

Safety In Mines Research Advisory Committee

Causes of Falls of Roof in South African Collieries

**J.N. van der Merwe
J.J. van Vuuren
R. Butcher
I. Canbulat**

**March 2001
Revised August 2001**

Executive summary

The main objective of this research was to determine the causes of falls of roof in South African collieries.

The research team was assembled from the CSIR Miningtek, ITASCA Africa (Pty) Ltd and Steffen, Robertson and Kirsten Consulting Inc to achieve a balanced view. The south African Colliery Manager's Association and the members of the South African National Institute of Rock Engineering who are employed by collieries lent full support to the work.

Sixteen major roof falls were investigated in detail and their causes established. This data was supplemented by mapping a large number of roof falls on four collieries in less detail. The total number of recorded roof falls was 182. The data bank simulated the data bank of roof falls that resulted in fatalities in terms of the total number of falls recorded and the thickness distribution of the collapses.

It was found that the causes of the falls differed for different thickness ranges of roof falls. The thin falls, classified as "Skin falls", which accounted for approximately 70% of all fatalities, were predominantly caused by ineffective joint support and excessive bolt spacings.

Ineffective joint support was found to be a dominant cause of roof falls for all thickness ranges, while the influence of excessive bolt spacings diminished with increasing fall thickness. However, it remained a significant cause of roof falls even for the major falls.

Burnt coal, dykes, bad mining practice, weathering and inferior materials were found to be minor causes, but nonetheless contributed to the falls. Horizontal stress was a minor cause, with increasing contribution for thicker falls. It was also found that indications of horizontal stress coincided with areas of reduced roof rock strength.

In general, roof falls occurred in areas where the roof rock was found to be less competent in terms of rock mass rating systems. It was concluded that the Coal Mine Roof Rating (CMRR) system developed by NIOSH in the USA held promise for application in South Africa and should be investigated further.

It was indicated that in South Africa, less bolts are installed for comparable roof conditions than in the USA and Australia. This supports the conclusion that excessive bolt spacing is a major cause of roof collapses in South Africa.

Acknowledgements

The investigation covered a wide area and entailed underground inspection at several mines. The topic itself, falls of roof, is one that poses a significant threat to mine supervisors at all levels. In addition to the obvious danger to workers posed by falls of roof, there is also the danger to a manager that the outcome of an investigation of this nature could be used for prosecution. Despite this, more invitations to investigate falls were received than could be handled.

This was due to the positive attitude adopted by the South African Colliery Manager's Association, SACMA. The organisation and its individual members are lauded for their contribution to this project. On every mine that was visited, the team was received cordially and given every assistance to complete the task. They are not mentioned by name to abide by the undertaking of confidentiality that was entered into.

The rock engineers employed by the mines assisted in similar fashion, accompanying the team on the visits without interfering or attempting to steer the investigation in any particular direction.

The team members, Jaco J. van Vuuren (ITASCA Africa), Richard Butcher (SRK Consulting) and before he left the country, Ismet Canbulat (CSIR Miningtek), were always prepared to change their schedules to accommodate urgent call-outs to falls of roof. The team members are thanked for their commitment and contribution to the task.

SIMRAC is gratefully acknowledged for sponsoring the project.

Contents

Executive summary	ii
Acknowledgements	iii
1 Introduction	1
2 Size Distribution of Fatal South African Roof Fall Accidents...	3
3 Method of investigation	4
4 Description of the database.....	6
5 Characteristics of All Roof Falls.....	8
5.1 Causes of Falls.....	8
5.2 Locality of All Falls.....	8
5.3 Timing of Falls	9
6 Characteristics of Skin Falls.....	12
6.1 Causes of Skin Falls.....	13
6.2 Localities of Skin Falls	14
6.3 Timing of Skin Falls	15
7 Characteristics of Large Falls.....	16
7.1 Causes of Large Falls.....	17
7.2 Localities of Large Falls	18
7.3 Timing of Large Falls	18
8 Characteristics of Major Falls.....	19
8.1 Causes of Major Falls	19
8.2 Localities of Major Falls	21
8.3 Timing of Major Falls	21

9	Discussion of Causes of Roof Falls	23
10	Influence of roof rock quality	24
10.1	Roof Rating Systems	24
10.2	Roof Rock Strength	25
10.2.1	Uniaxial Compressive Strength of Sandstone (UCS)	26
10.2.2	Modulus of Elasticity of Sandstone (E).....	26
10.2.3	Shear Strength of Sandstone.....	27
10.2.4	Uniaxial Tensile Strength (UTS) of Shale/Mudstone	27
10.2.5	Uniaxial Tensile Strength (UTS) of Sandstone	28
11	Role of Support Material	29
12	Discussion and Conclusions.....	31
12.1	Most Fatalities Are Caused By Thin Roof Falls	31
12.2	Characteristics of Roof Falls	31
12.3	Inadequate Joint Support Is The Most Frequent Cause Of Roof Falls.....	31
12.4	Bolt Spacing Is Too Wide.....	33
12.5	Under Performance of Support Material Is Not a Major Problem But Should Not Be Tolerated At All	34
12.6	Weathering Is Not a Major Cause of Roof Falls In Young Mines.....	36
12.7	Burnt Coal, Dykes And Bad Mining Are Relatively Minor Contributors to Roof Falls.	36
12.8	Horizontal Stress Signs Coincided with Weak Roof Material.....	36
	References	38
	Appendix I - Individual site investigation reports.....	39
	Appendix II – Summary of Mapped Falls.....	114

Tables

Table 1. Causes of roof falls for different thickness of falls.	25
---	----

Figures

Figure 1. Fall of roof fatality rate per thousand employees on South African collieries for the ten year period 1991 to 2000.	8
Figure 2. Distribution of thickness of fatal roof falls in South African collieries. The database consisted of 192 documented cases. More than 70% of all fatal roof falls were less than 0,51 m thick.	8
Figure 3. Comparison of the databases of mapped and fatal roof falls.	11
Figure 4. Comparison of the roof fall rates and size distributions of roof falls on the selected mines.	12
Figure 5. Causes of all roof falls	13
Figure 6. Localities of all roof falls	13
Figure 7. Timing of all falls	14
Figure 8. Size distribution of all falls.	14
Figure 9. Illustration of a roof fall with island bolts, indicating excessive bolt spacing.	15
Figure 10. Illustration of a fall with free hanging bolts, indicating insufficient tension or excessive bolt spacing. This roof is now unsupported.	15
Figure 11. Breakdown of the causes of Skin Falls, showing the dominance of excessive bolt spacing and insufficient joint support.	16
Figure 12. Example of a roof fall that was caused by insufficient joint support. The W-straps in the photo were installed after the roof fall had occurred.	16
Figure 13. Localities of Skin Falls.	17
Figure 14. Timing of Skin Falls relative to the timing of bolt installations.	18
Figure 15. Timing of Skin Falls with the old mine case excluded from the database.	19
Figure 16. Example of a Large Fall that occurred after the installation of support, caused by insufficient joint support. The joint plane without support is indicated by the ellipse.	19
Figure 17. Causes of Large Falls	19
Figure 18. Localities of Large Falls	20
Figure 19. Timing of Large Falls relative to the timing of support installation.	21
Figure 20. This fall was classified as Major. It occurred after support had been installed and was classified as being caused by horizontal stress. It occurred in the vicinity of a dyke.	22
Figure 21. Causes of Major Falls.	22
Figure 22. Causes of Major Falls, excluding weathering.	23
Figure 23. Localities of Major Falls.	24
Figure 24. Timing of Major Falls relative to the timing of support installation.	24
Figure 25. Causes of roof falls seen against the background of thickness of roof falls.	25
Figure 26. Test of correlation between the rating systems and road width.	27
Figure 27. Comparison of the amount of support installed for comparable roof conditions in South Africa, the USA and Australia, after Mark (1999).	28
Figure 28. UCS values that were obtained from roof sandstone samples collected at the fall areas.	28
Figure 29. Modulus of Elasticity of sandstone samples from the fall areas.	29
Figure 30. Shear strength of the sandstone samples.	30
Figure 31. Shear strength of Mudstone/Shale samples	31
Figure 32. Uniaxial Tensile Strength of the sandstone samples.	32
Figure 33. Even though this roof has been re-supported after a fall, the joint is still not adequately supported.	34
Figure 34. Unsupported joint that resulted in a major fall.	34
Figure 35. Skin fall caused by failure between bolts that were spaced too widely.	35
Figure 36. Major fall that was caused by a bolt system failing at the weakest link, the resin	35

contact. A denser bolt spacing would have prevented this fall.

Figure 37. Was the fall in the background caused by cutting lengths off cable anchors like the one shown in the photograph? **36**

Figure 38. The smooth appearance of the thread on this bolt was due to the nut having stripped off on it. **37**

1 Introduction

Despite several years of rock engineering research and application on South African collieries, fall of roof accidents continue to claim lives, as shown in Figure 1. There are several perceived reasons for this fact, including poor adherence to standards, poor design of support systems, poor performance of support elements, unknown nature of the stress regime, etc.

SIMRAC approved this research project to answer the following deceptively simple question: what causes the roof to fall? Analytical simulations of roof falls abound, with models ranging from simple beam behaviour to more complex numerical models simulating the complete stress regime with often uncertain input. Before this project, no major in situ underground investigation had ever been done in South Africa.

There were attempts in the past to obtain SIMRAC approval for a project of this nature, but it was never successful, mainly due to the difficulty in guaranteeing confidentiality. In 1999, however, the South African Colliery Managers Association (SACMA) lent their support for the project after the labour movements, the State and SACMA agreed that the results of this project would not be used for any purpose other than scientific investigation. The team therefore concentrated on the technical causes of roof falls and did not try to establish blame or negligence on the side of any individual or group of individuals.

The investigating team undertook not to identify any site, mine or individual that was connected to the investigation. It was also decided not to limit the investigation to accidents, but to include falls of roof that could have resulted in accidents. It was therefore important that the database of the cases that were studied should simulate the database of roof falls that did result in accidents as closely as possible. This was achieved after the SIMCOM sub committee of SIMRAC approved a change of scope. The initial plan was to study twenty falls of roof in great detail. However, after the first fifteen falls had been studied, it became clear that the database was heavily biased towards the major roof falls. To rectify this problem, a great number of smaller roof falls on selected mines were investigated in less detail and included in the database.

Obtaining suitable sites for research is often a problem. In this case, however, the opposite occurred: at one stage the data gathering threatened to get out of hand due to the number of sites that were offered by SACMA members. SACMA is to be lauded for their open attitude that contributed to this investigation in no small measure. In all cases, the team was given all the necessary information and mine plans and not in a single case did a manager attempt to steer the team in any particular direction. Likewise, the rock engineering practitioners employed by the mining companies lent their full support. The local rock engineers accompanied the investigating team on each investigation except one, in a case where a company did not employ a rock engineer.

The final outcome of the investigation did not yield a single cause for roof falls, but several. It was found that different size ranges of roof falls had different combinations of causes. It should be borne in mind throughout this report that where a single cause or type of cause is mentioned, it is the one that was considered to be the dominant one in any particular case, not the only one. For example, in the case where a laminated roof collapsed, it could have been prevented by installing longer bolts closer together to create a stronger beam or by restricting the road width. Therefore, was the cause of the collapse an inferior support system or an excessive road width? The team approached this problem in a pragmatic fashion: if the collapse occurred in an area where the road was wider than elsewhere in the section where the roof did not collapse, the cause was put down to excessive road width. However, if the road width was already restricted and several falls occurred in an area, the cause was put down to an inferior support system for the local conditions.

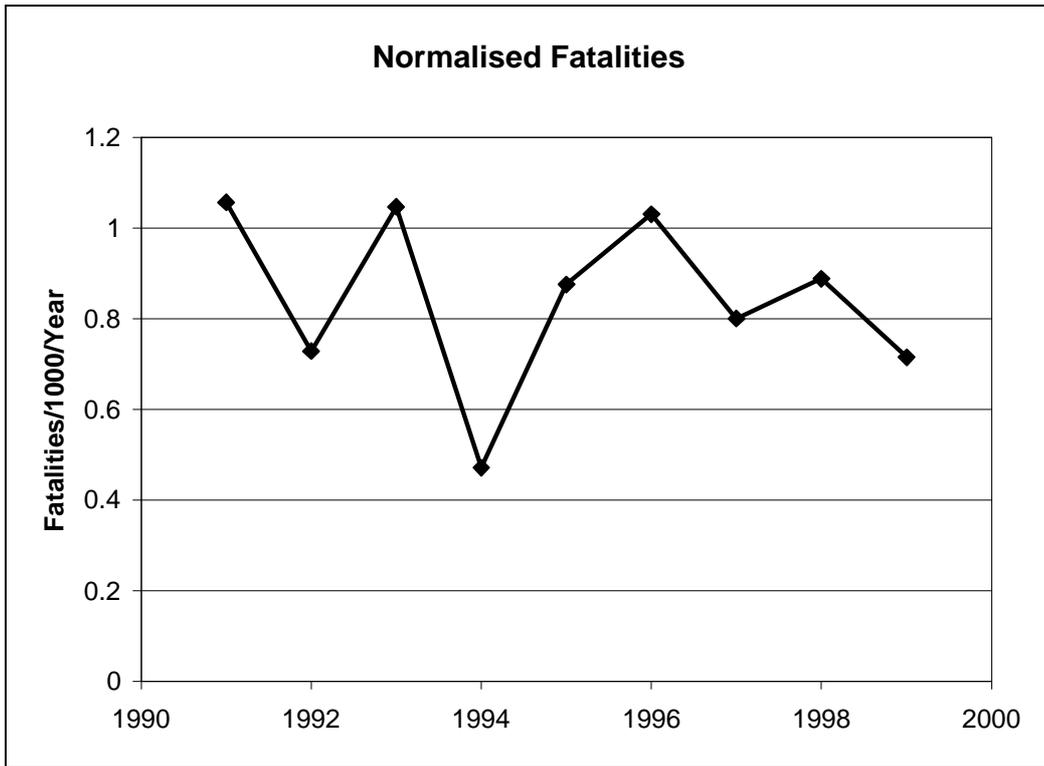


Figure 1. Fall of roof fatality rate per thousand employees on South African collieries for the ten-year period 1991 to 1999.

2 Size Distribution of Fatal South African Roof Fall Accidents.

Analyses of the sizes of roof falls that resulted in fatal accidents in South African collieries were done by Vervoort (1989) and later by Canbulat and Jack (1998). The former covered the period 1970 to 1988 and the latter the period 1989 to 1995. There were only minor differences in the distributions for those two periods. The main conclusion was that the majority of fatal accidents are caused by the smaller roof falls. Their data was combined and the resultant size distribution of fatal roof falls is shown in Figure 2.

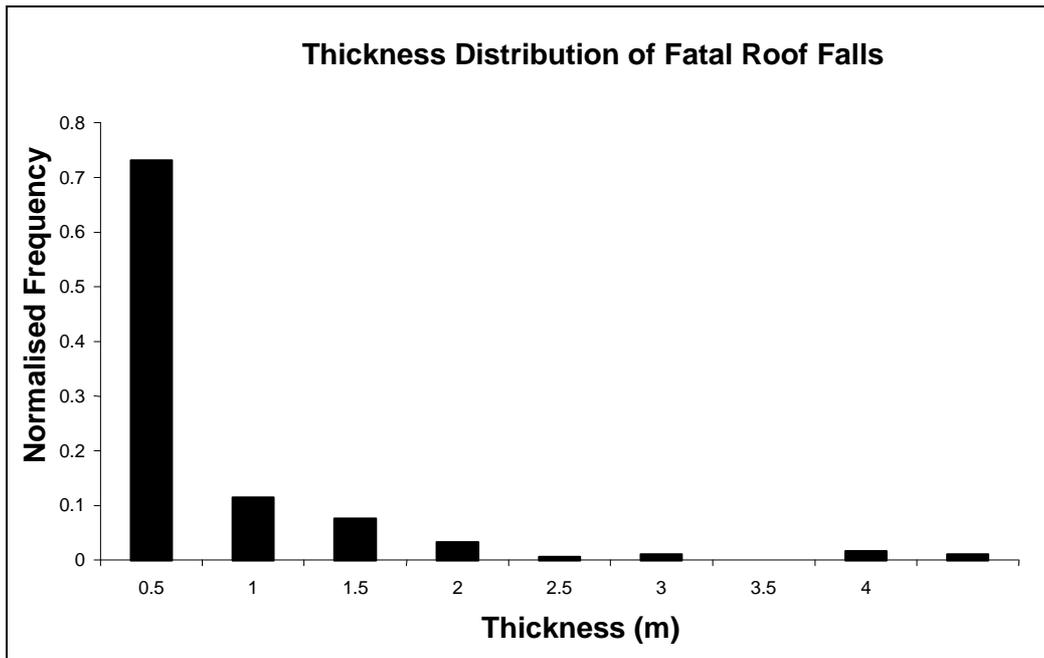


Figure 2. Distribution of thickness of fatal roof falls in South African collieries. The database consisted of 192 documented cases. More than 70% of all fatal roof falls were less than 0,51 m thick.

The trend in the USA is somewhat different. Bauer and Dolinar (2000) reported that in the USA, 82% of roof fall *injuries* were caused by falls that did not entail failure of the support system as a whole, but rather small falls between bolts. However, over the three-year period 1996 to 1998, only 3 out of 13 *fatalities* under supported roof resulted from skin falls, the remaining 10 being caused by massive failures.

According to Pappas et al (2000) 24% of roof fall injuries in the USA occurred under unsupported roof in the period 1996 to 1999. They also reported that a decade ago, approximately 50% of roof fall injuries occurred under unsupported roof. This is somewhat similar to the situation in South Africa, where reductions in roof falls under unsupported roof had been brought about by stricter discipline and enforcement.

More detailed descriptions of the characteristics of fatal roof falls in South Africa falls beyond the scope of this investigation, having been dealt with in detail by Canbulat and Jack (1998).

3 Method of investigation

According to the agreement with SACMA, the investigating team was invited by mine managers to investigate roof falls on their mines. Managers from all the groups participated in this effort and as a result, a wide geographical area was covered. The corners of the area that was included were the Parfuri coal field in the far North East of the country, East of Springs in the West, East of Middelburg in the East and South of Newcastle in the South. The emphasis was on the Witbank coalfield where most of the underground coal mining is being carried out. There, the No's 2, 3, 4 and 5 Seams were investigated.

Upon arrival at a mine, the team would be shown the mine plans and given a history of mining in the area. The geology would be discussed and the team had the opportunity to request any additional information.

The investigation would then proceed underground. Initially, the team would view the roof fall and hold a general discussion on the site. Thereafter, each team member would proceed on a detailed investigation of his area of speciality. This included:

Geophysics and geology, including stratigraphy, Rock Mass Rating (RMR), Bieniawski (1980), Modified Rock Mass Rating (MRMR), Laubscher (1985), and Coal Mine Roof Rating (CMRR), Molinda and Mark (1994);

Mining parameters, including road width, mining sequence, pillar size, equipment, mining method;

Roof support, including type of support, bolt length, spacing, installation quality, timing of support and

A general investigation of the surrounding area, including slips and faults, dimensions, other falls.

Following that, the team would reassemble and reach consensus on the cause of the roof fall. The team would then view specific areas of interest that were found during the detailed individual inspections. Finally, the team tested their diagnosis by establishing why the roof did not fall in the adjacent areas. During the underground investigation, mining personnel were at hand to assist and supply additional information.

Roof rock samples were collected from selected sites for laboratory testing. The tests included:

Uniaxial Compressive Strength,

Poisson's Ratio,

Modulus of Elasticity,

Uniaxial Tensile Strength and
direct Shear Strength.

In this way, fifteen major roof falls were investigated in detail in Phase 1 of the investigation. Those cases are summarised in Appendix A where the important information is contained for each case.

Following Phase 1, there was concern that the objective of simulating the fatal roof fall population would not be reached. There was an insufficient number of smaller roof falls, the mine managers having invited the team to investigate the more spectacular roof falls. This was rectified by changing the scope of the investigation. Four mines were selected and all falls were recorded in selected roadways. The dimensions and causes of the falls were recorded. The experience that was gained in Phase 1 was used to establish the causes of the falls, which were investigated in less detail than the ones in Phase 1. In this way, a total of 167 additional roof falls were investigated to bring the total to 182 falls. A total linear distance of 7,63 km was covered.

The following broad causes were established:

Excessive bolt spacing

Joints (including faults)

Weathering (including Acid Mine Drainage)

Incorrect or insufficient joint support

Burnt coal

Mining (e.g. cutting away pillar corners, breaking splits away at major discontinuities, poor horizon control, etc)

Excessive horizontal stress (including weak rock).

4 Description of the database

The database of the investigated roof falls simulated the database of fatal roof falls almost exactly. The comparison is shown in Figure 3.

Because the falls that were investigated were not necessarily those that resulted in an injury, it was important to create a comparable database. As Figure 3 shows, this objective was achieved.

The average density of the falls was 0,022 falls per linear metre, meaning that on average, there was a roof fall every 45 m. This includes minor roof falls.

In the USA, Molinda et al (2000) reported reportable roof fall rates ranging from 0,0003/m to 0,0001/m. These figures are not directly comparable with South Africa's, as the definition of a "reportable" roof fall according to US regulations is one that –

- Causes injury;
- Falls above anchorage;
- Blocks ventilation;
- Blocks escape or
- Stops production for 30 minutes or more.

Molinda et al (2000) also indicated that the vast majority of US collieries had a zero roof fall rate. This variation in roof fall rates for different mines in South Africa was also found by Canbulat and Jack (1998) who found that a small number of mines in South Africa was responsible for the majority of roof fall accidents.

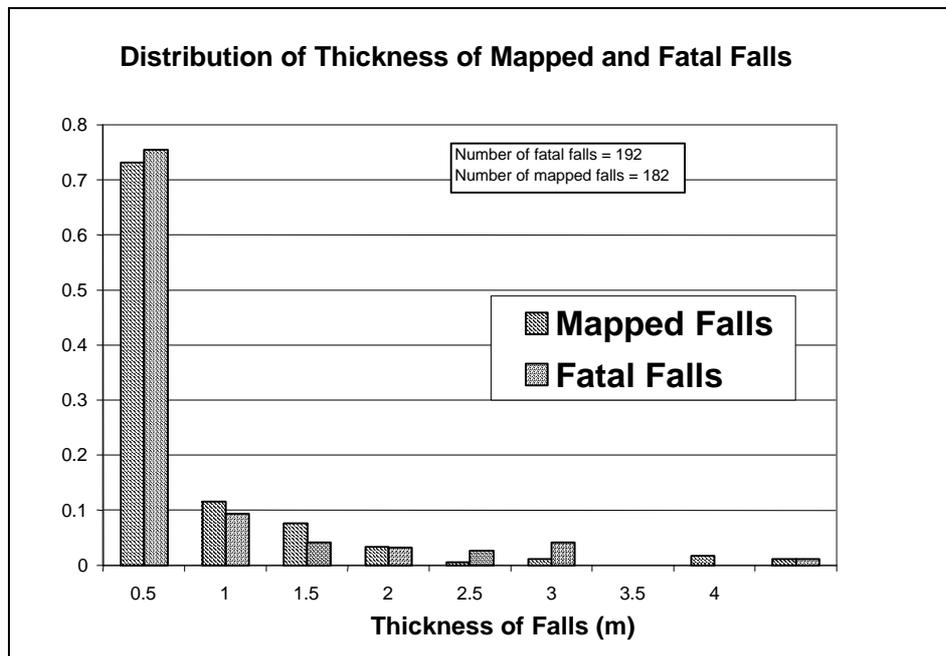


Figure 3. Comparison of the databases of mapped and fatal roof falls.

The roof fall rates on the mines selected for this investigation displayed some variation, as shown in Figure 4.

As seen on Figure 4, the characteristics with regard to the size distributions of roof falls on the four selected mines were largely similar. Mine DD had a slightly different distribution, displaying a

greater proportion of thicker roof falls. The selected area on Mine DD had been abandoned for about twenty years and the immediate roof consisted of coal. Over time, the roof had weathered and fallen in several areas. It is postulated that due to weathering, a great number of small falls had fallen out higher.

The ages of the other areas were between five and approximately ten years, but all the areas were ventilated and maintained. The highest density of roof falls was on Mine BB, which also had the greatest proportion of small falls.

5 Characteristics of All Roof Falls

5.1 Causes of Falls

If the population of roof falls is viewed as a whole, it was found that the dominant causes of falls were insufficient joint support (37%) and excessive bolt spacing (27%). These were followed by weathering (12%), horizontal stress/weak roof rock (9%), bad mining (9%), burnt coal (4%), and dykes (2%). This is shown in Figure 5.

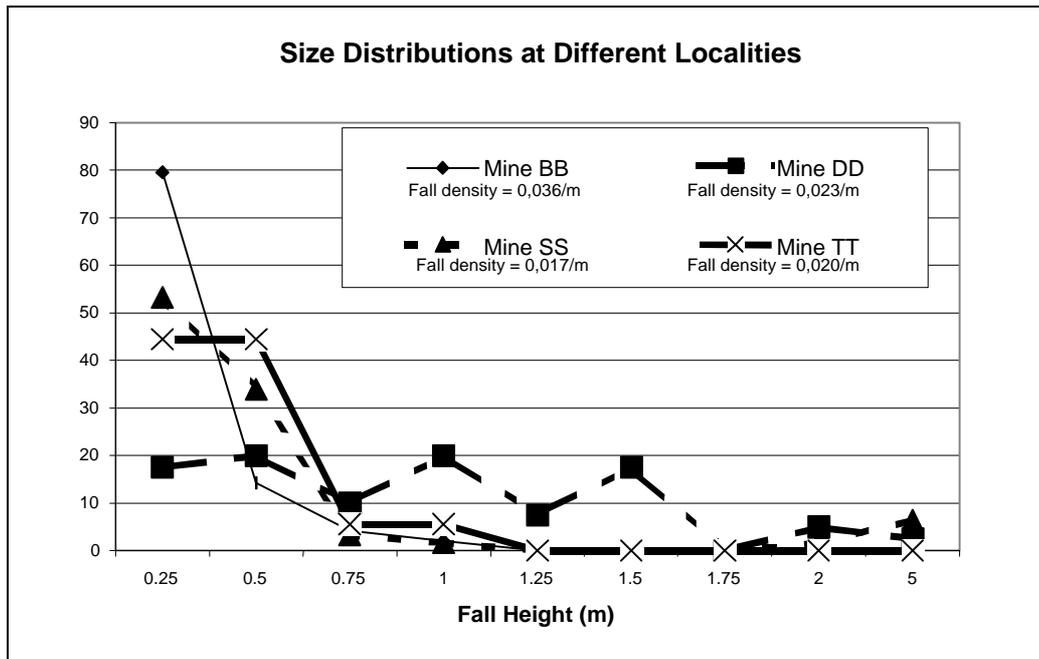


Figure 4. Comparison of the roof fall rates and size distributions of roof falls on the selected mines.

5.2 Locality of All Falls

Not surprisingly, it was found that the majority of all roof falls occurred at intersections, which were responsible for 66% of the total, shown in Figure 6. Bearing in mind that intersections account for approximately 30% of the total exposed roof, it means that one is more than four times as vulnerable to a roof fall injury in an intersection than in a roadway. According to Molinda et al (1998) the roof fall rate in the USA is eight to ten times greater in intersections than in roadways.

