The maximum cut out distance is determined by the mine manager and approved by the inspectors in different districts in the form of the "Managers Support Rules". These carry the full weight of the law in Australia.

Extensive underground monitoring programmes are conducted to determine the cut out distances and support requirements in Australia. Information which outlines the specific testing procedures for approval of extended cuts could not be found in literature.

The maximum cut out distance in Australia is 14 m with a remote controlled CM and a flooded-bed scrubber.

1.3 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 project investigates 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). They concluded that in nearly 40 per cent of all roof fall fatalities, the victim travelled inby permanent support (Bauer, 1998).

Bauer et al. (1995) statistically analysed MSHA accident data for 1990 and 1991 to compare reported accidents in extended-cut and nonextended-cut mines. The preliminary indication was that extended-cut mines had a 23 per cent higher reported accident rate at the face and a 12 per cent higher rate of accidents. Overall, the fatality rate was found to be 37 per cent lower for extended-cut mines. The authors also reported on a series of underground interviews of mine workers on extended-cut sections. The interviews addressed various aspects of the worker's experience, mine conditions, and their views on the safety of extended cuts. The results, in general, were that mine workers favoured extended cutting, and believing it to be safer because of fewer equipment moves (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 mine 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 compiled all his extended-cut case studies together with additional work in his PhD thesis in 1998.

Bauer 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 investigated 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.

In this study it was also found that there has been 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 also attempted to estimate safe cut out distances using a regression analysis package MINITAB. Bauer determined safe cut out distances for three stages. These stages and variables he used in the regression analysis are given in Table 1-1.

Table 1-1 Information available during various stages of mining for estimating safe depths-of-cut (After Bauer, 1998)

Mining Stage	Information available
Design stage	Core samples Laboratory rock strength values Outcrop/highwall inspection Overburden thickness Similar mine's experience
Pre-approval stage	All of the above, plus Roof fall history Roof strata characteristics Roof support methods Coal Mine Roof Rating (CMRR) Horizontal stress regimen Entry width Mining height
Post-approval stage	All of the above, plus Extended-cut experience/success

Bauer established the following equation for planning during the design stage;

$$\text{CutDepth} = 11.4 + 0.455\ (\text{CMRR}) - 0.00291\ (\text{Ovbd.}) \qquad (1\text{-}1)$$

where CutDepth = Safe cut out distance (ft)
CMRR = Coal Mine Roof Rating
Ovbd = Overburden thickness (ft)

A second regression analysis using just CMRR as a predictor of cut depth gave the following equation:

$$\text{CutDepth} = 10.1 + 0.442\ (\text{CMRR}) \qquad (1\text{-}2)$$

For a safe cut out distance during pre-approval stage, Bauer gave the following equation:

$$\text{CutDepth} = 8.1 + 0.564\ (\text{CMRR}) - 0.152\ (\text{EntWidth}) - 0.0029\ (\text{Ovbd.}) \qquad (1\text{-}3)$$

where EntWidth = Bord width (ft)

The best fit for the post-approval gave the following equation:

$$\text{CutDepth} = 35.4 + 0.164\ (\text{CMRR}) - 6.64\ (\text{Status}) \qquad (1\text{-}4)$$

Using just extended-cut status as the predictor of safe cut depths, the following equation was obtained:

$$\text{CutDepth} = 47.0 + 8.56\ (\text{Ovbd.}) \tag{1-5}$$

In this study Bauer stated that the best predictors of safe cut depths were CMRR and extended-cut status, however, these equations are not design equations and cannot be used to determine an exact safe cut out distance. They can be used simply as a starting point

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.

As mentioned before, one of the major concerns of extended cuts is the time to 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. Time based deterioration of roof strata is one function of the mobilisation of low friction parting planes. Where mobilisation occurs, the beds delaminate inducing tensile and shear forces which can cause the beam to fail through buckling. When the load acting on an unsupported beam exceeds the strength of the beam or plate in shear, failure occurs. If the cohesion, friction and clamping force are insufficient to prevent shear movement, the following can occur:

- the ply separates,
- the contact strength of the material at the two interfaces is exceeded and a failure, or surfaces develop within the rock mass,
- a combination of the above

The rock is 10 times weaker in tension than in compression, and strata failure is often initiated by tensile cracks at the centre 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, roofbolts 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 after the installation of permanent support.

Canbulat and Jack (1999) investigated the effect of time on roof behaviour in South Africa. The aim of this study was to investigate typical roof behaviour; therefore, unsupported roof per se was not monitored. The results obtained from 29 underground sites showed that close to a static face (within 0.5 m), the roof does not deform significantly. If a face remains static, the roof within its zone of influence (approximately 5.0 m away) experiences some degree of creep with time. The area of roof outside the zone of influence of the face (11 m away) is not affected by the face, irrespective of whether it is stationary or mobile.

Canbulat and Jack (1999) also investigated the roof deformations and effect of time in a road widening experiment. A site was located at a colliery where a section of roadway was widened from 5.1 m to approximately 12 m, Figure 1-1. While the first 5.1 m of the roof was supported, the second cut was not supported. This site was monitored for a period of 38 days. The results showed that there were relatively higher displacements in the skin of the roof at the centre of the roadway (3.5 mm) during the dynamic widening procedure and 24 hours thereafter. The total displacement measured in this site was 5.5 mm. These results indicated that roof deflection close to the unsupported section continued 38 days after the experiment, however, the displacement measured during this period was only 2.0 mm.

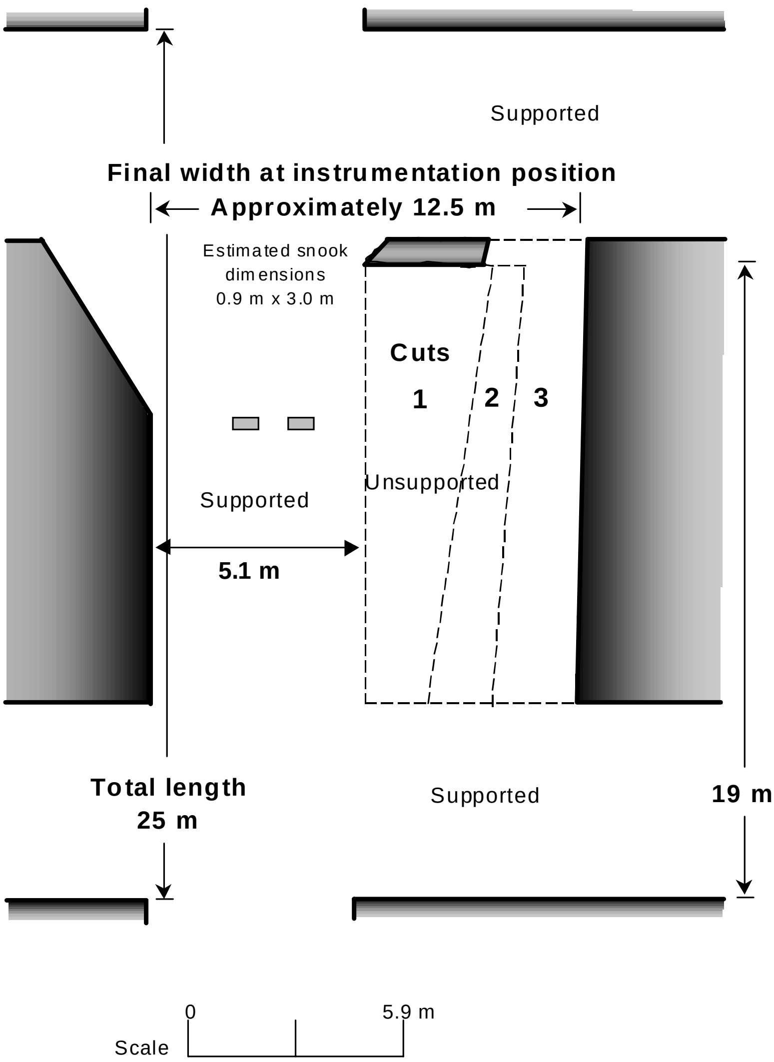


Figure 1-1 Cutting sequence and final roadway shape, (After Canbulat and Jack, 1999)

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 the 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 412. The following equation expresses the relationship for this range of tunnel spans (approximately the range encountered in underground coal mining), (Mark, 1999).

$$RMR = 13 + 1.4\ W_e \tag{1-6}$$

where W_e is the entry width, in feet.

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

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.

The CMRR weights the geotechnical factors that determine roof competence and combines them into a single rating on a scale of zero to 100. The CMRR can be calculated from underground exposures such as roof falls and air crossings, or it can be calculated from exploratory drill core (Mark and Molinda, 1996).

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/entry 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-entry width are given respectively by Mark:

$$CMRR_{crit} = 40.9 + (H/100) \tag{1-7}$$

$$CMRR_{crit} = 19.2 + 1.64\ W_e \tag{1-8}$$

where $CMRR_{crit}$ is the CMRR value below which instability may start to occur.
H is the depth below surface, in feet
W_e is the entry width, in feet

1.4 Conclusions

The literature review showed that in major coal producing countries regulations or guidelines state that no one should go under unsupported roof, therefore the cut out distances are limited to 6.0 m, which prevents the CM operator being under unsupported roof. 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.

Exemptions to allow extended cut out distances can be obtained from the district inspectorate in all the countries, including South Africa. Various procedures and methods have been adopted by the district inspectorates in order to grant the extended cut out distances. These are based on extensive underground monitoring, roof hazard plans together with the ventilation plans and risk assessment investigations.

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 reference 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.

2.0 Identification of geotechnically similar areas

2.1 Introduction

The objective of this component of the project was to quantify the geotechnical characteristics of the most commonly exploited coal seams in South Africa and thus obtain a regional subdivision into geotechnical areas to identify where the underground experiments should be conducted. This was achieved by identifying areas where the range in values of parameters showed distinctly different characteristics.

Geotechnical areas should be defined by specific combinations of geological factors comprising the rock mass, which in turn dictate the expected response of the rock mass to mining. Thus the crucial factor for discriminating such areas would be differences in the expected response of the rock mass to mining operations. Once this response is ascertained, the ultimate aim would be to adopt appropriate rock engineering strategies to minimise potential rock related hazards.

2.2 Pillar rating database

In 1993, as part of SIMCOL 021A project, over 300 panels in 19 seams at nine different coalfields were visited in order to establish individual seam strength formulae. The roof, support, discontinuities and pillar conditions were rated according to a system developed by Madden (1985).

Assessment of pillar performance was carried out in three stages. Firstly, the conditions of the pillar and the surrounding strata were described and recorded. Secondly, each observation was rated according to the relative importance of the parameters.

The classification process was based on detailed visual observations of bord and pillar conditions. A randomly chosen pillar in the centre of a bord and pillar panel was assumed to be representative of the area, and was rated. The following parameters were taken into account in the rating system

- Geology, including roof and floor thicknesses and overburden strata,
- Mining dimensions (pillar and bord dimensions and panel width),
- Pillar performance (pillar fracturing and scaling),
- Roof performance (density and height of roof falls),
- Support performance (efficiency of the installed support),
- Effects of structural discontinuities on the pillar stability.

As many as 45 different parameters were included in the database. These parameters include the measurements taken underground and surveyor offsets, safety factors calculated from these measurements, geological information, and visual underground observations that enabled discontinuity, roof, support and pillar ratings to be determined. From these parameters, only the discontinuity and roof ratings were thought to be relevant and representative of the different geological conditions in identifying the different geotechnical areas [other parameters can respond differently depending on the specific mining method or dimensions]. Therefore, these two ratings were used in this analysis.

In the rating of the immediate roof, roof competence, density and the height of falls were considered, with a maximum rating of 200 points. Table 2-1 was used in the roof rating system.

Table 2-1 Roof rating components.

Point		Roof Competence
100		No Falls/cracks
75		No falls but cracks
50		Occasional cracks
25		Falls to a competent layer
0		Falls to an in competent layer
		Density of Falls
100		None
75		Occasional on a slip/dyke
50		Associated with slip/dyke
25		Intersections only
0		Intersections and bords
		Height of falls
1.00		None
0.75		Slight 0.1 m
0.50		Moderate 0.1 - 0.5 m
0.25		Severe 0.5 - 2.0 m
0.00		Very severe >2.0 m

If the thickness of falls to an incompetent layer is greater than 2.0 m in bords and intersections, then the roof is rated zero.

The performance of the roof was calculated using the following equation:

Roof rating = Roof competence + (Density of falls x Height of falls).

The effect of structural discontinuities on the pillar strength was also investigated. Structural discontinuities such as slips, faults and dykes were mapped and their effects on pillar stability were rated. No effect is rated the highest (100). A severe situation where the discontinuities reduce the pillar area by approximately 30 % because of spalling is rated zero. Table 2-2 was used to determine the effect of discontinuities on pillars.

The analyses of discontinuity and roof ratings showed that one of these ratings alone is not sufficient to identify different geotechnical areas. However, combinations of the two ratings can be used to identify different geotechnical areas in South African seams.

The averages of these two ratings for different seams are plotted in Figure 2-1. This figure highlights that, while in many seams the effect of discontinuities on pillar performance was similar, the roof rating can be significantly different and may determine the stability of the excavations and indicate relative support requirements.

Table 2-2 Discontinuity rating components

Point	**Effect on the pillar**	
100	None	
75	Slight	Minor effects on corners
50	Moderate	Major effects on a corner or a sidewall
25	Severe	Major effects on corners or sides
0	Very severe	Feature reduces pillar area by 30

Based on this figure, it was concluded that in South African seams five different geotechnical conditions can be broadly identified, which have a similar roof and discontinuity condition. These seams and the groups are as follows;

Table 2-3 Identified geotechnical areas for South African coal seams.

Group No	Seam - Coalfield
1	Main - Zululand
2	Dundas - Utrecht 2 &3 - Vereeniging
3	Alfred - Utrecht
4	Top-Bottom - Klip River 4 - Witbank 5 - Highveld Dundas - Vryheid Gus - Vryheid Alfred - Vryheid 2 - Witbank 4 - Highveld
5	5 - Witbank 7 - Soutpansberg Gus - Utrecht CU+CL - Eastern Transvaal 1 - Witbank 2 - Highveld

The data shown in Figure 2-1 are also given in Table 2-4 together with a number of observation sites and the standard deviations of each seam for both discontinuity and roof ratings.

The trends of these areas can be explained as follows;

Area No:	Discontinuity rating	Roof rating
1	Poor	Very Good
2	Fair	Poor
3	Very Good	Fair
4	Very Good	Good
5	Very Good	Very Good

As can be seen in Figure 2-1, most of the collieries, which produce approximately 95 per cent of coal, are situated in Area 4. Therefore, the underground monitoring sites were concentrated in collieries in Area 4.

2.3 Conclusions

The analyses of discontinuity and roof ratings showed that using one of these ratings alone is not sufficient to identify different geotechnical areas. However, a combination of the two ratings can be used to identify different geotechnical areas in South African seams. Based on the analysis of these two ratings, five different geotechnical areas which have similar roof and discontinuity conditions have been identified. These seams and the groups listed in order of relatively good to relatively poor conditions are as follows;

Table 2-4 Average discontinuity and roof rating results for each seam.

SEAM	COALFIELD	NUMBER OF SITES	ROOF RATING	DISCON. RATING	STDEV. ROOF RATING	STDEV. DISCON. RATING
2	Vereeniging	7.0	13.9	61.1	19.7	37.8
Dundas	Utrecht	10.0	48.8	61.4	54.8	17.5
Alfred	Utrecht	7.0	67.9	92.9	76.0	12.2
4	Highveld	50.0	118.3	90.9	79.8	20.1
Gus	Vryheid	10.0	128.8	100.0	75.2	0.0
2	Witbank	111.0	129.5	93.2	78.2	12.8
Alfred	Vryheid	10.0	130.0	100.0	75.3	0.0
Dundas	Vryheid	8.0	146.9	100.0	76.1	0.0
Top-Bottom	Klip River	24.0	154.7	92.7	73.2	20.4
5	Highveld	2.0	156.3	100.0	61.9	0.0
4	Witbank	25.0	166.7	93.0	61.4	18.4
Main	Zululand	11.0	178.4	45.5	38.7	18.8
7	Soutpansberg	10.0	185.0	87.5	47.4	13.2
5	Witbank	12.0	198.1	86.5	7.2	16.9
Gus	Utrecht	7.0	200.0	92.9	0.0	12.2
CU+CL	Eastern Transvaal	18.0	200.0	98.6	0.0	0.0
2	Highveld	6.0	200.0	100.0	0.0	0.0
1	Witbank	12.0	200.0	100.0	0.0	0.0

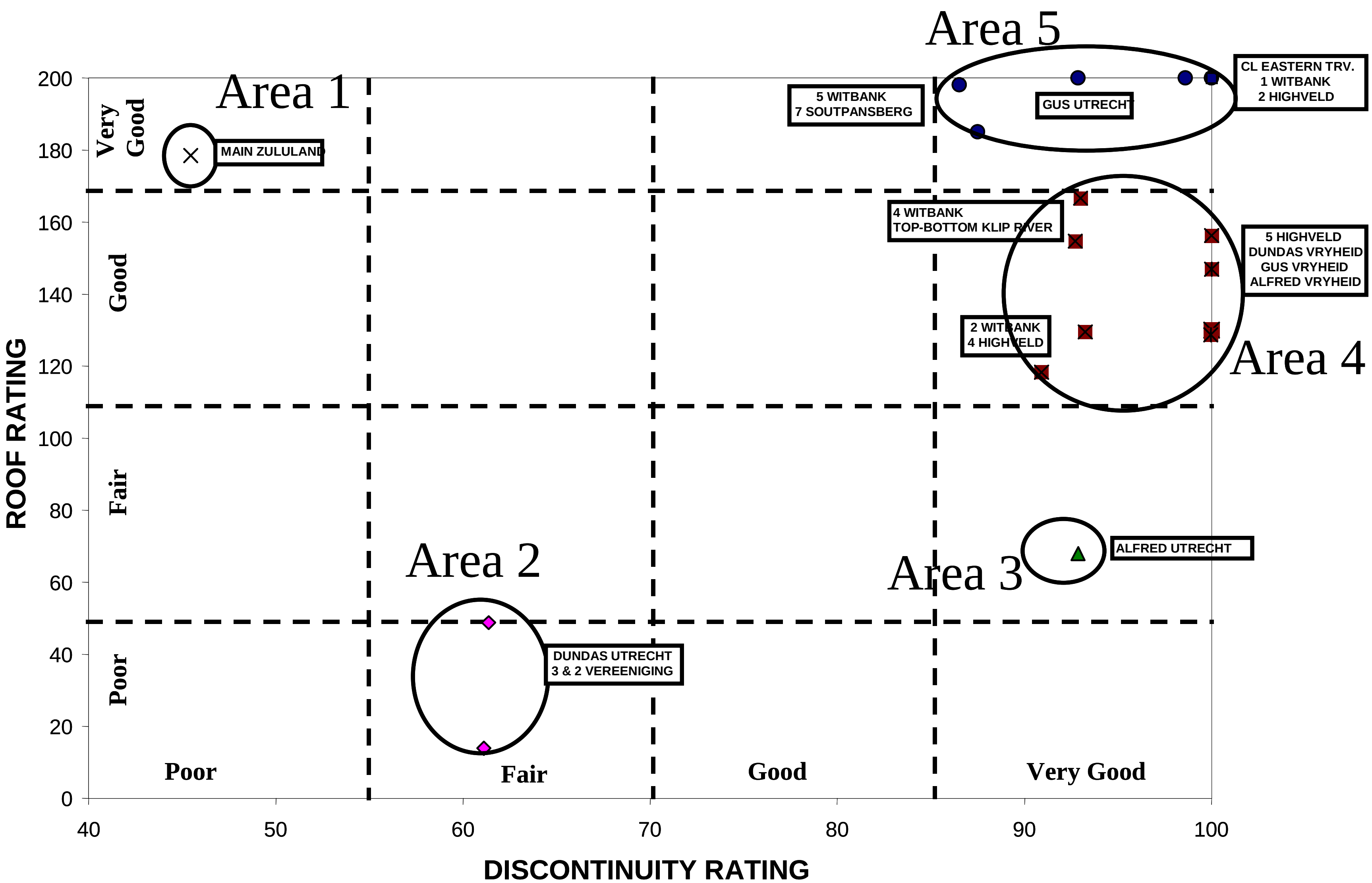


Figure 2-1 Geotechnical areas identified by discontinuity and roof ratings.

3.0 Underground monitoring

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 project. As mentioned above 95 per cent of coal production comes from Area 4, and therefore the monitoring sites were concentrated in this area in order to obtain data which represents the South African conditions.

Table 3-1 Distribution of test sites

Area	# of sites
1	
2	1
3	No colliery exists
4	11
5	1
Collieries	**# of sites**
A	4
B	3
G	1
K	3
S	1
T	1
Seams	**# of sites**
Witbank No 2	7
Witbank No 5	1
Highveld No 4	4
Vereeninging No 2B	1

Two sonic probe extensometers were 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.

The sonic probe extensometer system is a sophisticated electronic device. It generates a pulse that travels at the speed of sound, and is able to accurately determine the distance between magnetic fields, set up by magnets which are integral to the extensometer anchors.

The cylindrical magnetic anchors are locked in place at predetermined locations in a borehole and have a plastic tube inserted through their centres. This tube acts as a guide for a flexible probe that is then inserted through the entire string of anchors. The readout unit is connected to the probe and the distances between the magnetic fields are individually displayed and manually recorded. A schematic drawing of the complete sonic probe unit and the anchors that contain the doughnut magnets is presented in Figure 3-1.

By taking sets of readings over a period of time, any displacements occurring within the rock mass can be determined. Up to 21 anchors can be installed in one hole, thus very detailed information can be obtained as to the amount and location of deformation in the strata.

The readout unit is equipped with the option to measure the distances to all the anchors in the hole relative to the anchor closest to the collar of the hole, or the distances between adjacent anchors. The first option, measuring all the anchors relative to a common 'datum' anchor, was used during the monitoring programme. The reading values increase with the anchor depth into the hole making it easier to determine if a particular anchor point has inadvertently not been read.

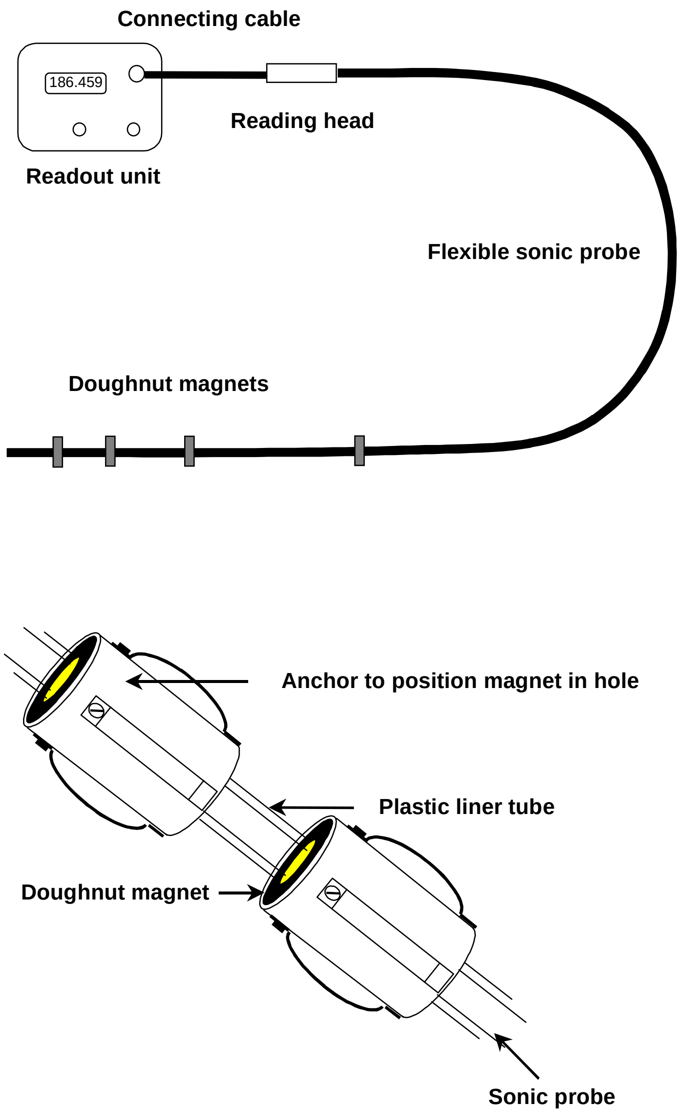


Figure 3-1 Schematic drawing of the sonic probe and magnetic anchors

The overall accuracy of the system has been determined to be of the order of 1.0 mm. This figure has been derived from borehole simulation tests in the laboratory, which showed that on a purely repetitive basis the majority of readings, with the exception of the occasional anomaly, taken on a set of static anchors falls within this range.

A customised computer program for processing the data and producing graphs was written by CSIR - Miningtek. A maximum of three sets of readings taken from up to 21 anchor positions can be input into the program. The output options include two different formats of time/displacement graphs as well as velocity and acceleration graphs.

The sonic probe measures the distance to each anchor relative to the reference anchor, which is the anchor closest to the collar of the hole. However, for graphical interpretation the computer program assumes the top anchor to be static and makes it the reference point, and then calculates the position of all the other anchors in the string relative to it.

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.

A. **CM sequence**: The full cut length was completed in four steps, Figure 3-2. 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.
B. **Road-header sequence**: The full cut length was completed in two steps, Figure 3-3. 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 of any roof support would necessitate the installation of instrumentation a few metres ahead of the face. Since this is clearly not possible the next best scenario is to install the instrumentation at the face. However, due to practicalities such as not working under unsupported roof and the limitations on how close machines such as roof bolters can get to the face, it is not usually possible to drill closer than approximately 1.0 m from the face. This results in the first monitoring holes being in or 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 could be monitored, as they were usually present at the face. This information is presented in each graph from each monitoring site. In the underground situation the quality of roof support installation is dependent on a number of factors. With resin bonded bolts the bond length and quality are dependent on the actual average hole diameter, the overdrilling of holes and deviations from the recommended resin spin and hold times. 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 3-4.

It is well known that the deformations in the roof will increase with increasing horizontal stresses. 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 carefully monitored using the technique developed by Mark and Mucho (1994), Figure 3-5. This technique is useful where the excessive stress signs can be seen. However, it is known that the damage caused by high horizontal stress in the roof is a function of the magnitude and direction of the horizontal stress as well as the strength of the rock. Therefore, it should be noted that when this technique shows no excessive horizontal stress, it does not necessarily indicate that the magnitude of horizontal stress is low. In other words, while the same magnitude and direction of horizontal stress can cause serious damage to the roof if the strength of the roof material is low, no signs of stress damage can be observed in the same conditions if the strength of rock is higher.

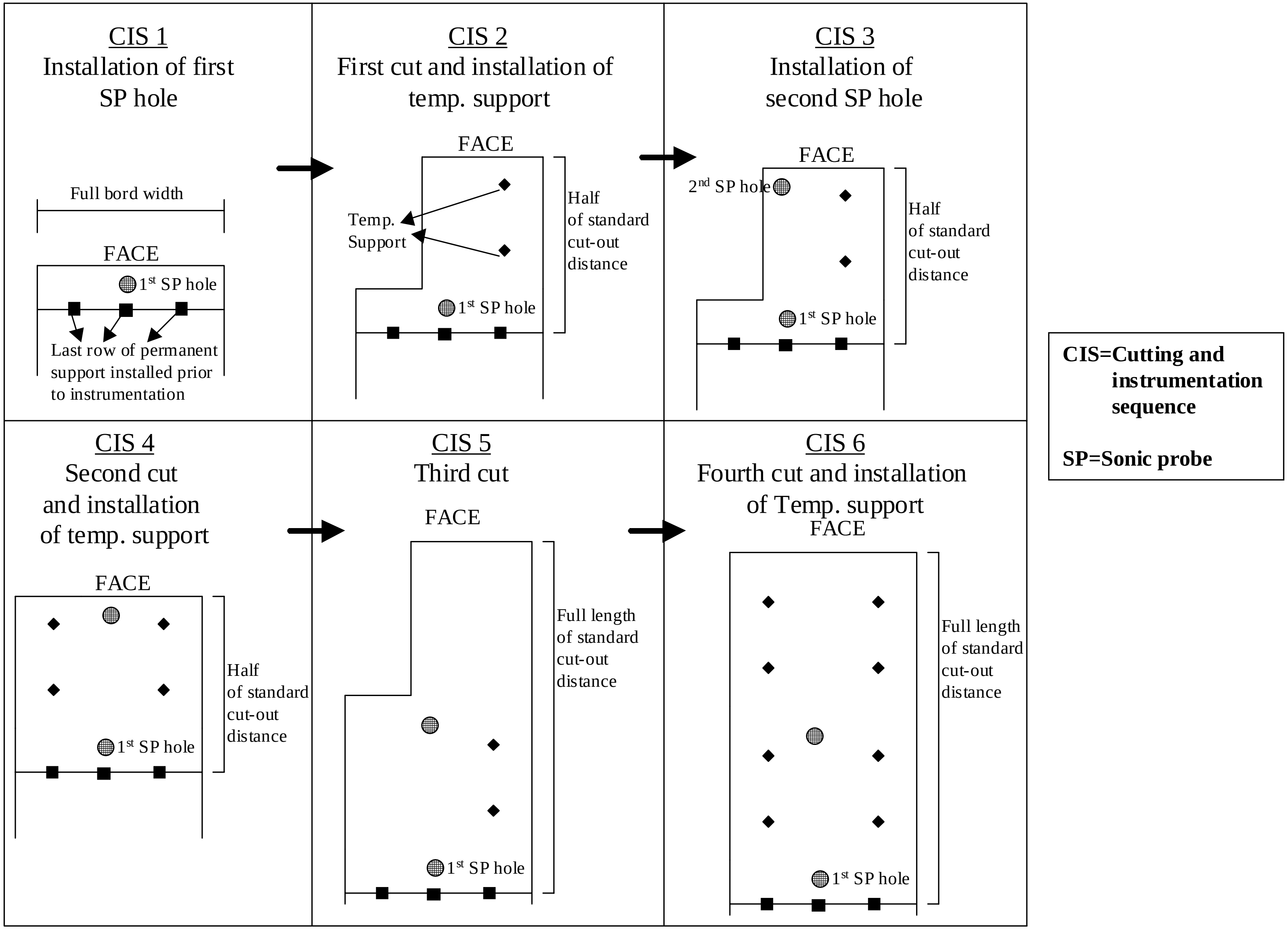
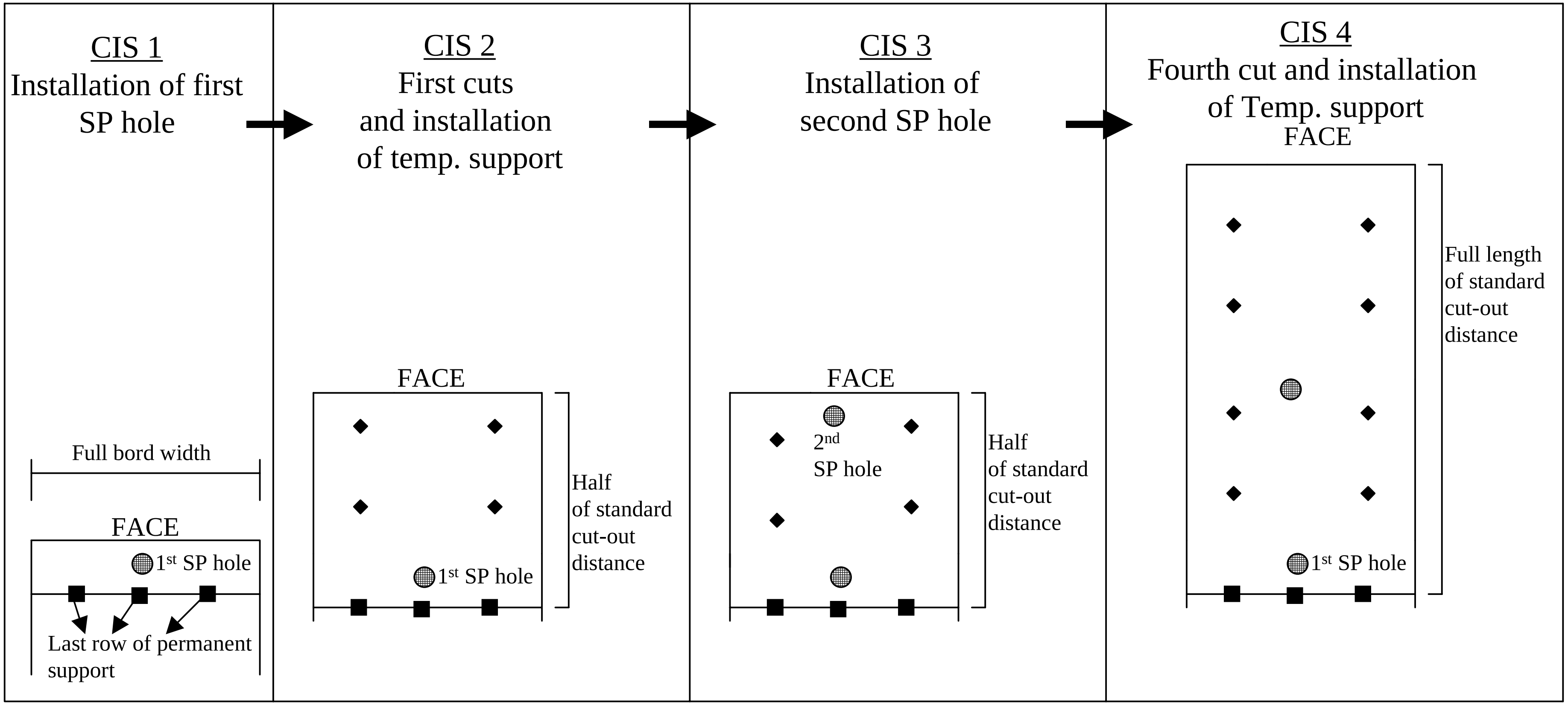


Figure 3-2 Cutting and instrumentation sequence in an CM sections



CIS=Cutting and instrumentation sequence

SP=Sonic probe

Figure 3-3 Cutting and instrumentation sequence in a road header sections

Observation \ Cause	Installation 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	System problems: 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	

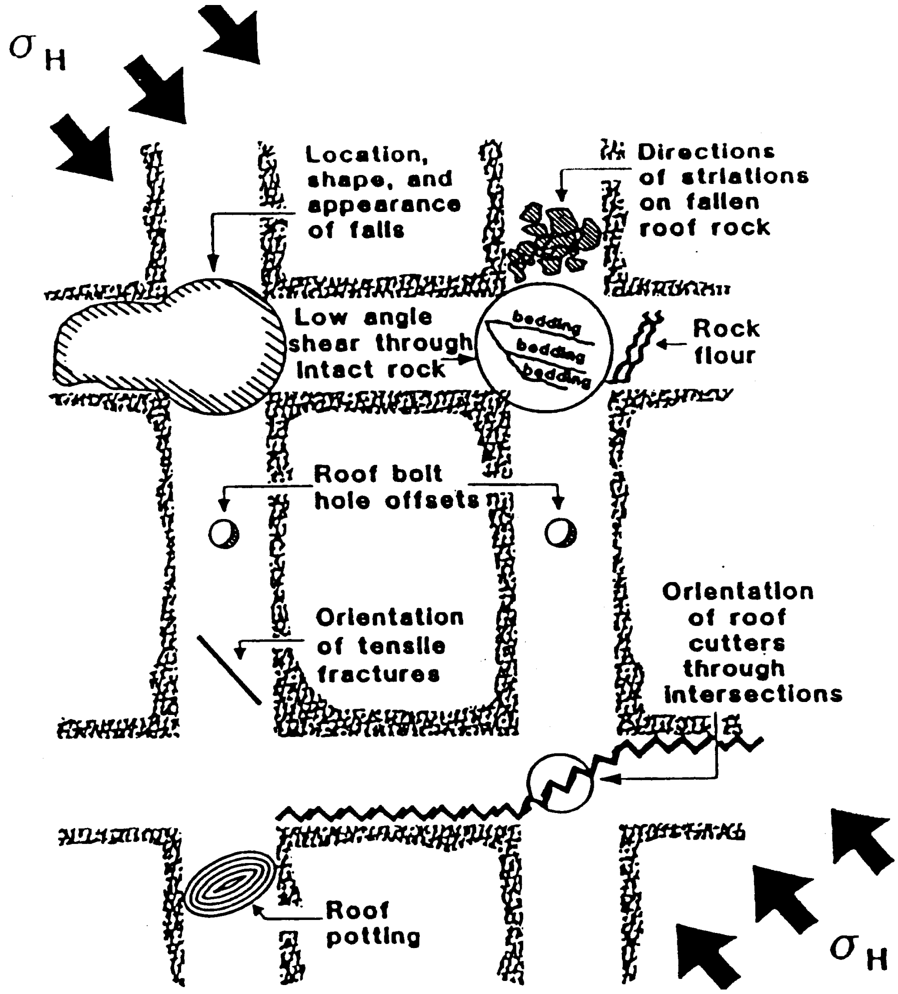
● Most likely cause ◑ Possible cause ○ Potential cause

Seriousness rating: 1 is the most serious non conformance

Alarm rating: 1 is the most dangerous situation

(b)

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



	Feature	Observation Noted	Relationship to G_H
1	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 G_H
2	TENSILE FRACTURES	Direction	Gives direction of G_H
3	ROOF POTTING	Direction of major and minor axes	Major axis gives direction of G_H
4	ROOFBOLT HOLE OFFSETS	Direction of roof movement	Roof layers move in direction of G_H
5	SHEAR PLANES AND ROCK FLOUR	Direction	Planes and rock flour lines are in the direction of G_H
6	STRIATIONS ON ROOF ROCK	Direction	Striations are parallel to G_H
7	ROOF FALLS	Location, shape and appearance	Location give 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 3-5 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 proposed 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, very 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.

3.3 Processing of information

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.

In all the monitoring sites the deepest displacements took place 2.0 m or less into the roof. Therefore, the graphs, where the results are presented, have been cropped at the 3.0 m elevation. This does not infer that displacements above the 3.0 m elevation are being discarded or ignored. All the results from each single anchor were used in the analyses. While a typical graph, which represents the full length of hole, is shown in Figure 3-6, Figure 3-7 shows the same graph but cropped at 3.0 m elevation. As can be seen from these figures the readings from each anchor were used in the final analysis of the results.

Although the displacements usually start at the roof skin and are evident for some distance into the roof, the section of the strata column under investigation does not extend right down to the roof skin. The reason for this is that the bottom magnetic anchor of the anchor string has to be approximately 0.2 m into the roof to allow the dummy anchor, used as a suspension point for the sonic probe, to be installed behind it.

3.4 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. This colliery falls under Area 4 in the geotechnical area category. The plans of the sections in Colliery 'A', where the experiments took place, are given in Appendix 1.

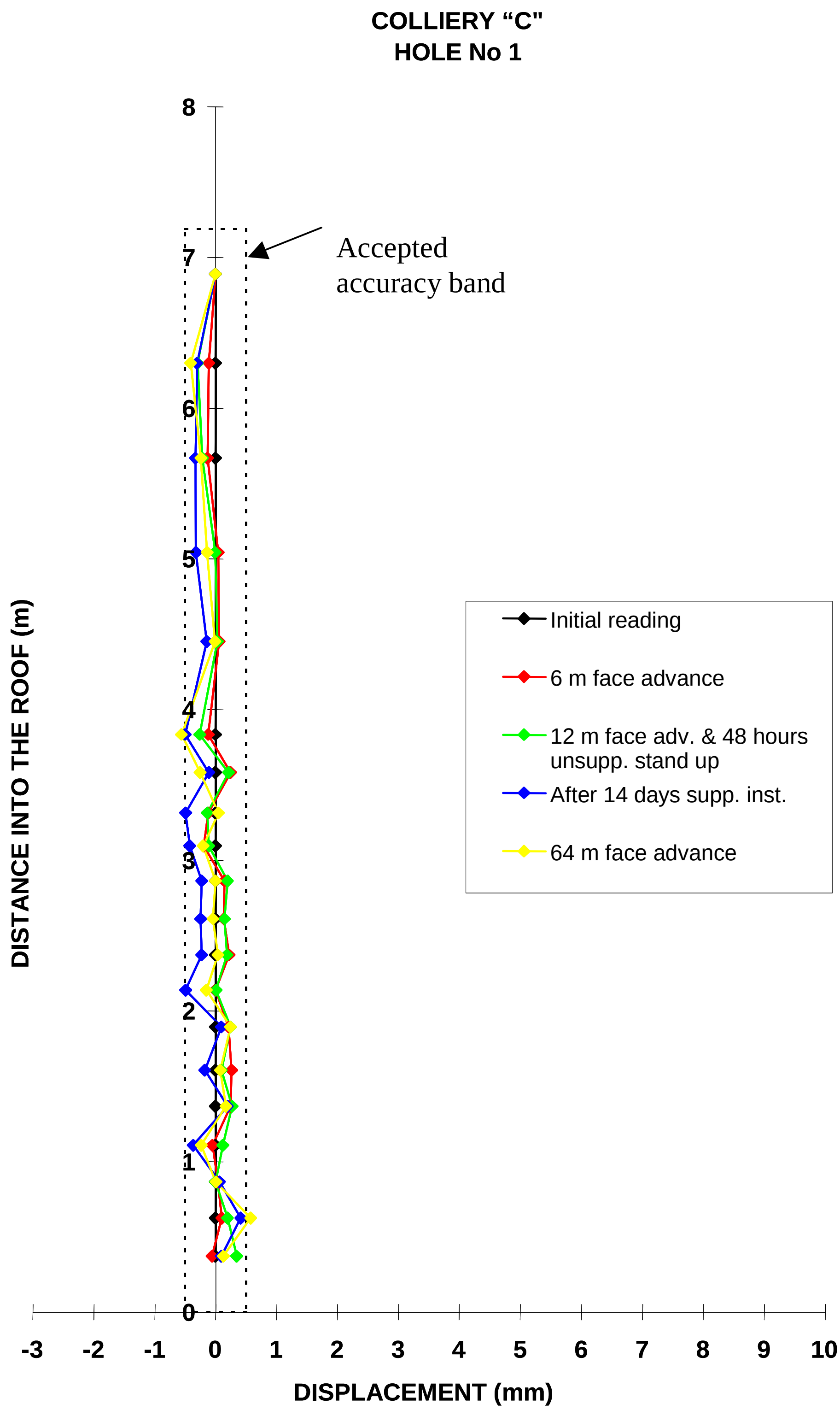


Figure 3-6 A typical sonic probe graph obtained from Colliery "C"

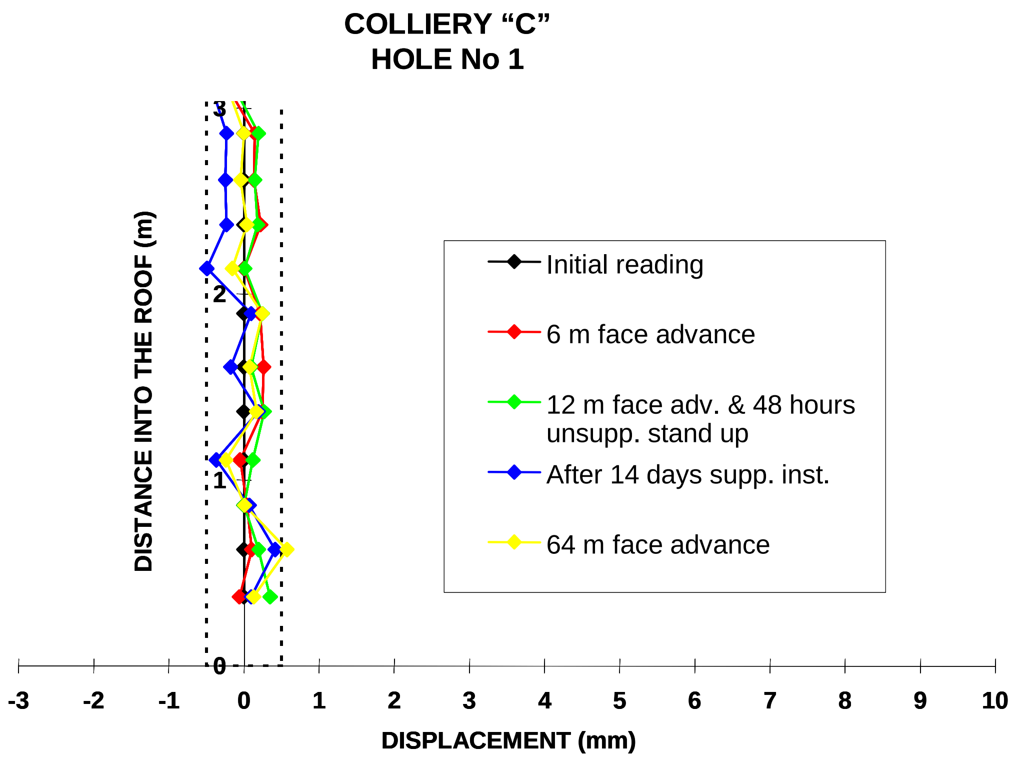


Figure 3-7 Sonic probe results from Figure 3-6 cropped at 3.0 m elevation

3.4.1 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. Because of a 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 a high magnitude of horizontal stress. This was confirmed by the stress mapping technique. Installation and performance of support were found to be excellent in the section.

The CM experiment sequence was used in both experiments. 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 3-8. The summary of site performance in this site is given in Table 3-2.

Table 3-2 Site performance Colliery "A" Site 1, Test 1

Colliery	"A"
Site	1
Coalfield	Witbank
Seam	No 2B
Depth below surface (m)	32
Production method	CM & shuttle cars
Type	Bord & Pillar primary production
Pillar width (m)	9
Bord width (m)	5.8
Mining height (m)	3
Imm. roof lithology	0.13 m grit, 0.65 m grit/coal, and 0.08 m coal overlain by 0.74 m thick sandstone
Support type	OZ-Bar
Bolt diameter (m)	20
bolt length (m)	1.5
Support density(bolt/m^2)	0.57
Resin type	Slow & Fast
Number of resin capsules	3
Resin capsule diameter (mm)	19
Cut out distance (m)	16
Geotechnical area	Area 4
Performance & remarks	In No 1 hole deformation was negligible approximately 0.5 mm, 0.5 m into the roof, after the face had advanced by 16 m. No further displacement was recorded during the 48 hours unsupported waiting period. The total dilation recorded in No 2 hole was 3.0 mm, which occurred 1.0 m into the roof at the grit/coal and sandstone contact. 2.0 mm dilation took place as soon as the face was advanced by 8.0 m (16 m full length completion), and no further dilation was recorded after 48 hours stand-up time. A further 1.0 mm dilation at the skin took place 5 days after the installation of support (face advance was approximately 16 m). However, the roof stabilised and no further dilation was recorded 13 days after support installation (face advance was approximately 30 m).

3.4.2 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 3-9. 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 ($\pm$ 0.5 mm). The summary of the site performance in this site is given in Table 3-3.

Table 3-3 Site performance Colliery "A" Site 1, Test 2

Colliery	"A"
Site	1
Coalfield	Witbank
Seam	No 2B
Depth below surface (m)	32
Production method	CM & shuttle cars
Type	Bord & Pillar primary production
Pillar width (m)	9
Bord width (m)	5.8
Mining height (m)	3
Imm. roof lithology	0.13 m grit, 0.65 m grit/coal, and 0.08 m coal overlain by 0.74 m thick sandstone
Support type	OZ-Bar
Bolt diameter (m)	20
Bolt length (m)	1.5
Support density(bolt/m^2)	0.57
Resin type	Slow & Fast
Number of resin capsules	3
Resin capsule diameter (mm)	19
Cut out distance (m)	16
Geotechnical area	Area 4
Performance & remarks	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. A 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 laminae and sandstone, was 2.5 mm. Initial 2.0 mm dilation was recorded after 8.0 m face advance (16 m full length completed), and no further dilation took place during the 48 hours stand-up time. A further 0.5 mm dilation, 0.75 m into the roof, took place after 542 m face advance. The second hole behaved in a very similar manner to first experiment, second hole.

3.4.3 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. In some localised areas in the section, floor heave and scaling of roof-pillar contact indicated a high horizontal stress. The underground dimension control, installation and performance of support were excellent in the section.

The road-header experiment sequence was used in the experiment. 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 in 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 3-10. 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 3-10), 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 3-4.

Table 3-4 Site performance Colliery "A" Site 2

Colliery	"A"
Site	2
Coalfield	Witbank
Seam	No 2B
Depth below surface (m)	59
Production method	CM & shuttle cars
Type	Bord & Pillar primary production
Pillar width (m)	7.5
Bord width (m)	6.58
Mining height (m)	3.4
Safety factor of pillars	1.89
Imm. roof lithology	0.54 m sandstone overlain by 1.8 m shale
Support type	OZ-Bar
Bolt diameter (m)	20
Bolt length (m)	0.9
Support density(bolt/m^2)	0.41
Resin type	Slow & Fast
Number of resin capsules	2
Resin capsule diameter (mm)	19
Cut out distance (m)	16.7
Geotechnical area	Area 4
Performance & remarks	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.

3.4.4 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 high horizontal stress 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 in the experiment. 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 3-11. 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 3-11), however, no further displacement took place after this stress change.

The summary of the site performance in this site is given in Table 3-5.