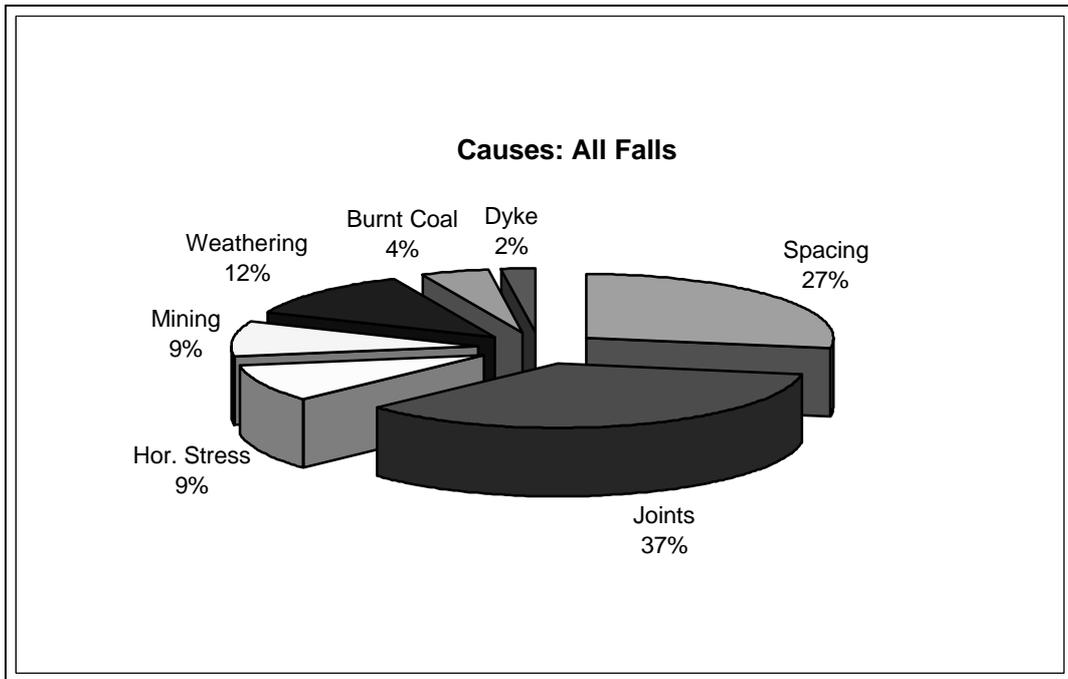


Figure 5. Causes of all roof falls

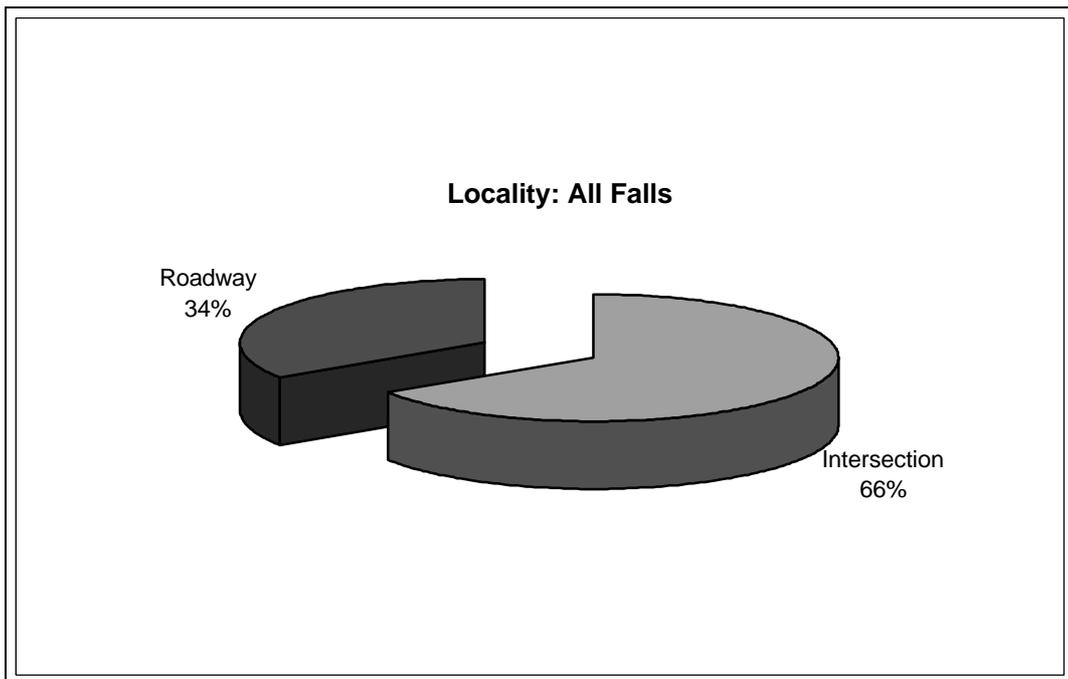


Figure 6. Localities of all roof falls

5.3 Timing of Falls

The timing of roof falls were tested against the timing of support installation. It was found, as shown in Figure 7, that the majority of falls – 68% - occurred after support had been installed and in 20 % of all cases, no systematic support was installed. In this classification of timing, the term “No support” means that the roof was considered to be sufficiently stable by itself and no artificial support was installed. The term “Before support” means that systematic support was in fact used in

the area, but that the roof fell before the intended supports were installed. This is cause for concern, indicating that the effort and cost incurred in supporting the roof were not effective. The large proportion of roof falls in areas where no support was installed, indicates a lack of judgement. Analysing all roof falls as a single group may be misleading. The smaller falls mostly occurred between bolts with a different mechanism than the major falls. It was then decided to split the database into smaller groups, with the following classification based on thickness of the falls:

- Skin falls – less than 0,3 m thick
- Large falls – 0,31 to 1,0 m thick
- Major falls – thicker than 1,01 m.

The size distribution is shown in Figure 8.

As the population of mapped falls is a good simulation of the population of falls that resulted in fatalities, it is clear that the most important group is the “Skin” group, as it accounts for 62% of the total number of roof falls.

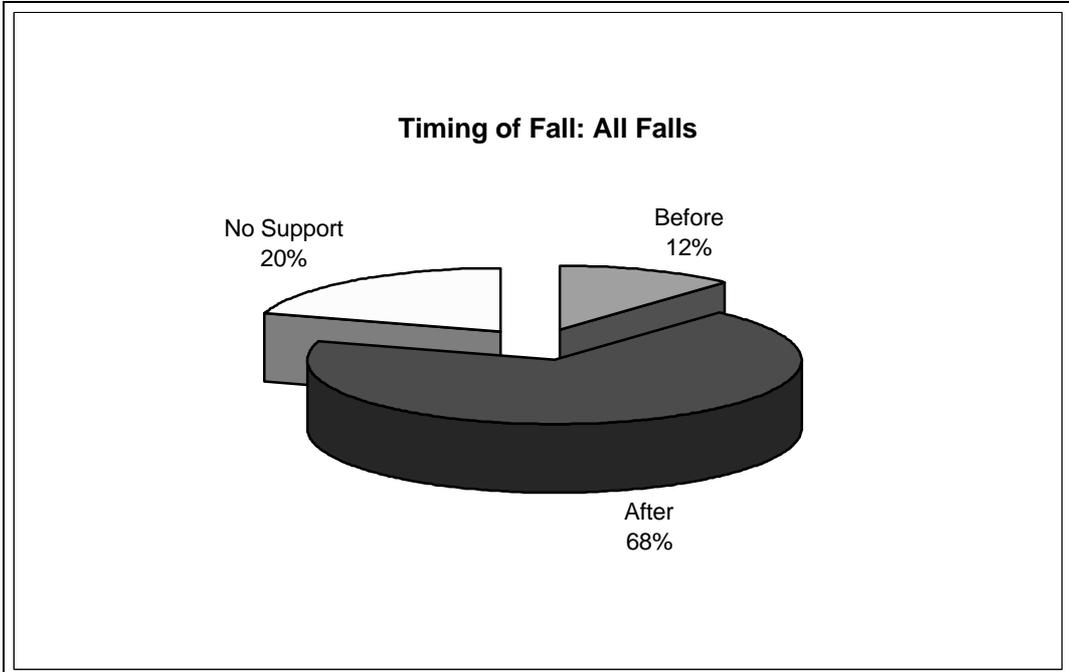


Figure 7. Timing of all falls

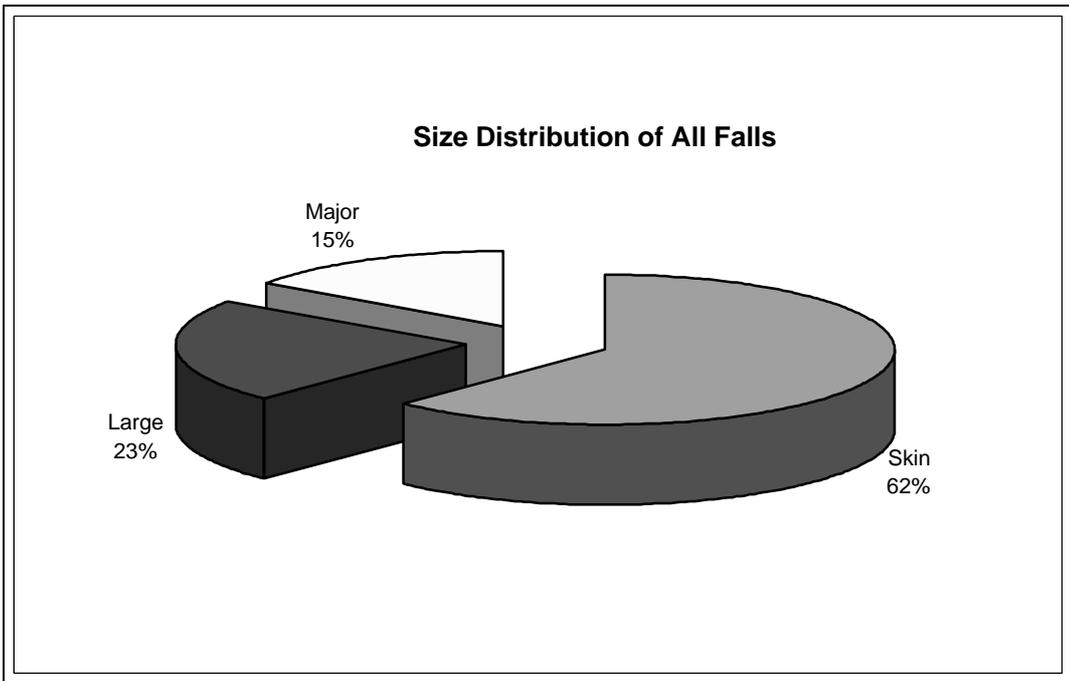


Figure 8. Size distribution of all falls.

6 Characteristics of Skin Falls

Most of the skin falls occurred as falls with island bolts, like the example shown in Figure 9, or as falls with free hanging bolts, as shown in Figure 10. According to van der Merwe (1998), the most likely causes of such falls are excessive bolt spacing and insufficient bolt tension. This should be seen in the context of roof rock strength and thickness of the layers. A spacing that is adequate at the time of bolt installation, may not be sufficient after a long time of exposure when the roof material has been allowed to weather. Also, there is no single correct bolt spacing, it primarily being a function of the thickness of the individual roof layers.



Figure 9. Illustration of a roof fall with island bolts, indicating excessive bolt spacing.



Figure 10. Illustration of a fall with free hanging bolts, indicating insufficient tension or excessive bolt spacing. This roof is now unsupported.

6.1 Causes of Skin Falls

Figure 11 shows the breakdown of causes of Skin Falls. Excessive bolt spacing and joints together account for 68% of all these minor, yet most dangerous roof falls. An example of a Skin Fall caused by a joint is shown in Figure 12.

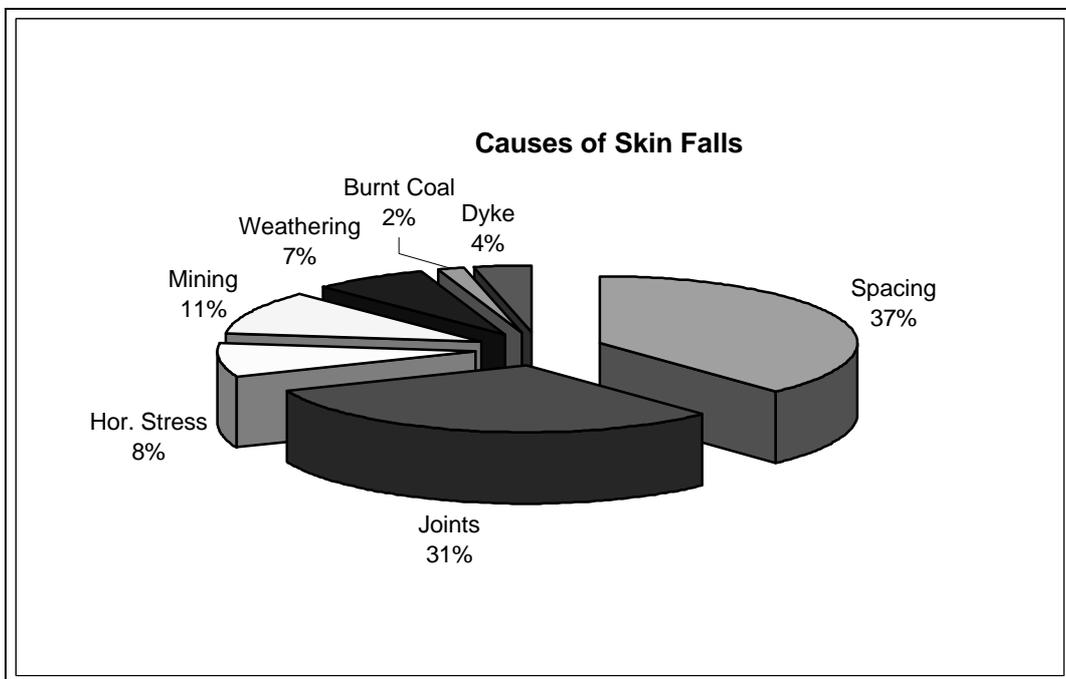


Figure 11. Breakdown of the causes of Skin Falls, showing the dominance of excessive bolt spacing and insufficient joint support.



Figure 12. Example of a roof fall that was caused by insufficient joint support. The W-straps in the photo were installed after the roof fall had occurred.

6.2 Localities of Skin Falls

The localities of skin falls is shown in Figure 13. It is seen that 71% of the falls occurred in intersections. If this is normalised for the relative area of intersections as opposed to roadways, it means that on an equal length basis, Skin Falls are four times more likely to occur at intersections than in roadways.

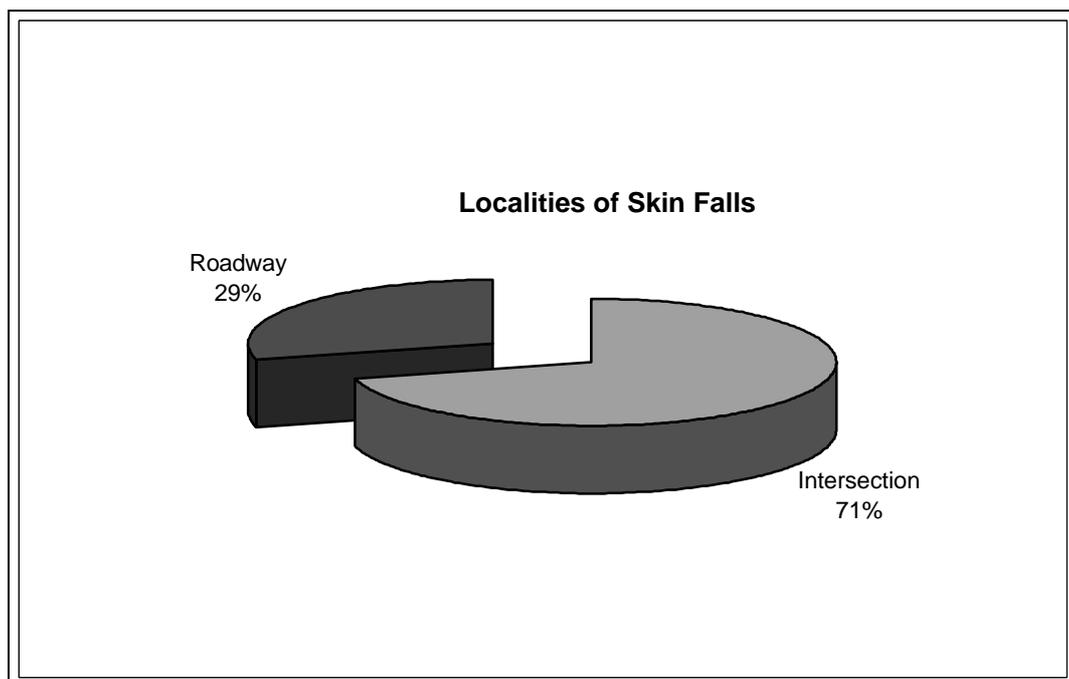


Figure 13. Localities of Skin Falls.

6.3 Timing of Skin Falls

Figure 14 shows that 68% of Skin Falls occurred after support had been installed. In 19% of the cases, no support was installed and only 13% occurred before the installation of support. If everything had been done correctly, there should be no falls after support had been installed.

Due to the inclusion of one old, abandoned area in the database, it may be argued that the database is biased. It is possible that at the time of support installation, the support was in fact adequate but that the roof rock lost strength due to weathering. Also, personnel are unlikely to work in abandoned areas. To test this argument, the Skin Falls were also analysed with the exclusion of the old mine case. This result is shown in Figure 15.

It is seen that the result is not significantly affected, as 72% of Skin Falls occurred after the installation of support. Therefore, it is concluded that roof support systems are not adequate to prevent the occurrence of Skin Falls. The most important issues to address to rectify this situation are bolt spacings, which should be smaller, and joint support.

7 Characteristics of Large Falls

Compared to Skin Falls, Large Falls occur less seldom. The density of Skin Falls is 0,015 falls/m, while that of large falls is 0,006 falls/m. There are more than twice as many falls per metre in the thickness range 0 - 0,3 m than in the range 0,31 m to 1,0 m. An example of a Large Fall is shown in Figure 16.

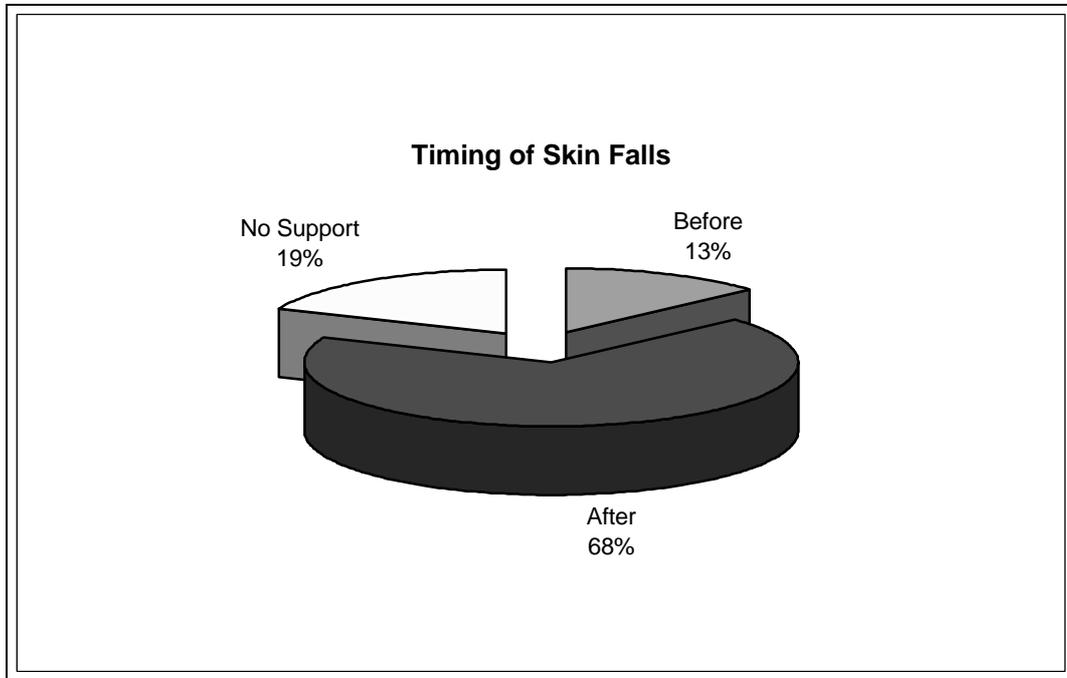


Figure 14. Timing of Skin Falls relative to the timing of bolt installations.

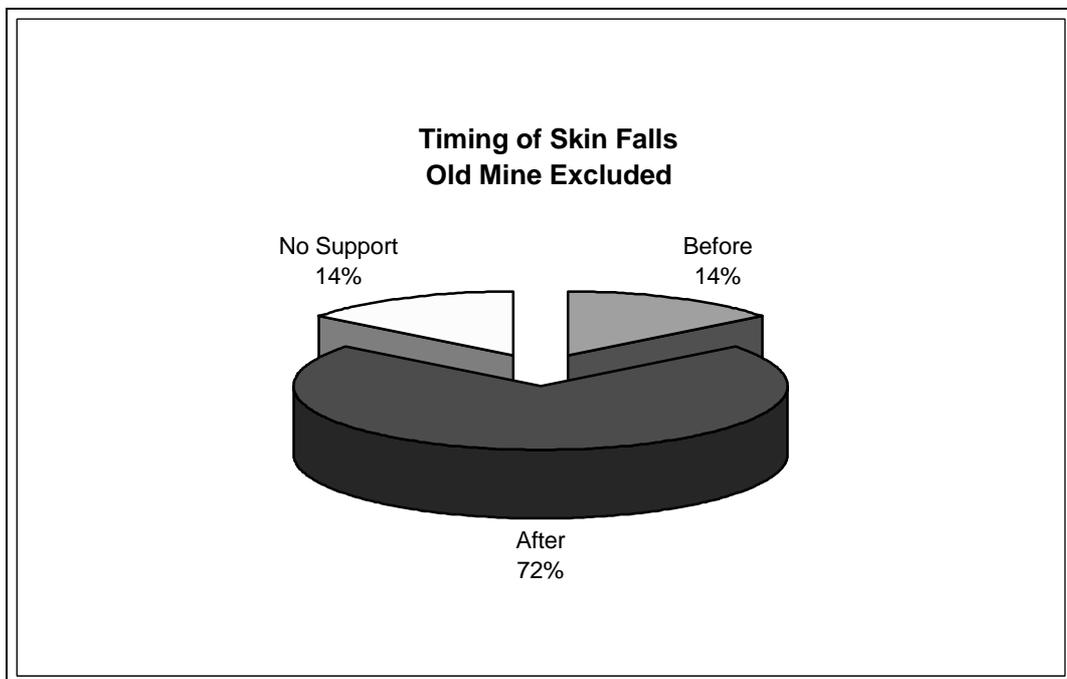


Figure 15. Timing of Skin Falls with the old mine case excluded from the database.

7.1 Causes of Large Falls

As Figure 17 indicates, Large Falls are also primarily caused by insufficient joint support and excessive bolt spacing, although the relative proportion has come down to 49%. Excessive spacing of bolts is down to 10% while weathering has increased to 24% compared to 7% for Skin Falls. Insufficient joint support increased from 31% to 39%.

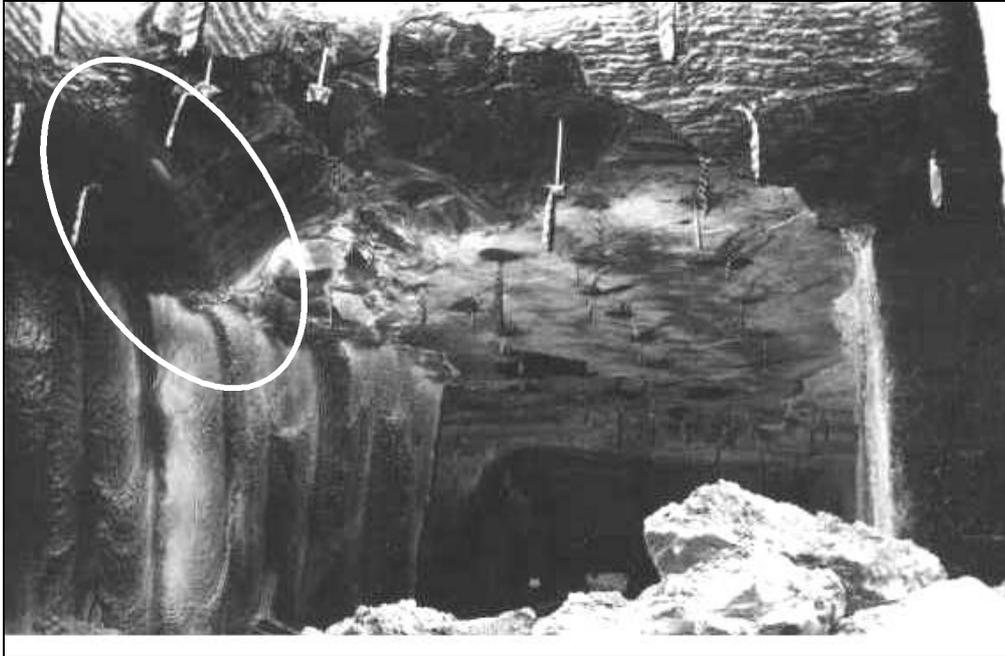


Figure 16. Example of a Large Fall that occurred after the installation of support caused by insufficient joint support. The ellipse indicates the joint plane without support.

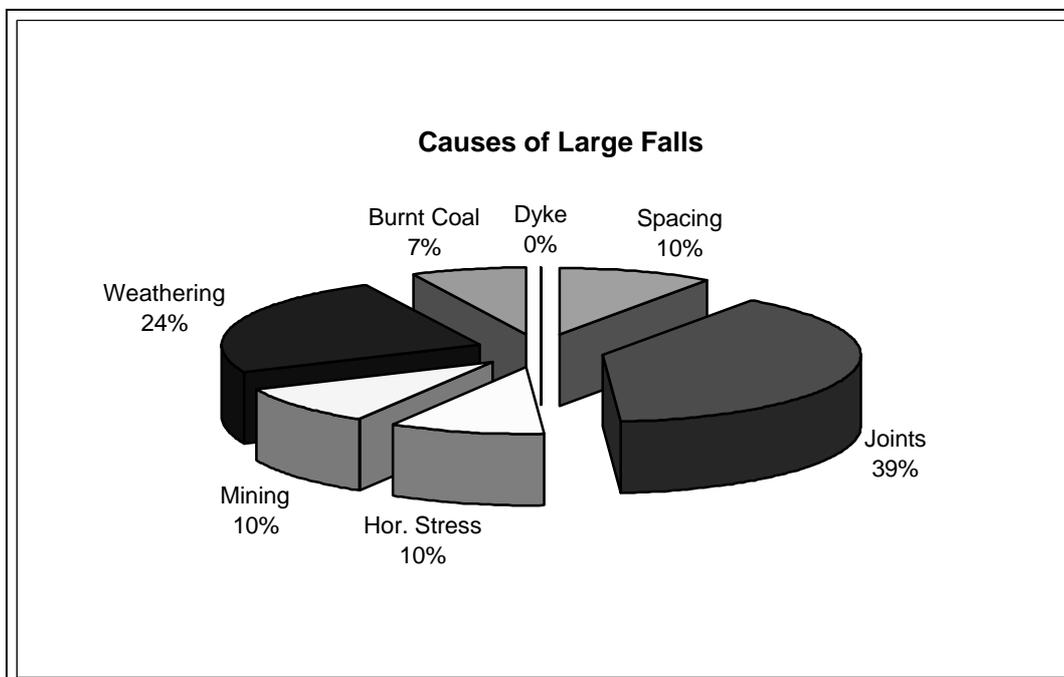


Figure 17. Causes of Large Falls

7.2 Localities of Large Falls

There is a slight shift in the localities of large falls. Intersection falls are still in the majority, although the relative proportion of intersection/roadway falls has come down to 61/39 from 71/29. The breakdown is shown in Figure 18.