Table 3-5 Site performance Colliery "A" Site 3

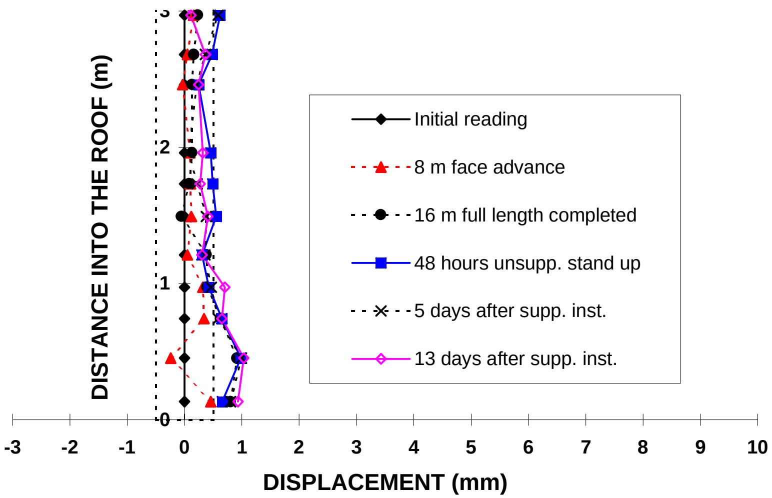
Colliery	"A"
Site	3
Coalfield	Witbank
Seam	No 2B
Depth below surface (m)	53
Production method	Road header & shuttle cars
Type	Shortwall development section
Pillar width (m)	30 x 10
Bord width (m)	6.1
Mining height (m)	3
Safety factor of pillars	5.1
Imm. roof lithology	0.28 m laminated sandstone overlain by 0.85 m grit which overlain by thick sandstone (>2.0 m)
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.44
Resin type	Slow & Fast
Number of resin capsules	3
Resin capsule diameter (mm)	19
Cut out distance (m)	16
Geotechnical area	Area 4
Performance & remarks	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

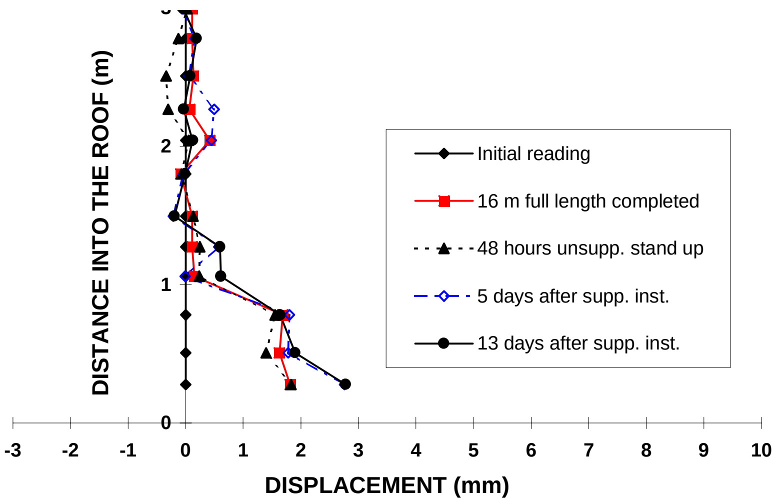
COLLIERY "A" - SITE 1 (TEST1)

(CUT OUT DISTANCE 16 m)

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

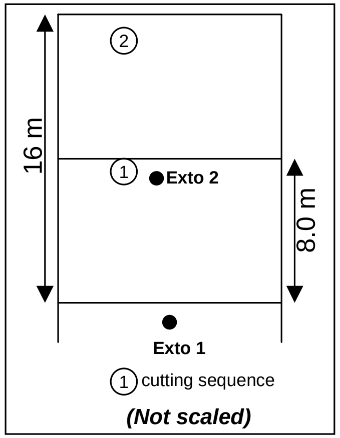


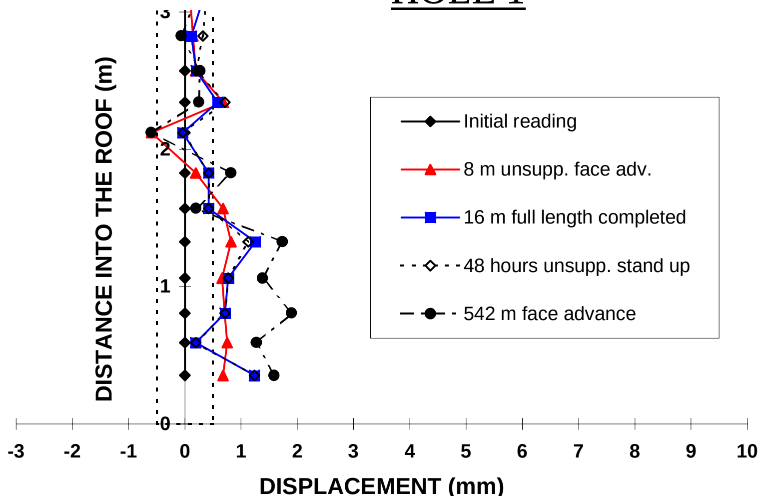
Figure 3-8 Colliery "A" Site 1, Test 1

WITBANK COALFIELD
No 2B Seam

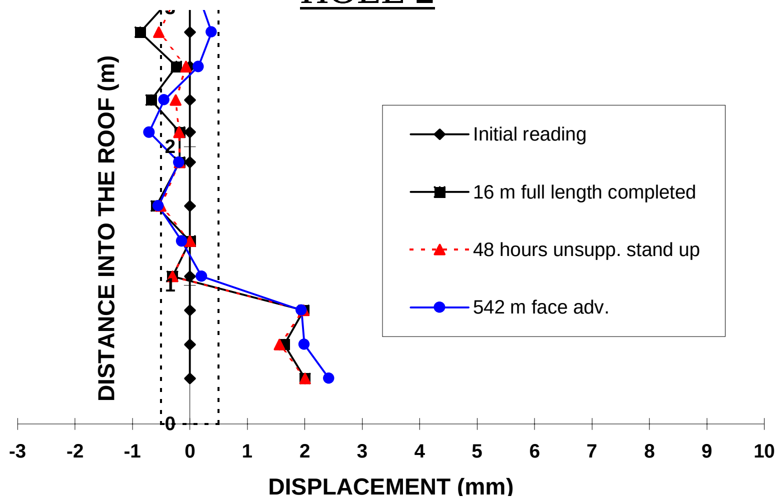
COLLIERY "A" - SITE 1 (TEST2)

(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

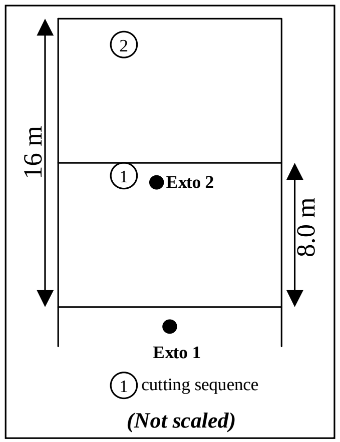


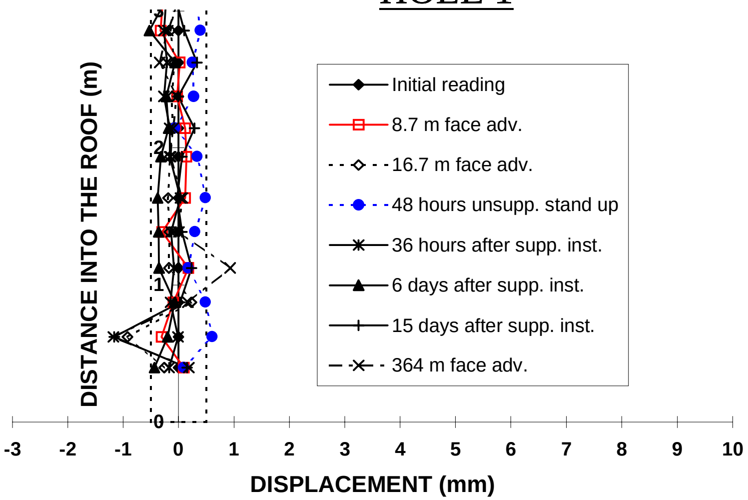
Figure 3-9 Colliery "A" Site 1, Test 2

WITBANK COALFIELD
No 2B Seam

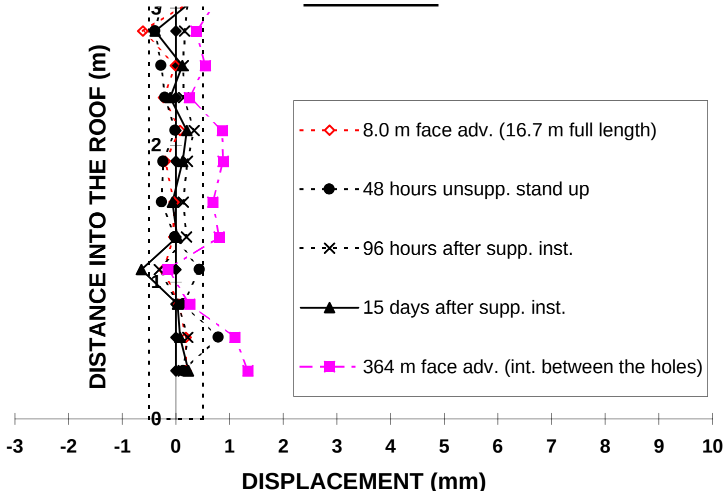
COLLIERY "A" - SITE 2

(CUT OUT DISTANCE 16.7 m)

HOLE 1



HOLE 2



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

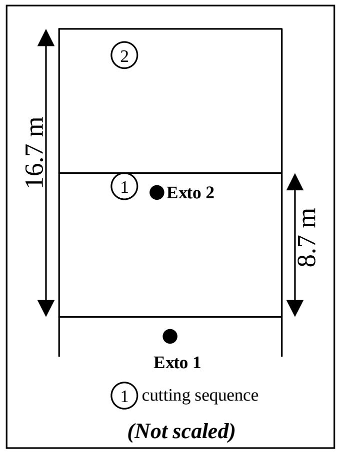


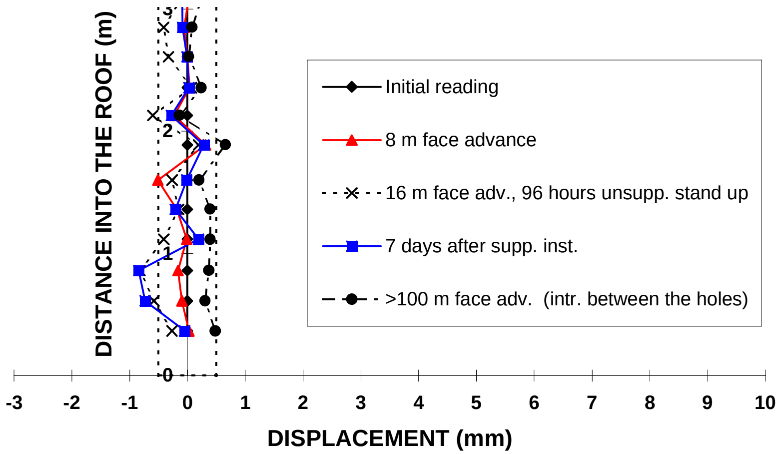
Figure 3-10 Colliery "A" Site 2

WITBANK COALFIELD
No 2B Seam

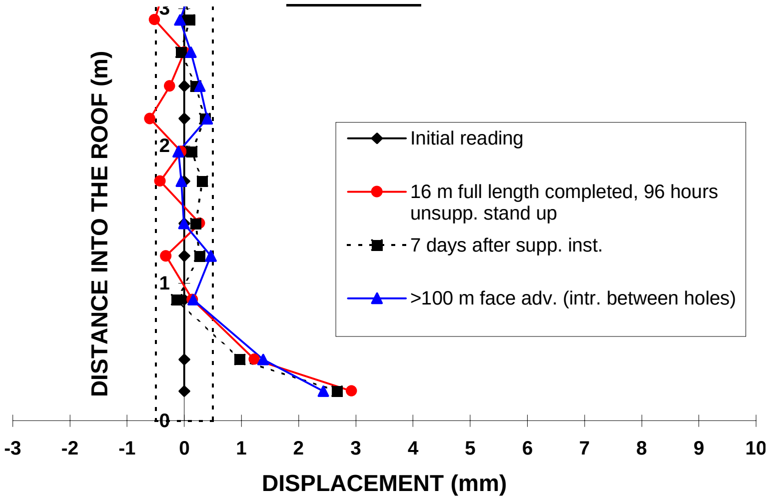
COLLIERY "A" - SITE 3

(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

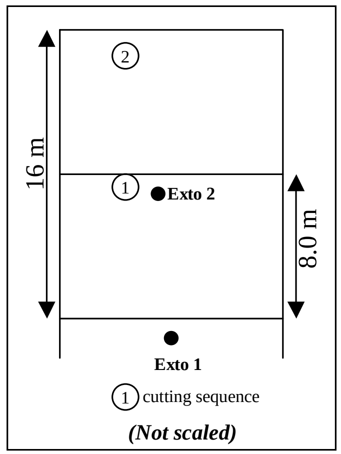


Figure 3-11 Colliery "A" Site 3

3.5 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. While the first two underground monitoring sites (Site 1 and 2) on No 4 Seam fall under Area 4, Site 3 where No 5 is being mined falls under Area 5. The plans of the sections in Colliery 'B', where the experiments took place, are given in Appendix 1.

3.5.1 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 in the experiment. 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. This is thought to be due to coal left in the roof. The results obtained from both holes during the first experiment in Colliery 'B', Site 1 are presented in Figure 3-12. The summary of site performance in this site is given in Table 3-6.

Table 3-6 Site performance Colliery "B" Site 1

Colliery	"B"
Site	1
Coalfield	Witbank
Seam	No 2
Depth below surface (m)	75
Production method	CM & shuttle cars
Type	Bord and pillar primary production
Pillar width (m)	10.5
Bord width (m)	6.5
Mining height (m)	4.4
Safety factor of pillars	1.86
Imm. roof lithology	1.0 m coal overlain by 1.05 m thick sandstone
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.15
Resin type	Slow & Fast
Number of resin capsules	3
Resin capsule diameter (mm)	19
Cut out distance (m)	24

Geotechnical area	Area 4
Performance & remarks	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.

3.5.2 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 in 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 3-13. The summary of site performance in this site is given in Table 3-7.

Table 3-7 Site performance Colliery "B" Site 2

Colliery	"B"
Site	2
Coalfield	Witbank
Seam	No 2
Depth below surface (m)	44
Production method	Road header & shuttle cars
Type	Bord and pillar primary production
Pillar width (m)	15.7
Bord width (m)	6.7
Mining height (m)	4.2
Safety factor of pillars	4.85
Imm. roof lithology	0.85 m coal overlain by 0.65 m thick shale/siltstone
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.15
Resin type	Slow & Fast
Number of resin capsules	3
Resin capsule diameter (mm)	19

Cut out distance (m)	31
Geotechnical area	Area 4
Performance & remarks	Both holes showed no dilation during the experiment.

3.5.3 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 the section raised possibility of high horizontal stress. Detailed stress mapping technique also showed that the horizontal stress was high. However, while excessive horizontal stress caused damage in the roof in some areas, there was no movement in the roof in the experiment site.

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 in full width, the second hole drilled and instrumented and initial readings 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 roofbolting could not be monitored.

While in general, the roof and the pillar conditions were good, underground dimension control (pillar width, bord width and direction of cutting) was poor.

The results obtained from both holes during the first experiment in Site 1 are presented in Figure 3-14. The summary of site performance in this site is given in Table 3-8.

Table 3-8 Site performance Colliery "B" Site 3

Colliery	"B"
Site	3
Coalfield	Witbank
Seam	No 5
Depth below surface (m)	43.7
Production method	CM & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	6.4
Bord width (m)	6.2
Mining height (m)	1.7
Safety factor of pillars	3.56
Imm. roof lithology	0.27 m thick interlaminated sandstone overlain by 1.27 massive sandstone.
Support type	OZ-Bar
Bolt diameter (m)	20
Bolt length (m)	0.9
Support density(bolt/m^2)	0.24
Resin type	Slow & Fast
Number of resin capsules	2
Resin capsule diameter (mm)	19

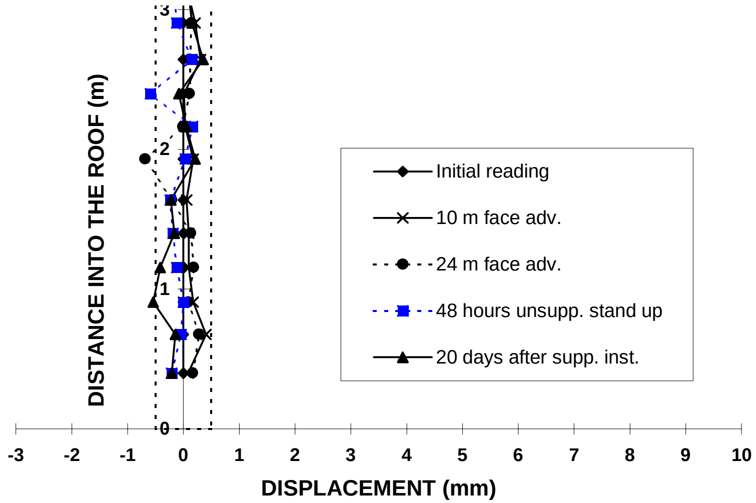
Cut out distance (m)	12
Geotechnical area	Area 5
Performance & remarks	Both holes showed no dilation during the experiment.

WITBANK COALFIELD
No 2 Seam

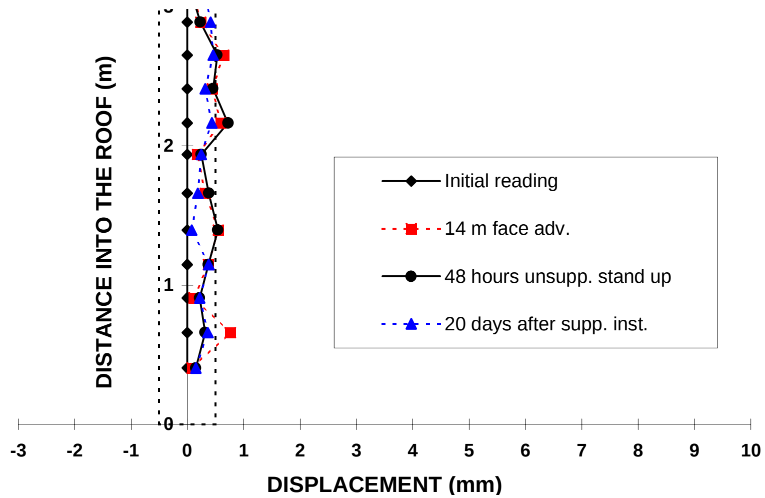
COLLIERY “B” - SITE 1

(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

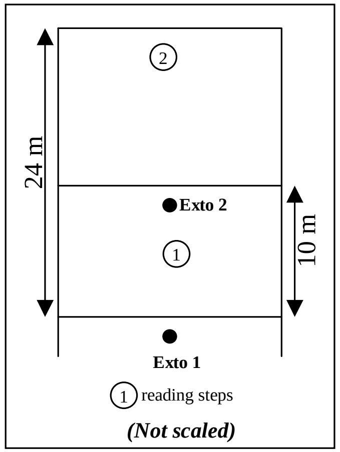


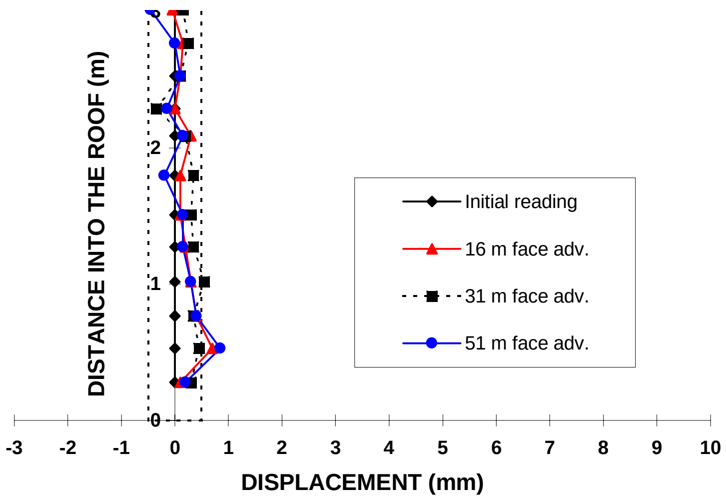
Figure 3-12 Colliery "B" Site 1

WITBANK COALFIELD
No 2 Seam

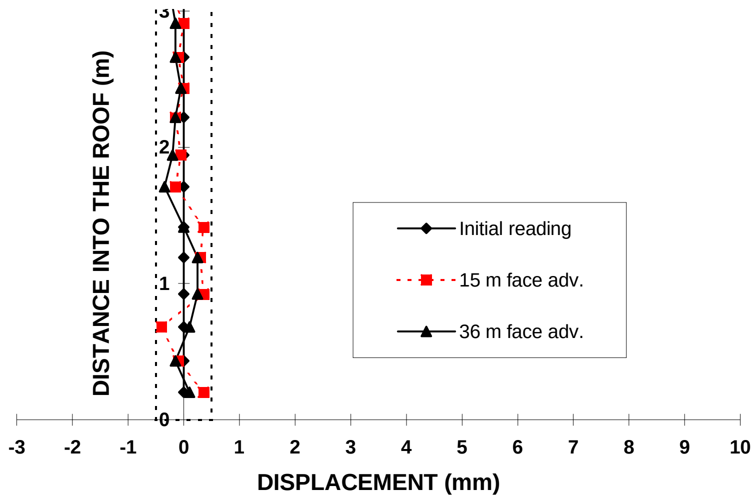
COLLIERY "B" - SITE 2

(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

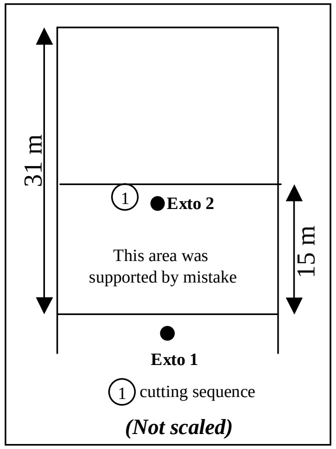


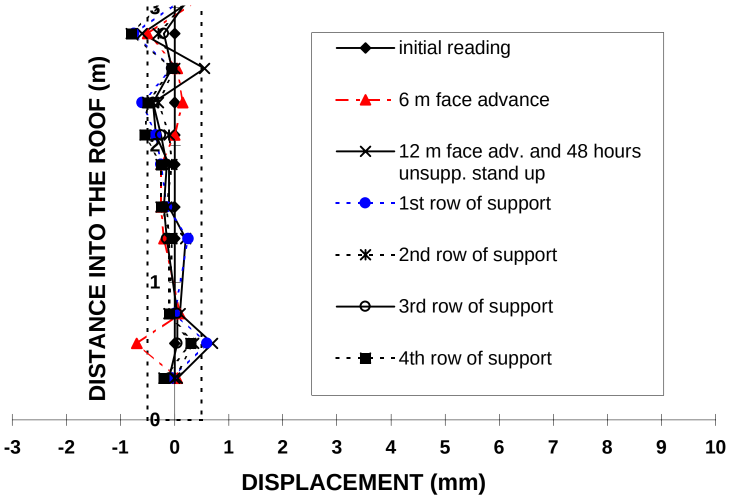
Figure 3-13 Colliery "B" Site 2

WITBANK COALFIELD
No 5 Seam

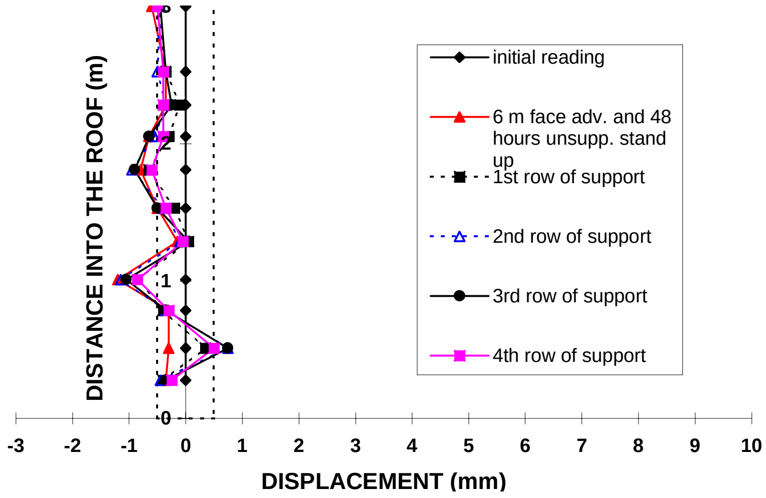
COLLIERY "B" - SITE 3

(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
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

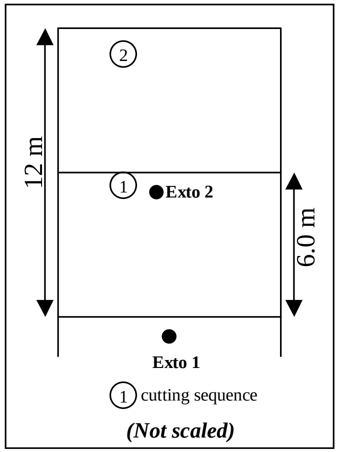


Figure 3-14 Colliery "B" Site 3

3.6 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. All three sections fall under Area 4 in the geotechnical area analysis. The plans of the sections in Colliery 'C', where the experiments took place, are given in Appendix 1.

3.6.1 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 to mine No 4 Seam in the Highveld Coalfield. There were no excessive stress indications in the section. The underground dimension control, roof and pillar conditions were good.

The road header experiment sequence was used in the experiment. 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 3-15. The summary of site performance in this site is given in Table 3-9.

Table 3-9 Site performance Colliery 'C' Site 1

Colliery	"C"
Site	1
Coalfield	Highveld
Seam	No 4
Depth below surface (m)	54.2
Production method	Road header & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	10
Bord width (m)	7.03
Mining height (m)	4.3
Safety factor of pillars	2.33
Imm. roof lithology	0.94 m thick coal/mudstone laminae overlain by 2.25 m thick shale/sandstone
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.14
Resin type	Slow & Fast
Number of resin capsules	2

Resin capsule diameter (mm)	19
Cut out distance (m)	12
Geotechnical area	Area 4
Performance & remarks	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.