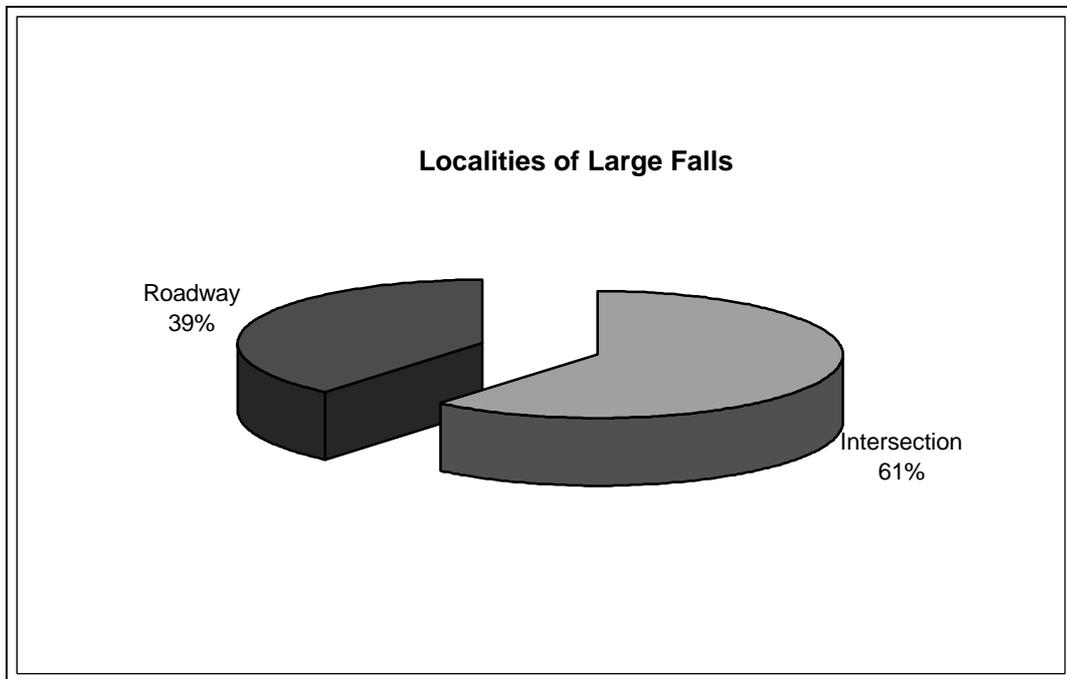


Figure 18. Localities of Large Falls

7.3 Timing of Large Falls

As Figure 19 shows, the tendency for Large Falls to occur after the installation of support is still apparent. There is no significant difference between the tendencies for Skin and Large Falls to occur after support installation. 27% of the falls occurred where no support was installed and 12% before support was installed.

8 Characteristics of Major Falls

Major Falls occur less frequently than Skin and Large Falls, at a density of 0,004 falls/m. They are also responsible for a smaller proportion of fatal accidents. An example of a major fall is shown in Figure 20.

8.1 Causes of Major Falls

Figure 21 shows the breakdown of causes of Major Falls. Excessive bolt spacing is no longer one of the primary causes, but insufficient joint support retains a dominant position, accounting for 50% of the falls. The second most frequent causes, horizontal stress and weathering, are smaller contributors at 14% each.

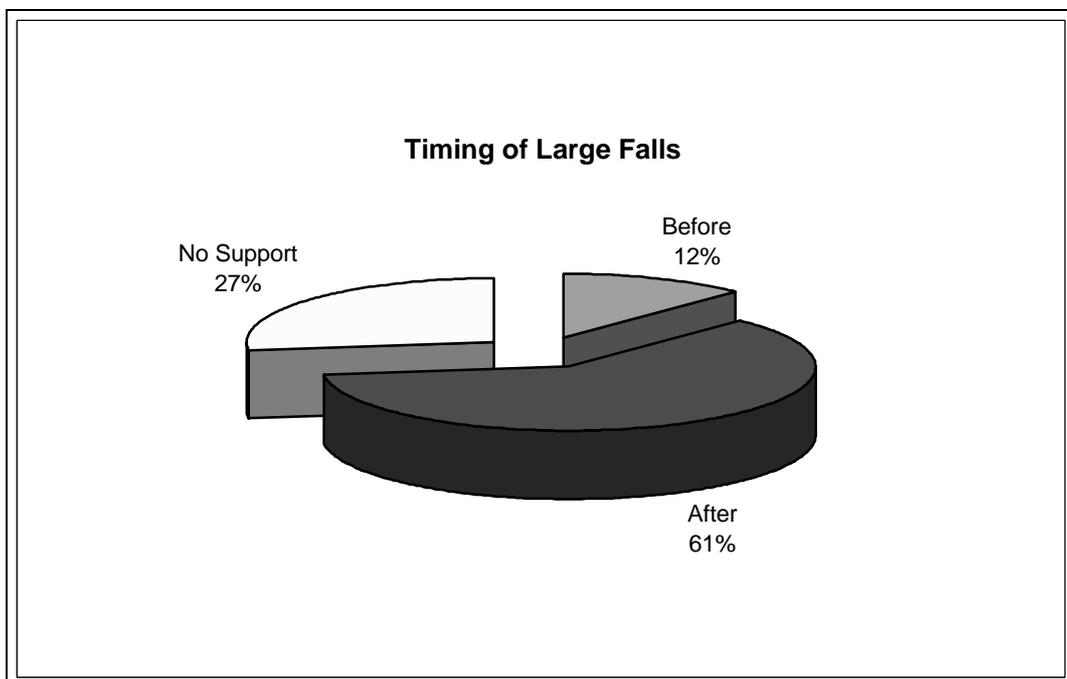


Figure 19. Timing of Large Falls relative to the timing of support installation.

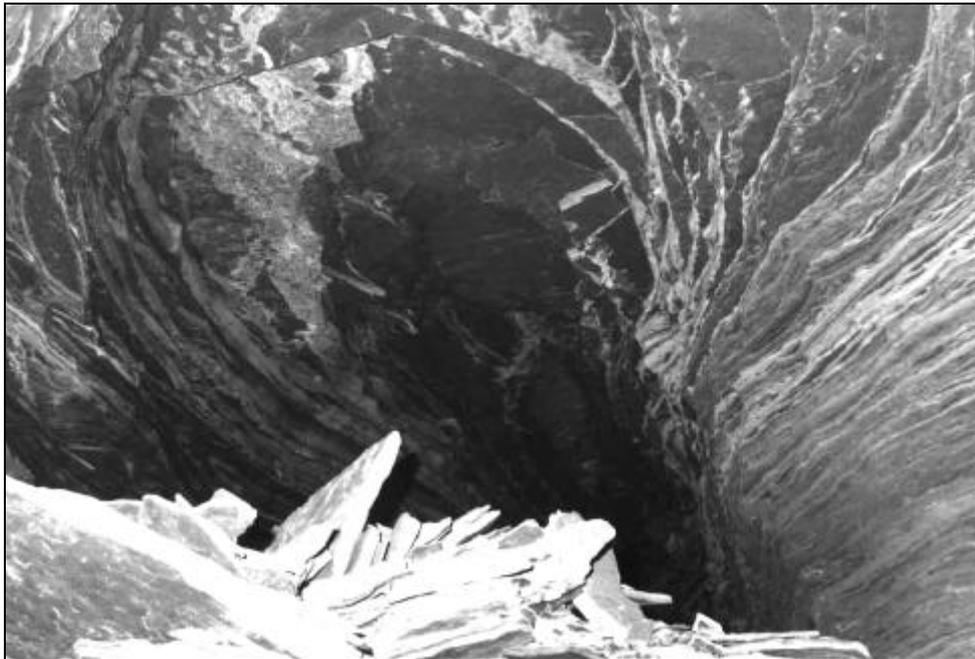


Figure 20. This fall was classified as Major. It occurred after support had been installed and was classified as being caused by horizontal stress. It occurred in the vicinity of a dyke.

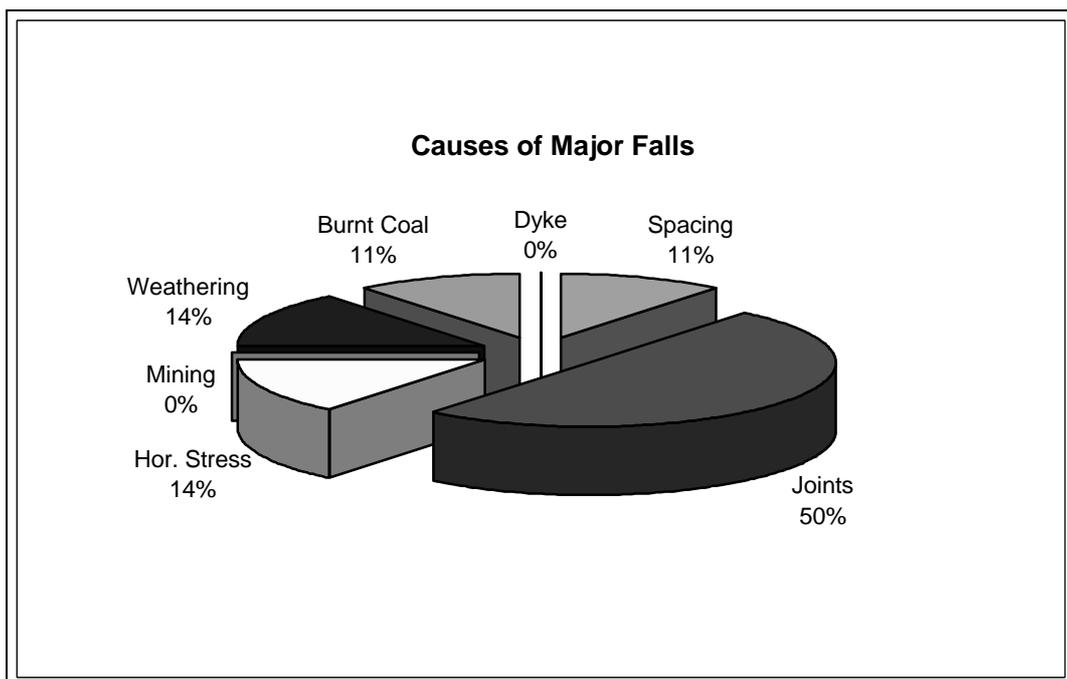


Figure 21. Causes of Major Falls.

The prominence of weathering as a cause of roof falls may be due to the inclusion of an old mine in the database. If those falls are excluded from the database, the relative contributions shift somewhat. This is shown in Figure 22.

Horizontal stress and excessive spacing then increase to 24% and 18% respectively, but insufficient joint support is still dominant at 40%.

8.2 Localities of Major Falls

As Figure 23 indicates, even Major Falls tend to occur more frequently at intersections than in roadways, although the tendency is somewhat reduced. The intersection frequency is 54% and 46% of Major Falls occurred in roadways. A Major Fall is thus almost three times as likely to occur in an intersection than in a roadway on an equal distance basis.

8.3 Timing of Major Falls

As expected, Major Falls occur more frequently than with skin or large falls after support had been installed, see Figure 24. Major Falls are usually thicker than the length of roof bolts and bolting has less effect. It was noticed that in most cases, additional support had been installed before the Major Falls occurred, indicating that there was at least some indication that a problem could be expected. However, the additional supports were not always effective.

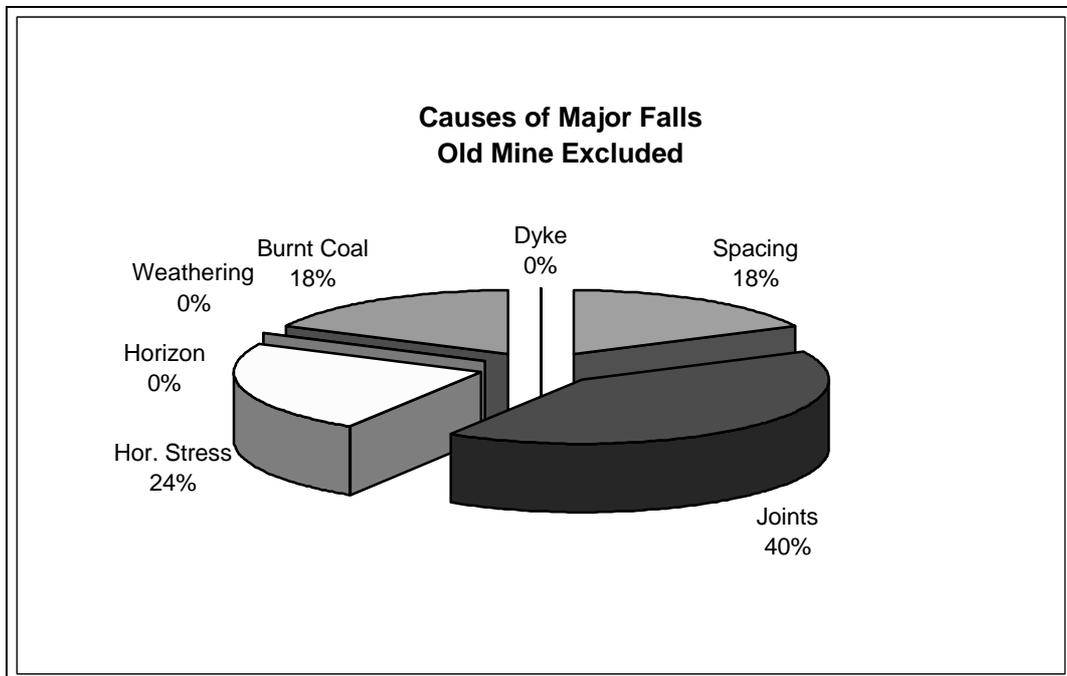


Figure 22. Causes of Major Falls, excluding weathering.

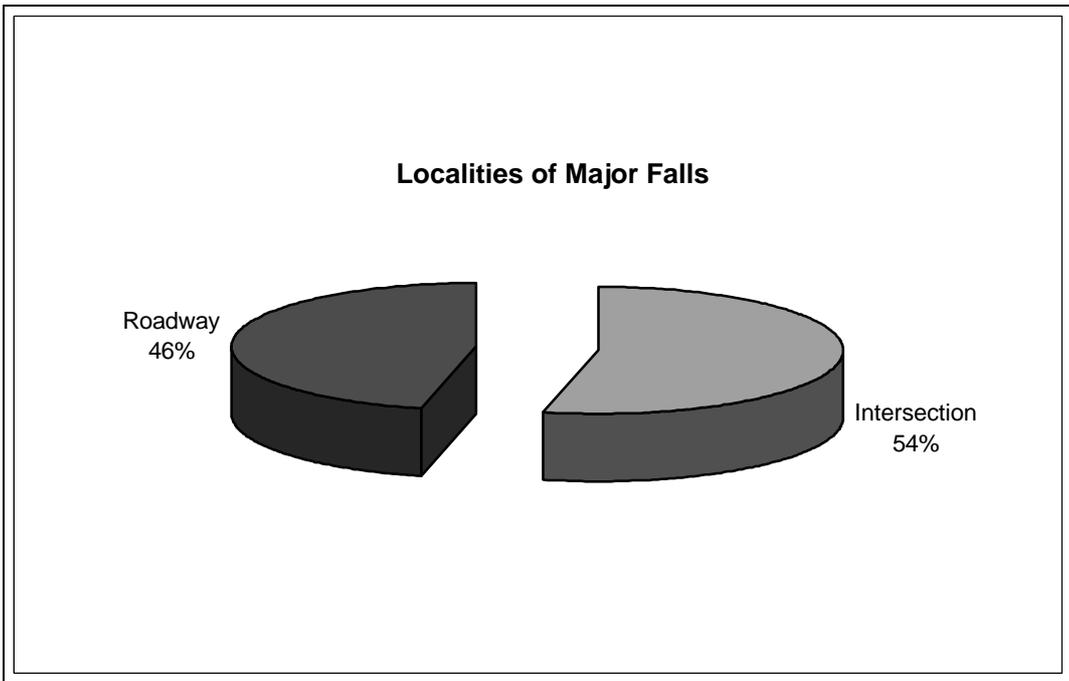


Figure 23. Localities of Major Falls.

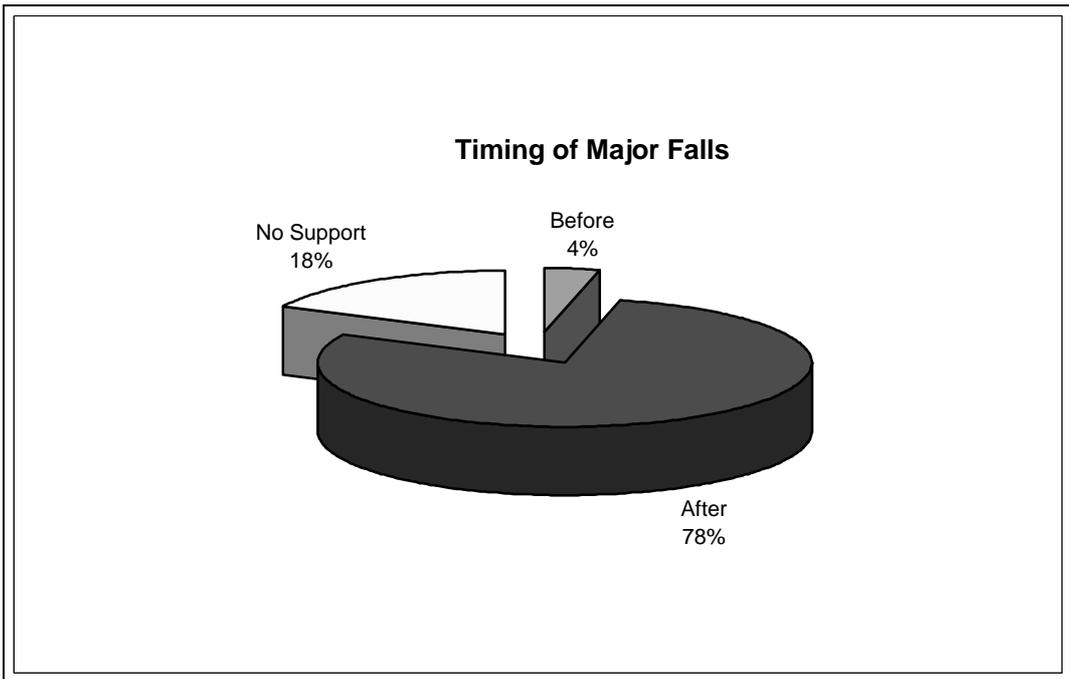


Figure 24. Timing of Major Falls relative to the timing of support installation.

9 Discussion of Causes of Roof Falls

Table 1 is useful to obtain an overview of the contributions of the identified causes of roof falls in the different thickness categories.

Cause	% Contribution to.....			
	All falls	Skin Falls	Large falls	Major falls
Insufficient joint support	38	31	39	50
Excessive bolt spacing	27	38	10	11
Weathering	12	7	24	14
Horizontal stress	9	8	10	14
Bad mining	9	11	10	0
Burnt coal	4	2	7	11
Dykes	2	4	0	0

Table 1. Causes of roof falls for different thickness of falls.

This information is also shown in Figure 25. Inspection of the figure indicates that insufficient joint support and excessive bolt spacing can be considered major contributors to falls of roof, especially in the more frequently occurring thinner falls. It is by attending to these two causes that the greatest improvement will be made to fall of roof accident rates. Insufficient joint support is a consistently high contributing factor, even increasing in importance for the thicker roof falls.

Horizontal stress, weathering and bad mining each contribute at around the ten percent mark to roof falls and should not be neglected. Horizontal stress tends to be a more frequent cause of major roof falls, but is still less of a problem than insufficient joint support. Weathering is, as can be expected, a more important cause of roof falls in older mines, especially where the immediate roof is a material that is prone to weathering like coal, shale or mudstone.

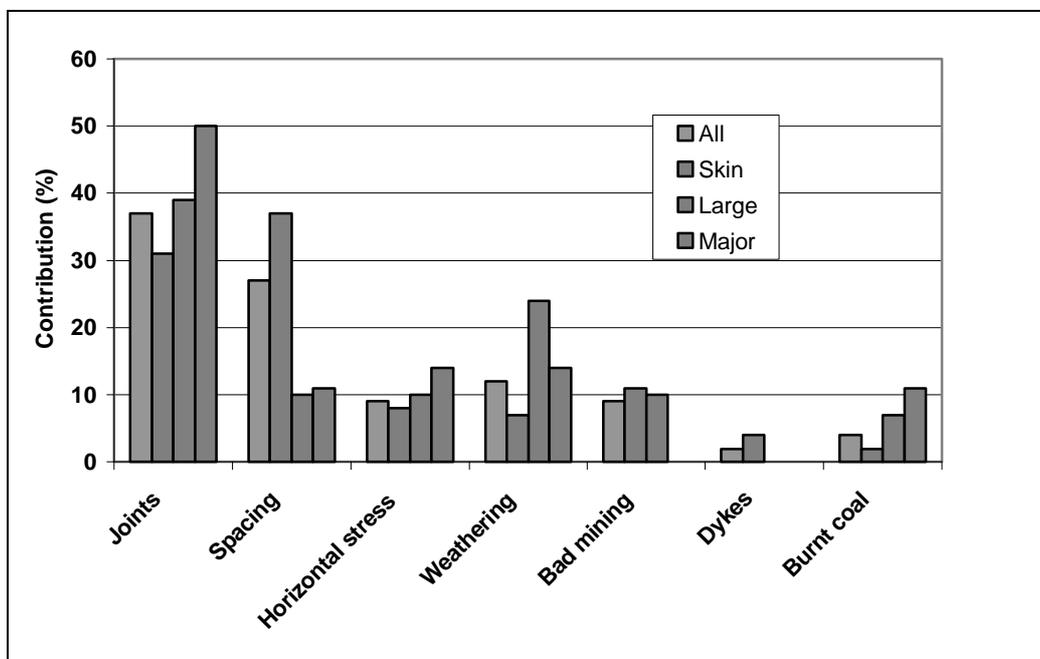


Figure 25. Causes of roof falls seen against the background of thickness of roof falls.

10 Influence of roof rock quality

The influence of roof rock quality was investigated by two methods, namely different rating systems and laboratory testing. This was only done for Phase 1 of the investigation, where the falls were investigated in greater detail.

10.1 Roof Rating Systems

Three rating systems were used, namely the Rock Mass Rating or RMR, Bieniawski (1980), the Modified Rock Mass Rating or MRMR, Laubscher (1990) and the Coal Mine Roof Rating or CMRR, Molinda and Mark (1994).

It is postulated that miners often know when the roof quality deteriorates and that they then take some form of action. Most often, they will reduce the road width. In that case, there should be a correlation between the rating systems and the road widths that were measured. This possibility was tested against the data for the roof falls that were investigated in Phase 1. The results are shown in Figure 26.

It is seen from the figure that the regression line between road width and the MRMR has a negative slope, indicating a negative correlation. The other two rating systems both displayed positive slopes, with R^2 values of 0,44 and 0,52 for the RMR and CMRR systems respectively. It is not surprising that the CMRR correlated better to the data, it having been developed specifically for coal mine roof evaluation.

Mark (1999) reported on CMRR values for South African collieries. He visited a number of collieries in the same geographical area that was investigated in Phase 1 of this investigation and reported CMRR values in the range 45 to 87, average 66. His data was not gathered at roof fall sites and can be seen as representing “healthy” roof. The ratings found in this investigation, performed at roof fall sites, were in the range 8 to 55, average 31, which are considerably lower.

This indicates that although it was developed for USA coal mines, the CMRR has at least some merit for application or further development in South Africa.

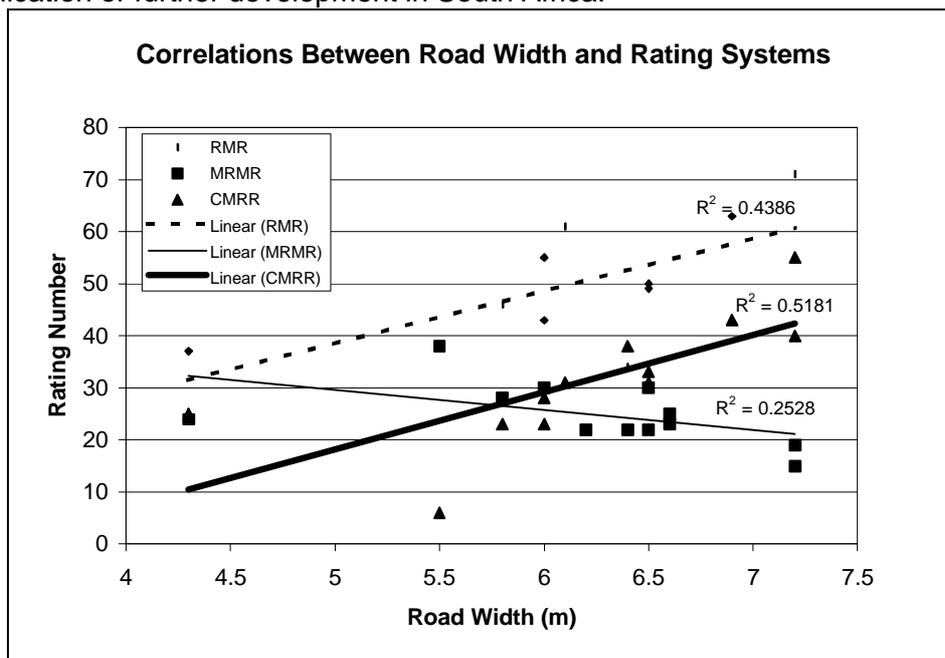


Figure 26. Test of correlation between the rating systems and road width.

Mark (1999) also reported on the correlation between the CMRR and a factor representing the amount of support installed in a mine roof, the PSUP. The PSUP is based on the spacing, length and capacity of bolts and road width; the higher the number, the greater the amount of support. Figure 27 is compiled from Mark (1999). Even though based on limited South African data, the figure is clear: in South African collieries, less support is used for comparable roof conditions than in either the USA or Australia. This supports previous conclusions that in South African collieries, the density of supports needs to be increased.

10.2 Roof Rock Strength

Rock samples were collected from a number of the fall sites, primarily those that were investigated during Phase 1 of the investigation. The following laboratory tests were conducted:

Uniaxial compressive strength

Poison's ratio

Modulus of Elasticity

Uniaxial tensile strength

Direct shear strength

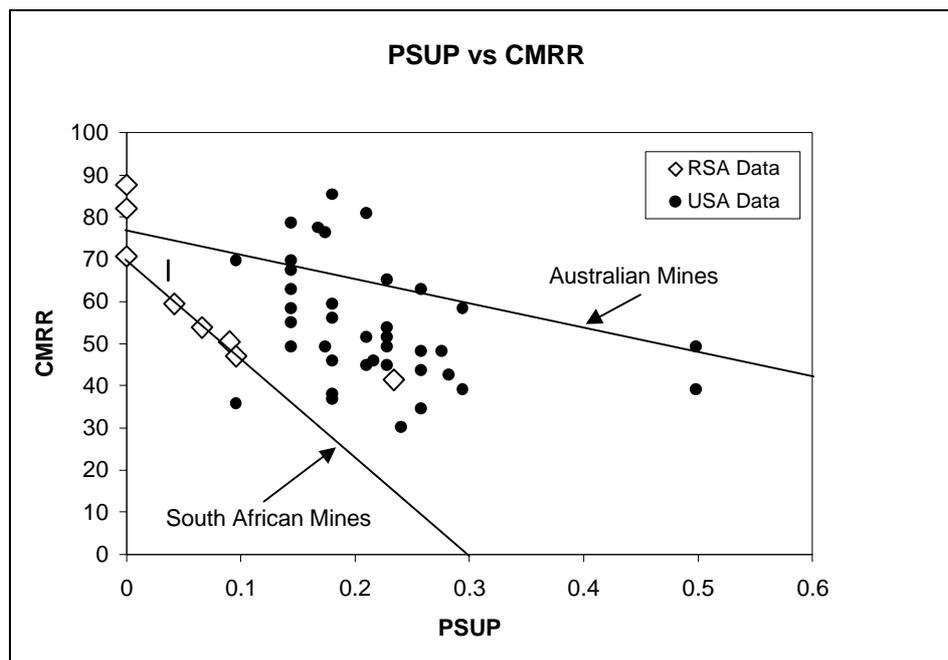


Figure 27. Comparison of the amount of support installed for comparable roof conditions in South Africa, the USA and Australia, after Mark (1999).

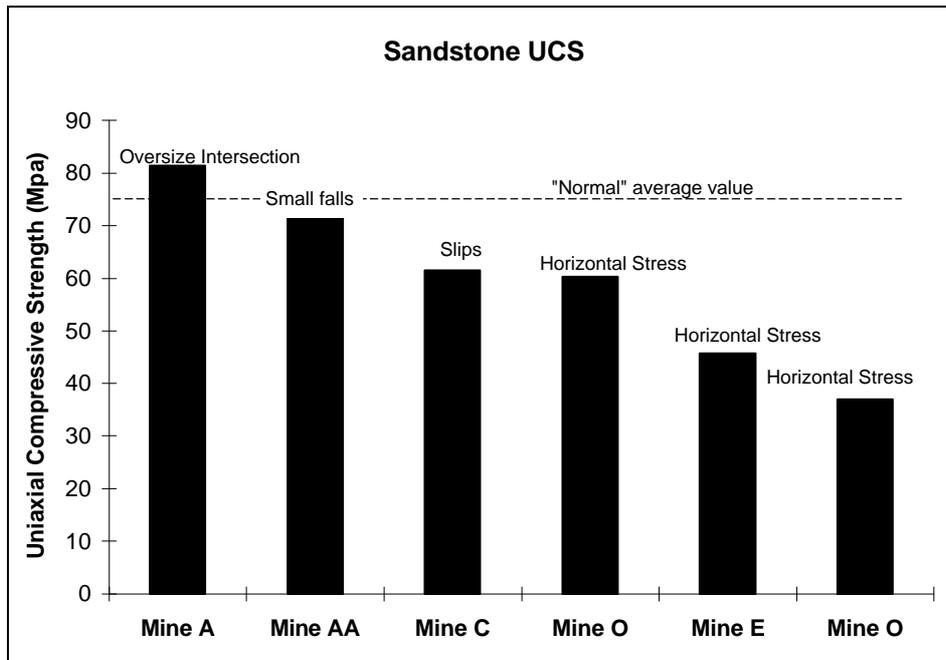


Figure 28. UCS values that were obtained from roof sandstone samples collected at the fall areas.

10.2.1 Uniaxial Compressive Strength of Sandstone (UCS)

The UCS values for sandstone ranged from less than 40 MPa to 80 MPa, see Figure 28. This is less than the “normal” average value of approximately 75 MPa and indicates a reduced compressive strength in the fall areas.

The causes of the falls that were determined by the investigating team are also indicated on the figure. It is significant that the falls that were categorised as having been caused by horizontal stress had the lowest UCS of all the samples. This confirms that horizontal stress is more likely to become a problem in areas with reduced strength.

10.2.2 Modulus of Elasticity of Sandstone (E)

Figure 29 shows the E values that were obtained from the set of samples, again seen against the background of the diagnosed causes of the roof falls.

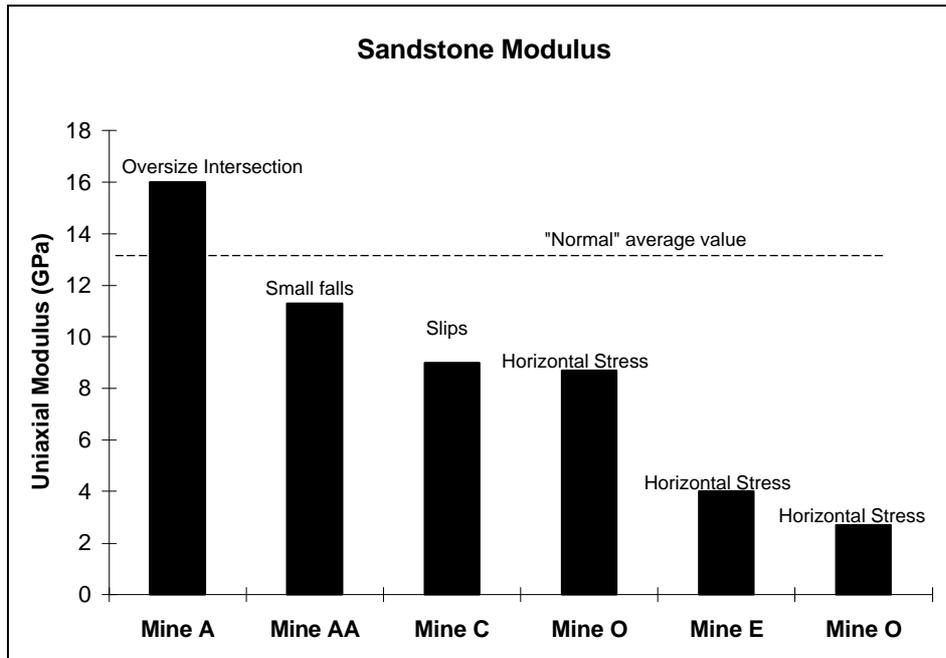


Figure 29. Modulus of Elasticity of sandstone samples from the fall areas.

The previous observation that the “horizontal stress falls” are characterised by reduced numbers, in this case stiffness, is repeated here. With the exception of one case, all the stiffness values are below the “normal” value of 13 GPa.

10.2.3 Shear Strength of Sandstone

The shear strengths of the sandstone samples are shown in Figure 30 in similar format as the previous two figures. The reduction in strength is again apparent, the “normal” shear strength of sandstone being approximately 15 GPa. All the samples had reduced strength, the greatest reduction again being apparent from the samples where horizontal stress was the dominant cause of the roof falls.

10.2.4 Uniaxial Tensile Strength (UTS) of Shale/Mudstone

Curiously, the Shale/Mudstone samples, shown in Figure 31, did not suffer the same shear strength reduction as the sandstone samples. However, the “horizontal stress” samples are again at the low end of the range.

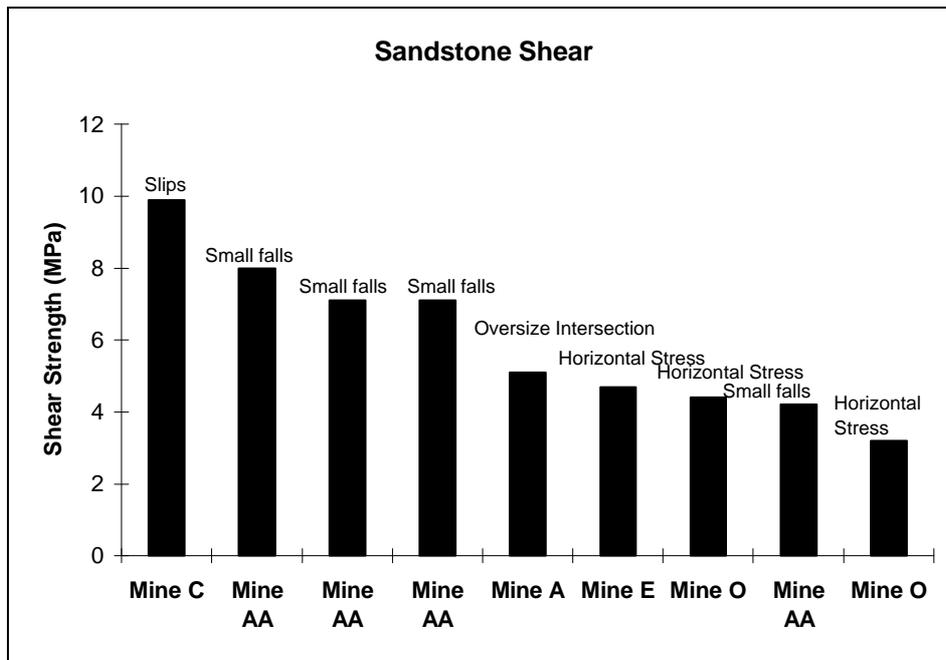


Figure 30. Shear strength of the sandstone samples.

10.2.5 Uniaxial Tensile Strength (UTS) of Sandstone

The UTS of the sandstone samples were obtained by performing the standard Brazilian strength test. The results are shown in Figure 32. In this case, there is no general strength reduction, although the “horizontal stress” samples maintain their consistent position at the low end of the scale.

11 Role of Support Material

The quality of support material was found to be a possible problem in isolated cases. In two cases, bolts that had been rejected by major companies with in house testing facilities were found at the sites of roof falls on small mines without those facilities. While it could not be proven that the inferior material caused the falls, it is suspected to have contributed.

In two other cases, suspicious lengths of cut-off cable were found in places where cable anchors had been installed by outside contractors. The lengths of the cables to be installed are usually specified by contract and are delivered in the correct length. There is no reason to cut cable to the required length underground. However, it was not possible to retrieve full lengths of installed cables from the fall debris to check the installed length of the cables.

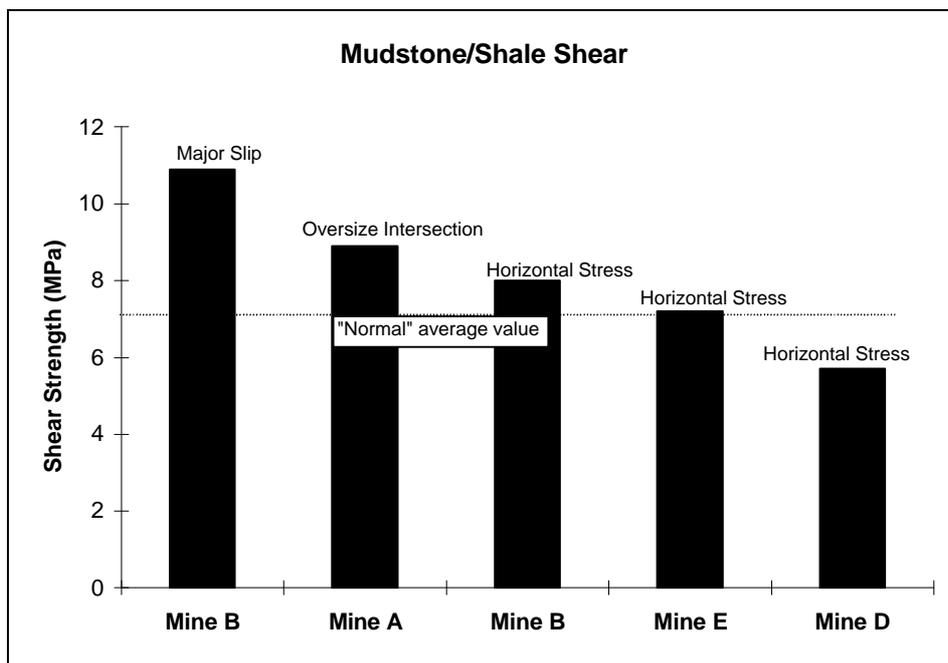


Figure 31. Shear strength of Mudstone/Shale samples

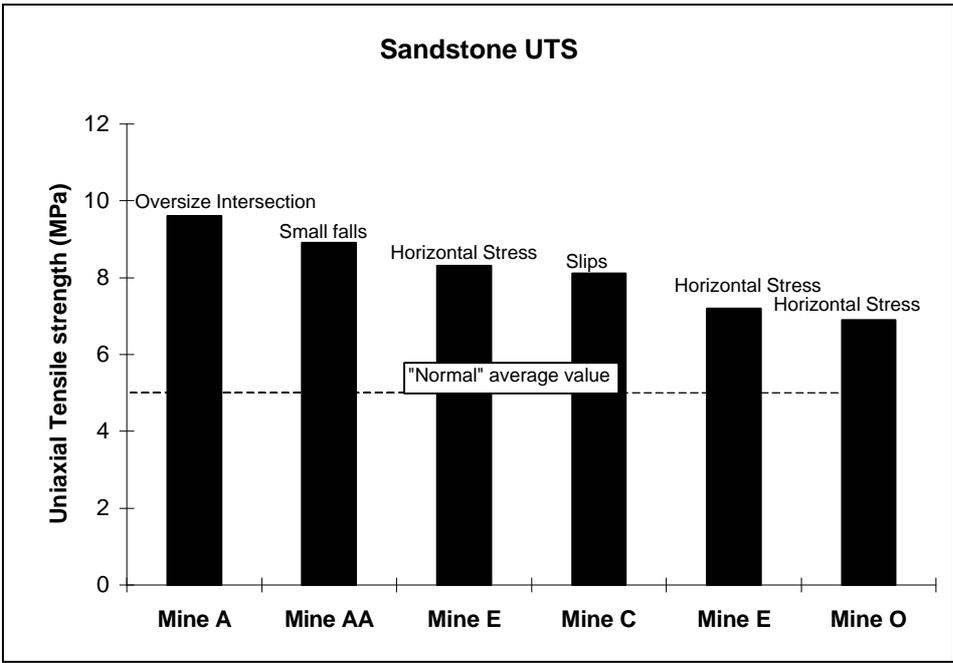


Figure 32. Uniaxial Tensile Strength of the sandstone samples.

12 Discussion and Conclusions

12.1 Most Fatalities Are Caused By Thin Roof Falls

The majority of fatalities in South African collieries is caused by relatively thin slabs of roof rock falling out between bolts. More than 70% of the roof falls are less than 0,5 m thick. The frequency of fatal roof falls diminishes rapidly as the thickness increases. The database of falls that were investigated, is a good simulation of the database of fatal roof falls.

12.2 Characteristics of Roof Falls

Viewing all roof falls, irrespective of thickness, the dominant causes are insufficiently supported joints and bolt spacings that are too wide. Together, these two causes account for 64% of all roof falls. This is followed by a number of contributory causes, namely weathering, horizontal stress and bad mining, each contributing around 10% to the total. Burnt coal and dykes can be considered as minor causes.

Roof falls occur predominantly at intersections where 66% of the total falls occurred, the remaining 34% occurring in roadways. This implies that workers are approximately four times as vulnerable to roof falls in intersections than in roadways.

Almost 70% of all falls occurred after support had been installed, just over 10% before support had been installed and in 20% of the cases no support had been installed.

Slightly different patterns were found for three thickness ranges of roof falls, namely skin falls (up to 0,3 m thick), large falls (between 0,31 m and 1,0 m thick) and major falls (thicker than 1,0 m). The dominance of insufficiently supported joints increased with increasing fall thickness while excessive bolt spacing became less important as a cause of roof falls. Horizontal stress combined with weak roof rock increased as a contributor with increasing fall thickness, being responsible for 14% of major roof falls.

Intersections are the most popular location for roof falls in all size categories, although there is a relative shift in favour of roadways with the thicker falls.

In all categories, the majority of falls occurred after support had been installed, indicating the inadequacy of the support systems. It would appear that the bolts are too widely spaced to prevent thin falls between bolts and that the systems are unable to bear the weight of the major falls. Both of these problems can be addressed by installing bolts on a denser pattern.

12.3 Inadequate Joint Support Is The Most Frequent Cause Of Roof Falls

Despite several years of attention to the support of joints, lack thereof continues to be the single most frequent cause of roof falls in all thickness categories. Whether the solution lies in training, awareness or enforcement is not the subject of this investigation, which is only aimed at establishing the causes of roof falls.

There are several well publicised methods to support joints but they were simply not found to have been implemented underground. In a number of cases, when unsupported joints were pointed out to underground personnel, some reactions were that that particular joint was a stable one and

would not fall, that it was too far from the face to be re-supported, that there was insufficient labour to do it or that if all joints had to be supported the roof would fail under the weight of the steel.

If individuals who intentionally move in under unsupported roof can be dismissed while they endanger only their own lives, then surely the same drastic method of enforcement can be applied to personnel who neglect to support joints, thereby endangering the lives of several other people.

Figures 33 and 34 are further examples of unsupported joints.

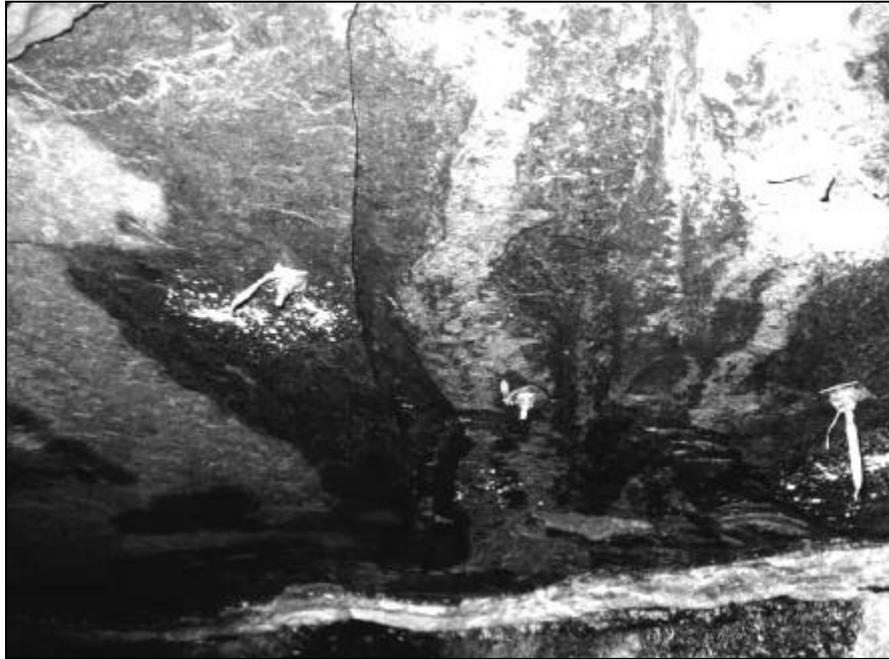


Figure 33. Even though this roof has been re-supported after a fall, the joint is still not adequately supported.



Figure 34. Unsupported joint that resulted in a major fall.

12.4 Bolt Spacing Is Too Wide

The second most frequent cause of roof falls in all categories was found to be excessive bolt spacing. As a result of excessive spacings, a major cause of skin falls is dislodgement of relatively thin roof plates between and around bolts, like the example shown in Figure 35. A second consequence is that bolts are unable to sustain the weight of larger falls, as shown in Figure 36.

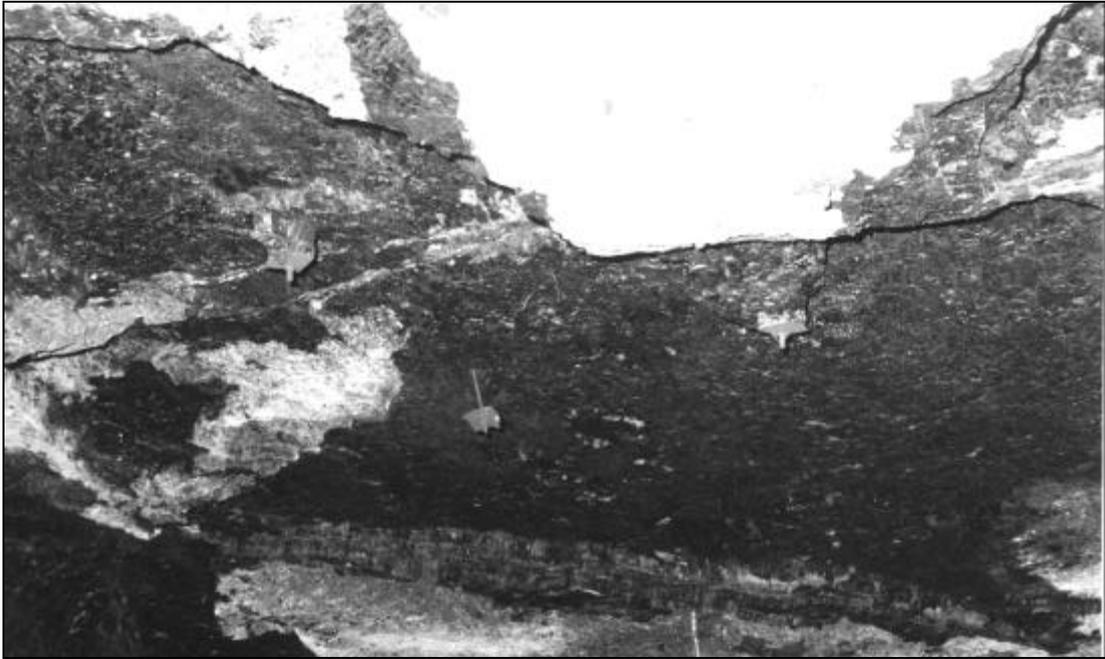


Figure 35. Skin fall caused by failure between bolts that were spaced too widely.



Figure 36. Major fall that was caused by a bolt system failing at the weakest link, the resin contact. A denser bolt spacing would have prevented this fall.

It was also indicated that in South Africa, less bolts are used for comparable roof conditions than in the USA and Australia. Although this indication is based on limited data, it conforms to visual observation over several years.

This problem can be overcome by increasing the density of bolts or by applying some form of aerial cover - see Bauer and Dolinar (2000) - like wiremesh, wider head plates, synthetic material mesh, spray-on membrane or even the "safety net" that is under development for use in gold mine stopes. Addressing this problem will also alleviate the problem of inadequate joint support.

12.5 Under Performance of Support Material Is Not a Major Problem But Should Not Be Tolerated At All

While under performing materials were only found in isolated cases, the frequency should be zero. The fact that material that was known by the supplier to be sub standard was found at two small mines without the benefit of a full time rock engineering practitioner, was disturbing.

In two other areas, cable cut-offs that were found at the sites of collapses raised suspicion but could not be proven to have contributed to the fall because it was not possible to determine the lengths of cables that were installed. Figure 37 shows one of the cable cut-offs.

In one fall, it was found that the nuts had stripped off the bolts, shown in Figure 38, resulting in a major fall of roof. The thread should not be the weakest part of the bolt assembly. It is suspected that in this case, either the diameter of the thread was too small or the diameters of the nuts were too wide.



Figure 37. Was the fall in the background caused by cutting lengths off cable anchors like the one shown in the photograph?



Figure 38. The smooth appearance of the thread on this bolt was due to the nut having stripped off on it.

The use of sub standard material can be prevented by implementing a regular and scientifically designed testing procedure on all mines. This is a requirement of the Guidelines for the Compilation of Codes of Practice to Prevent Rock Related Accidents on Collieries, but it is not always implemented.

When appointing sub contractors for special support installation, price should not be the only consideration. There should also be strict supervision during the construction process.

12.6 Weathering Is Not a Major Cause of Roof Falls In Young Mines

In the single old section that was investigated, weathering was found to be the dominant cause of roof falls. In other cases, it played a minor role. However, there were often signs of Acid Mine Drainage (AMD) at roof falls, even in younger mines. It is recommended that roof rocks' susceptibility to acid attack be evaluated.

12.7 Burnt Coal, Dykes And Bad Mining Are Relatively Minor Contributors to Roof Falls.

Burnt coal and dykes are known hazards in coal mining. Perhaps for that reason, dyke areas and the surrounding burnt coal areas are usually well supported.

Under the category of "Bad mining" are included matters like poor horizon control with continuous miners, misdirected blast holes in conventional mining and patently wrong mining like breaking intersections away underneath obvious joints. The incidence of this was found to be low in the investigation. However, it did occur, and where it did, it tended to persist for long distances. This may indicate development by a less disciplined crew.

12.8 Horizontal Stress Signs Coincided with Weak Roof Material.

It has often been said that failure of anything, a mine roof included, is the result of an imbalance between the load on the object and its strength. Therefore, for roof failure to occur there are always two matters to be considered, namely the strength of the roof material and the stress acting on it. It is misleading to consider only one of the two sides of this equation.

There has been a recent trend in South Africa to over emphasise the role of horizontal stress in colliery roof failures. What this investigation found, is that the areas where horizontal stress was deemed to have been the major cause of roof falls, were also characterised by significant weakening of the roof material.

The conclusion reached from this is that even where the magnitude of horizontal stress is low, it may result in failure in areas where the roof material is weak. Therefore, under the conditions of relatively shallow coal mining as in South Africa, the problem is not as much the magnitude of the stress as the local strength reduction of the roof material.

Fortunately, it is easier to monitor the strength of the roof than it is to monitor the magnitude of the stress. There are several ways to monitor the roof strength, including laboratory tests, simple on site point load tests, splitter index tests on core, etc.

The incidence of roof falls due to the manifestation of horizontal stress signs was relatively low. Where it was apparent, the falls tended to be major ones.