3.6.2 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 to mine No 4 Seam in the Highveld Coalfield. There were no excessive stress indications in the section. A major problem observed in support installation was the overdrilling of boltholes. Underground dimension control, roof and pillar conditions were good.

The road header experiment sequence was used in the experiment. 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 3-16. The summary of site performance in this site is given in Table 3-10. Figure 3-16 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 roofbolting, which took place after the unsupported stand-up time. However, the roofbolting should affect the roof skin first before influencing movement further into the roof. As can be seen from the figure, the roof skin was 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 hours stand-up time. No further dilation was recorded in hole No 2.

Table 3-10 Site performance Colliery 'C' Site 2

Colliery	"C"
Site	2
Coalfield	Highveld
Seam	No 4
Depth below surface (m)	70
Production method	Road header & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	12
Bord width (m)	6.7

Mining height (m)	4.7
Safety factor of pillars	2.16
Imm. roof lithology	0.8 m mudstone/coal/sandstone laminae overlain by 2.3 m thick shale/sandstone
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.15
Resin type	Fast
Number of resin capsules	1
Resin capsule diameter (mm)	19
Cut out distance (m)	12
Geotechnical area	Area 4
Performance & remarks	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.

3.6.3 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. Underdrilling of bolt holes was observed, however, in general support installation and performance were good. Underground dimension control, roof and pillar conditions were also found to be 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 3-17. The summary of site performance in this site is given in Table 3-11. Figure 3-17 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 3-11 Site performance Colliery 'C' Site 3

Colliery	"C"
Site	3
Coalfield	Highveld
Seam	No 4
Depth below surface (m)	61
Production method	Road header & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	9

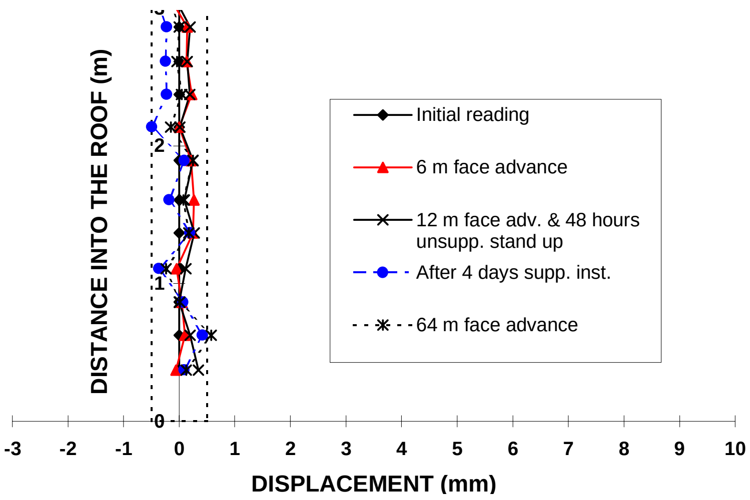
Bord width (m)	6.1
Mining height (m)	4.3
Safety factor of pillars	2.06
Imm. roof lithology	0.83 m shale/coal/sandstone laminae overlain by 2 m thick shale/sandstone
Support type	Re-bar
Bolt diameter (m)	16
Bolt length (m)	1.8
Support density(bolt/m^2)	0.16
Resin type	Fast
Number of resin capsules	1
Resin capsule diameter (mm)	19
Cut out distance (m)	18
Geotechnical area	Area 4
Performance & remarks	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

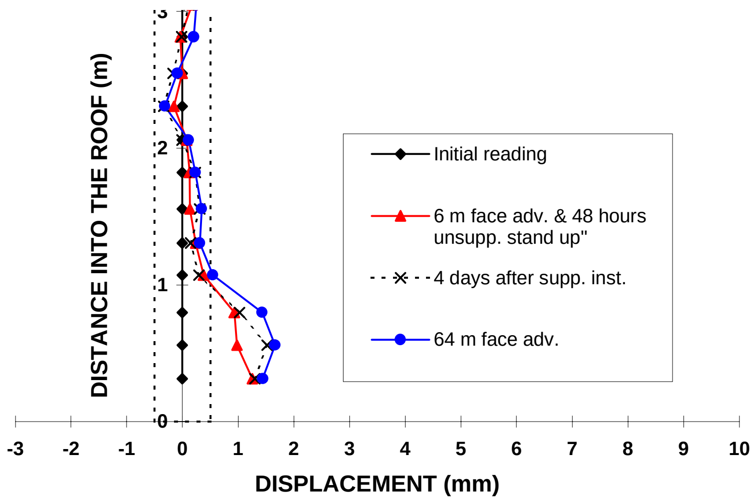
COLLIERY "C" - SITE 1

(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

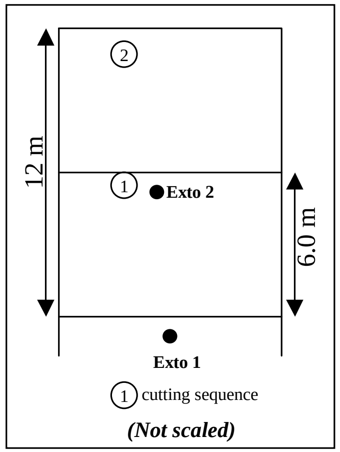


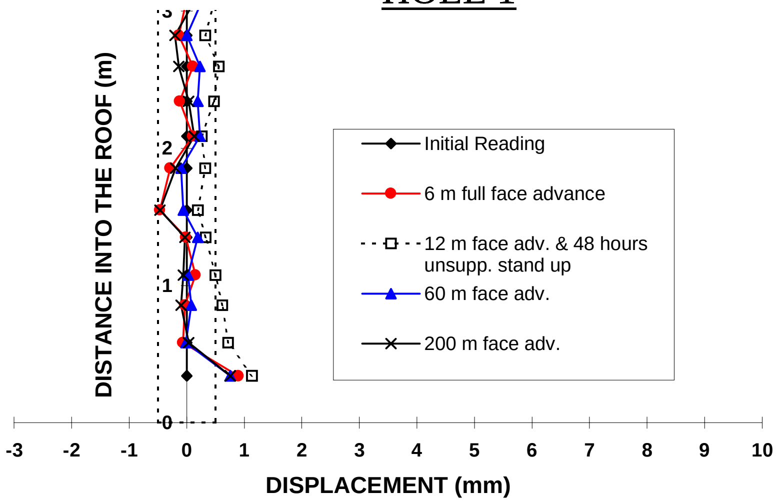
Figure 3-15 Colliery 'C' Site 1

HIGHVELD COALFIELD
No 4 Seam

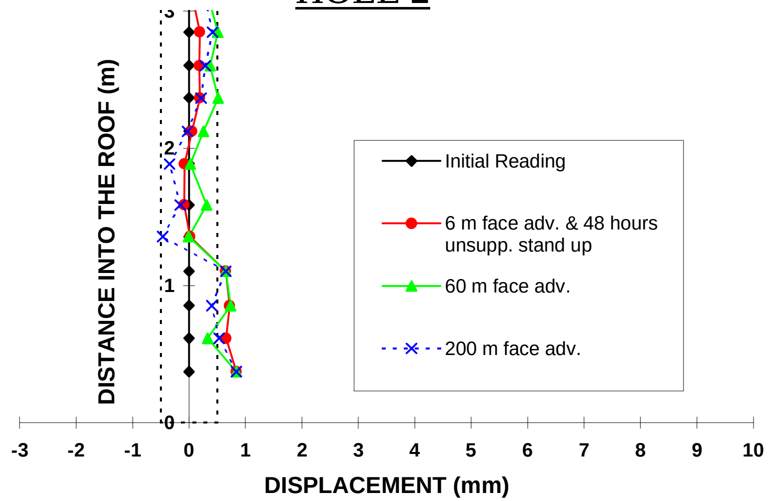
COLLIERY "C" - SITE 2

(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

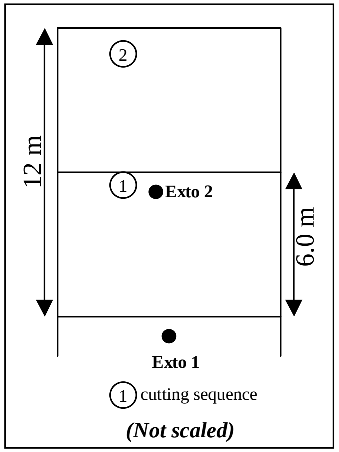


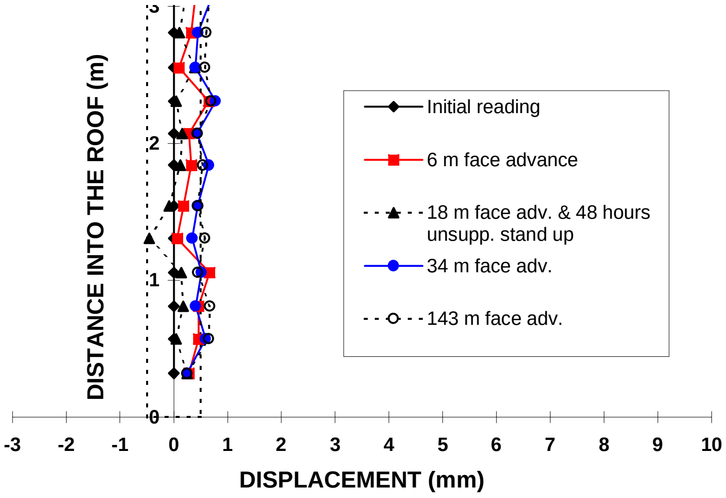
Figure 3-16 Colliery 'C' Site 2

HIGHVELD COALFIELD
No 4 Seam

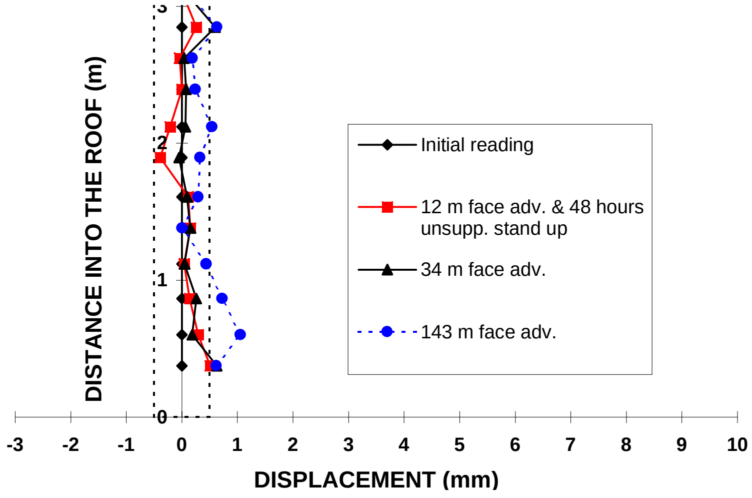
COLLIERY "C" - SITE 3

(CUT OUT DISTANCE 18 m)

HOLE 1



HOLE 2



Immediate roof lithology (not scaled)

DEPTH INTO THE ROOF (m)	SECTION	WIDTH (m)	RECORD OF STRATA
0.83		>2	SHALE/SANDSTONE, laminated, greyish, f.g.
0.79		0.12	COAL
0.59		0.06	SANDSTONE, light green fine grained
0.32		0.13	COAL
0		0.34	SHALE, dark greyish-brown

Site performance

BOLT TYPE:	RE-BAR
BOLT DIAMETER (mm):	16
HOLE DIAMETER (mm):	24
BOLT LENGTH (mm):	1800
HOLE LENGTH (mm):	1900
NUMBER OF BOLTS IN A ROW:	2
DISTANCE BETWEEN THE ROWS (m):	2
BOLT/m^2	0.16
RESIN CAPSULE DIAMETER (m):	19
RESIN TYPES	FAST
NUMBER OF RESIN CAPSULES:	1
BORD WIDTH (m):	6.1
PILLAR WIDTH (m):	9
DEPTH (m):	61
MINING HEIGHT (m):	4.3
SAFETY FACTOR:	2.06

Cutting sequence

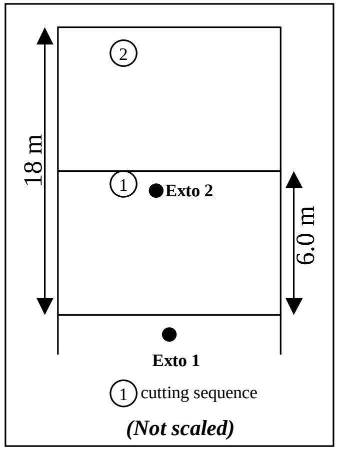


Figure 3-17 Colliery 'C' Site 3

3.7 Colliery 'D'

One site was monitored at colliery ‘D’. The colliery is situated in the Witbank Coalfield and mining is underway on the No 2A Seam, which falls under Area 4 in the different geotechnical area analysis. The plan of the sections in Colliery 'D', where the experiment took place, are presented in Appendix 1.

3.7.1 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 to mine No 2A Seam in the Witbank Coalfield. 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 installation and performance of support were also 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 roofbolting. 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 3-18. The summary of site performance in this site is given in Table 3-12.

Figure 3-18 shows that while hole No 1 showed the least dilation of 2.0 mm, 2.5 mm dilation were recorded in both holes: No 2 and No 3. 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 is important to note that the dilation was recorded further into the roof in this experiment than in all the other sites monitored during the project. This was investigated in detail, however, no obvious reason could be found for this behaviour in the area. Experience in UK collieries showed that any site can experience instability at some time, which can lead to deformation above the bolted height and in turn lead to roof falls, if additional support is not set. This is found to be due to stress changes, unseen geological discontinuities, changes in the roof lithology and weathering. Therefore, a detailed investigation was initiated in the section. However, time constraints, and face advances limited the investigation. 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 roofbolting and/or tensioning in the roof. While it was not possible to observe the effect of roofbolting in the No 1 or No 3 holes, because the initial displacements took place at the bolt horizon as a whole beam, Hole No 2 presented an ideal site to attempt to determine the effect of the installation of the roofbolts. The installation of the bolts appeared to have little if any effect on the bed separation already evident within the bolt horizon as can be seen in Figure 3-18. 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 roofbolts has no remedial effects on pre-existing openings within the bolt horizon.

Table 3-12 Site performance Colliery 'D' Site 1

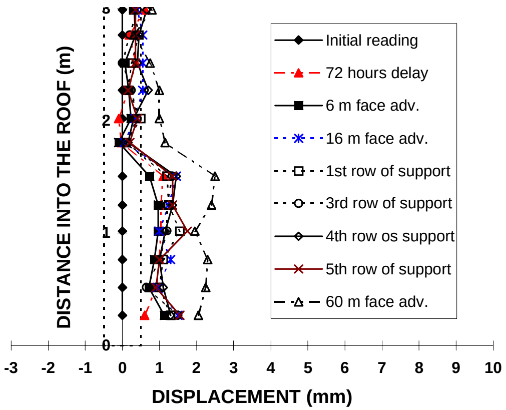
Colliery	"D"
Site	1
Coalfield	Highveld
Seam	No 4
Depth below surface (m)	53
Production method	Road header & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	8.2
Bord width (m)	6.6
Mining height (m)	4.2
Safety factor of pillars	2.03
Imm. roof lithology	1.83 thick coal/shale laminated
Support type	Resin point anchor
Bolt diameter (m)	16
Bolt length (m)	1.5
Support density(bolt/m^2)	0.23
Resin type	Fast
Number of resin capsules	2
Resin capsule diameter (mm)	19
Cut out distance (m)	16
Geotechnical area	Area 4
Performance & remarks	While No 1 and No 3 holes showed 2.5 mm, No 2 hole showed 2.0 mm displacement.

WITBANK COALFIELD
No 2A Seam

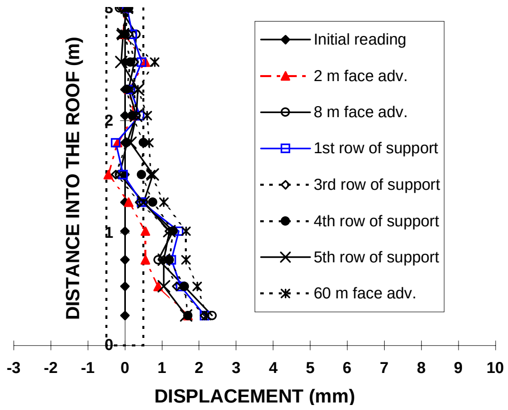
COLLIERY "D" – SITE 1

(CUT OUT DISTANCE 16 m)

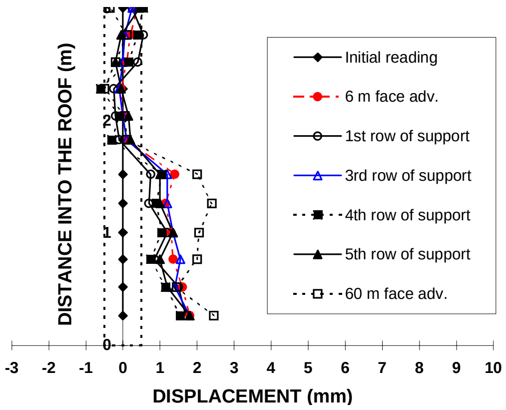
HOLE 1



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

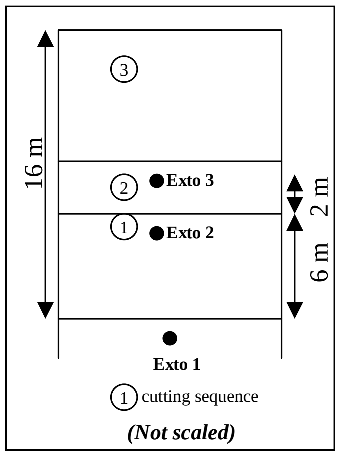


Figure 3-18 Colliery 'D' Site 1

3.8 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, which falls under Area 4 in the geotechnical areas analysis. The plan of the sections in Colliery 'E', where the experiment took place, is given in Appendix 1.

3.8.1 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 to mine No 4 Lower Seam in the Highveld Coalfield. There were no excessive horizontal stress indicators in the section. The pillar, roof and underground dimension control were excellent. The installation and performance of support was also excellent.

The road header experiment sequence was used in the experiment. 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 3-19. The summary of site performance in this site is given in Table 3-13.

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 3-13 Site performance Colliery 'E' Site 1

Colliery	"E"
Site	1
Coalfield	Highveld
Seam	No 4 Lower
Depth below surface (m)	178
Production method	CM & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	25
Bord width (m)	7.0
Mining height (m)	3.8
Safety factor of pillars	1.91
Imm. roof lithology	0.35 m thick coal/sandstone laminae overlain by 0.65 m thick siltstone, and thick gritstone
Support type	Re-bar
Bolt diameter (m)	20
Bolt length (m)	1.5
Support density(bolt/m^2)	0.24
Resin type	Slow & Fast
Number of resin capsules	2

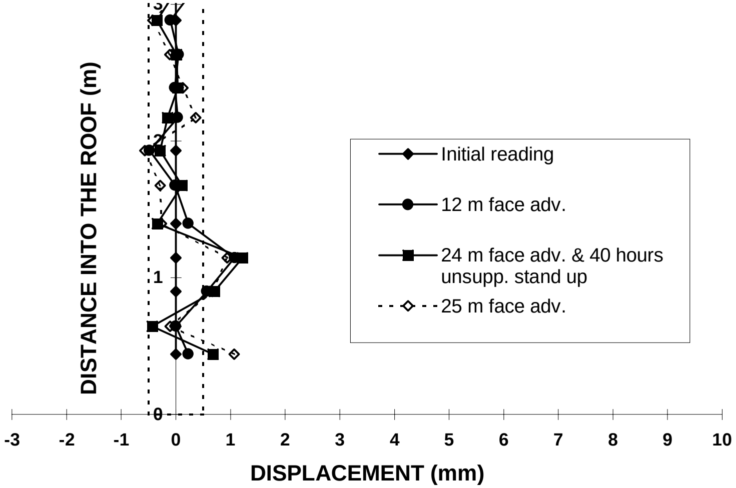
Resin capsule diameter (mm)	19
Cut out distance (m)	24
Geotechnical area	Area 4
Performance & remarks	A maximum of 1.0 mm was recorded in No 1 hole. No 2 showed 5.0 mm displacement, which was the largest amongst all the monitoring sites. Displacements in both holes took place 0.3 m into the roof at the coal/sandstone and siltstone interface.

HIGHVELD COALFIELD
No 4 Lower

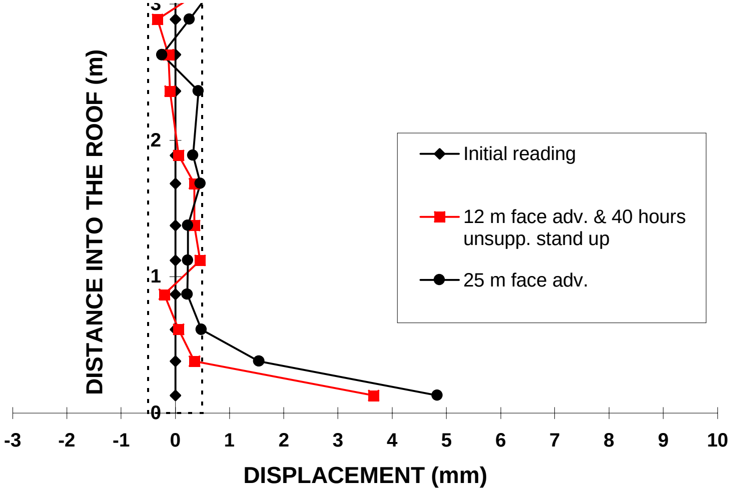
COLLIERY "E" - SITE 1

(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

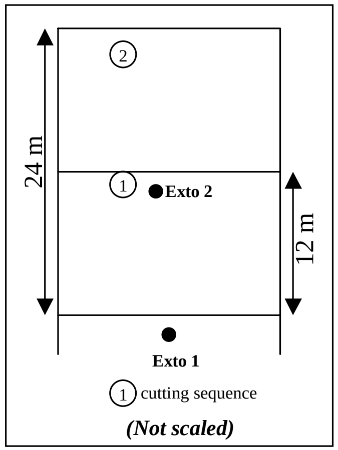


Figure 3-19 Colliery 'E' Site 1

3.9 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, which falls under Area 2 in the geotechnical areas analysis. The plan of the sections in Colliery 'F', where the experiment took place, is given in Appendix 1.

3.9.1 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 to mine No 2B Seam in the Vereeniging Coalfield. The pillar, roof and underground dimension control were excellent. The installation and performance of support were also excellent.

The CM experiment sequence was used in the experiment. 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 3-20. The summary of site performance in this site is given in Table 3-14.

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. This was expected because of the high density of support used in the section (0.77 bolts/m^2).

Table 3-14 Site performance Colliery 'F' Site 1

Colliery	"F"
Site	1
Coalfield	Vereeniging
Seam	No 2B
Depth below surface (m)	44.1
Production method	CM & shuttle cars
Type	Primary bord and pillar production
Pillar width (m)	28.8
Bord width (m)	5.2
Mining height (m)	2.6
Safety factor of pillars	12.3
Imm. roof lithology	1.41 m thick coal/shale layers overlain by 1.1 m thick coal/shale
Support type	Re-bar
Bolt diameter (m)	20
Bolt length (m)	1.8
Support density(bolt/m^2)	0.77
Resin type	Slow & Fast
Number of resin capsules	3

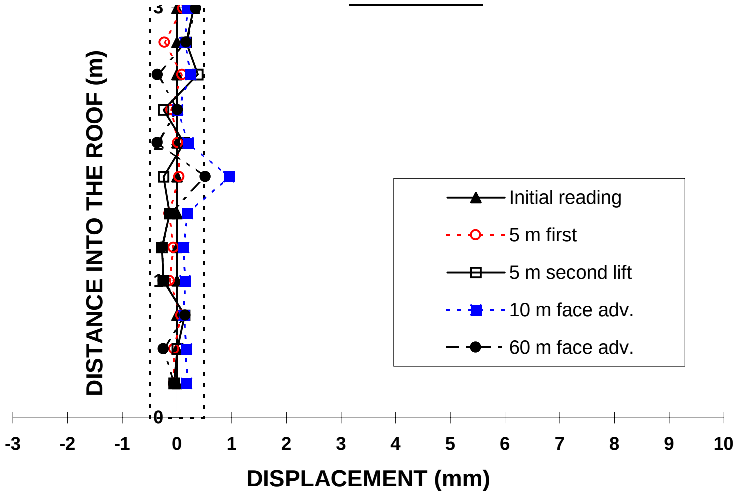
Resin capsule diameter (mm)	19
Cut out distance (m)	10
Geotechnical area	Area 3
Performance & remarks	Both holes were stable throughout the experiment, and no displacements were recorded in either hole.