References

- Bauer, R. and Dolinar, D. (2000).** Skin Failure of Roof and Rib and Support Techniques in Underground Coal Mines. Proc. New Technology for Coal Mine Roof Support. NIOSH IC9453.
- Bieniawski, Z.T. (1976).** Rock Mass Classification in Rock Engineering. Proc Symp Exploration for Rock Engineering, Ed Z.T. Bieniawski. AA Balkema, Cape Town.
- Canbulat, I and van der Merwe, J.N. (2000).** Safe mining face advance and support installation practice in mechanical miner workings under different geotechnical conditions. Safety in Mines Research Advisory Committee, Department of Minerals and Energy, Johannesburg, South Africa. March 2000.
- Canbulat, I. and Jack, B. (1998).** Review Of Current Design Methodologies To Improve The Safety Of Roof Support Systems, Particularly In The Face Area, In Collieries. Safety in Mines Research Advisory Committee, 1998.
- Jack, B. (2000).** Roof Behaviour in South African Coal Room and Pillar Panels. 19th Conference on Ground Control in Mining, Morgantown, West Virginia.
- Laubscher, D.H. (1990).** Geomechanical Classification System for the Rating of the Rock Mass in Mine Design. South Afr. Inst. Min. Metall. Volume 10 1990 pp 257 – 273.
- Mark, C. (1999).** Ground Control in South African Mines – a US Perspective. 18th Conference on Ground Control in Mining, Morgantown, West Virginia.
- Mark, C. and Molinda, G. M. (1994).** Evaluating Roof Control in Underground Coal Mines With the Coal Mine Roof Rating. 13th Conference on Ground Control in Mining, Morgantown, West Virginia.
- Moebs, N. and Stateham, R.M. (1986).** Coal Mine roof Instability: Categories and Causes. United States Bureau of Mines IC 9076.
- Molinda, G., Mark, C., Bauer, E., Babich, D. and Pappas, D. (1998).** Factors Influencing Intersection Stability in US Coal Mines. 17th Conference on Ground Control in Mining, Morgantown, West Virginia.
- Molinda, G.M, Mark, C. and Dolinar, D. (2000)** Assessing coal Mine Roof Stability Through Roof Fall Analysis. Proc. New Technology for Coal Mine Roof Support. NIOSH IC9453.
- Van der Merwe, J.N. (1998)** Simple User's Guide to the Installation of Resin Roofbolts. Safety in Mines Research Advisory Committee, Department of Minerals and Energy, Johannesburg, South Africa. March 1998.
- Vervoort, A. (1989).** A Review of Fall of Ground Fatalities in South African Collieries During the Period 1970 – 1988. Sangorm Symposium: Advances in Rock Mechanics in Underground Coal Mining. South African National Group on Rock Mechanics.
- Vervoort, A. (1989).** Support Design in Coal Mine Roadways and Intersections. Sangorm Symposium: Advances in Rock Mechanics in Underground Coal Mining. South African National Group on Rock Mechanics.
- Vervoort, A. and Jack, B. (2000).** Three-Dimensional Simulations of the Roof Behavior in Coal Room and Pillar Panels. 19th Conference on Ground Control in Mining, Morgantown, West Virginia.

Appendix I - Individual site investigation reports

Case A Summary: Dimensional imbalance

Mine locality: Highveld coalfield

Seam: No 4

Mining method: Bord and pillar with 70° turn-off (Joy 12HM31 with continuous haulage). Pillar centres in direction of roadways were 28 m, road widths nominally 7,2 m.

Mining depth: 120 m

Mining height: 2,9 m

General description of roof: Sandstone beds (0,28 and 0,15 m thick) separated by laminated shale. Shale content of fall 82%. In general, beds varied in thickness from 4 to 30 cm – see Figure 1.



Figure 1. Illustration of the laminated nature of the roof. Note the variation in thickness of the light sandstone layer at the bottom of the roof.

Discontinuities present: Small joints, possibly inconsequential

Sandstone MRMR: 28

Sandstone hydraulic radius: 8 m

Sandstone UCS: 7,4 MPa

Shale MRMR: 19

Shale hydraulic radius: 2,3 m

Shale UCS: 4,3 MPa

Combined CMRR: 40 (Weak roof)

Combined MRMR: 19

Combined hydraulic radius: 10 m

Combined UCS: 4,1 MPa

Time of fall: Several weeks after development

Supported/unsupported: Supported, after beds had separated.

Support: 1,5 m M20 Ausbar full column dual speed resin, 25 mm holes, 4 bolts per row 2 m apart.

Cut-out distance: According to standard 16 to 24 m depending on roof conditions. Manager prefers 18 m. Uncertain what it was at fall position.

Support quality: Good

Monitoring: Tell tales in general area, none found at fall position. Tell tales on mine are 1,8 m long, no marks found in roof after fall (1,6 m high), assumed there was no tell tale at fall position. According to mine overseer no warning signs day before fall.

Locality: Intersection

Horizontal stress indicators: Non detected

General description of fall: Intersection fell to height of 1,6 m, lateral dimensions 14,8 m by 13,6 m. Area of fall 155 m² – see Figures 2 and 3.



Figure 2. General view of the fall area.

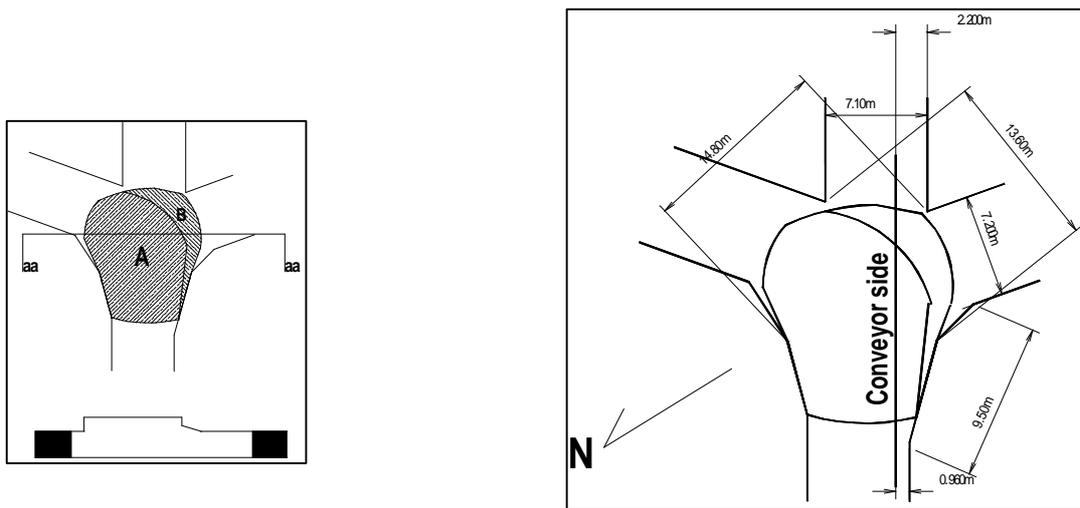


Figure 3 Plan view of the fall

Cause of collapse

The roof collapsed due to a combination of three adverse factors that were present. Firstly, the intersection was cut to excessive dimensions for the roof composition in this locality. Secondly, the supports were too short – being 1,5 m long in a 1,6 m high fall of roof. Thirdly, there was evidence of the supports only being installed after the roof beds had separated – see Figure 4. It is possible that the fall could have been prevented had any one of the three factors not been present.

This was an isolated fall and the only discernible difference between the fall locality and the immediate surrounding area was that all the other intersections were smaller, averaging 11 by 10 m and with a maximum single dimension of 12 m. It was not possible to determine the thicknesses of the roof layers in the surrounding area, although they were seen to vary over small distances at the fall – 15 cm to 7 cm over a lateral distance of 12 m.



Figure 4. *The white material on the roof next to the top end of the bolt is resin, indicating that the bolts were only installed after the roof layers had separated.*

There were 18 other intersection falls in the wider area that was not visited. These occurred over a traverse of several hundred metres and were not inspected. There were no other falls within 200 m of the inspected one.

The roof composition is given, but the length and spacing of supports, the time of support installation and the mining dimensions are controllable. Roof stability is achieved by combining the controllable parameters in such a way that the disturbing forces in the roof do not exceed the strength of the material, or by containing the failed material and preventing it from falling.

Effect of dimensions

The tensile stress generated in a roof is directly proportional to the square of the minimum dimension. Therefore, in this case, the tensile stress generated in the roof was 44% greater than that at any of the other intersections in the area.

Effect of bolt length

Longer bolts would have been able to suspend the roof, even though the roof material had failed. Assuming a bolt density of 3 m² per bolt and 25 mm hole diameters, bolts of 1,8 m length would have suspended this roof.

Effect of time of support installation

If the support can be installed before the beds separate, it is possible to combine the layers into a thicker plate. The tensile stress generated in a roof is inversely proportional to the beam thickness. There was evidence that the beds had already separated by the time the bolts were installed (see Figure 4) and therefore the roof behaved like a number of independent plates rather than a thicker, single one. Because of this, the generated tensile stress was more than twice as high as it could have been, had the bolts been installed sooner.

Conclusion

This fall occurred because there was an imbalance between the thicknesses of the roof layers, the size of the intersection and the length of the bolts.

Given the circumstances, the most important immediate controllable factor was the intersection size. However, against the background of:

- unknown roof composition but seen to be variable,
- mining method that often results in large intersections to accommodate the continuous haulage, and
- the other 18 falls that had occurred with the same equipment in the same traverse,

The simplest prevention would have been to install 1,8 m long bolts.

Case B Summary: Slips

Mine locality: Highveld Coalfield

Seam: No 4

Mining method: Double header 16 road bord and pillar with Joy 12HM17 and 12HM21 continuous miners, right hand side 2 lines ahead. Road width was 6.6 m.

Mining depth: 57 m

Mining height: 4.5 m

General description of roof: 15 to 60 cm fine grained muddy sandstone (also logged as fine grained sandstone with sandy mudstone) followed by +/- 2 m of mudstone with sandy mudstone with a thin (15 cm) sandstone layer at the base. Note: the mine geologists do not distinguish between mudstone and shale.

Discontinuities present: A thin dyke (0.5 m) is still to be exposed where the falls occurred and has been exposed by the leading roadways to the right. A major slip, vertical displacement 0.5 m, with sympathetic smaller slips was present at the major fall site.

Time of fall: During shift, as CM was in the process of pulling back.

Supported/unsupported: Unsupported

Support: M16 x 1.5 m mechanical anchors, 3 per row at 2 m spacings close to the fall, elsewhere 2 bolts per row at 2 m row spacing.

Cut-out distance: 25 m

Support quality: Medium to good

Monitoring: None

Locality: Intersection with major slip

Horizontal stress indicators: None

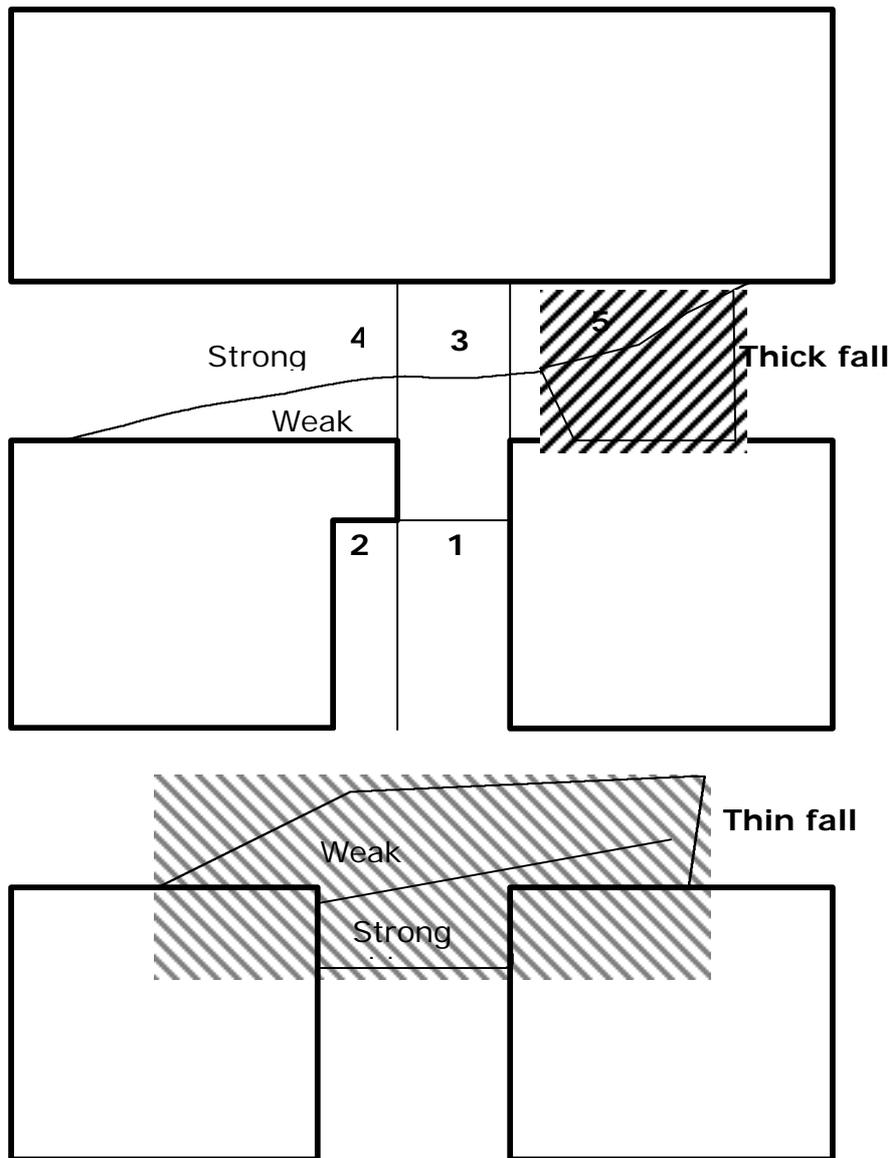


Figure 1 Diagram showing the positions of the two falls and the cutting sequence.

General description of fall: The fall positions and mining sequence are shown in Figure 1. There were two falls. The thin one was approximately 0.1 m to 0.3 m high by 11 m by 7 m (77 m²), either side of a small slip with approximately 0.1 m displacement. The exposed roof above showed signs of dampness but there was no free flowing or dripping water, see Figure 2.

The thick fall, shown in Figure 3, was approximately 0.5 m high by 12 m long and on average 4 m wide (48 m²). It occurred on the weak side of the major slip. On the previous shift, the slip was uncovered in the roadway. The second lift was then not cut, in order to stabilise the slip. The split was then cut from left to right, through the slip and beyond to hole in the next roadway. As the CM pulled out, the roof collapsed. The fall occurred between the position where the exposed weak side of the slip was approximately 3 m wide to where it disappeared into the ribside.

Cause of collapse

The small fall occurred prior to support being installed in an area where the nether “sandy mudstone” was thin. The minor slip running through the fall was sufficient to split the beam into two

cantilevers. Without the presence of the slip, the maximum tensile stress at the edges of the beam would have been:

$$s = \frac{gL^2}{2t} \quad [B.1]$$

where g = unit weight of rock
 L = length of beam
 t = thickness of beam,

or approximately 3 MPa. However, with the slip in place, the clamped beam became a cantilever, with maximum tensile stress:

$$s_c = \frac{3gL^2}{t}, \quad [B.2]$$

or approximately 18 MPa, which is more than the tensile strength of about 5 MPa.

At the position of the large fall, the bottom layer had become substantially thicker and thus stronger. Once again, however, the slip turned the beam into a cantilever.

At the far end of the split, where the fall started, the entire road width was on the weak side of the slip. At a beam length of 6.6 m, the generated tensile stress was approximately 6 to 7 MPa which is just greater than the tensile strength. The fall then occurred, the plate breaking back dynamically to where the beam length was about 3 m.

Where the slip was first exposed in the roadway, it's strike was parallel to the split. When the split was then cut, the CM operator possibly assumed that it would maintain a



Figure 2 *Detail of the roof of the thin fall.*



Figure 3 Overall view of the thick fall

constant direction. He may not have been able to see that it had turned closer to the left-hand ribside.

The split was followed in other roadways across the section, where it had not resulted in a fall, see Figure 4. In all the other positions, it either daylighted in a roadway (in other words, crossing the excavation as opposed to running along it) or had a short exposed weak side. It was also well supported, indicating that personnel had become wary of it.

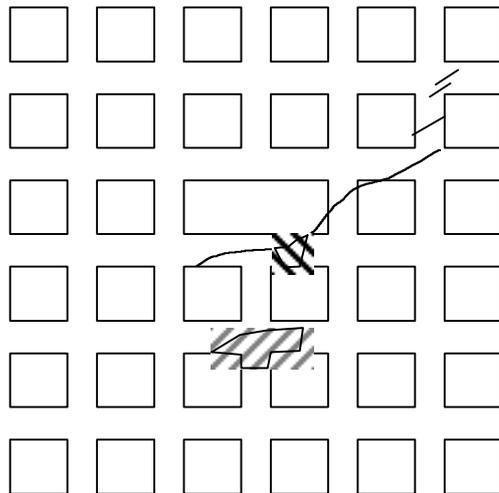


Figure 4 Sketch plan showing the continuation of the slips through the section.

Conclusion

Both of these falls are fully explained by the presence of the unsupported slips, changing the basic behaviour of the roof to that of a cantilever with the accompanying drastic increase in tensile stress.

Prevention

The fall could have been prevented if the slip had been supported at short intervals, i.e. by reducing the cut out distance. On this mine, it was seen that operator awareness of roof hazards was at a high level – yet they were possibly lulled into a false sense of security when the slip did not fall after its initial exposure.

Case C Summary: Poor support and directional control in weak roof, complicated by horizontal stress.

Mine locality: Witbank Coalfield

Seam: No 2

Mining method: Continuous miner bord and pillar, 9 road section operating as two sections feeding onto a common belt, the right hand 4 roads leading by one split.

Mining depth: 47 m.

Mining height: 2 – 2.5 m, following the seam height. Numerous floor rolls, restricting mining height at short intervals.

General description of roof: Bottom roof is 1.0 to 0.9 m thick laminated coaly shale and sandstone bands, average 0.2 m thick, overlain by jointed weathering shale, see Figure 1. Several discontinuous sandstone lenses (more like brecciated plates) in roof, shown in Figure 2.

Discontinuities present: Numerous flatly dipping slips (see Figure 3) affecting the bottom 1.5 m of roof. Average trace length 12 m. Prominent slip at fall position, 20 m long, 30° dip, rough undulating surface.

Roof CMRR: 31 (zero stand up time)

Roof MRR: 61 (1 month stand-up time)

Roof UCS: 18 MPa (DRMS)

Roof MRMR: 25 (Unstable)

Internal Angle of Friction: 30° – 35°

Cohesion: 100 – 150 kPa

Time of fall: Between shifts.

Supported/unsupported: Supported.

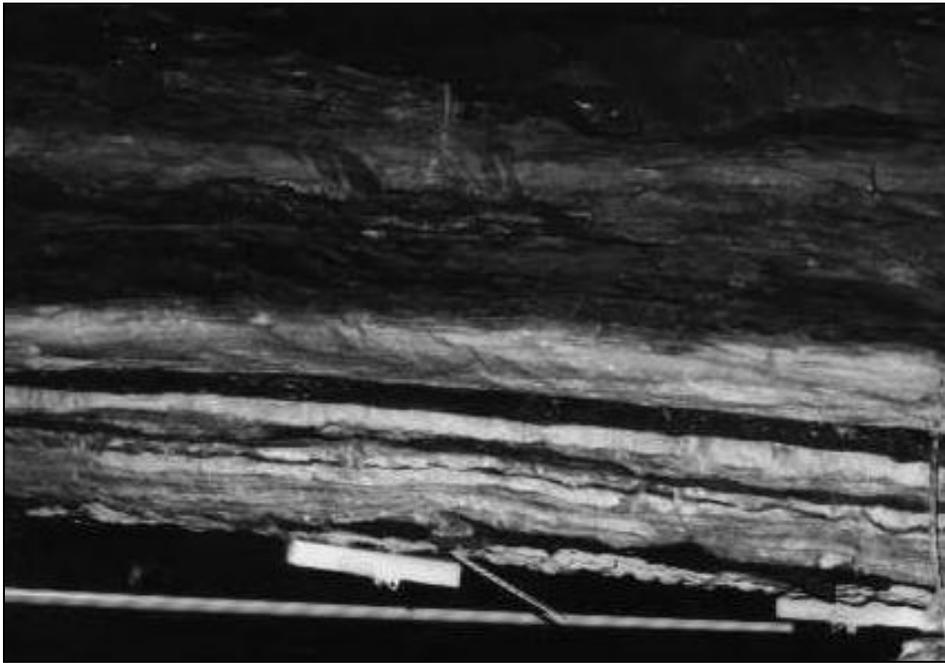


Figure 1 *Laminated nature of the roof.*



Figure 2. *Discontinuous sandstone remnant in the roof, arguably the result of differential settlement of the floor of the seam.*

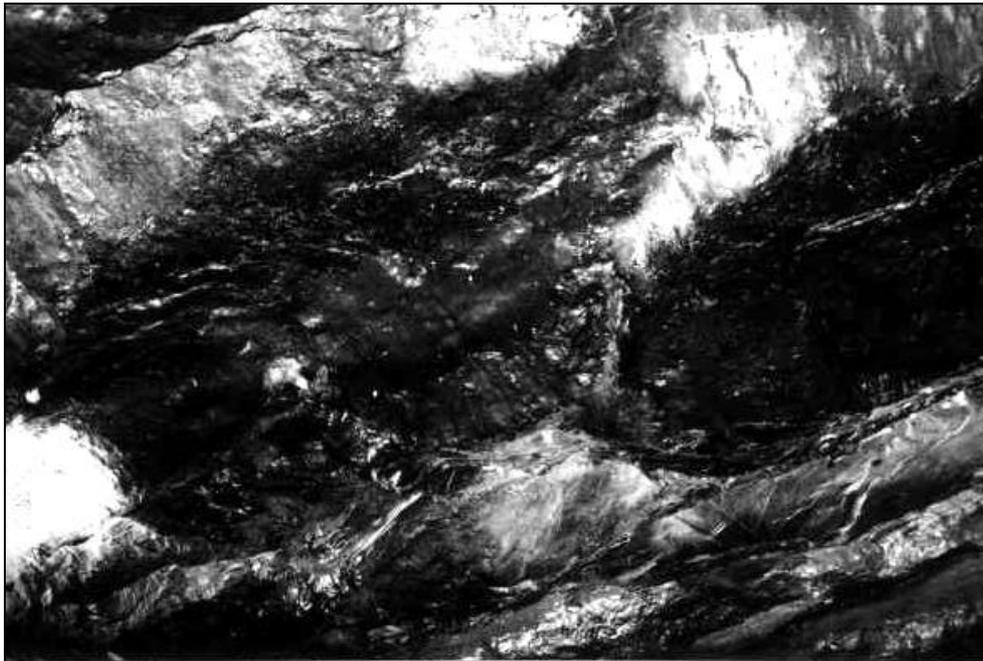


Figure 3. Dense pattern of flatly dipping joints.

Support: 1.5 m long 14 mm twisted square bar in 22 mm holes, semi-full column resin (1.0 m anchor as per standard). Pattern 3 bolts per row spaced 1.5 m, first lift 8 m long supported before second lift is cut. At fall and most other slips, 1.8 m long by M16 rebar full column resin was used at dense (< 1 m) spacing. Headboards were common.

Cut-out distance: 8 m.

Support quality: Poor. Holes either over drilled or not enough resin capsules used, resulting in anchors only being 1.0 to 0.4 m long. Insufficient tension observed on several bolts (loose washers). Number of bolts with stripped nut thread on them found at fall. *(Note: similar to previous experience with twisted square bar. Grooves run through threads, which misalign when torque applied to bolt, impeding crimp nut passage, excessive torque on bolt which develops micro steel fractures, fails at low loads later. Alternatively, thread on nuts stripped off. Suppliers informed of problem several years ago.)*

In general, the directional control over drivage was poor resulting in irregularly sized pillars and intersections see Figure 4.

Monitoring: None

Locality: Intersection



Figure 4. Mine plan of the general area in which the fall occurred.

Horizontal stress indicators: Sporadic horizontal stress indicators evident, see Figure 5, sufficient to determine direction of maximum to be NW-SE. No stress indications within two pillars of fall (20 m) and no over thrusting or other horizontal stress indicators seen in sides of fall.



Figure 5. One of the sporadic occurrences of guttering.

General description of fall: The fall was approximately 1.4 m high in an intersection that was oversized (8.5 m x 10.5 m) due to a corner of a pillar being cut out, see Figure 6. Figure 7 is a general view of the fall. Two slips intersected in the central region of the fall.

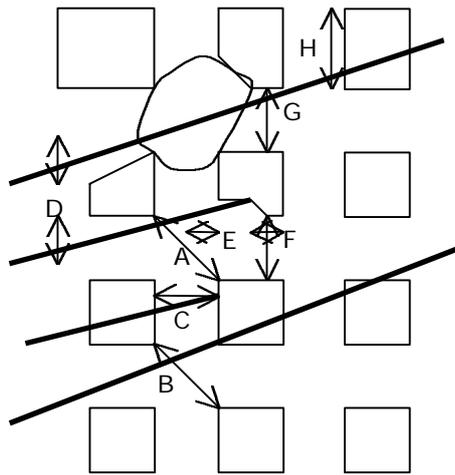


Figure 6. Sketch of the fall. The following dimensions were measured at the fall:

$A = 10.5\text{ m}$, $B = 8.4\text{ m}$, $C = 5.9\text{ m}$, $D = 3.0\text{ m}$, $E = 2.7\text{ m}$, $F = 6.1\text{ m}$, $G = 6.1\text{ m}$.



Figure 7. General view of the fall

Cause of collapse

This roof was at the lower extreme of South African roof conditions, comparable with “poor” roof in the USA. Under the circumstances, a dense, stiff support system would have been more appropriate, coupled with strict control over mining directions and dimensions.

The contribution of the horizontal stress indicators could not be established. Given the flatly dipping slips, it must have played a role in destabilising the roof at the slip positions, even though it is assumed that the magnitude was not very high. With roof as weak as was observed, even a moderate stress would have resulted in more damage than was observed. But whether the fall would have occurred without any horizontal stress at all is also possible.

The headboards per se did not contribute to the fall, rather, conditions would have been worse without their additional areal coverage. Over the longer term, as the timber deteriorates, they may have a negative influence, but over the short term they are better than merely using steel washers.

Conclusion

It is concluded that this fall was due to poor support, poor directional control and possibly sub standard support materials under the conditions of a weak, flatly jointed roof with the added complication of horizontal stress.

Case D Summary: Fall 1 unsupported slip, Fall 2 well installed but incorrect type of support.

Mine locality: Witbank Coalfield

Seam: No 2

Mining method: Conventional bord and pillar followed by selective top coaling.

Mining depth: 75 m

Mining height: Initially +/- 3.5 m, after top coaling 4.5 m to 6.5 m.

General description of roof: Coal of varying thickness (0 to 2 m) followed by 0.5 to 1.0 m of alternating shale/sandstone bands of 10 to 30 cm thickness, then laminated shale.

Discontinuities present: Number of calcite filled continuous NW-SE striking slips spaced 15 – 20m apart. Numerous shorter NW-SE striking joints, spacing varied from 2 m too less than 0.5 m.

Time of fall: Unknown. Area was mined +/- 20 years ago.

Supported/unsupported: Fall 1 was not supported, fall 2 was supported.

Support: 1.5 m M16 mechanical anchors.

Cut-out distance: Unknown, but as the mining method was drill and blast, not more than 2 m.

Support quality: Good at time of installation.

Monitoring: None

Locality: Fall 1 at an intersection, fall 2 intersection and roadway.

Horizontal stress indicators: None

General description of falls

Figure 1 is a plan view of the falls.

Fall 1: Wedge shaped at intersection against slickensided joint, see Figure 2. Fall height appeared to be 0.5 m. It could have been more than a metre on one side, although the higher area on the southern side was where the top coaling stopped. The extra height may have been top-coaled and not fallen.

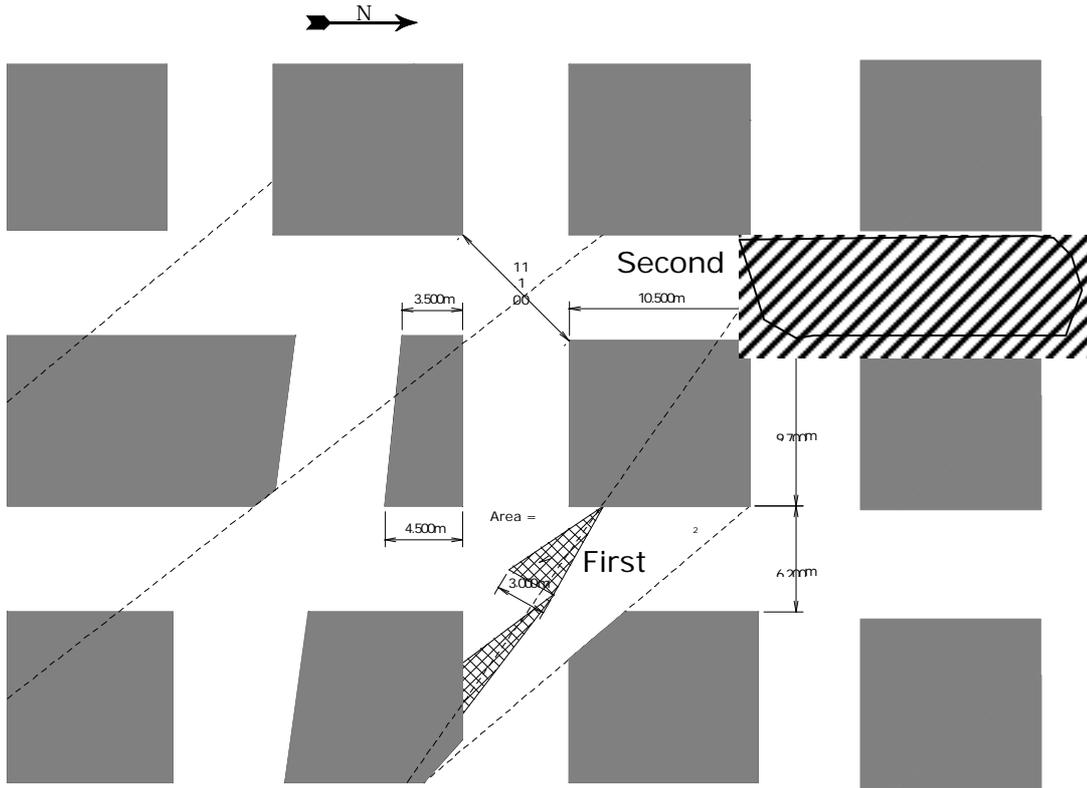


Figure 1 Plan view of the two falls.



Figure 2. View of the slip that bounded Fall 1 on the northern side.

Fall 2: Fall 2 covered a wide area, interspaced with top coaling. The fall height was exactly 1.5 m, the length of the bolts. Each bolt position could be seen, indicated by brown circular discoloration, see Figure 3. The bolts appeared to have been well tensioned, indicated by the rock columns on top of washers, shown in Figure 4.

One of the fallen rocks contained a roofbolt hole. Radial stress fractures emanated from the hole position, shown in Figure 5, indicating that at the time of installation the expansion of the anchor resulted in a localised stress that exceeded the compressive strength of the shale.

Cause of collapse

Fall 1 was caused by the slip resulting in a cantilever. It was possibly triggered by blasting during the top coaling operation. It had the appearance of being aided by horizontal stress, but no other indications of horizontal stress could be found. In the soft roof, such signs should have been visible if the horizontal stress had been higher than normal.

Fall 2 appeared to have been caused by the bolts. It is postulated that the mechanical anchors caused stress fracturing at the ends of the holes when they were expanded. The bolt density was seen to have been dense (less than a metre spacing in places) and over time resulted in a plane of fracturing at the anchor horizon. Rust marks on the roof indicate that water was also present, or at least that moisture accumulated in the minute openings caused by the stress fracturing. Eventually a loose layer was formed with the upper boundary at the anchor horizon and the collapse occurred.

The collapse may have been aided or even initiated by the top coaling operations in the section. That this collapse was not part of the top coaling mining operation, is indicated by the fact that no material was loaded out, as was done elsewhere.

Conclusion

Fall 1 was caused by the unsupported slip.

Fall 2 was caused (or aided) by the wrong choice of support for the existing conditions. Resin anchors would have afforded better anchorage without inducing the radial stress in the surrounding rock. From all appearances, the bolts were well tensioned and installed at a dense spacing. While this is normally sound mining practice, and commendable in the light of the observed jointing, it aggravated the situation in this case.

Prevention

Fall 1 could have been prevented by supporting the slip and Fall 2 by using resin instead of mechanical anchors.



Figure 3. *View of the discoloration at the anchor positions of the bolts, also showing the high density of slips.*



Figure 4. *View of a rock column on top of the washer, indicating sufficient pretension on the bolt.*



Figure 5. View of the stress damage around a roof bolthole.

Case E Summary: Horizontal stress

Mine locality: Witbank Coalfield

Seam: No 5

Mining method: Continuous miner bord and pillar, overmining No 2 seam with 30 m parting. Bord width 6.9 m.

Mining depth: 40 m

Mining height: 2.4 m

General description of roof: Silty sandstone (0.36 m), followed by laminated sandstone (0.2 m), then laminated silty sandstone (0.24 m), then shale (0.15 m) and finally sandstone with shale bands (0.1 m). Roof slightly weathered, susceptible to weathering. Dripping water was evident, see Figure 1.



Figure 1. *Water dripping from the roof.*

Discontinuities present: Major slip, planar and slickensided, through centre of fall.

Roof MRMR: 22

Roof UCS: 6 MPa

Roof CMRR: 43

Roof RMR: 63

Time of fall: Not known.

Supported/unsupported: Supported

Support: Full column resin M16 by 1.5 m long rebar, 3 bolts per row at 2 m spacing.

Cut-out distance:

Support quality: Medium. Holes not filled, powdery resin observed in fall, indicating over mixing of resin.

Monitoring: none at fall.

Locality: Intersection.

Horizontal stress indicators: A stress damage survey of the workings indicated that the maximum horizontal stress has a direction of 185degrees. The area where the fall of ground occurred shows signs of stress induced damage. This mainly took the form of scaling of a roof siltstone bed (that bounded the fall) and roof guttering, see Figure 2. This cutter extended some 14m into the corner of a pillar. At this position rock flour and shearing of a seam joint was noticed. Over thrusting of the nether roof beds was evident, see Figure 3. The stress damage rating (SDR) values for the fall of ground area ranged from 3 to 6 (Deep cutter to roof fall conditions). The SDR values for the area surrounding the fall ranged from 2 to 3 (slight to deep cutter conditions). All stress damage was confined to the roof, with all ribs being stable.

General description of fall: A plan view of the fall is shown in Figure 4 and a general view in Figure 5. The fall originated in the intersection and migrated into the split to the right. From there it crossed the next roadway and caused a cutter on the opposite ribside. The cutter/overthrusting continued for at least another roadway where it caused damage to a pillar corner. The fall height was 0.7 m at the intersection. The continuation fall was confined to the nether 0.3m skin of the roof.



Figure 2. *Buckling of the nether roof layer*



Figure 3. *Over thrusting of roof layers.*

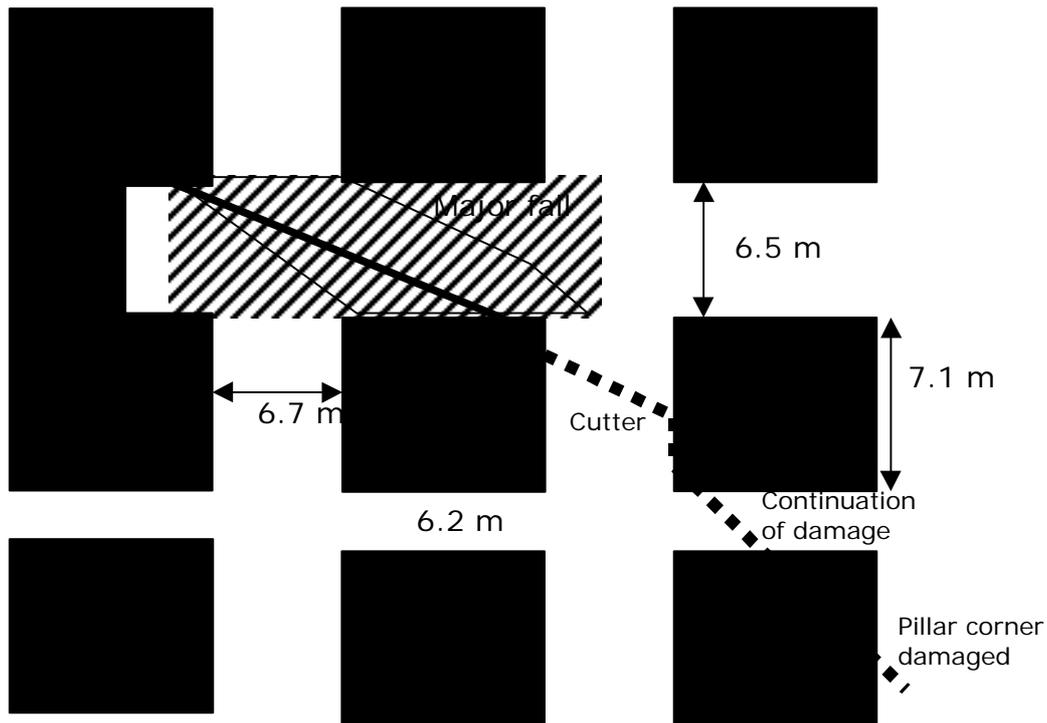


Figure 4. *Sketch plan of the fall.*

Cause of collapse

This fall was caused by high horizontal stress. The joint was the weak spot where it originated. Once mobilised, the roof strata continued to deform, driven by the high horizontal stress. The difference between the magnitudes of the maximum and minimum horizontal stresses are estimated to be small, as there was evidence of minor over thrusting in a direction perpendicular to the axis of the main damage, see Figure 6.

Conclusion

The cause of the horizontal stress could not be established.

Prevention

It is possible that the fall may have been prevented by installing a stiffer support system, i.e. thicker bolts at a denser pattern and with higher quality of bolt installation. The time lapse between exposure and support of the roof is not known, but it should be a minimum in a case where high horizontal stress exists.



Figure 5. *General view of the major fall.*



Figure 6. *Minor over thrusting in a direction perpendicular to the main guttering.*

Case F Summary: Insufficiently supported wedge

Mine locality: Highveld Coalfield

Seam: No 4

Mining method: Bord and pillar with Voest Alpine ABM30 Bolter miner. Road width 6.6m.

Mining depth: 84 m.

Mining height: 4.2 m

General description of roof: The roof was coal, 1.5 m to 2.0 m thick. It was followed by a sequence of muddy sandstone and mudstone with sandy mudstone with a thin (15cm) sandstone layer at the base. Note: the mine geologists do not distinguish between mudstone and shale.

Discontinuities present: The fall occurred at a 1.2m wide dyke striking NE-SW. The mining direction was due West. The dyke was steeply dipping towards the East. There were numerous sympathetic slips.

Time of fall: Between shifts.

Supported/unsupported: Supported

Support: M16 by 1.5m long mechanical anchors, 2 bolts per row with 1.5m spacing. In the vicinity of the dyke the spacing was decreased to 3 bolts per row at 1m spacing. Additional support was 3.5m cable anchors and W-straps.

Support quality: The bolts were well installed, but there was some doubt regarding the quality of the cable anchors. This did not contribute to the fall, as they were installed on the other side of the dyke as where the fall occurred. The tension indicator rings were not exposed – see Figure, implying that either the pretension was insufficient or that the wrong indicator rings were installed. A 2 m long cut-off was found, see Figure 2, which is always cause for concern as it may mean that some of the cables were cut shorter prior to installation.

Monitoring: None.

Locality: Intersection.



Figure 1. *Detail of the tensioning assembly of the cable anchor, showing that the load indicator had not come out. Note also that there is only one grouting tube instead of two.*



Figure 2. *Worker holding the cable cut-off that was found in the section.*

Horizontal stress indicators: None.

General description of fall: The fall was wedge shaped, +/- 1.5 m high by 12 m long and 2 m wide at the wide end. The top was flat. A plan view of the fall area is shown in Figure 3 and a view of the slickensided edge of the fall in Figure 4. As the roof had not yet been re-supported, a detailed inspection could not be done. The roadway through the dyke had been mined a few days prior to the fall. On the night before the fall, a cubby was cut at the dyke/roadway intersection. No additional support appeared to have been installed in the newly exposed roof.

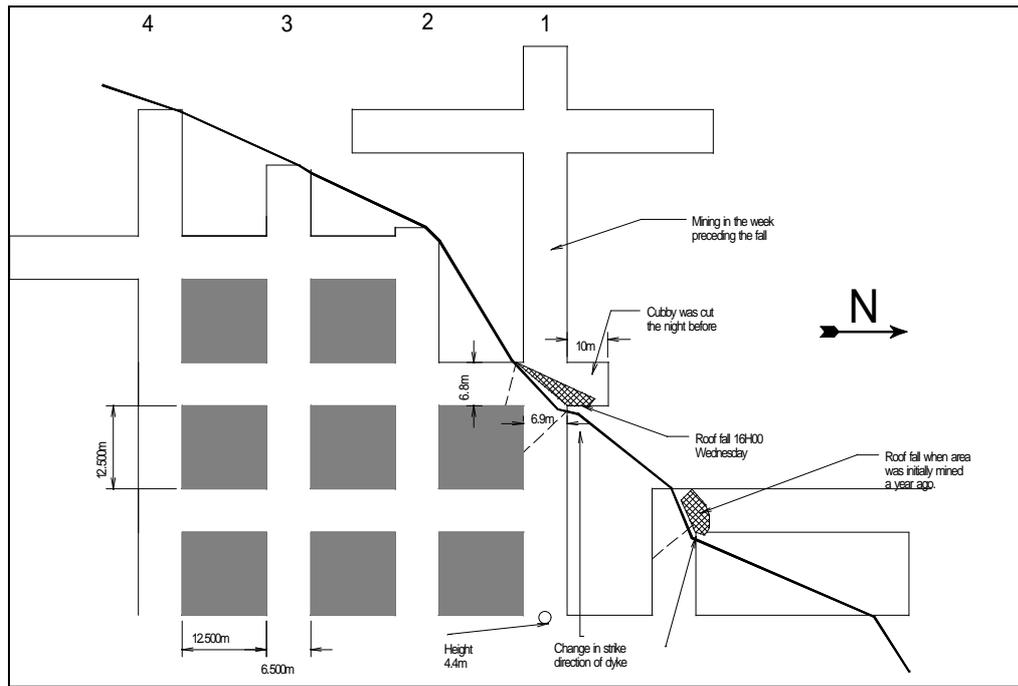


Figure 3. Plan view of the fall.

Cause of collapse

The new cubby exposed new roof that enlarged the area of the roof on the western side of the dyke. This is the weak side (the dyke was steeply dipping towards the East). A slip almost parallel to the dyke formed a wedge with the dyke. The support was insufficient to contain the wedge – no roofbolt holes were visible in the slickensided sides of the fall. The bolts were both too short, and did not penetrate the plane of the joint that formed the wedge.



Figure 4. View of the slickensided joint forming the one end of the wedge.

Conclusion

This fall was caused by an insufficiently supported wedge that was formed by joints associated with the dyke. As the dyke had been mined before, and appeared to be stable, consideration was possibly not given to the effects of enlarging the area that was mined at the dyke.

Prevention

This fall could have been prevented by better support (i.e. longer bolts at a denser spacing) and ensuring that bolts penetrate the joint plane or by installing W-straps across the joint plane. Primarily, however, it could have been prevented by not cutting the cubby at the dyke/roadway intersection.

Case G Summary:

Mine locality: Witbank coalfield

Seam: No 1 Seam

Mining method: Bord and pillar with drill and blast, 14 roadways, 6.5m wide roadways.

Mining depth: 70 m

Mining height: 2.1 m to 2.5 m, sometimes less as frequent rolls in the floor occur (No 1 Seam is directly on the paleo floor).

General description of roof: The immediate roof was 0.25m laminated shale/sandstone layer, followed by a 0.25m thick coal stringer, 0.3m laminated shale, another 0.1m coal stringer, 0.4m laminated shale, .1m coal/shale contact and a 0.3m sandstone.

Discontinuities present: The fall occurred on a slickensided fault with dip of 50° and trace length of 16 m +. Displacement was 1.2 m. A number of sympathetic joints were found, in particular two to the West of the fall that appeared to form a wedge.

Roof MRMR: 23 (poor)

Roof UCS: 8 MPa

Angle of friction: 25 – 30°

Cohesion: 200 – 300 kPa

RQD: 90%

Roof RMR: 50 (fair)

Roof CMRR: 33 (weak)

Time of fall: Between shifts

Supported/unsupported: Supported

Support: M16 full column resin V-bar, 1.2 m long, 3 bolts per row at 2 m spacings,.

Support quality: Poor. A number of bolts with at least 200 mm free space at the bottom were found. Torque was less than 150 N-m, indicating poor pre-tension. Two V-bars were found to be broken. (Note: the suppliers were informed of inherent problems with M16 V-bar strength some years ago. Fractures in the steel tend to form at the positions where the bar is pinched, weakening the steel.)

Monitoring: None

Locality: Intersection.

Horizontal stress indicators: None

General description of fall: A plan view of the fall is shown in Figure 1. The height was 1.5 to 1.75 m, with a flat roof and inclined sides, see section in Figure 2. The base area of the fall was 90 m². The contact of the fault plane with the roof rock at the top of the fall was seen to be open, indicating lack of compression, perhaps tensile horizontal stress. Water was dripping out of the opening. The southern extension of the fault was also open, +/- 2 to 5 mm. Bed separation on the western side of the fall was also seen.

Three roofbolt holes were found in the fault plane, indicating that bolts had penetrated the plane. However, judging from their lateral position, 1 m from the edge of the fall, the length of the bolts, 1.2 m and the inclination of the fault plane, 45° – 50°, the anchor length on the strong side of the fault was about 0.2 m. Resin was seen inside the holes, indicating that the V-bar had pulled out of the resin. Usually, with more suitable bolts, the failure occurs on the resin/rock plane. This type of failure has previously been observed with V-bar.

Cause of collapse

The fault changed the clamped beams in the roof to cantilevers, thereby increasing the induced tensile stress from 1.8 MPa to 10.8 MPa (6 m long beams, .25 m thick). This is clearly in excess of the tensile strength of the roof material of around 5 MPa.

The support that was installed through the fault plane was inadequate. With the liberal assumption that the bolt spacing was 1 m all along the fault plane, then the total resistance offered by the bolts was as follows (assuming resin/steel cohesion of 2000 kPa):

Length of anchor = 200mm
Bolt diameter = 16mm
Contact area = 10 053mm²
Resistance = 20.1 kN

The total weight to be supported was as follows:

Length = 6 m

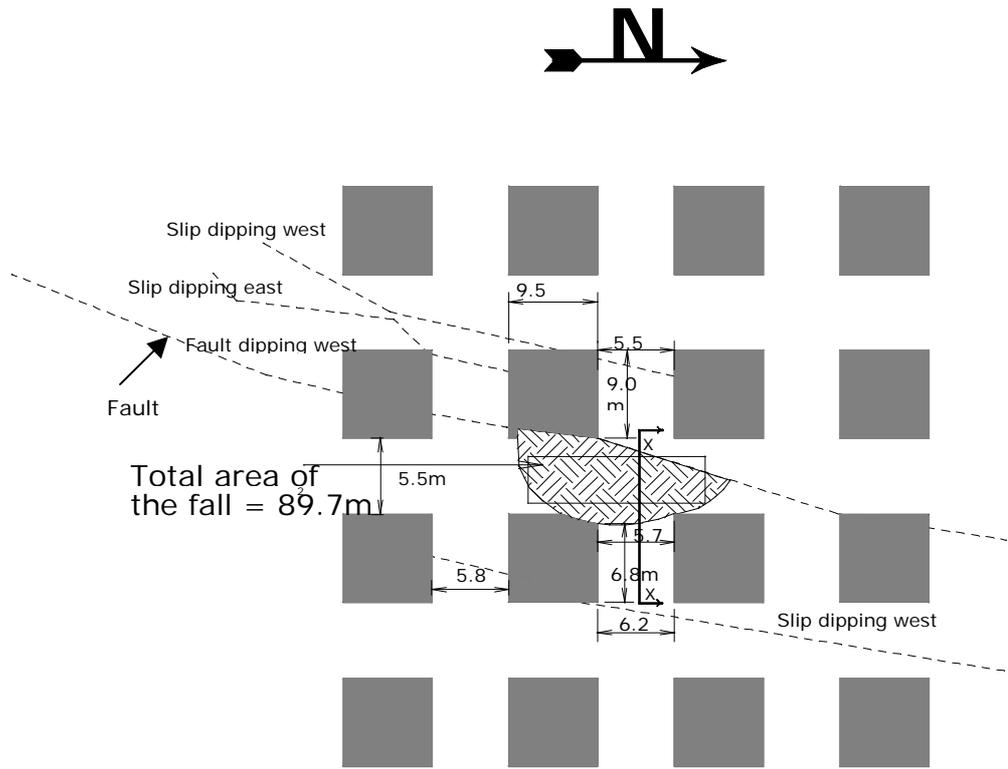


Figure 1. Plan view of the fall.

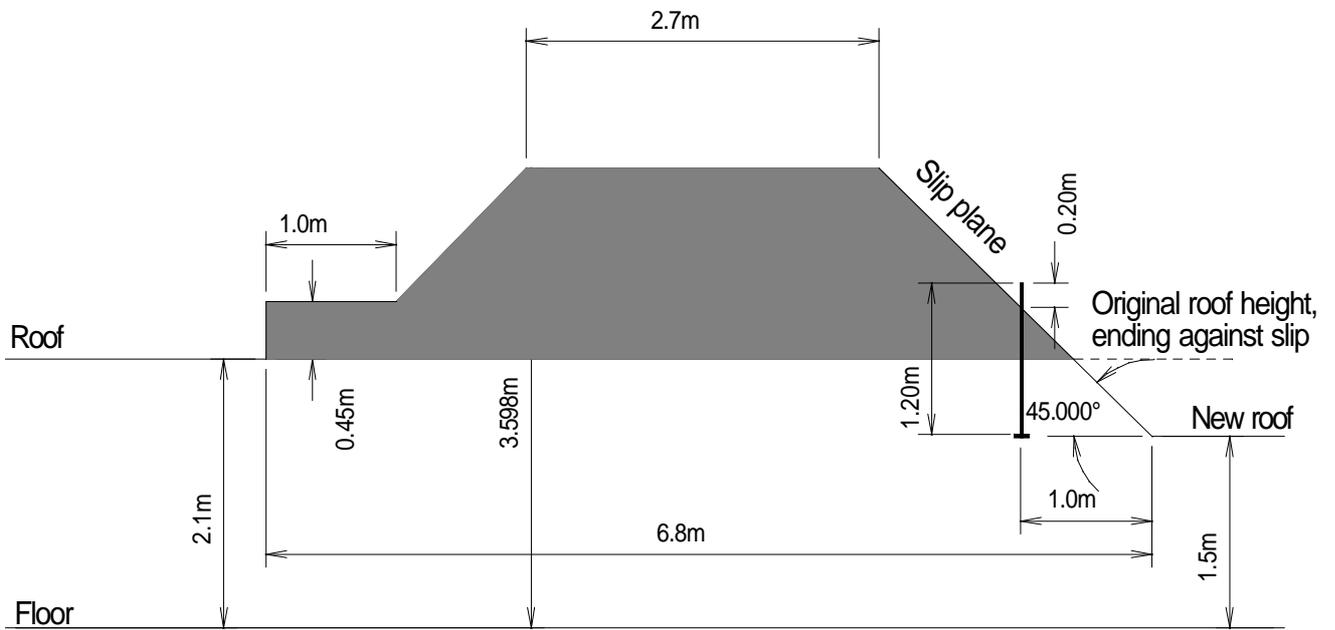


Figure 2. Cross-section through the fall.

Width = 1 m
Height = 1.5 x .66666 (assuming 2/3 height due to profile of fall)
= 1 m
Volume = 6 m³
Weight = 150 kN

(Note that if more suitable steel tendons were used, the resistance would have been somewhat better:

*Length of anchor = 20 mm
Hole diameter = 22mm
Contact area = 13 823mm²
Resistance = 27.6 kN*

This is a 37% improvement)

Conclusion

This fall was caused by the fault that changed the roof strata behaviour from that of a clamped beam to a cantilever. The split at the fault position enlarged the exposed area of the weak side of the fault and the support that was provided through the fault plane was inadequate.

Prevention

The first point to note is that the fall would possibly have been prevented if the split had not been mined. The split was mined under the weak side of the fault, with the fault at the furthest end – this is the worst possible position for a split to be in.

Secondly, with regard to bolting, the resistance per bolt on a 1 m spacing would have had to be 150 kN. This is in excess of the strength of a M16 bolt, so that either M20 bolts would have been required or the spacing along the fault plane would have had to be reduced to 0.5m.

If the spacing had been reduced to 0.5m, the anchor length would have had to be 0.5m, see Figure 3. This would have been possible if the bolts were installed 0.5 m from the fault plane. To summarise, a support configuration consisting of the 1.2m bolts spaced at 0.5m apart and installed 0.5m from the fault, would have been sufficient to contain the fall. Note that this presupposes a suitable support tendon. If V-bar had been used, with failure occurring on the steel/resin interface, the spacing would have had to be reduced to 0.37m.

The other alternative, which is preferable because it is independent of the dip of the fault, is to install bolts on either side of the fault and to join them with short lengths of W-strap.

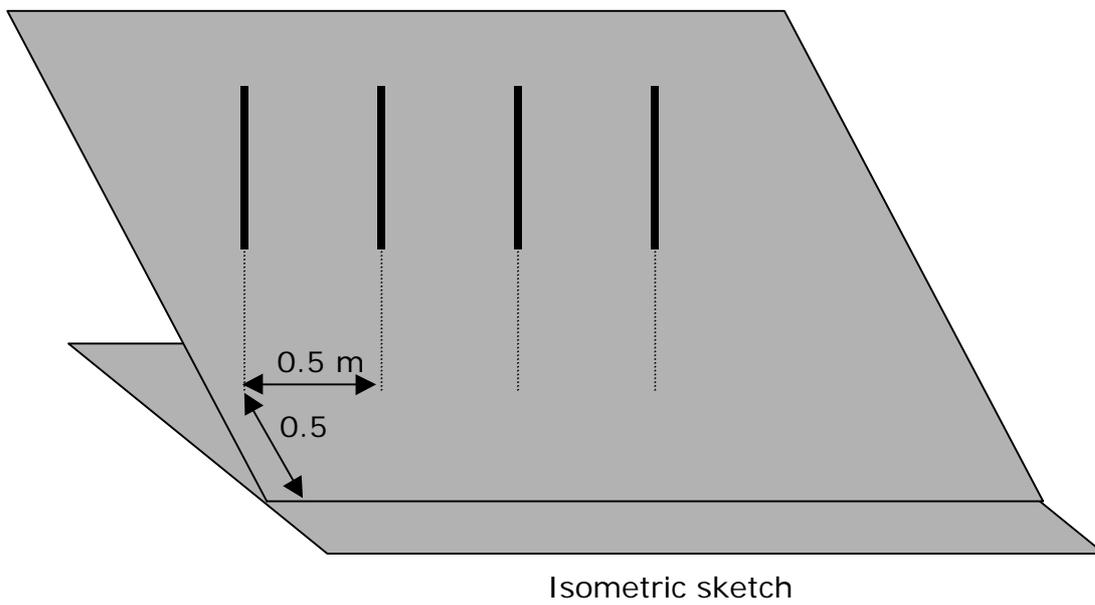
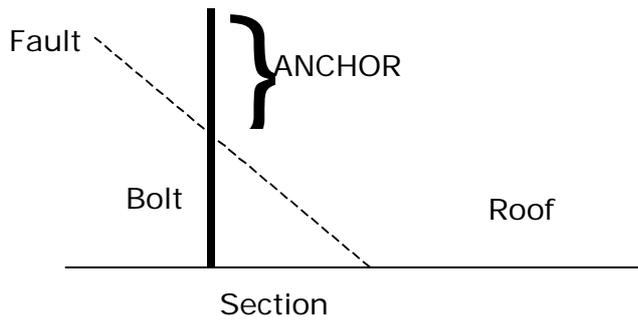


Figure 3. Section and isometric view to explain bolting configuration through fault plane.