VEREENIGING COALFIELD
No 2B Seam

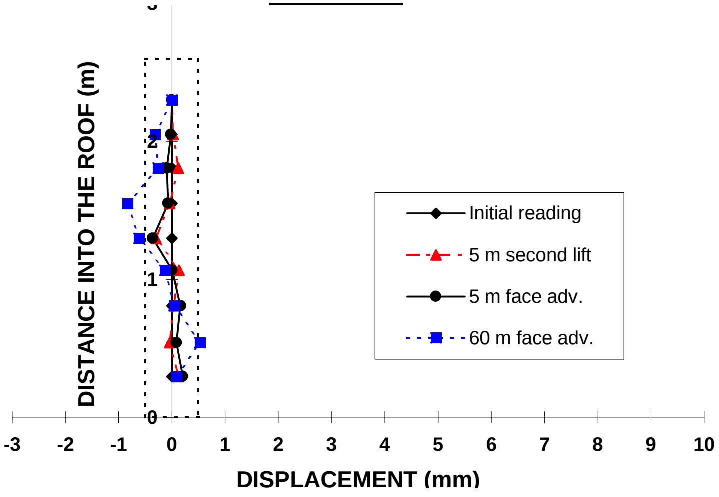
COLLIERY “F” - SITE 1

(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

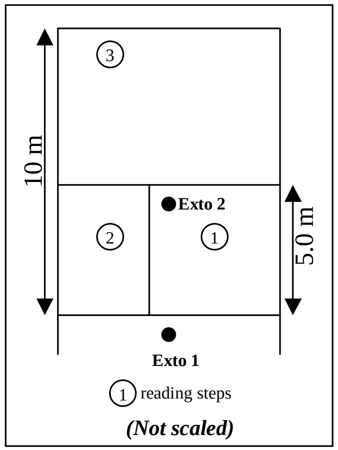


Figure 3-20 Colliery 'F' Site 1

3.10 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 was as stable as for the significantly smaller 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 3-15 and Table 3-16. 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.

The relationships between the maximum dilation and the support density, roof lithology, bord width and cut out distance are shown from Figure 3-21 to Figure 3-24. 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 3-25. 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 can assist mining personnel in selecting the correct support system and bolt length.

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. Therefore, it is recommended that the support should be installed immediately after cutting. Also, as recommended by Bauer (1993), the length of unsupported cut outs that will be left standing over extended periods (weekends and holidays) should not exceed the bord width.

Two attempts were made to identify the effect of roofbolting and tensioning of roofbolts. 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 roofbolts. The installation of the bolts appeared to have little if any effect on the bed separation already evident within the bolt horizon as can be seen in Figure 3-18. 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 roofbolts has no remedial effects on pre-existing openings within the bolt horizon.

Although the effect of the installation of pre-tensioned roofbolts was specifically monitored in only one site, in general in the monitoring holes, where the displacements occurred within the roofbolt 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 roofbolts 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 conventional gravity loaded beam (or plate) theory. As the length of the roadways exceeded twice their width in all cases, the valid simplification of using beam formulae was introduced. The maximum deflection formula was used in the form:

$$\eta = \frac{\gamma B^4}{32Et^2} \tag{3.1}$$

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 3-26. 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 actual roof that was monitored consisted of plates rather than beams. 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 3-15 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	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	Sandstone	0.54
A	3	0	0.44	6.1	16	0	Sandstone & grit	1.13
B	1	0	0.15	6.5	24	0	Coal	1
B	2	0	0.15	6.7	31	0	Coal	0.85
B	3	0	0.24	6.2	12	0	Sandstone	0.27
C	1	0	0.14	7.03	12	0	Mudstone/coal	0.94
C	2	0.5	0.15	6.7	12	1	Mudstone/coal/sandstone	0.85
C	3	0	0.16	6.1	18	0	Shale/coal	0.65
D	1	1.8	0.23	6.6	16	2	Coal/shaly coal	1.83
E	1	0.5	0.24	7	24	1	Coal/sandstone	0.35
F	1	0	0.77	5.2	10	0	Coal/shale	0.54

Table 3-16 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	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	Sandstone & grit	1.13
B	1	0	0.15	6.5	24	0	Coal	1
B	2	0	0.15	6.7	31	0	Coal	0.85
B	3	0	0.24	6.2	12	0	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	Mudstone/coal/sandstone	0.85
C	3	0.7	0.16	6.1	18	1	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	Coal/sandstone	0.35
F	1	0	0.77	5.2	10	0	Coal/shale	0.54

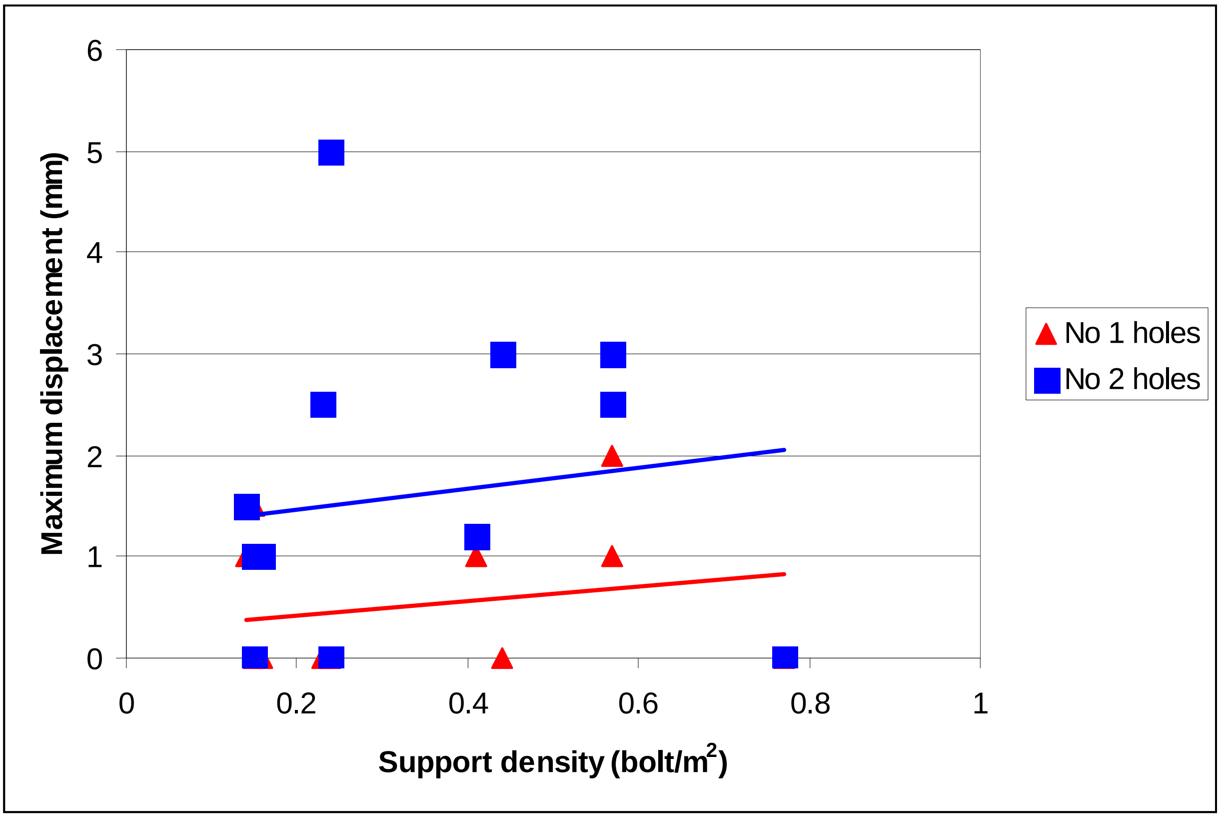


Figure 3-21 The relationship between the support density and total displacement

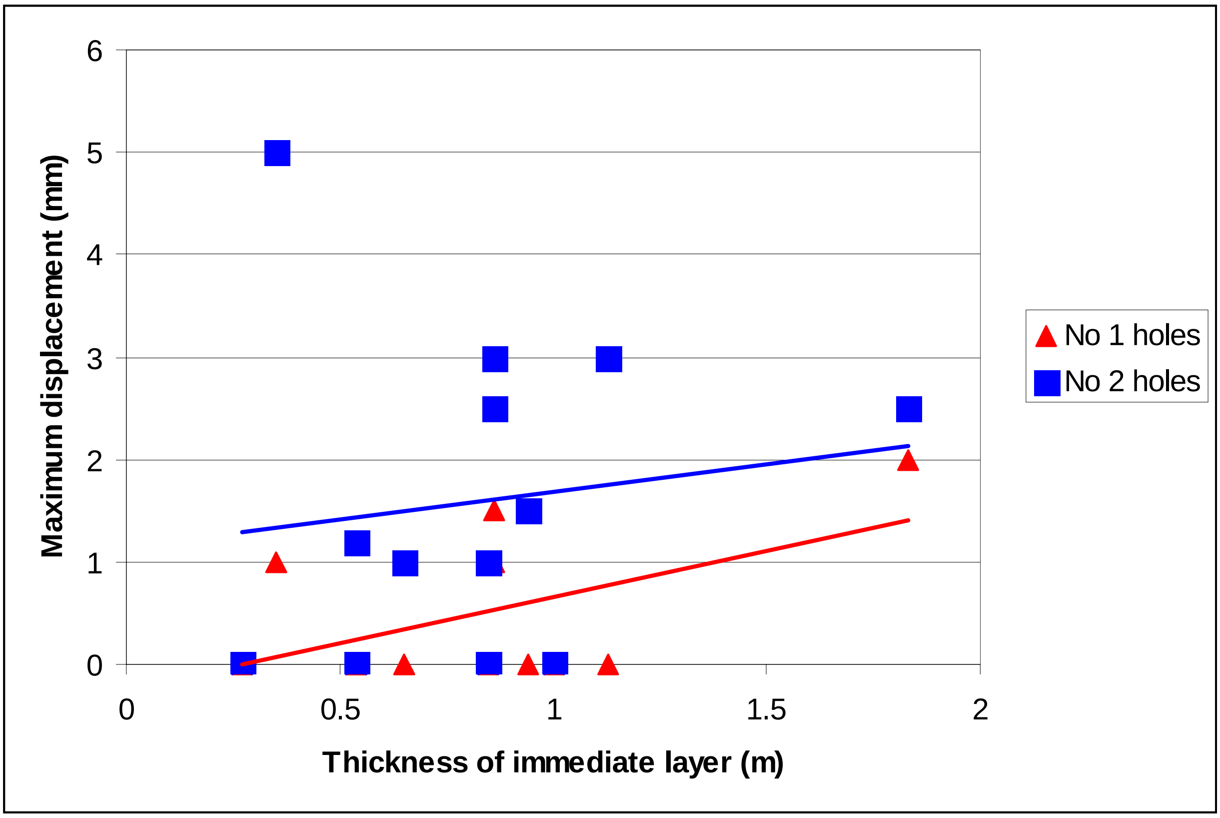


Figure 3-22 The relationship between the thickness of the immediate layer and total displacement

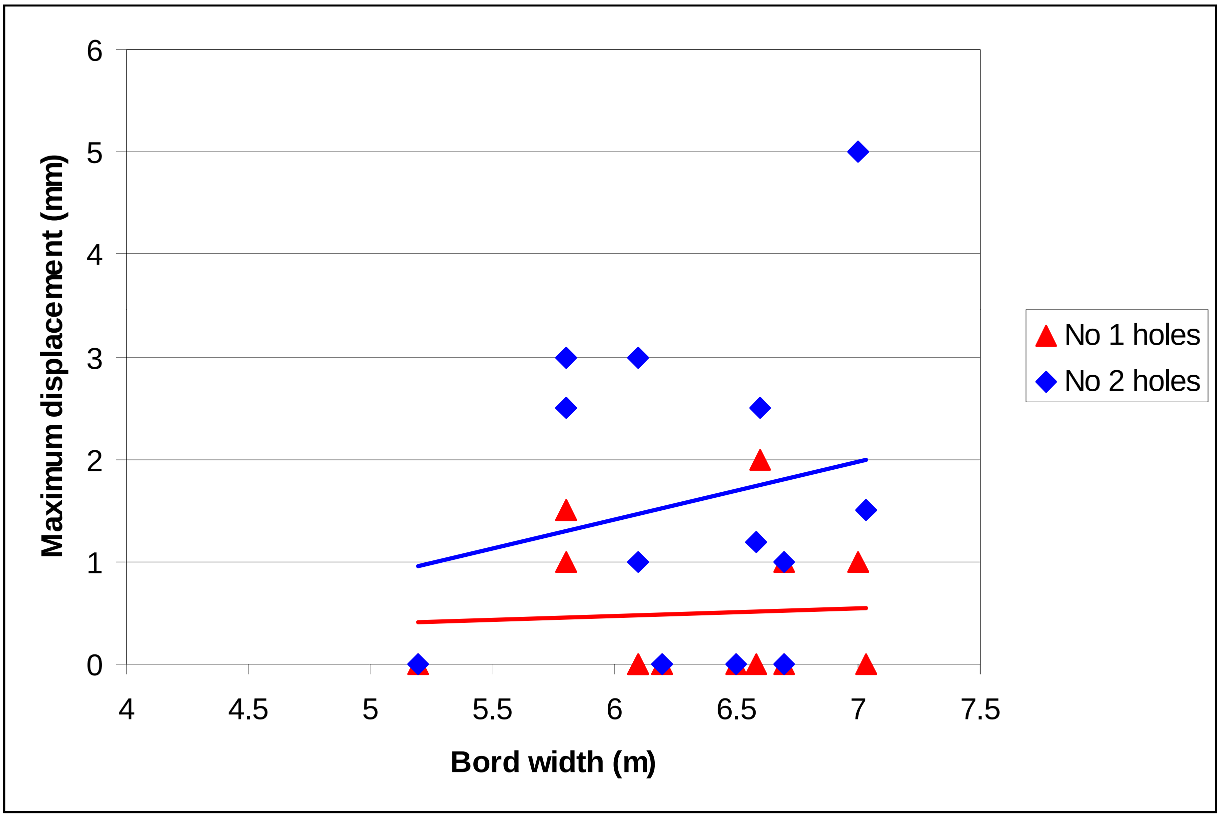


Figure 3-23 The relationship between the bord with and total displacement

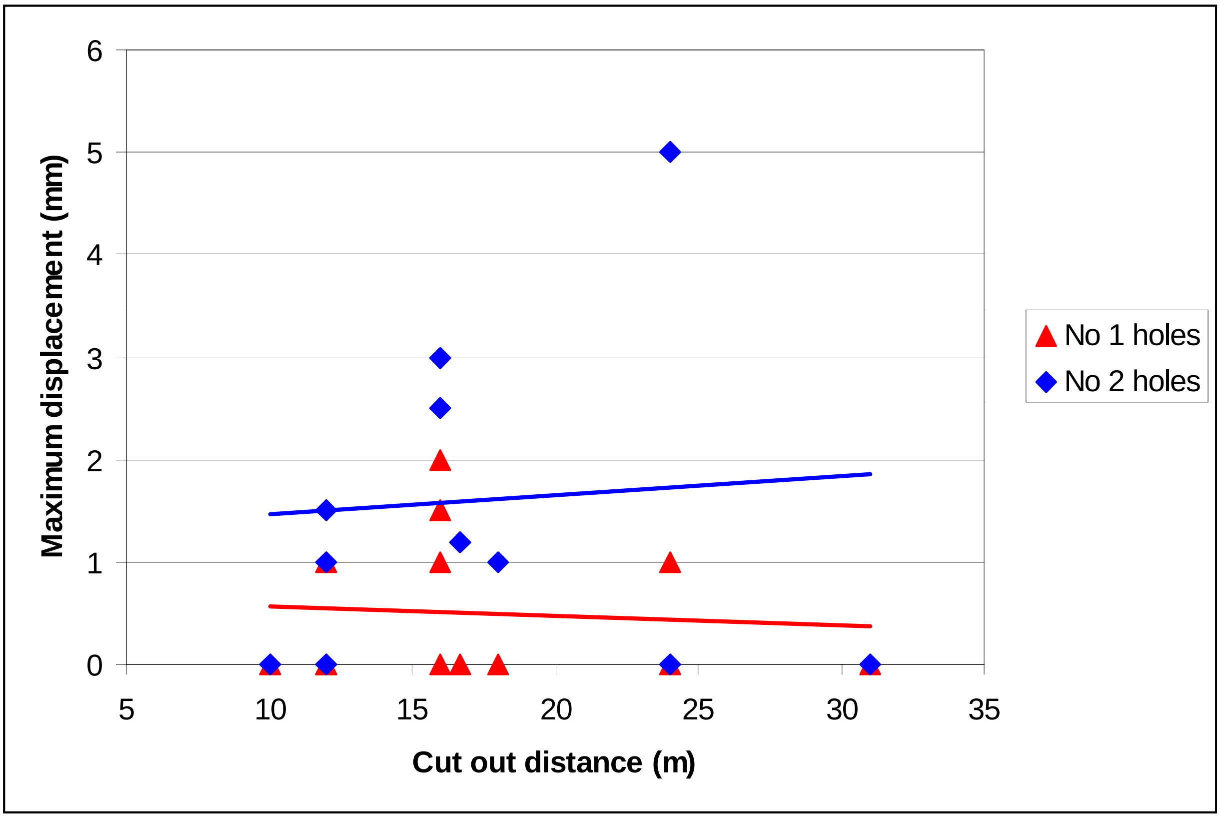


Figure 3-24 The relationship between the cut out distance and total displacement

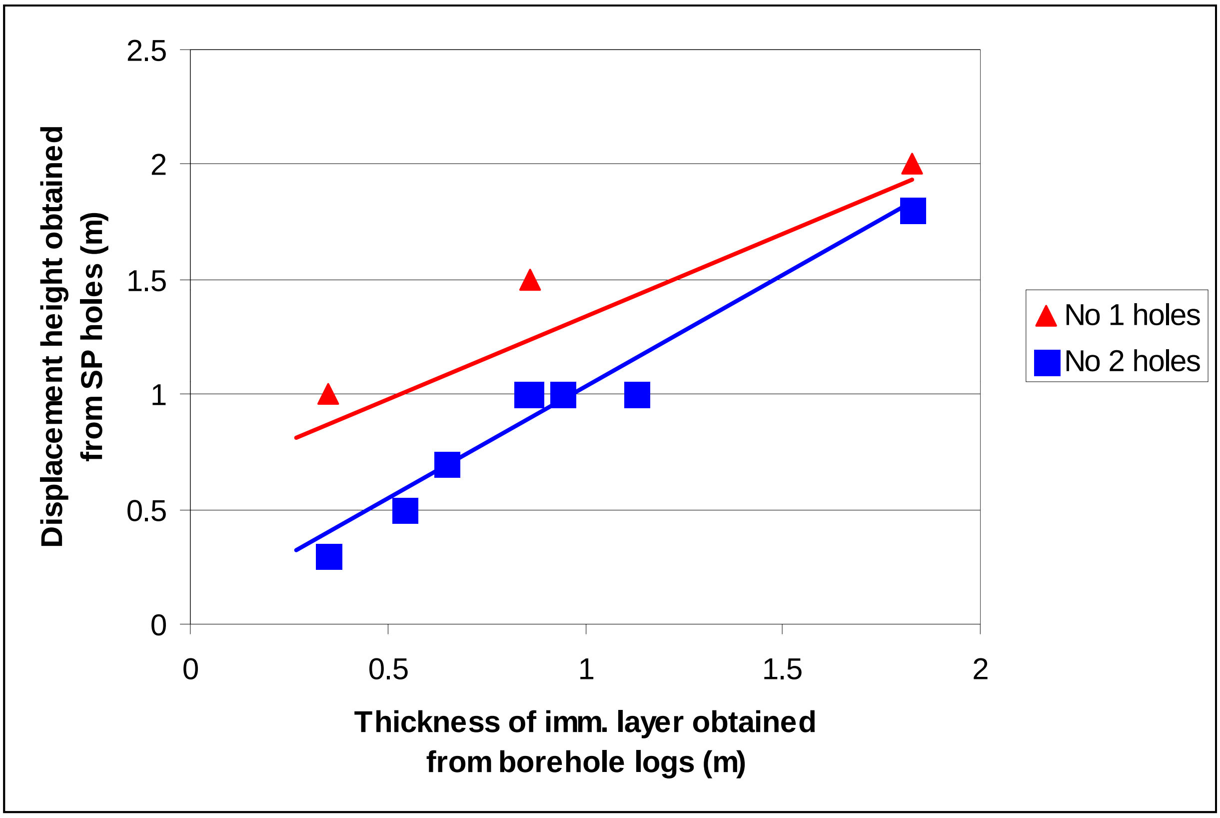


Figure 3-25 The relationship between the thickness of the immediate layer obtained from the borehole logs and height of the displacement obtained from underground sites where some degree of dilation was recorded

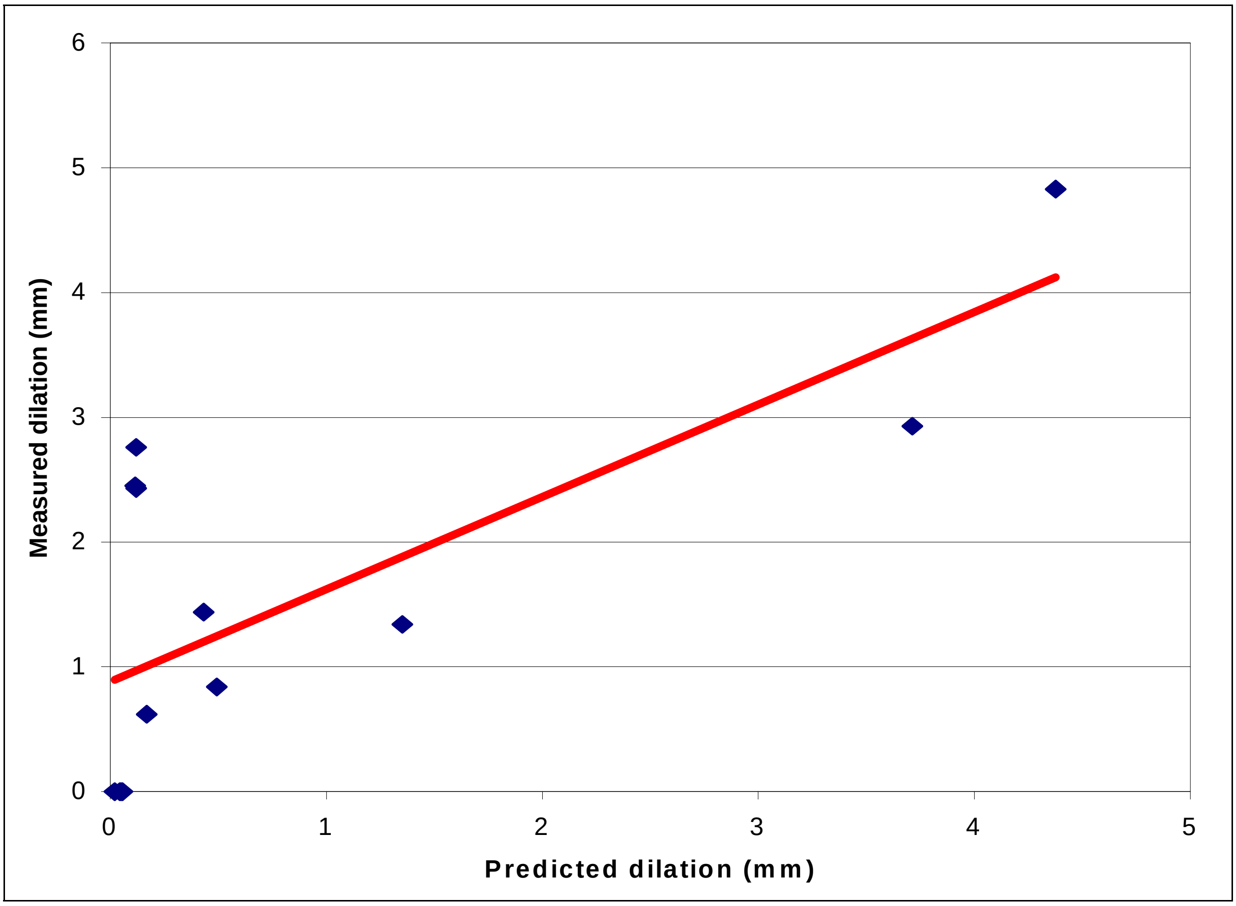


Figure 3-26 Relationship between measured and predicted dilation

3.11 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 derived. To investigate these further, a numerical modelling code was added to the research program. The three dimensional boundary element code Map3D was used in the analysis. The basic three dimensional model that was used is shown in Figure 3-27.

The first variable that was investigated was bord width, Figure 3-28. The trend is clear – the greater the bord width, the greater the roof deflection. This is not an unexpected result.

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 middle 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 3-29. 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.

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 3-30. This trend was also apparent in the monitoring, indicated by the good correlation between measured deflections and the beam predictions.

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 3-27 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 3-31. As can be seen from this figure, the properties of the immediate layer have a major effect on the deformations in the roof. The lower the modulus, the more deflection is predicted, again confirmed by the measurements.

Dilations in the roof as the mining steps take place were also investigated using Map3D. The results are shown in Figure 3-32. A 6.0 m bord width was modelled with 6.0 m mining steps. This figure indicates that a significant portion of the displacements (80 per cent) takes place within the first 12 m of the face advance. This is an important trend, as it was seen underground that very little additional deflection occurred once the face had advanced beyond a distance equal to twice the bord width.

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 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.

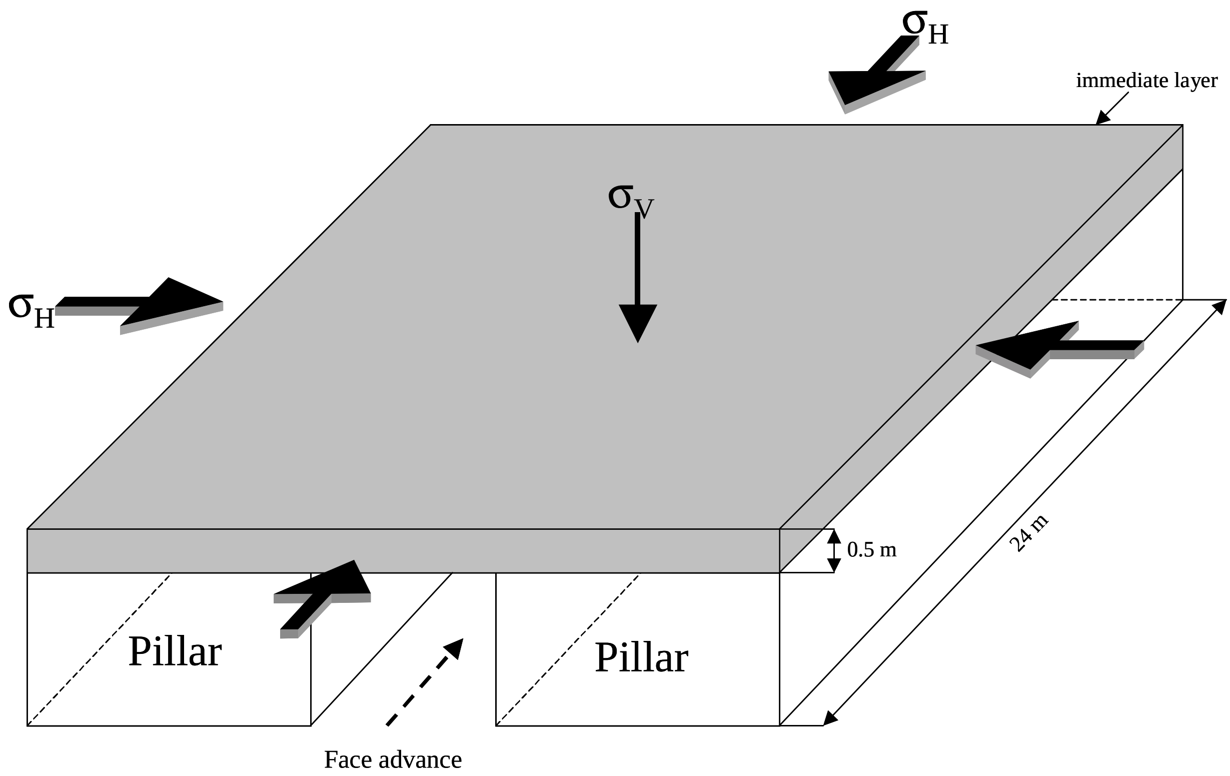


Figure 3-27 The basic MAP3D Model that was used in the numerical modelling analysis

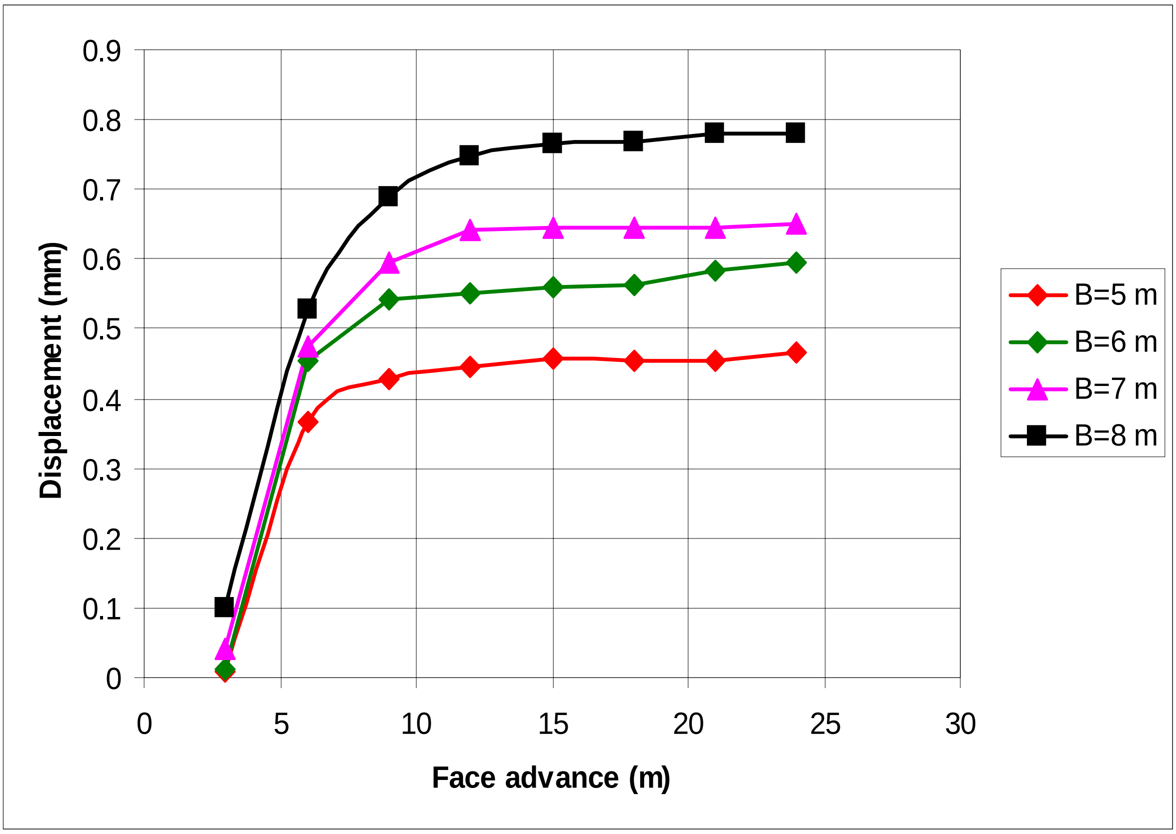


Figure 3-28 Effect of bord with on dilation

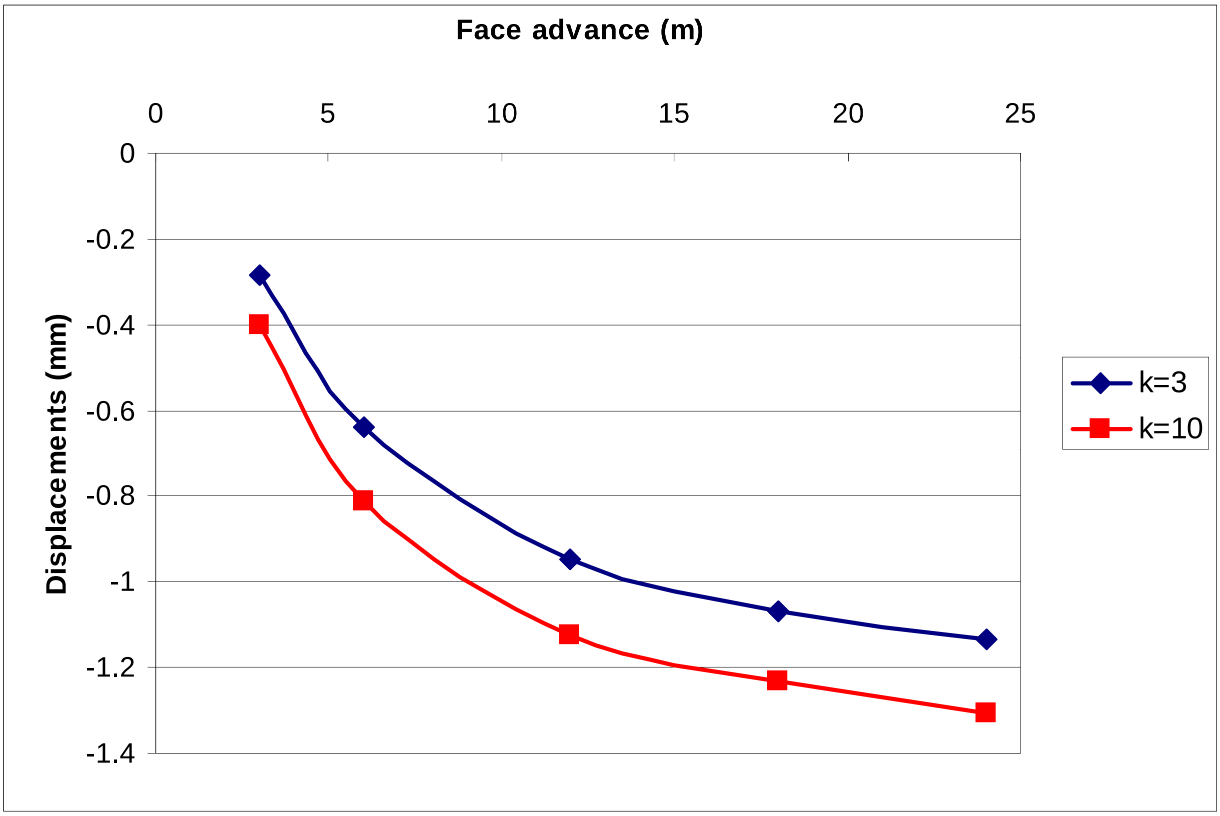


Figure 3-29 Effect of k-ratio on roof deformations

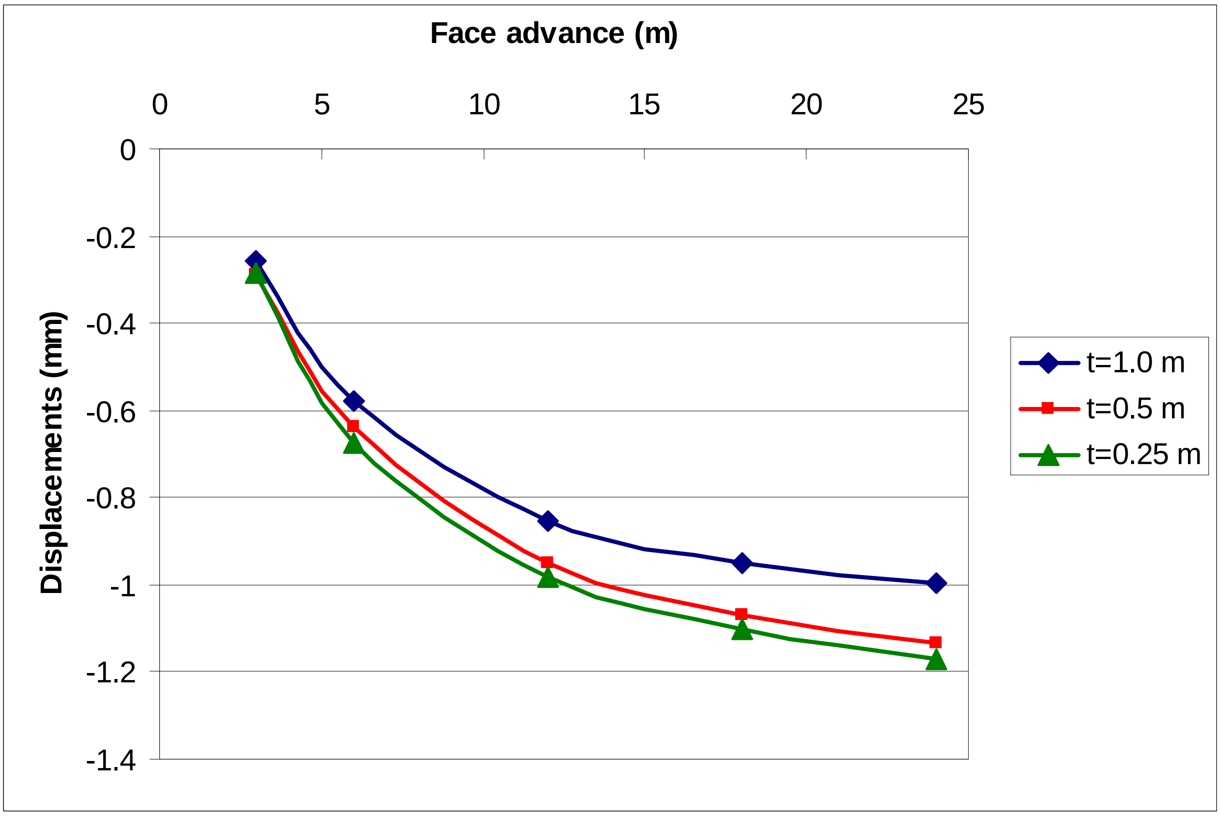


Figure 3-30 Effect of the thickness of the immediate layer on roof deformations

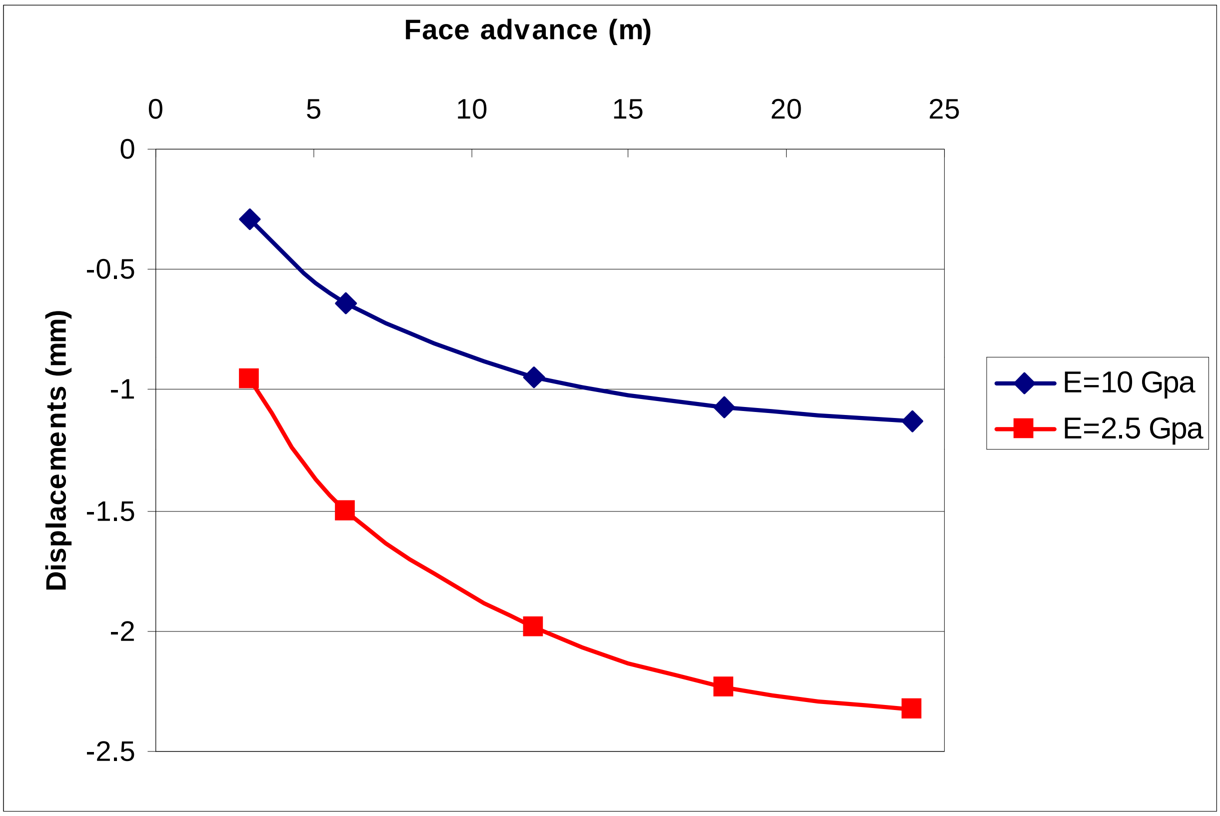


Figure 3-31 Effect of the strength of the immediate layer on roof deformations

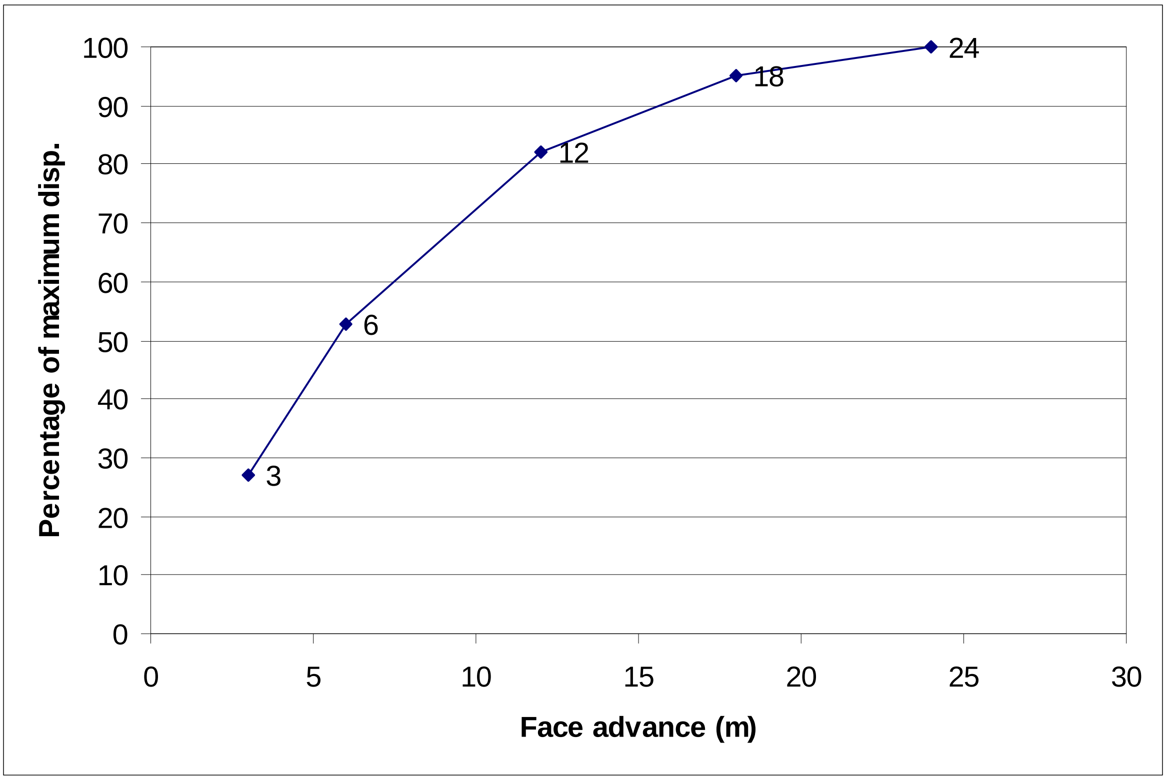


Figure 3-32 Effect of face advance in a 6.0 m bord width, obtained from numerical modelling

4.0 Conclusions and recommendations for future research

4.1 Conclusions

The recently introduced restrictions on cut out distances sparked major debate in the coal mining industry. There were several reasons for the restrictions, roof stability being one of them. The others included methane and dust control and reducing the total area of unsupported roof in an underground section at any given time. This investigation covered the roof stability aspect only. While it thus did not address the overall issue, it attempted to quantify the effect of cut out distance on roof failure.

4.1.1 The literature survey indicated no significant negative results from extended cut out distance

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, mainly Bauer (several references, see list at back of report) seems to indicate that extending the cut out distance in the USA had little effect on roof stability, mainly because operators tended not to reduce the cut out distance under adverse roof conditions and only extend it if roof conditions were good.

4.1.2 The research methodology had to be adapted to suit practical requirements.

The ideal research methodology from a cold 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.

4.1.3 Extended cut out distance does not contribute significantly to roof deflection.

The most important conclusion from the 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 the typical South African condition, with bord widths in the range of 5.5 m to 7.2 m, it implies that roof stability would not be adversely affected by advancing further than 11 m to 14 m. Whatever the roof deflection was going to occur, 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.

From the investigation it appears that it is unnecessary to place this onerous restriction on the underground mines as a general rule. If roof failure had occurred within the 12 m cut out distance, it may have been warranted, but in not a single case did roof failure occur. This indicates that in general, the roof is strong enough to withstand the effects of the deflections that occur during face advance.

4.1.4 The effects of time could only be studied for a limited time period.

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 the minute fractures resulting from the additional deflection may change the roof behaviour and eventually result in failure.

4.1.5 The role of roofbolting is to prevent further dilation.

Results from one sonic probe monitoring hole showed that roofbolting had no remedial effect on roof deformations. Although the effect of roofbolting was specifically monitored in only one sonic probe monitoring hole, 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 roofbolts. This indicates that the roofbolt 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 roofbolting 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).

4.1.6 The roof behaves like a gravity loaded composite beam in which roof lithology and road width play the dominant roles.

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 plates 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 area (three of the sites exhibited obvious signs of high horizontal stress), it appears not to have 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.

The 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.

4.1.7 Exemption from the 12 m cut out distance

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 a 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 assessments plans.

4.1.8 The major limitation to the conclusions is that the probability of encountering unexpected discontinuities in extended cut outs has not been addressed.

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 gremlin." 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.

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. It needs to be stressed that the investigation showed that no significant effect on roof stability in general will result from extending cut out distances.

4.1.9 Technology transfer

Findings of this project were presented to rock engineer practitioners and mining personnel at various mining houses.

4.2 Recommendations for future research

The roof behaviour in South African collieries has been investigated over many years. In general, the results showed that the elastic deformations in the roof are relatively low compared to other major coal producing countries with comparable conditions, namely Australia and the USA. Although the deformations are low, the fall of ground fatality rate is still very high compared to those countries (Mark, 1999).

4.2.1 Roofbolting mechanism

It is recommended that a comprehensive study on the roofbolting mechanism should be conducted. This study should include the material properties of roofbolts that are being used in South African collieries, the effect of pre-tension and general principals of roofbolt design specifically for South African conditions. It has been shown previously that the majority of fatalities are caused by relatively small falls between bolts, which indicates that the spacing of bolts should perhaps receive more attention.

4.2.2 Properties of roof material

This study indicated that the composition of the roof, specifically the thicknesses of the lithological units and their mechanical properties, play the dominant role in roof deflection which is the pre-cursor to failure. Current analysis is done using standard beam theory, which in turn depends on interalia the following assumptions:

i) Each stratum is homogeneous, elastic and isotropic,
ii) There is no or weak bonding between the strata
iii) Each stratum is subjected to uniform loading in both the transverse (due to self weight) and axial (due to horizontal stress) directions simultaneously
iv) When the upper stratum loads onto the lower stratum, the deflections of the two strata are equal at each point along the roof span and
  a) The upper beam loads the lower beam with a uniform load per unit length of beam,
  b) The lower beam supports the upper beam with an equal load per unit length.

These assumptions and their effects on roof behaviour require confirmation and refinement. Due to a lack of data, generalised mechanical properties are usually used in analyses. Research is required to determine more specific properties of roof strata.

4.2.3 Numerical modelling

The numerical modelling indicated the usefulness of modelling to determine trends even though detailed site specific modelling was not done. More detailed modelling will yield more useful, site specific results incorporating the effects of increased levels of high horizontal stress. Research is required to provide guidelines with respect to modelling techniques and even the types of model to be used, i.e. continuum vs. discontinuum, linear vs. non linear, etc.

4.2.4 Early detection of discontinuities and control over dimensions

During the underground experiments it was observed that there are two major problems within the mining cycle, which require detailed investigations. These are detection of geological discontinuities and underground control of dimensions (bord width, pillar width, mining height and direction of mining). These should be investigated and a probabilistic design should be established in order to take into account the variations in mining dimensions and effect of geological discontinuities.

5.0 References

Bauer, E. R., Chekan, G. J. and Steiner, L. J., 1997. Stability evaluation of extended cut mining in underground coal mines. NYRockas'97, 36th U.S. Rock Mech. Symp., Linking Science to Rock engineering, June 29 - July 2, New York, NY, Int. J. Rock Mech. & Min. Sci. 34 : 3-4, Paper No. 302, 10 pp.

Bauer, E. R., Pappas, D. M. and Listak, J. M., 1993. Ground control safety analysis of extended cut mining. BuMines IC 9372, 24 pp.

Bauer, E. R., Steiner, L. J. and Hamrick, C. A., 1995. Extended cut mining and worker safety in underground coal mines. SME Preprint 95-60, 7 pp.

Bauer, E. R. 1998. The impact of extended depth-of-cut mining on coal mine ground control and worker safety. PhD Thesis, The Pennsylvania State University, Department of Mineral Engineering, August.

Bieniawski, Z. T., 1984. Rock mechanics design in mining and tunnelling. A.A. Balkema, Rotterdam, 272 pp.

Bieniawski, Z. T. 1989. Engineering rock mass classifications. Wiley, NY, 251 pp.

Buddery, P. (1989). Fundamental principles of roofbolting in South African Collieries. Advances in rock mechanics in underground coal mining. SANGORM Symposium, September.

Campbell, J. A. L., Riffey, R. L. and Risner, P. D. 1989. Deep cut remote control mining. Proceedings of AIME Annual Meeting, Denver, CO, September, pp. 101-112.

Canbulat, I. and Jack, B. W., 1998. Review of current design methodologies to improve the safety of roof support systems, particularly in the face area, in collieries. SIMRAC Final Project Report, COL 328. 212 pp.

Davis, H., 1977. Hand held radio transmitter runs continuous miner. Coal Age, v. 82, No. 8, August, pp. 66-68.

Divers, E. F. , Jayaraman, N. and Custer, J., 1982. Evaluation of a combined face ventilation system used with remotely operated mining machine. BuMines IC 8899, 7 pp.