Case H Summary: Thin wedge / beam created by joints

Mine locality: Highveld Coalfield

Seam: No 2

Mining method: Continuous miner bord and pillar, 5 road section. Road width 7.2 m

Mining depth: 125 m

Mining height: 4 m

General description of roof: The general condition of the roof can be described as good. The immediate roof strata consisted of a laminated sandy mudstone contact with an upper laminated sandy micaceous mudstone bed. The laminated mudstone is sometimes known locally as shale.

Discontinuities present: There was a fault with 0.7 m displacement at the far end of the fall, and two or more minor slips at the near end of the fall.

Roof MRMR: 22 (poor)

Roof UCS: 9.9 MPa

Roof CMRR: 55 (moderate, stand-up time 4 days)

Roof RMR: 71 (good, 15 days stand-up time)

Time of fall: During cutting

Supported/unsupported: Unsupported

Support: 0.9 m long M16 mechanical anchors, 4 bolts per row at 2 m spacings. Joint support is 1.5 long mechanical anchors.

Cut-out distance: 15 m

Support quality: Excellent. 90% of bolts had torque in excess of 200 N-m.

Monitoring: None

Locality: Roadway

Horizontal stress indicators: None

General description of fall: The fall occurred over the full width of the roadway, from the exposed fault at the far end of where the CM was cutting to a position 4 m back where it was bounded by two or more minor joints. A plan view is shown in Figure 1. Figure 2 is a general view of the fall area. The fall was 0.5 m thick at the far end, thinning to approximately 0.1 m at the near end. The thinning of the fall was due to thinning of the immediate roof stratum caused by a dome structure.

The same fault was present in the adjacent roadway. There, no fall occurred. The shiftboss reduced the cut out distance to install support closer to the fault before it was exposed.

Cause of collapse

The fault and the joints caused cantilever behaviour of the roof, resulting in a failure plane at the right hand side, see Figure 3. The length of the beam was approximately 7 m and the average thickness was 0.3 m. Then, the induced tensile stress is:

$$s = \frac{3gL^2}{t} = 12.3 \text{ MPa}, \quad [\text{H.1}]$$

where g = unit weight of beam
 L = length of beam
 T = thickness of beam.

This is in excess of the tensile strength of the roof rock of approximately 5 MPa.

This was exacerbated by the thinning of the roof strata back from the fault. The orientation of the fault is as favourable as it could be and in general this type of situation seldom results in falls of roof. The instability was aided by the other minor joints and the dome structure of the roof strata.

Alternatively, and equally acceptable, is that the combination of fault and the joints formed a wedge in the roof. The far end of the fall could not be examined for the presence of small joints as the fall had not been re-supported at the time of the investigation.

Conclusion

The fall was caused by the excessive cut out distance resulting in mining underneath an unfavourable jointing configuration in the roof without support. The underground crew's reaction in the next roadway, i.e. to project the position of the fault and then to reduce the cut out distance accordingly, is laudable.

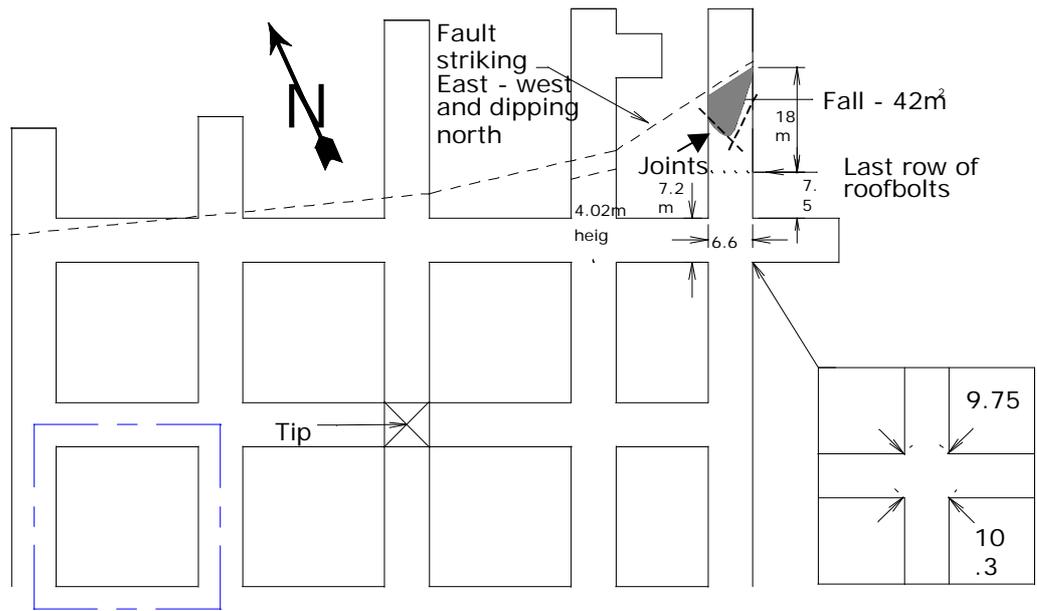


Figure 1. Plan view of the roof fall.

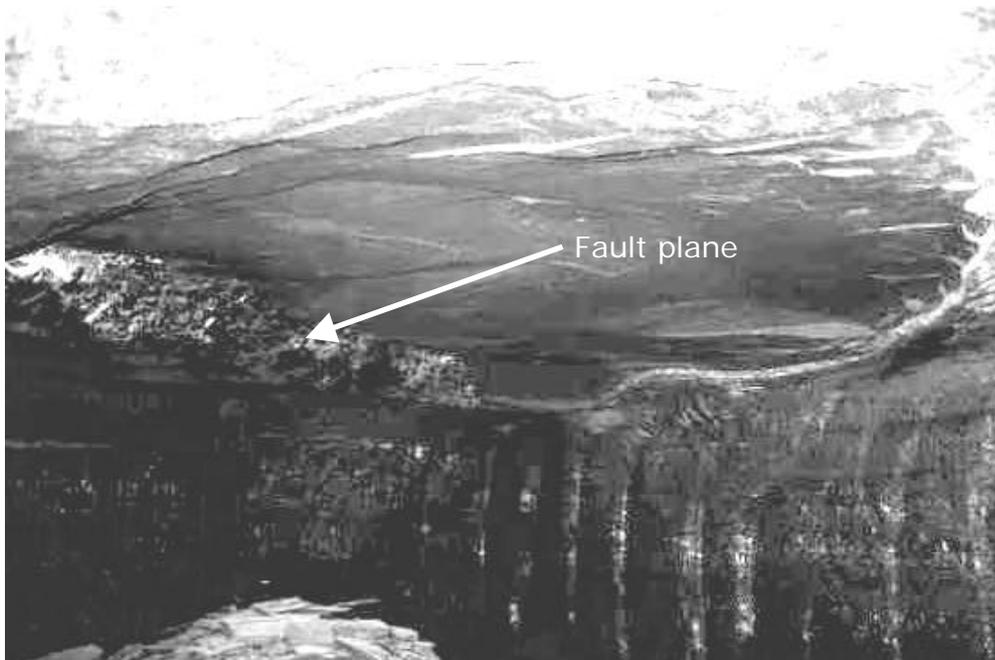


Figure 2. General view of the roof fall, showing the fault at the far end.

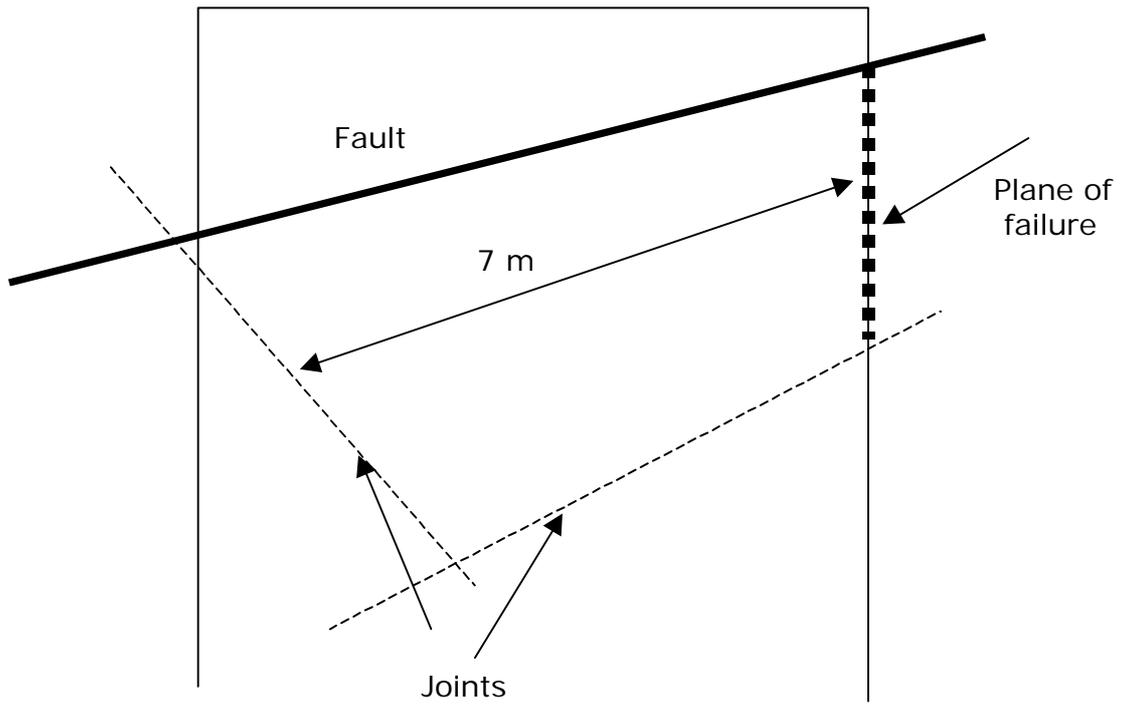


Figure 3. *Simplified diagram of the beam formed by the configuration of the fault and the minor joints.*

Prevention

This fall could have been prevented by a reduced cut out distance or by the implementation of a method to detect discontinuities ahead of the face. This method does not exist yet and needs to be developed.

Case I Fall 1/2 Summary: Weak, weathered strata, joints.

Mine locality: Pafuri Coalfield

Seam: No.7

Mining method: Bord and pillar drill and blast. Road widths average 5.5 m. Irregular pillar sizes due to dip and several discontinuities.

Mining depth: 50 m

Mining height: 2.7 m

General description of roof: 2.5 m highly weathered laminated shale followed by 1.5m moderately weathered laminated shale.

Discontinuities present: Densely jointed, close proximity to dyke. Some joints slightly open.

Roof MRMR: 15 (very poor)

Roof RMR: 38 (poor)

Roof CMRR: 6 (weak)

Roof UCS: 3 MPa

Roof cohesion: 100 kPa

Angle of friction: 30°

RQD: 25%

Time of fall: Small falls between bolts during mining, bigger falls at unknown time some weeks after mining.

Supported/unsupported: Supported

Support: M16 rebar 2 m long resin point anchor with W-straps, additional support 2.4 m long M16 resin point anchor 0.5 m apart on joints. Most bolts installed parallel to joints.

Support quality: Medium to poor, bolts not fully tensioned. Indications that roof had separated prior to support installation, see Figure 1.1 showing resin bulbs on bolts.

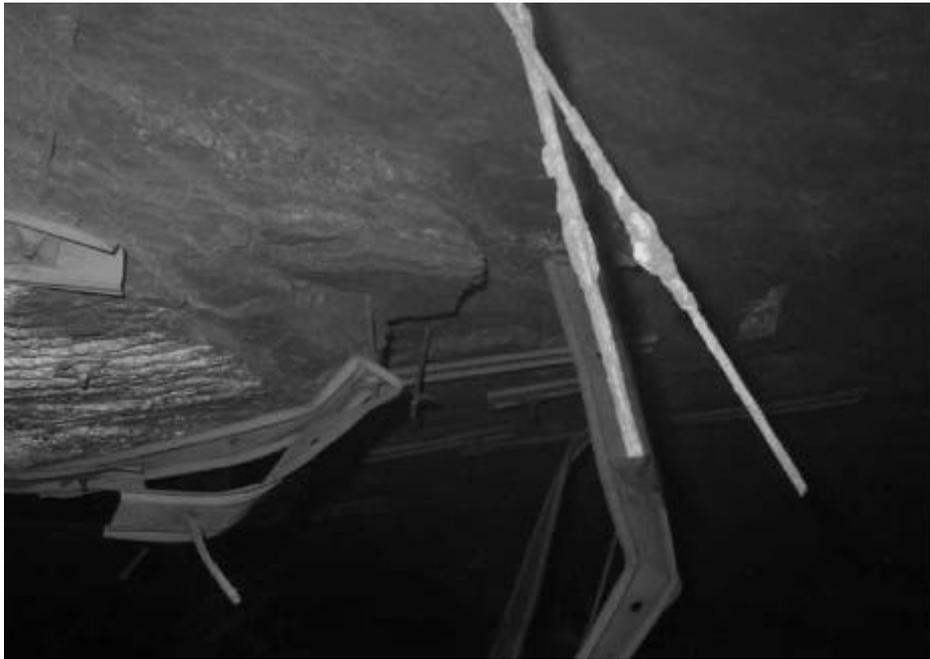


Figure 1.1. *The resin bulbs on the exposed bolts indicate that bed separation had already taken place by the time the bolts were installed.*

Cut-out distance: N/A

Monitoring: None

Locality: Intersection

Horizontal stress indicators: None at fall, potting some distance away. Direction of maximum horizontal stress estimated at 120° . Open joints in approximate direction of major horizontal stress indicate lack of compression, or even tensile minor horizontal stress. At a small (unmapped) fall approximately 200 m west of this area, a roofbolt showing post installation horizontal movement was seen, see Figure 1.2.

General description of fall: See Figure 1.3 for plan view. Fall height estimated 1.5m for Fall 1 and 6.5 m for Fall 2. Figure 1.4 shows dome shape of Fall 2 and Figure 1.5 the small fragmented rubble, indicative of the weak nature of the roof.

Cause of collapse

Poor rock quality, weathering, joints. Falls do not align with direction of maximum horizontal stress, which is probably incidental due to the proximity of the dyke. Appears that the contribution of horizontal stress may have been to open some joints, increasing instability.

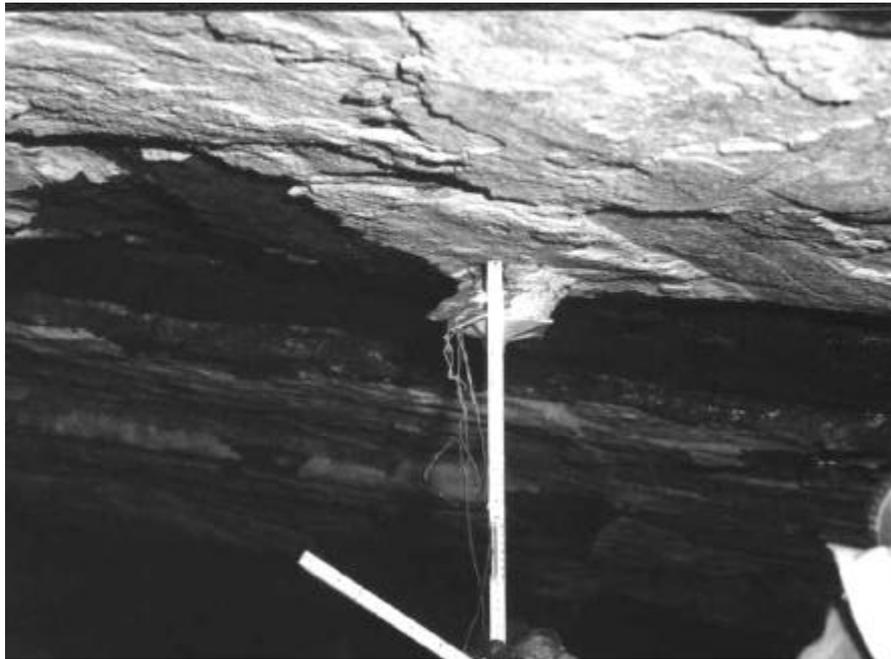


Figure 1.2. Post installation horizontal displacement of bolt or bending beam?

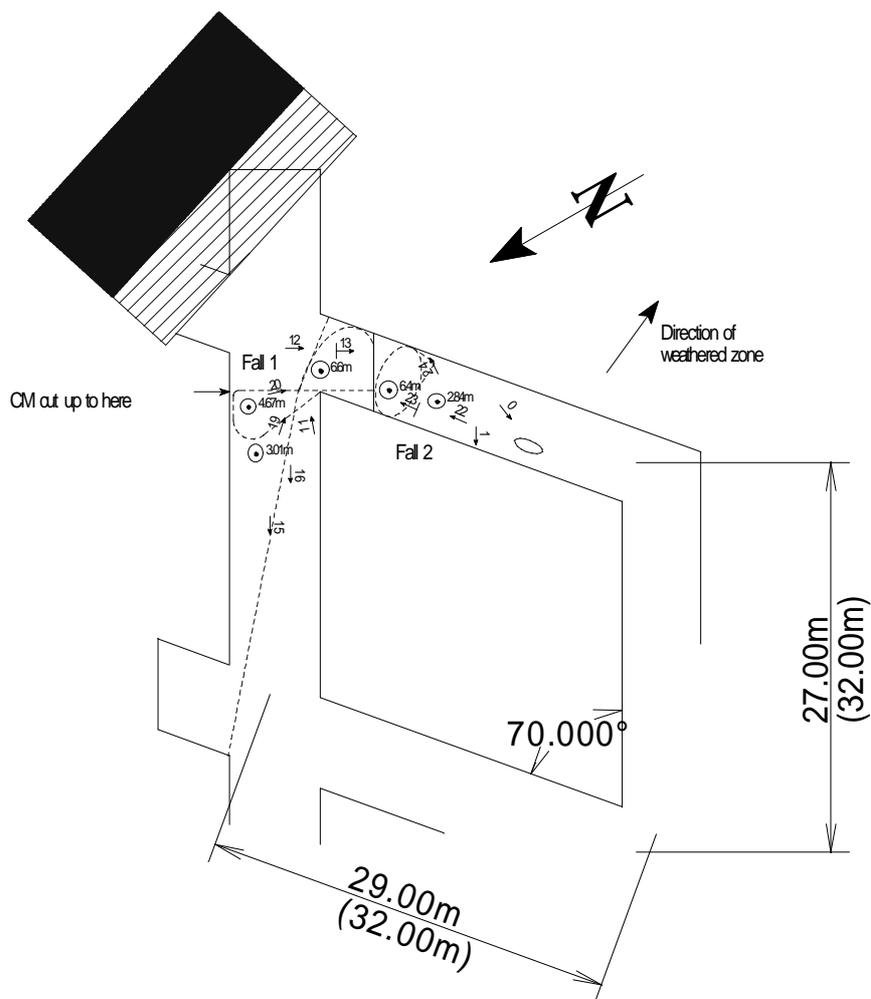


Figure 1.3. Plan view of Falls 1 and 2.

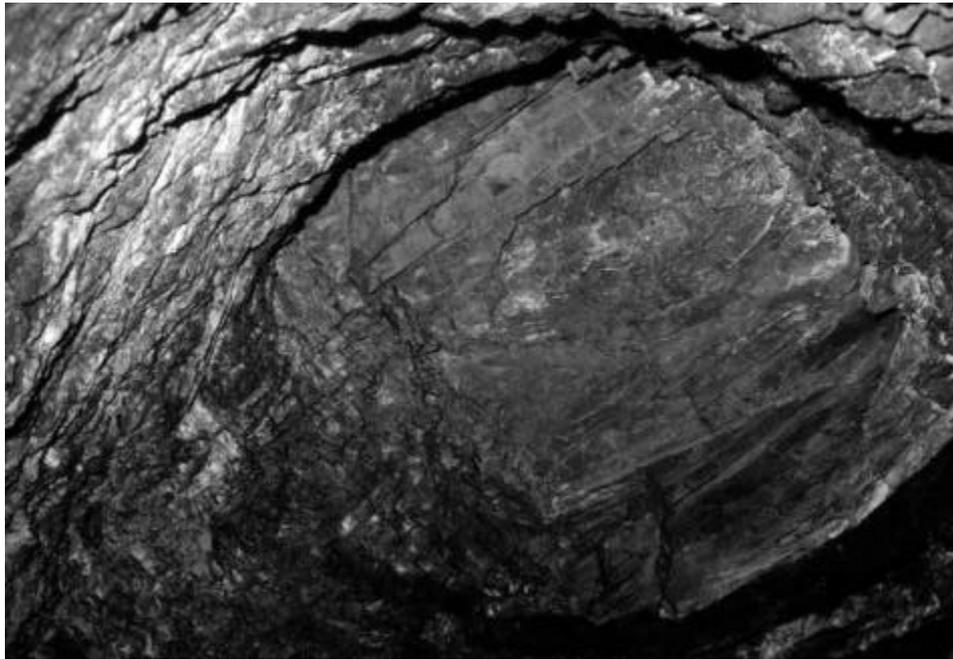


Figure 1.4. *Dome shape of Fall 2. Note the density of the joints.*



Figure 1.5. *Note the small fragments of the fall rubble.*

Conclusion

Falls caused by joints.

Prevention

Area support comprising of 2m long full column resin anchors supplemented by W-straps as standard.

Case I Fall 3/4 Summary: Weak, jointed roof with water and horizontal stress.

Mine locality: Pafuri Coalfield

Seam: No 7

Mining method: Two-road bord and pillar drill and blast development. Bord width 6m.

Mining depth: 340 m

Mining height: 2.7 m

General description of roof: 2 m severely weathered laminated shale.

Discontinuities present: There was a dyke less than 30 m away. Two joint sets were present, the first one with dip/dip direction $50^{\circ}/270^{\circ}$ and spacing .5 m, and the second with dip/dip direction $80^{\circ}/180^{\circ}$.

Roof MRMR: 38

Roof RMR: 43 (fair)

Roof CMRR: 23 (weak)

Roof UCS: 8.5 MPa

Roof cohesion: 150 – 200 kPa

Angle of friction: 35°

RQD: 96%

Time of fall: Not known, after support.

Supported/unsupported: Supported

Monitoring: None

Locality: Roadway.

Horizontal stress indicators: Deep cutters and one severe roof potting, see Figure 2.1. The SDR was estimated at 4 – 6 and the direction of the maximum horizontal stress was 179° .



Figure 2.1. Roof potting in the vicinity of Fall 3/4

General description of fall: Figure 2.2 is a plan view of Falls 3 and 4. Figures 2.3 and 2.4 are general views of Falls 3 and 4 respectively. Water came out of the roof. At fall 4 there was a borehole.

Cause of collapse

Falls 3 and 4 were caused by the intense jointing in the vicinity of the dyke, aggravated by horizontal stress and the weathered nature of the roof. The roof at Fall 4 was weakened by water accumulating in a borehole prior to being mined.

At a depth of 340 m, the horizontal stress with a k-ratio of 2.5 is 21 MPa, which is greatly in excess of the stress measured at mines where horizontal stress is perceived to be a major problem.

Conclusion

The falls were caused by weak, jointed roof in the vicinity of a dyke, aggravated by water and horizontal stress.

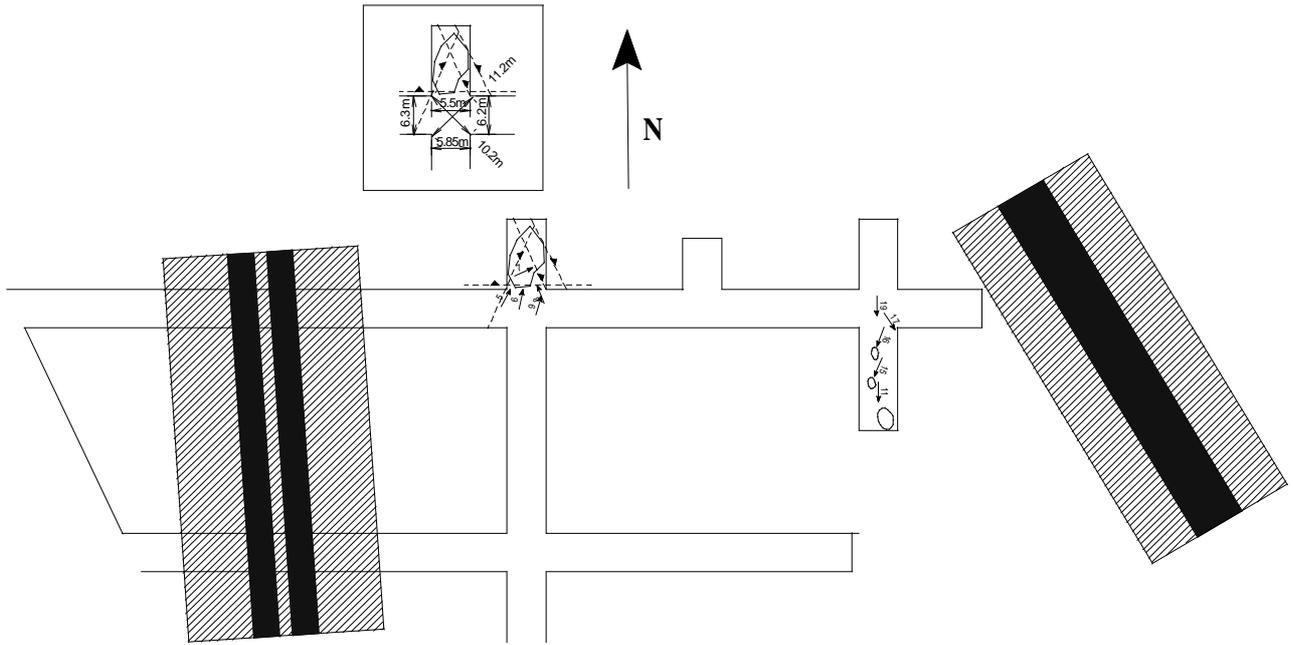


Figure 2.2. *Plan view of fall 3 and 4.*

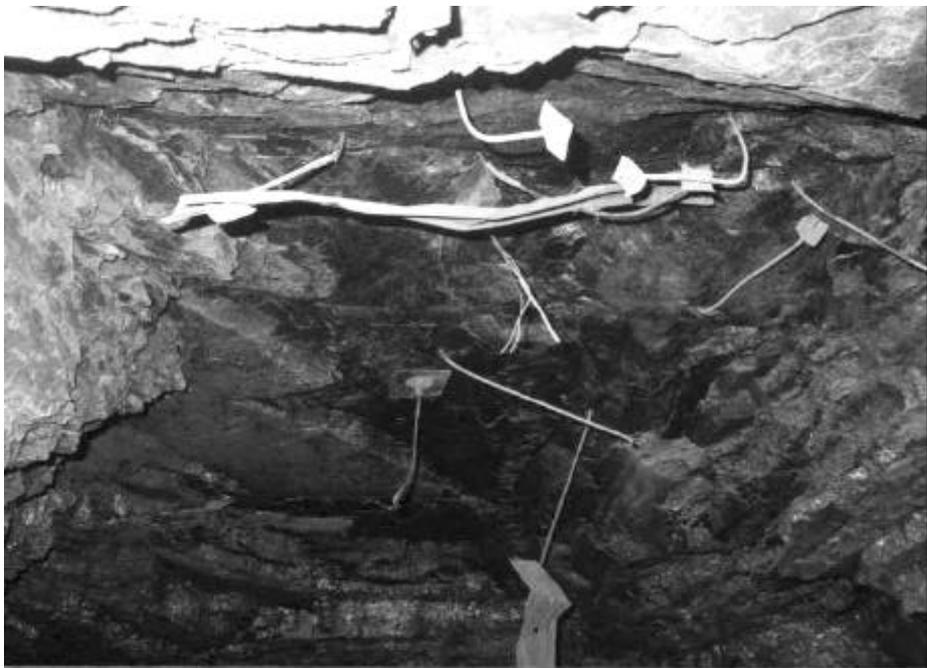


Figure 2.3 *General view of Fall 3, showing the blocky nature of the roof.*



Figure 2.4. General view of Fall 4, showing water flowing from the borehole.