Galvin, J. Personnel communication.

Grau, R. H. and Bauer, E. R. 1997. Ground control worker safety during extended cut mining. Proceedings of the 16th International Conf. on Ground Control in Mining, West Virginia Univ., Morgantown, WV, August 5-7, pp. 283-288.

Guidance on the use of rockbolts to support roadways in coal mines. Deep Mines Coal Industry Advisory Committee, Health and Safety Commission, 1996.

Guideline for the compilation of a mandatory code of practice to combat rockfall accidents in collieries. Mine Health and Safety Inspectorate, Department of Minerals and Energy, 1996.

Jayaraman, N. I. Volkwein, J. C. Divers, E. F. and Babbit, C A., 1987. Enhancing continuous miner production with scrubbers and exhaust face ventilation. Proceedings, 3rd International Conf. on Innovative Mining Systems, Rolla, MO, November 2-4, pp. 116-128.

King, R. H. and Frantz, R. L., 1977. Health and safety aspects of automated, remote controlled, continuous underground coal mining in the U.S.A. Paper presented at the International Conf. on Remote Controlled and Monitoring in Mining, Birmingham, England, October 11-13, 17 pp.

Lindsay, G.C., 1973. Remote control mining comes of age. Coal Mining and Processing, v. 10, No 9, September, pp. 30-44, 41.

Love, A. C. and Randolph, R. F., 1991. Visibility and task demands for remote control continuous mining machine operators. Proceedings, 35th Annual Meeting of the Human Factors Society, San Francisco, CA, v. 1, September 2-6, pp. 872-879.

Love, A. C. and Randolph, R. F., 1992. Radio remote control continuous miner operator positioning. Chapter 34, Proceedings of the Symposium, New Technology in Mine Health and Safety, A. Wahab Khair, Ed., SME Annual Meeting, Phoenix, AZ, February 24-27, pp. 357-360.

Mark, C., 1999. Application of coal mine roof rating (CMRR) to extended cuts. Mining Engineering, pp 52-56, April.

Mark, C., 1999. Ground control in South African coal mines - A US perspective. Proceedings of 18th International Conference on Ground Control. Morgantown, WV. August.

Mine Health and Safety Act, 1996. Parliament of South Africa No. 29.

Molinda, G. M. and Mark, C. 1994. The coal mine roof rating (CMRR): A practical rock mass classification for coal mines. Proceedings of the 12th International Conf. on Ground Control in Mining, West Virginia Univ., Morgantown, WV, August 3-5, pp. 92-103.

Radcliffe, D. E. and Raymond, M. S. 1980. Long-term response of a coal mine roof to time lapse between exposure and support. Bureau of Mines Coal Mine Health and Safety Program. Technical Progress Report. No. 110, January.

Randolph, R. F., 1992a. Human factors of high technology mining. Chapter 28, Proceedings of the Symposium, New Technology in Mine Health and Safety, A. Wahab Khair, Ed., SME Annual Meeting, Phoenix, AZ, February 24-27, pp. 287-295.

Sanders, M. S. and Kelly, G., 1981. Visual attention locations for operating continuous miners, shuttle cars and scoops. USBM Contract J0387213, BuMines OFR 29 (1) - 82, 140 pp.

Taylor, C. D., Goodman, G. V. R. and Vincze, T., 1992. Extended cut face ventilation for remotely controlled and automated mining systems. Chapter 1, Proceedings of the Symposium, New Technology in Mine Health and Safety, A. Wahab Khair, Ed., SME Annual Meeting, Phoenix, AZ, February 24-27, pp. 3-11.

U.S. Code of Federal Regulations, 1994a. Title 30 - Mineral Resources; Chapter I - Mine Safety Health Administration, Department of Labour; Subchapter O - Coal Mine Safety And Health; Part 75 - Mandatory Safety Standards - Underground Coal Mines; Subpart C - Roof Support; July 1, pp. 511-520.

Van der Merwe, J. N. (1995). Practical coal mining strata control. A guide for managers. Johannesburg, South Africa, November.

Volkwein, J. C., Thimons, E. D. and Halfinger, G. 1985. Extended advance of continuous miner successfully ventilated with a scrubber in a blowing section mine ventilation. Proceedings of the 2nd U.S. Mine Ventilation Symposium, Univ. of Nevada-Reno, Reno, NV, September 23-25, pp. 741-745.

Warner, E. M., 1973a. Cable and wireless remote control of continuous miners. Paper presented at the Mining Convention of The American Mining Congress, Denver, CO, September 9-12, 14 pp.

Appendix 1

Section plans where experiments took place

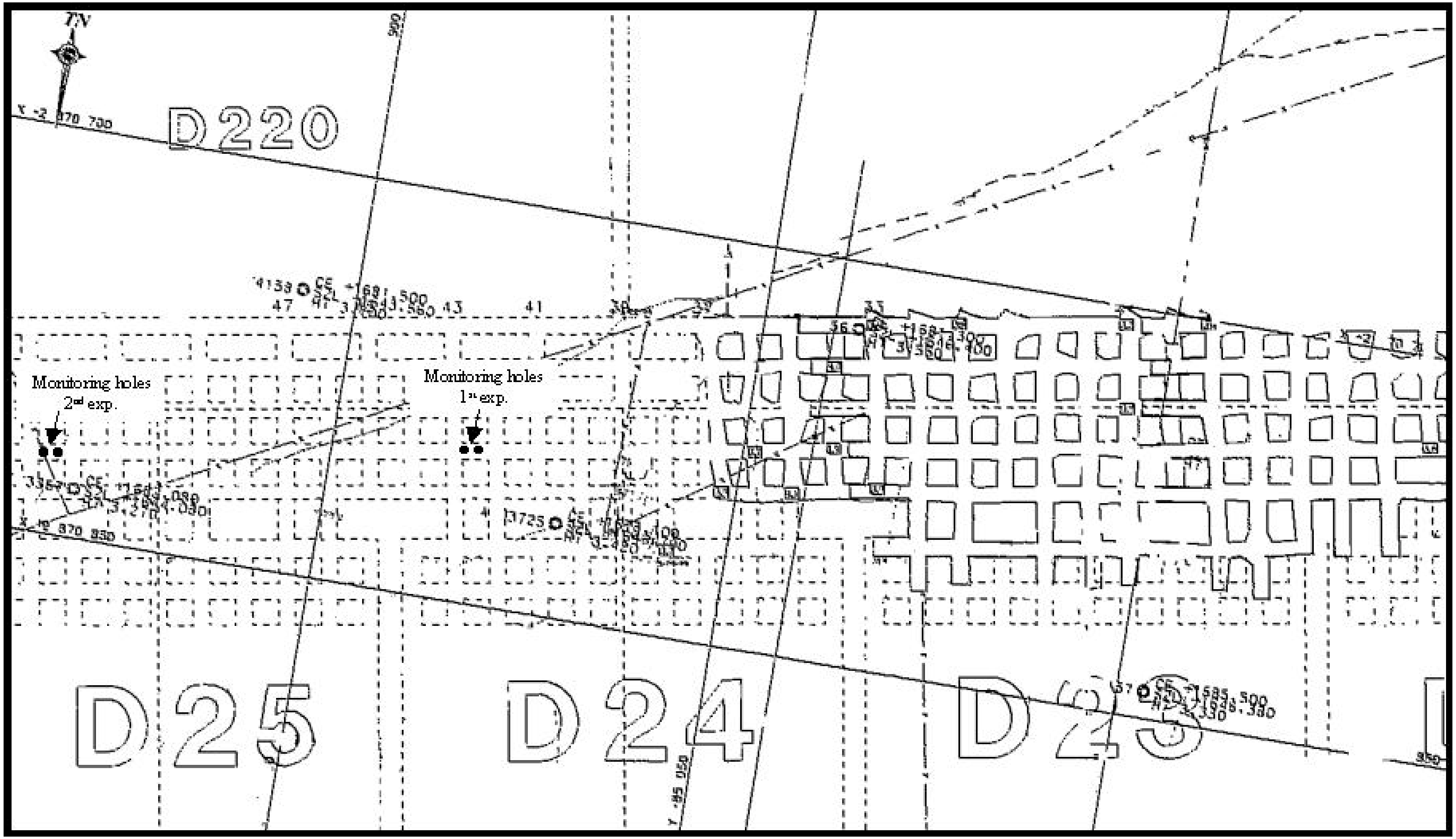


Figure A-1. Section plan Colliery 'A' Site 1 (not scaled)

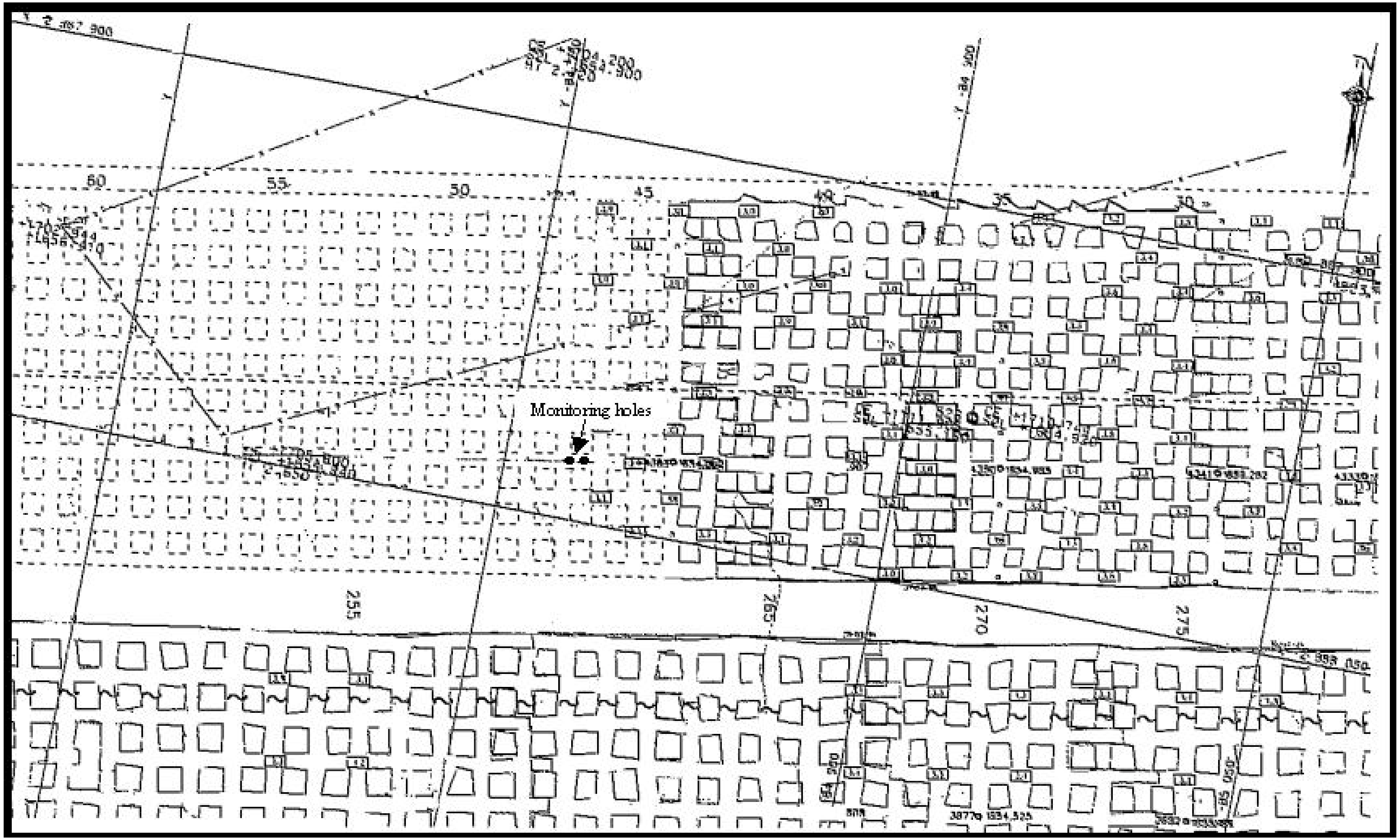


Figure A-2. Section plan Colliery 'A' Site 2 (not scaled)

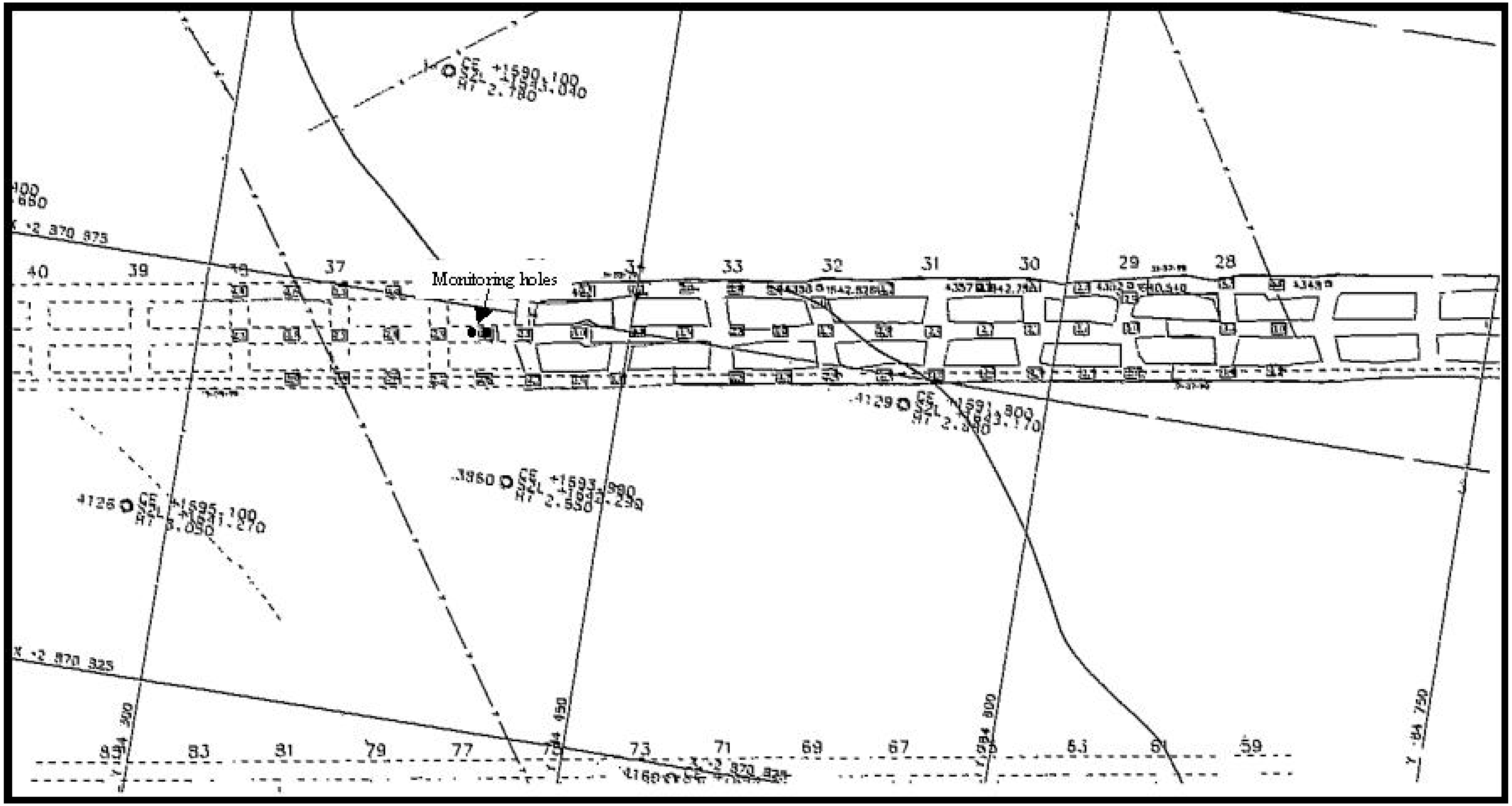


Figure A-3. Section plan Colliery 'A' Site 3 (not scaled)

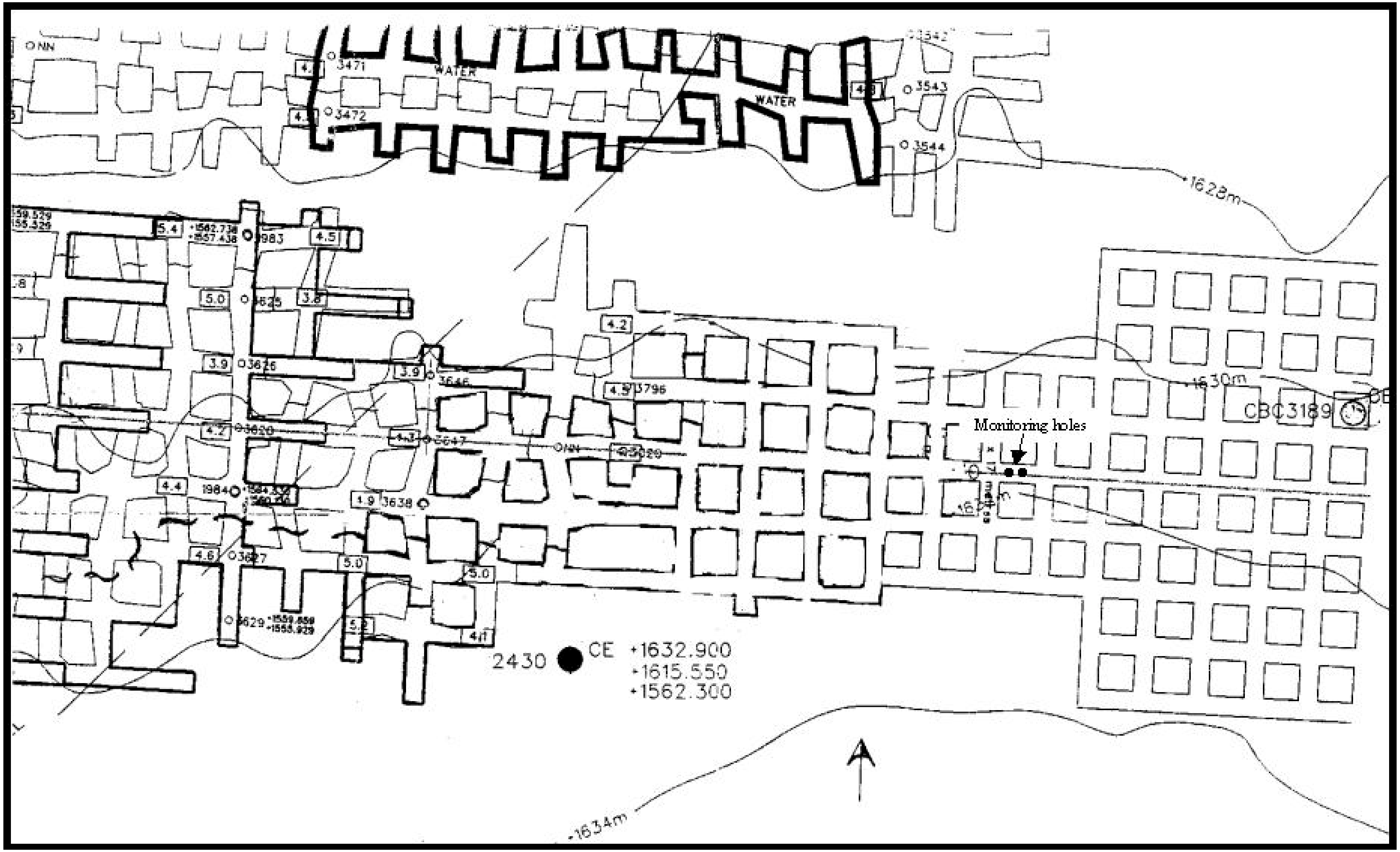


Figure A-4. Section plan Colliery 'B' Site 1 (not scaled)

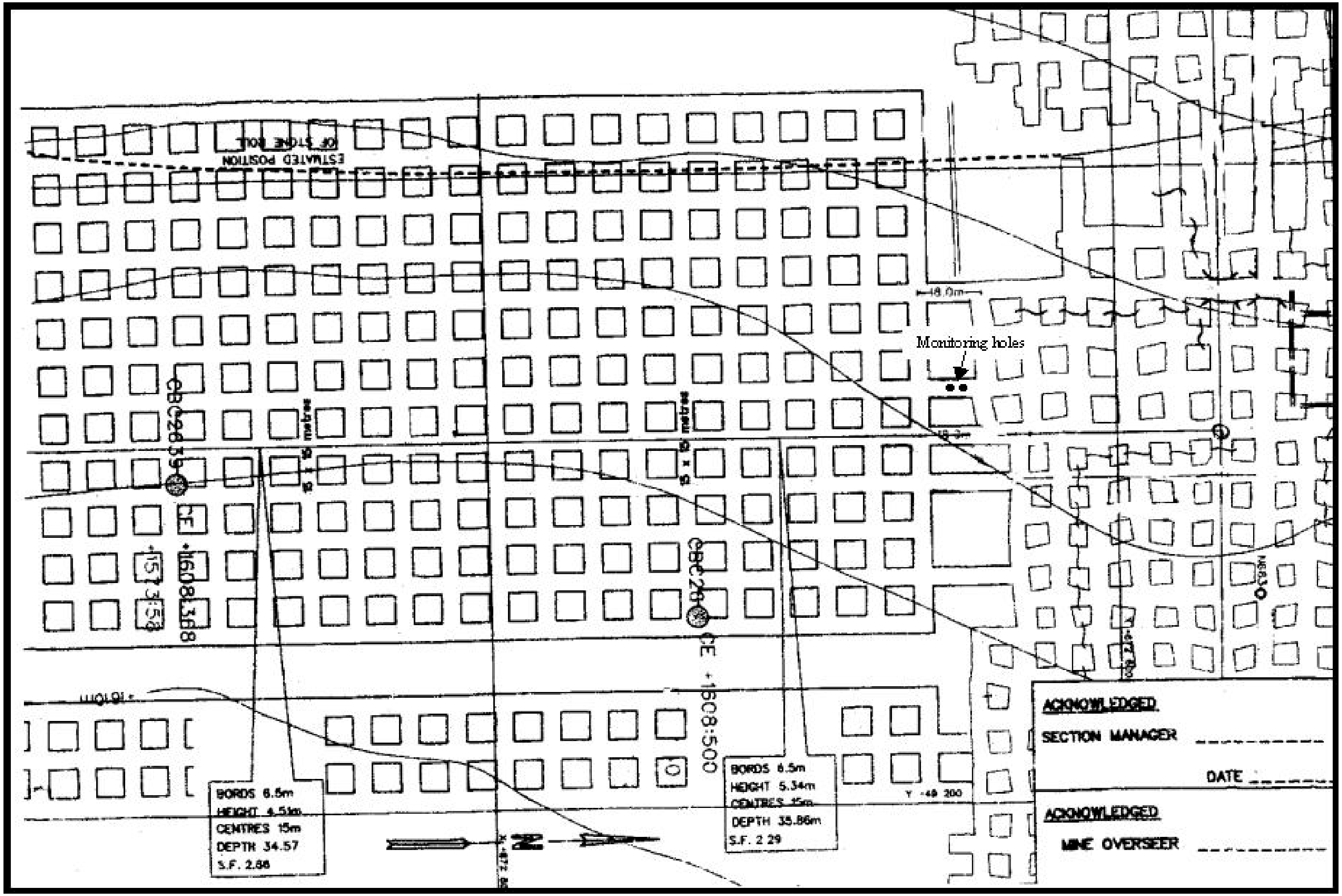


Figure A-5. Section plan Colliery 'B' Site 2 (not scaled)

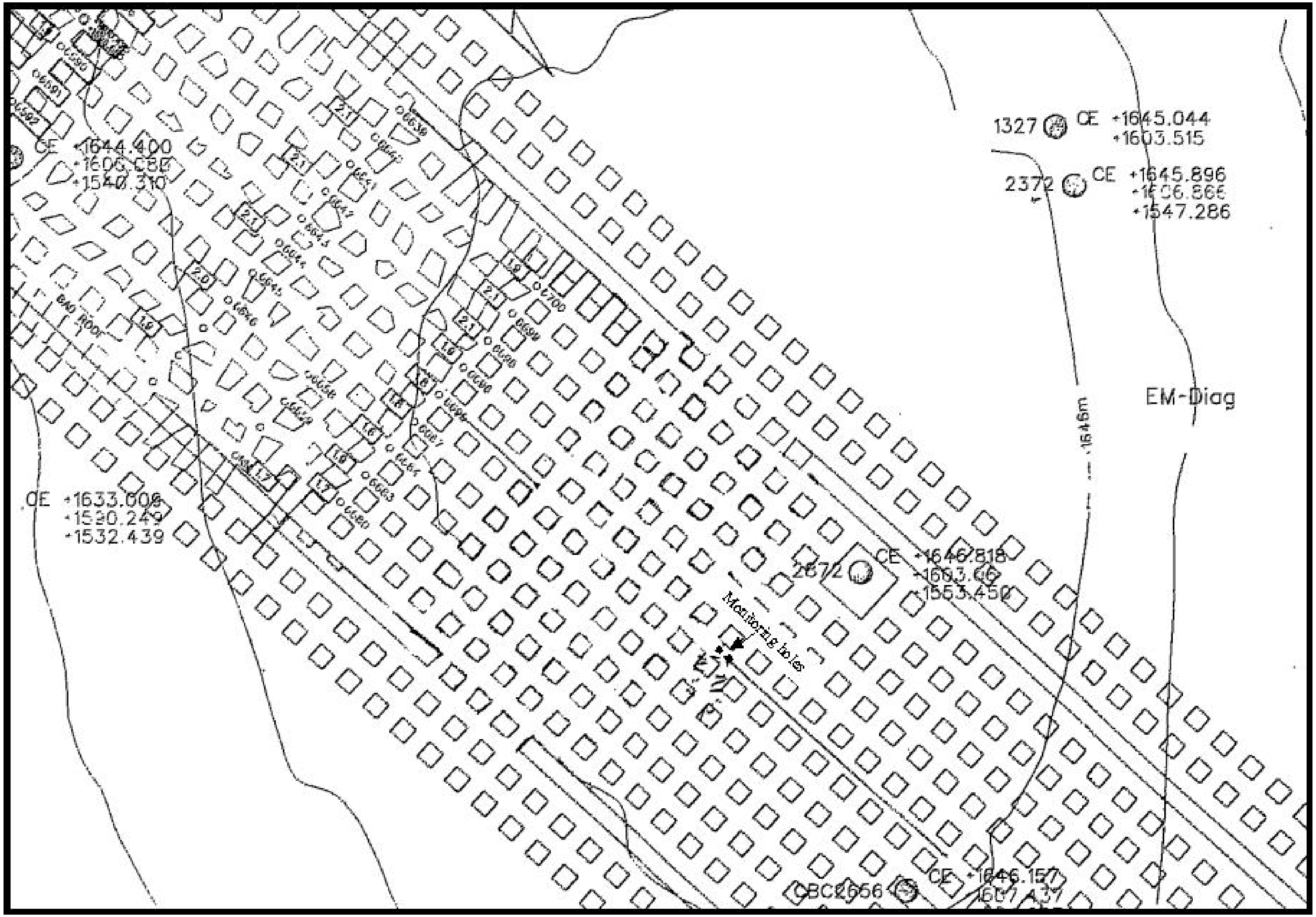


Figure A-6. Section plan Colliery 'B' Site 3 (not scaled)

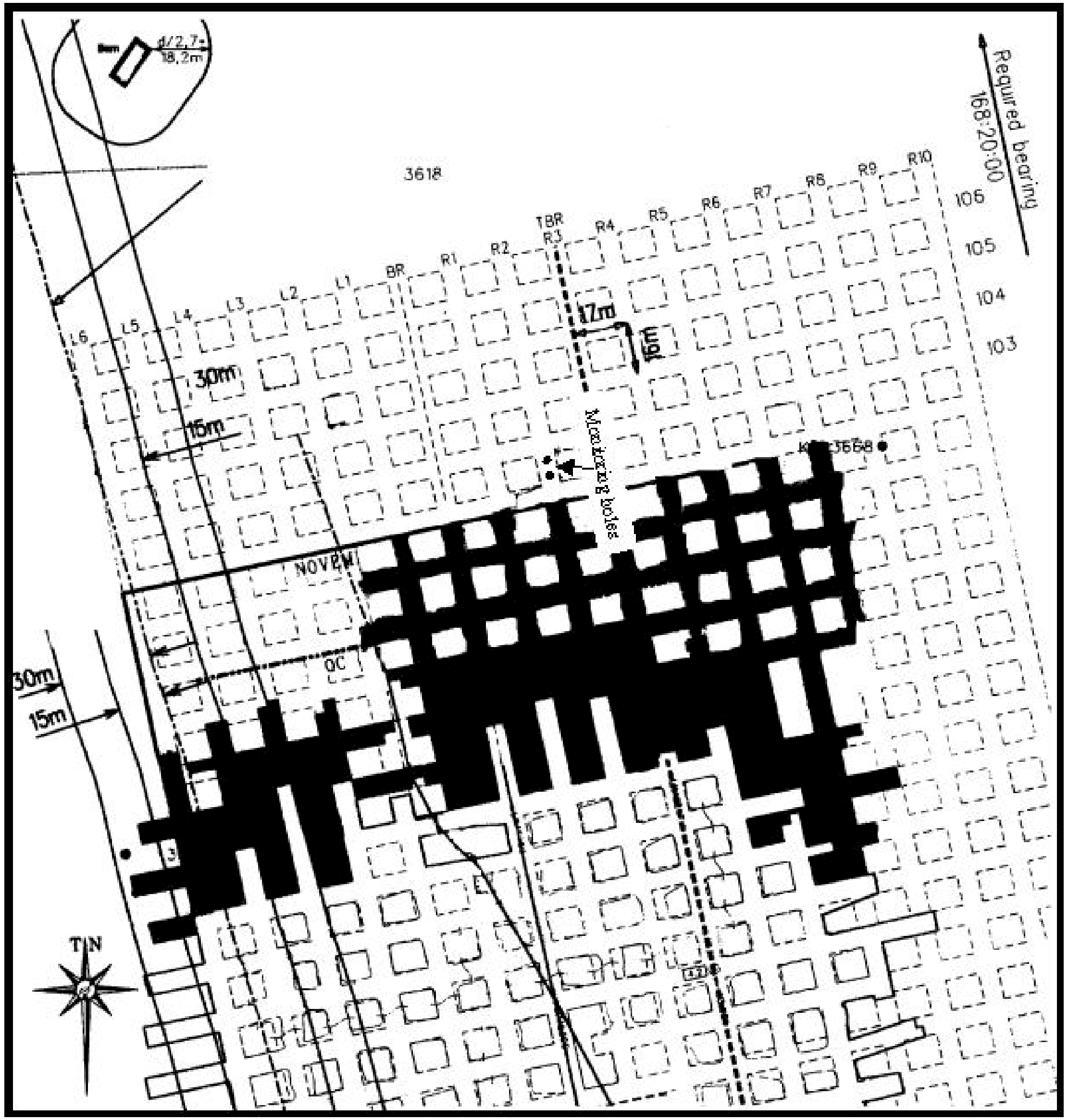


Figure A-7. Section plan Colliery 'C' Site 1 (not scaled)

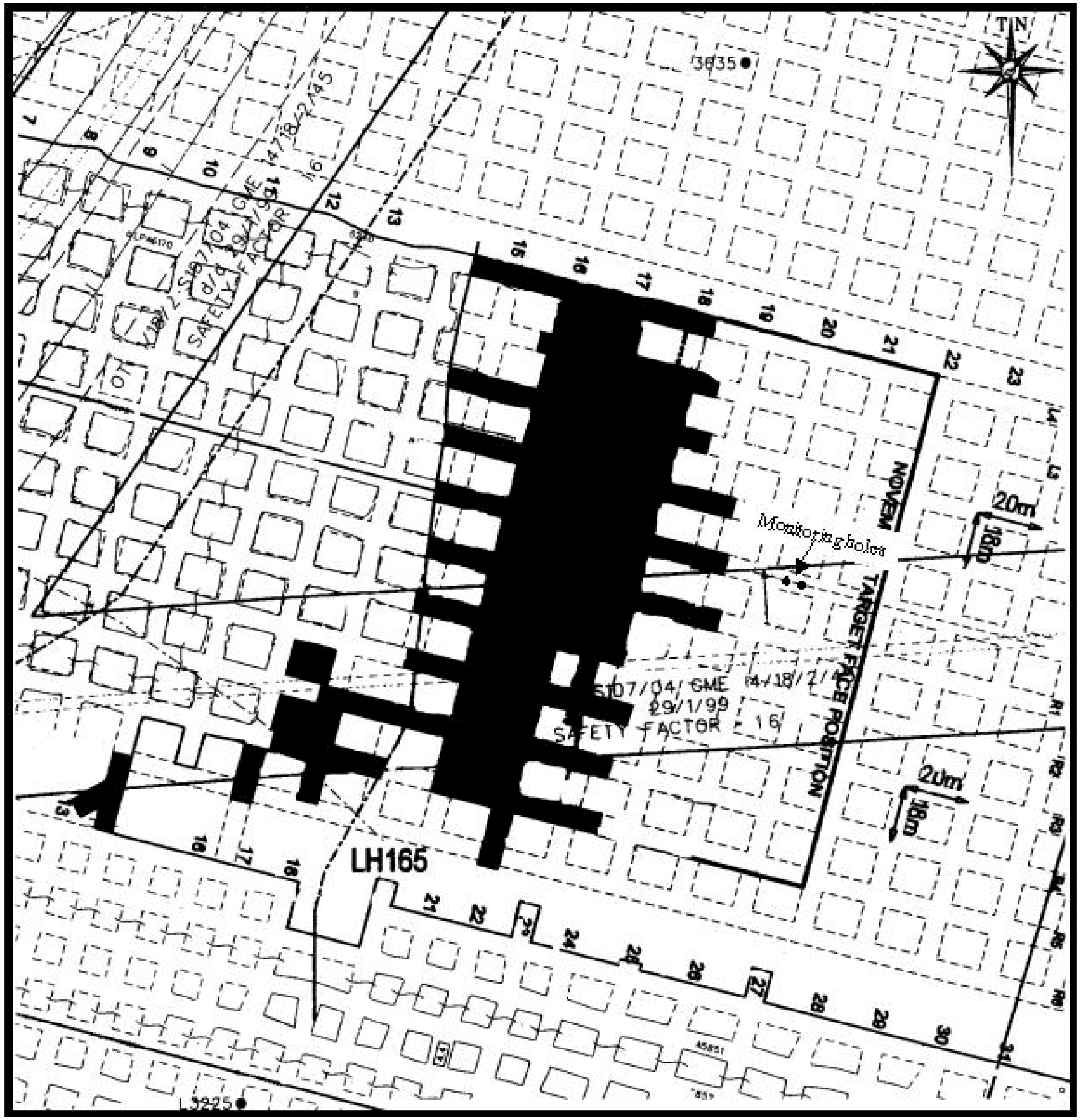


Figure A-8. Section plan Colliery 'C' Site 2 (not scaled)

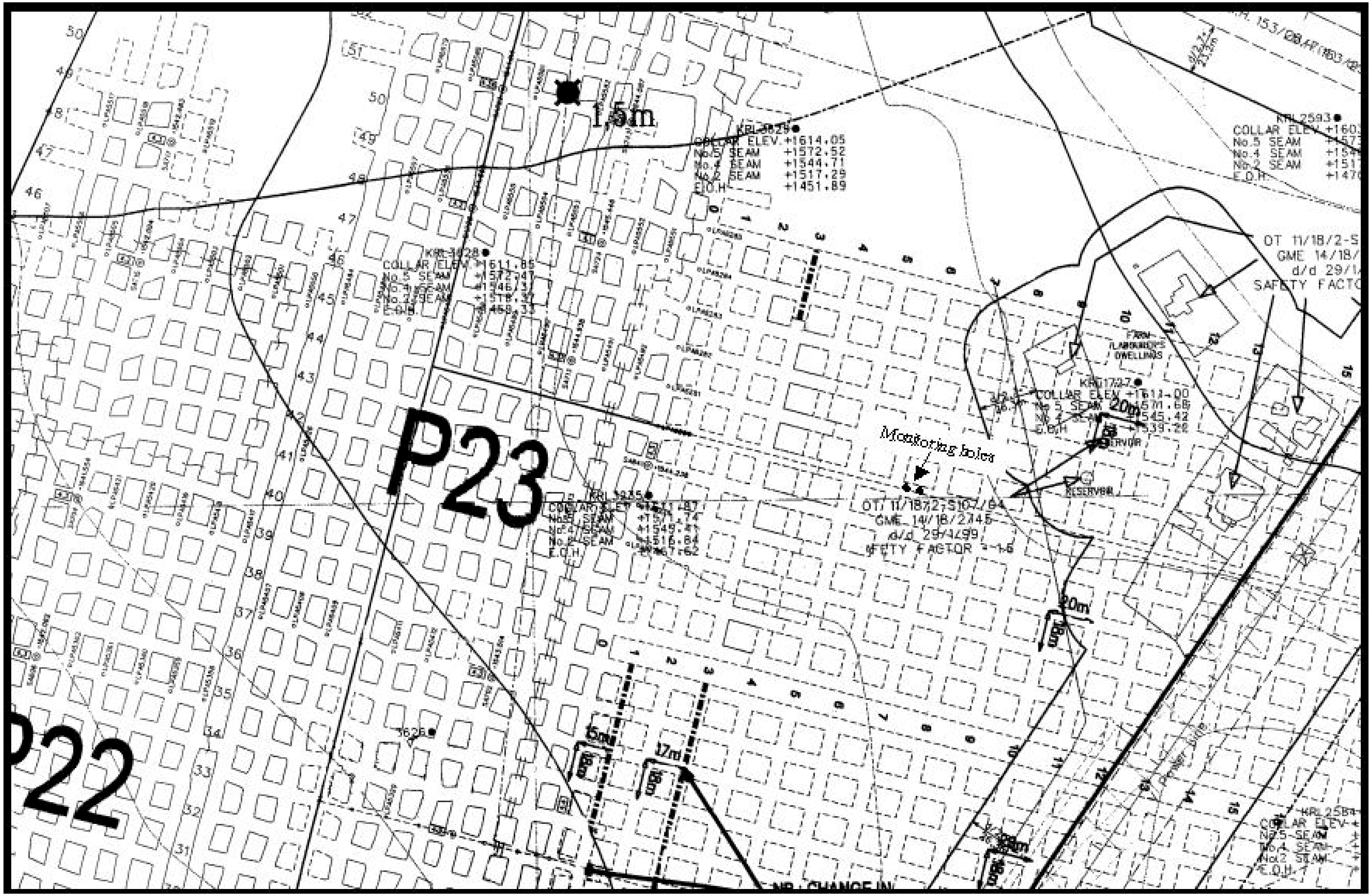


Figure A-9. Section plan Colliery 'C' Site 3 (not scaled)

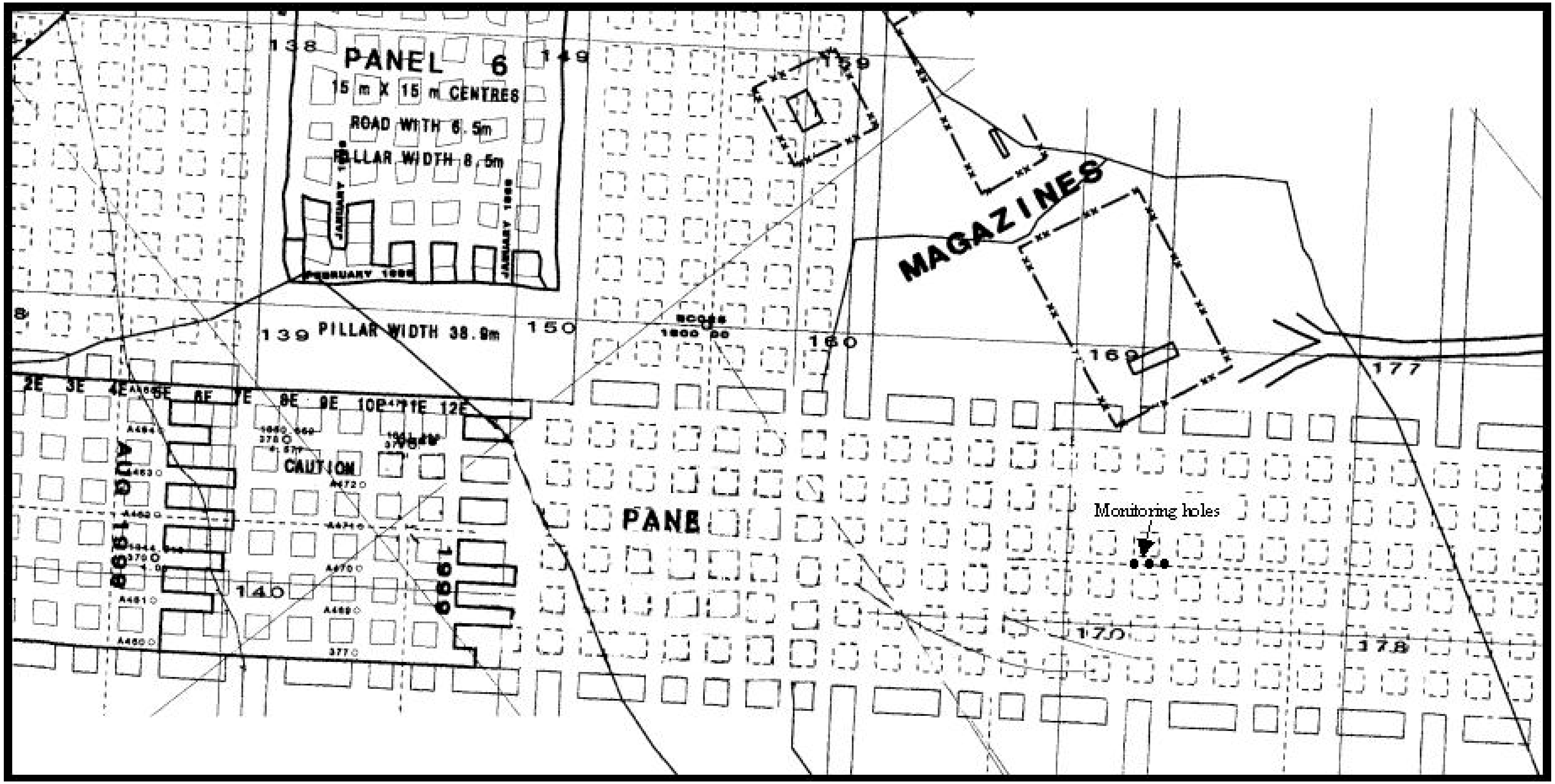


Figure A-10. Section plan Colliery 'D' Site 1 (not scaled)

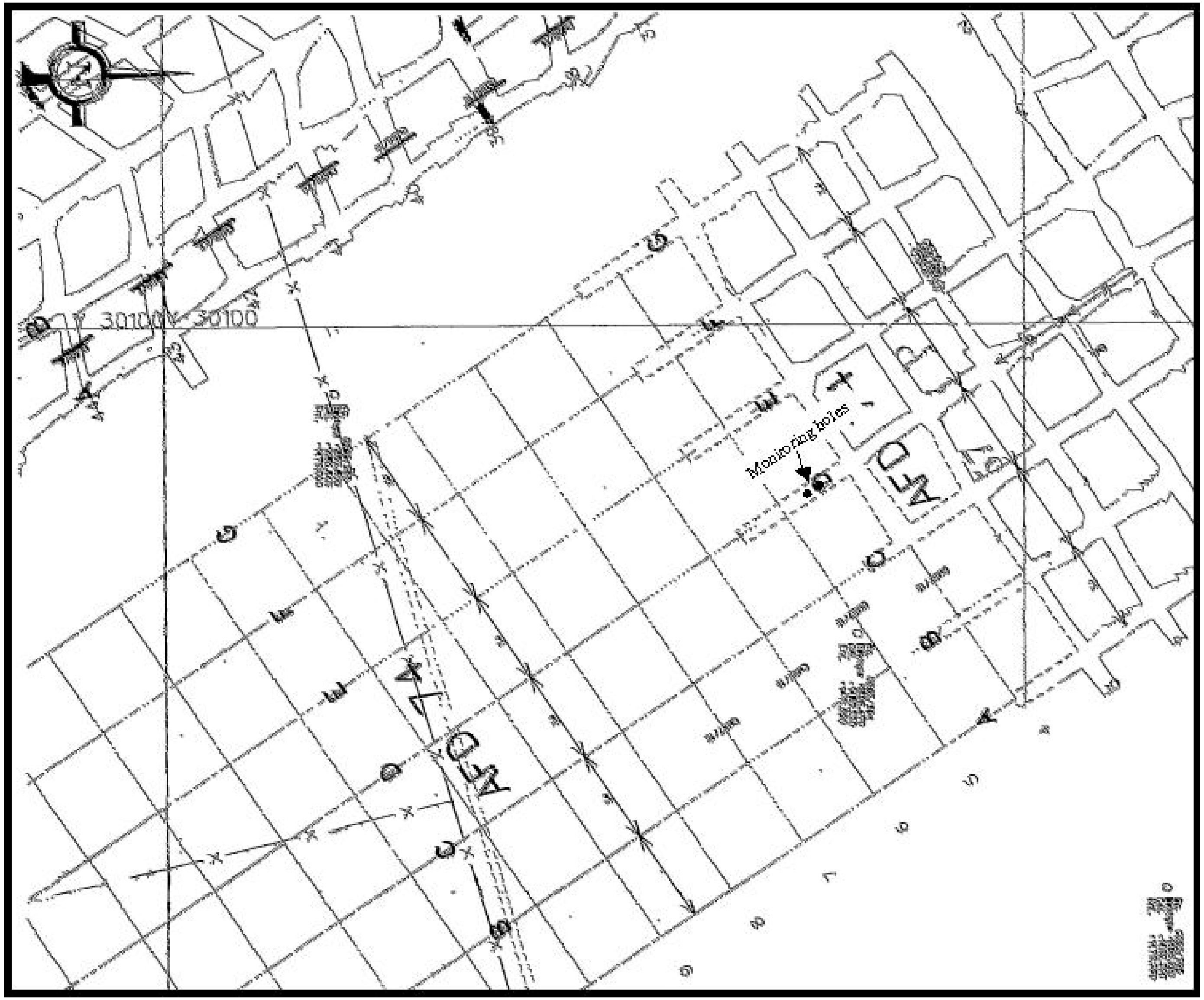


Figure A-11. Section plan Colliery 'E' Site 1 (not scaled)

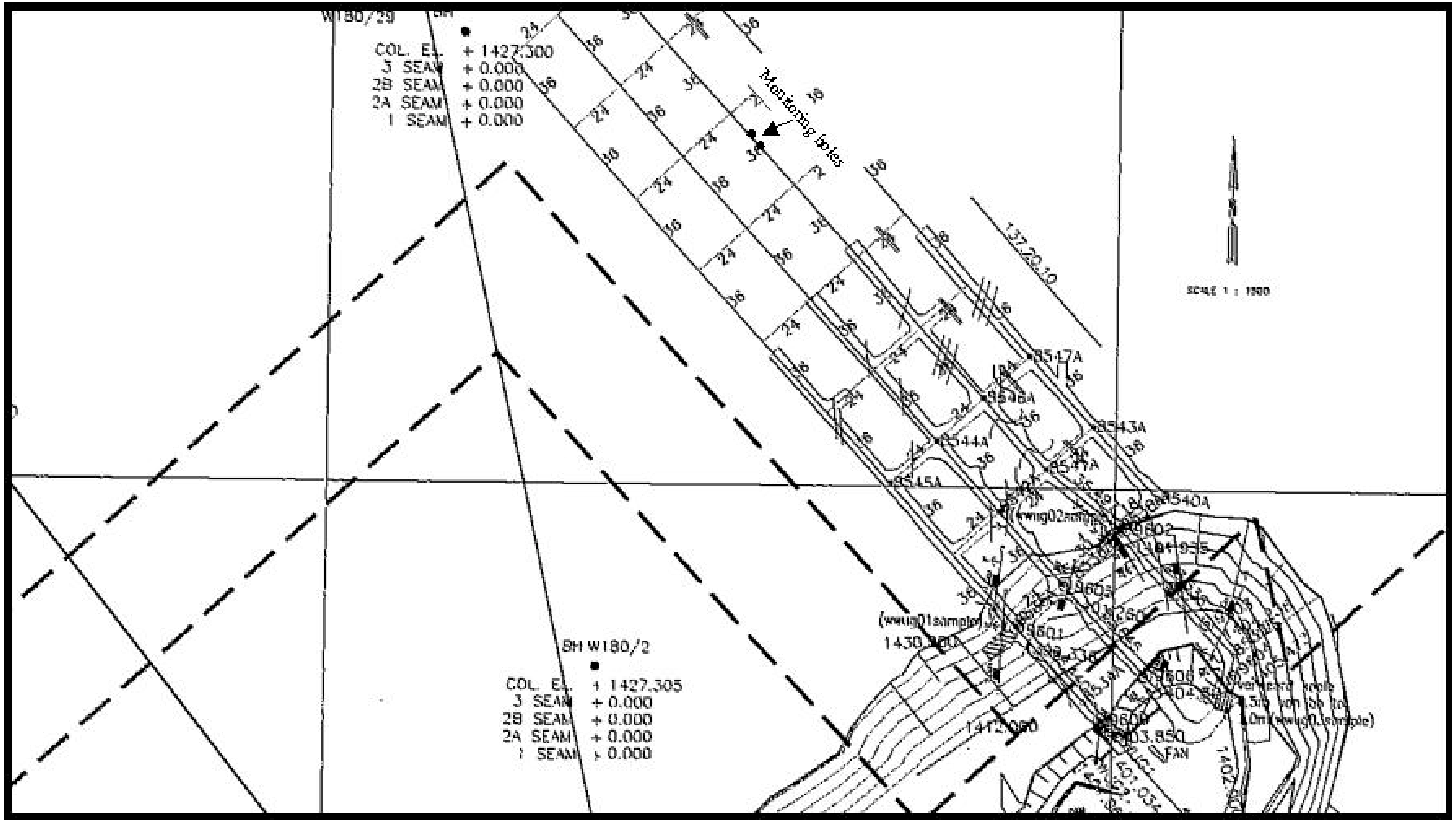


Figure A-12. Section plan Colliery 'F' Site 1 (not scaled)