Case I Fall 5 Summary: Burnt roof, joints. (3 falls in close proximity)

Mine locality: Pafuri Coalfield

Seam: No 7

Mining method: Continuous miner bord and pillar development, stooping in area. Road width designed to 5 m, average measured at 4.3 m.

Mining depth: 70 to 85 m

Mining height: 2.3 m

General description of roof: Baked shale, bedded .2 m to .6 m thick beds. Coal is burnt.

Discontinuities present: Close to dyke, heavily jointed.

Roof MRMR: 30

Roof RMR: 37 (poor)

Roof CMRR: 21 – 28 (weak)

Roof UCS: 5.9 MPa

Roof cohesion: 100 kPa

Angle of friction: 30°

Time of fall: Started during development, continuous.

Supported/unsupported: Supported

Support: M16 rebar, 2 m long, resin point anchor. Supplemented by W-straps and cable anchors.

Support quality: Medium.

Monitoring: None

Locality: Intersection, continued into roadway.

Horizontal stress indicators: None. Joints are loose, indicating lack of compression in roof.

General description of falls: Figures 3.1 and 3.2 are plan views of two of the falls. The fall heights ranged from 1.5 m to over 4 m. Figures 3.3 and 3.4 show typical damage to the roof. Note the column of rock on top of the W-strap in the background of Figure 3.4, indicating the weakness of the roof and its inability to support itself even over very short distances. Figure 3.5 shows the intensity of the jointing in the area. Figure 3.6 is a general view of the fall shown in plan in Figure 3.2.

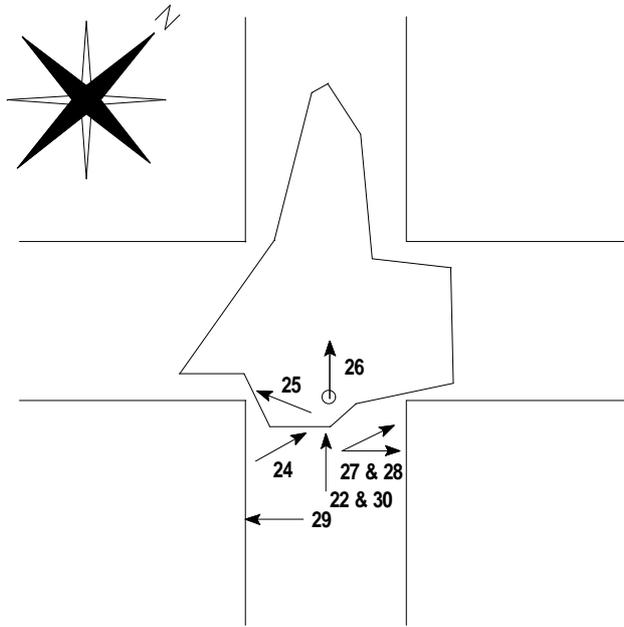


Figure 3.1. Plan view of one of the falls in the area, indicating the aerial extent of the fall. The fall had an estimated height of 1.5 m.

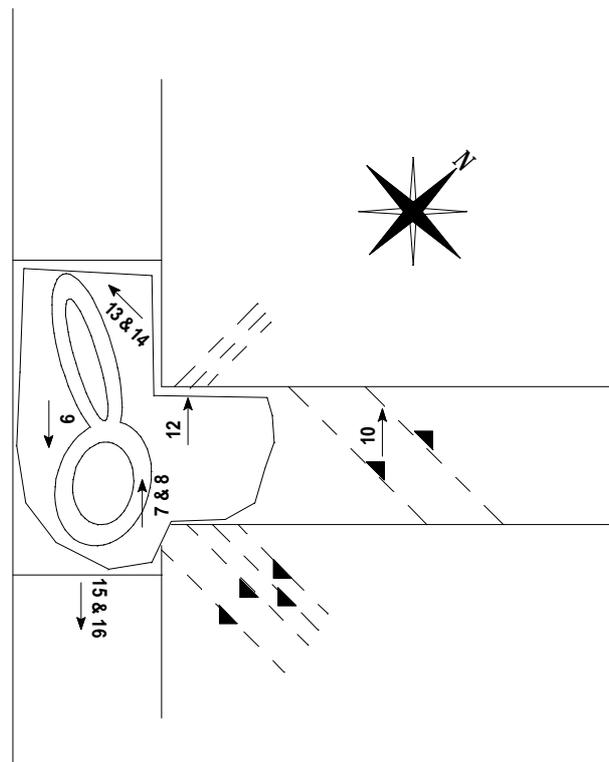


Figure 3.2. Plan view of the fall that was more than 4 m high.



Figure 3.3. *General view of one of the burnt coal falls. The channel shape of the fall was caused by parallel joints, forming a wedge that fell.*



Figure 3.4. *Note the rock on top of the W-strap in the background, indicative of the roof's inability to support itself over short distances.*



Figure 3.5. *Note the intensity of jointing in the roof.*



Figure 3.6. *General view of the 4 m high fall, shown in plan in Figure 3.2.*

Cause of collapse

These falls were caused by the high intensity of jointing and weak roof that is typical of heat induced damage in the vicinity of some dykes.

Conclusion

Heat generated by the dykes weakened the roof, causing the falls of ground.

Prevention

Standard support consisting of cable anchors and W-straps would have prevented these falls. This would require a cut-out distance of say 3 m and very low advance rates.

Case I Fall 6 Summary: Dyke, burnt coal, wedge

Mine locality: Pafuri Coalfield

Seam: No7

Mining method: Bord and pillar drill and blast. Road widths 6.3 to 6.6m.

Mining depth: 57 m

Mining height: 3.0 m

General description of roof: Bedded, laminated shale. Beds are 200 – 600 thick.

Discontinuities present: Split mined semi-parallel to dyke, sympathetic joints present. Wedge forming joints spaced at 1.8 m to 0.5 m.

Roof MRMR: 24

Roof RMR: 34

Roof CMRR: 38 (weak)

Roof UCS: 5.4 MPa

Roof cohesion: 150 – 200 kPa

Angle of friction: 35° to 40°.

RQD: 98 %

Supported/unsupported: Supported.

Support: M16 rebar, resin point anchor.

Monitoring: None

Locality: Intersection and roadway.

Horizontal stress indicators: None

General description of fall: Figure 4.1 is a plan view of the fall , which consisted of two parts. It was not known which occurred first. The fall in the intersection was in the burnt coal, and was approximately 4m high. The fall in the split was between joints that formed a 3m high wedge that fell out. Figure 4.2 shows one of the joints that formed the wedge.

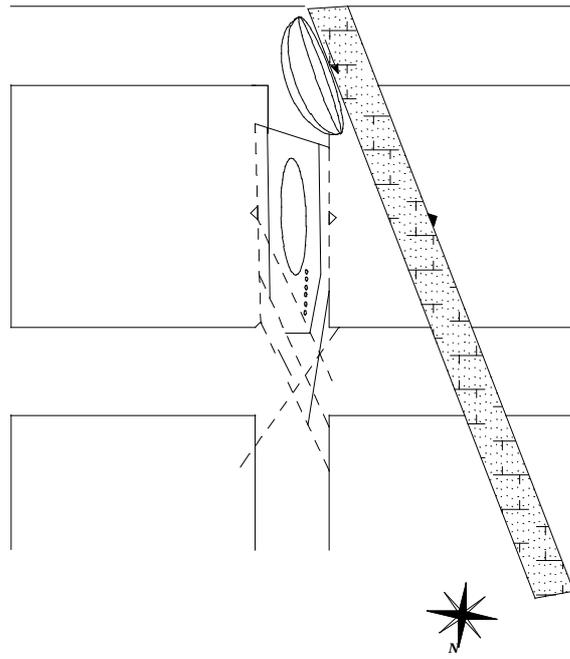


Figure 4.1 *Plan view of the two parts of the fall.*



Figure 4.2. *One side of the wedge formed by intersecting joints.*

Cause of collapse

The fall was caused by intersecting joints forming a wedge and by the weakened roof due to the heating effects of the dyke.

Conclusion

The fall was caused by intersecting joints forming a wedge and by the weakened roof due to the heating effects of the dyke.

Prevention

The fall in the intersection could have been prevented by installing cable anchors and W-straps concurrent with mining at short cut-out distances, perhaps also by not creating the split in that position. The fall in the split could have been prevented by firstly not mining the split in close proximity to the dyke or by installing thicker bolts at closer spacings through the planes of the two joints that formed the wedge.

Case L Summary: Horizontal stress, weak zone

Mine locality: Witbank Coalfield

Seam: No 2

Mining method: Bord and pillar 11 road panel with Joy 12HM9. Designed bord width is 6.5 m, although measured widths in the section were only 5.5 m to 6.0 m..

Mining depth: 45 m

Mining height: 3.1 m to 3.5 m (average 3.3 m)

General description of roof: The immediate roof is gritstone, which was 120 mm thick at the major fall site. Further south it is thicker than 500 mm and to the north of the fall it is 400 mm thick. On top of this is a weak mudstone contact, increasing in thickness from 35 mm in the north to 200 mm in the southern end of the panel. At the major fall site it is 70 mm thick. Overlying the contact is laminated shale, bed spacing 60 to 200 mm.

Discontinuities present: No major features, moderate flat dipping jointing with dip/strike direction of 10°/234°.

Roof MRMR: 22 (on verge of being stable)

Roof RMR: 46 (fair – 3 days stand up time)

Roof CMRR: 23 (weak – immediate collapse))

Roof UCS: 5.5 MPa

Roof angle of friction: 35° to 40°

Roof cohesion: 150 – 200 kPa

RQD: 60%

Time of fall: Between shifts.

Supported/unsupported: Fall occurred in two phases. The first fall was unsupported. The roof was then supported and then the major fall occurred.

Support: M20 Ausbar 1.5 m long 3 bolts per row at 2 m spacings in 25 mm holes, full column dual speed single capsules. Bolt density was increased at the fall.

Cut-out distance: 12 m at major fall, reduced to 5 m after fall occurred.

Support quality: Medium at fall, good elsewhere. At fall, bolts were installed late after first fall occurred. Due to height, equipment could not reach roof and bolts not installed to full length. The torque on a number of bolts was less than 150 N-m and several holes were not filled with resin.

Monitoring: None

Locality: Intersections and roadways

Horizontal stress indicators: The excavations where the fall occurred showed signs of damage that could be attributed to horizontal stress. The near-by workings showed signs of horizontal stress damage. The horizontal stress damaged (roof) was confined to the southwest side of the panel. This damaged zone was defined by a line bearing 302/122° (magnetic). To the north east of this line, the roof was not damaged. In the southwest section, horizontal stress damage took the form of potting and cutter roof, see Figure 1. The cutter damage to the roof varied from slight to deep. Minor floor lifting was seen in the undamaged areas, see Figure 2. Minimal pillar stress induced damage was observed. Where this occurred it normally took the form of scaling of the pillar corners, with shear failure of the top pillar/roof contact being also seen. The falls were seen to migrate around corners, see Figure 3. One of the roofbolt holes was seen to be deformed as shown in Figure 4.

A stress damage survey gave SDR rating values of 6 to 2.5 for the damaged section (fall of ground to slight cutter roof). In the undamaged section values of 1 were recorded (stable). Stress damage surveys indicated that the maximum horizontal stress has a direction in the region of 240°. Underground stereographic surveys show that the maximum horizontal stress acts (in plan) at a 10° angle to the dip of the bedding and at a 20° angle to the main boundary between different thickness of sandstone (<0.5m) in the immediate roof. From the above facts it can be ascertained that higher than normal roof shear stresses are generated in these areas.

General description of fall: Figure 5 shows a layout plan of the section with most of the falls, potting, cutters, floor heave and guttering indicated on it. A distinct channel, in which the roof falls occur, can be plotted from this. The direction of this channel is ± 300° to 320°. The Mine overseer stated that this channel has been following them from some distance back and slowly crept across the panel as they advanced.

Figure 6 shows a more detailed plan of the area around the major fall (Fall L) and Figure 7 shows the mining sequence.



Figure 1 **Minor potting.**

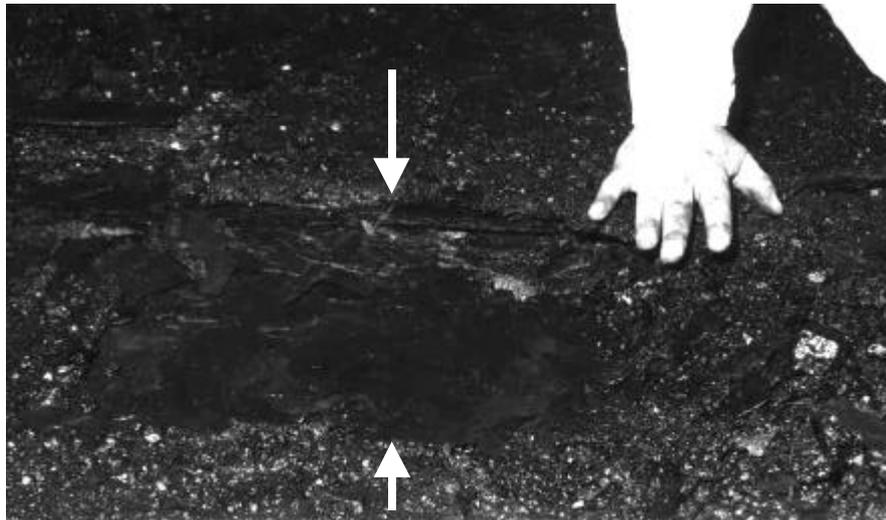


Figure 2. **Floor heave**

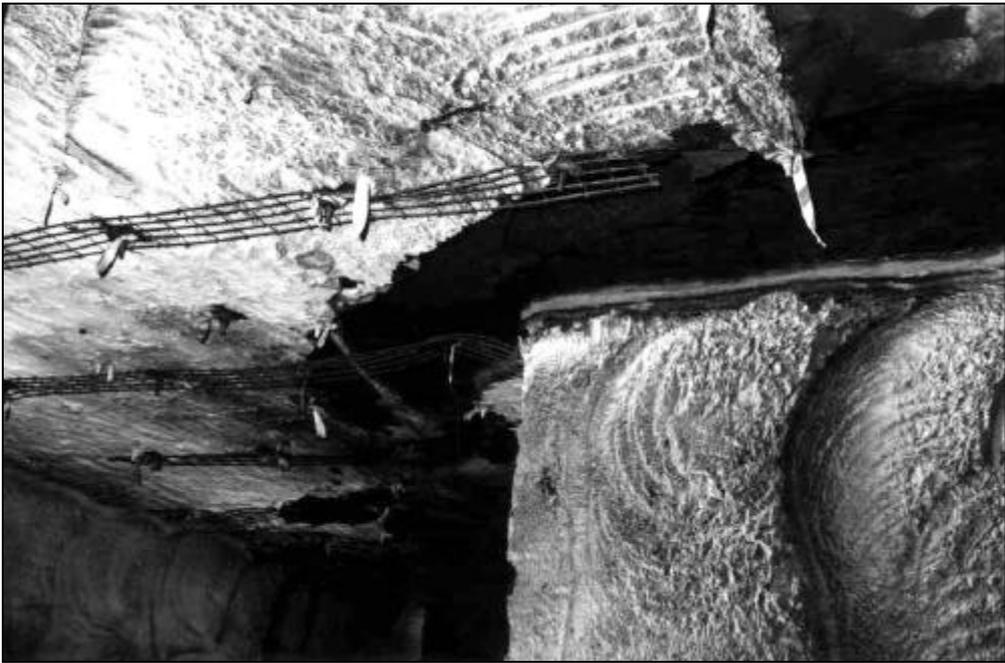


Figure 3. *Fall progressing around a corner in the form of a cutter.*

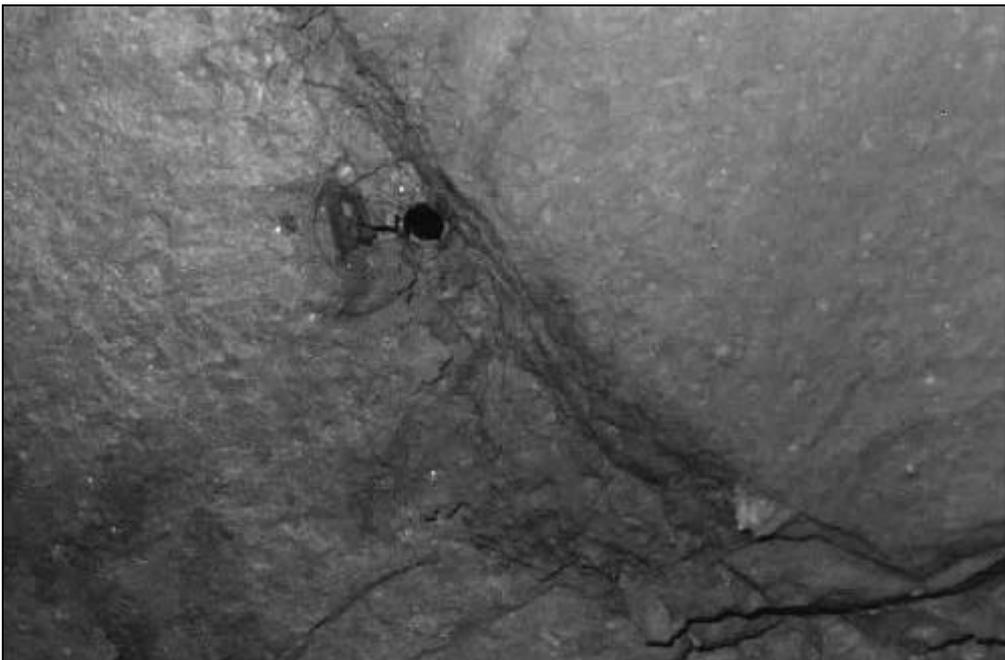


Figure 4. *Deformed roof bolt hole.*

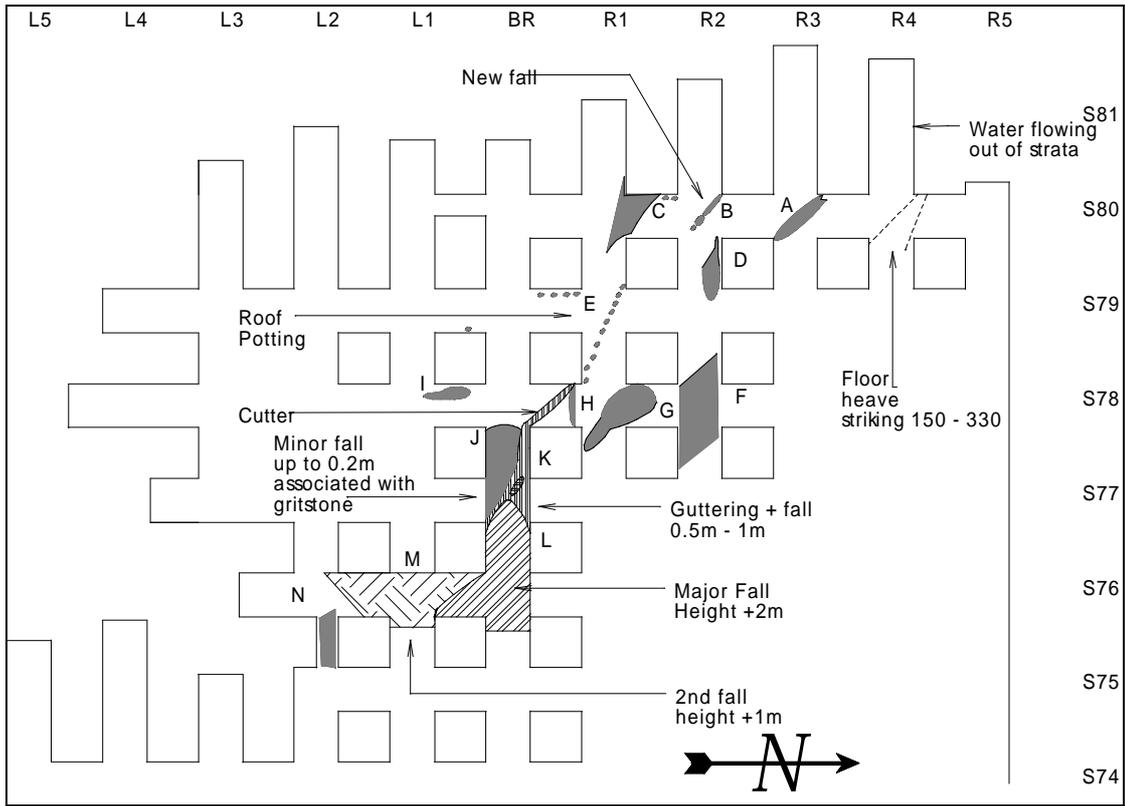


Figure 5. General key plan of the falls.

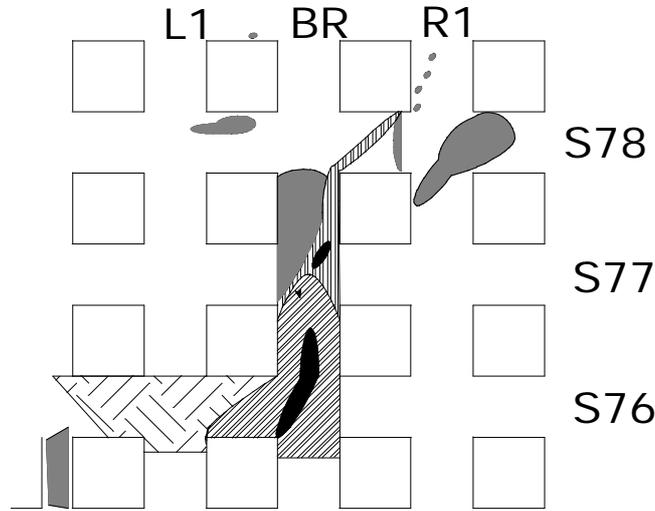


Figure 6. Detail of the major fall, Fall L in Figure 5.

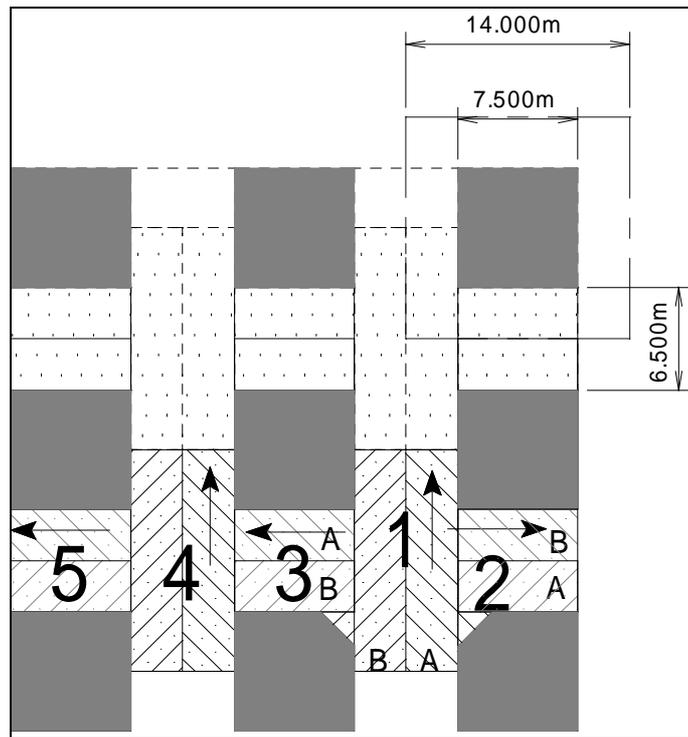


Figure 7. The cutting sequence.

The contour plan of the seam floor shows no dipping of the seam.

In split 80, road no. 4, floor heave was observed. The floor heave was $\pm 100 - 150\text{mm}$ high and ran in two lines more or less diagonally across the intersection. The lines were not parallel with the strike direction being $\pm 320^\circ$ and 340° respectively. The intersection that was formed, was larger than the other intersections throughout the section due to the splits being started of in this roadway. Thus the pillar corners were cut away to enable the CM to turn. The corresponding intersection, one split back was also inspected and no floor heave was observed. The position of the floor heave corresponds to the channel of weakness.

From the sidewall on the northern side in road 4, water was running out of a concentrated area in the strata above the coal, see Figure 8. It appeared to be coming from the bedding plane between the coal and the Gritstone. The flow was more than 3 l/min . The sidewall was stained reddish brown.

The Mine Overseer reported that prior to intersecting the water in road no. 4, water was coming out of the faces mined towards the western side. Once they passed the water in road no. 4, the water in the faces stopped. The direction of the Paleo ridge is towards the north-western side.



Figure 8. *Position where water was coming out at the contact between the roof and the ribside.*

Fall A had an elliptical shape diagonally across the intersection with a height of $\pm 0.3\text{m}$ and it occurred just after cutting the roadway. The dome of the fall is more towards the west side of the fall.

In the next intersection at Fall B, potting occurred diagonally across the intersection at $\pm 0.1\text{m} - 0.3\text{m}$. This happened in the last week prior to the investigation as the Mine Overseer reported it was the first time he had noticed the fall.

Fall C was ± 0.1 up to 0.4m thick and occurred whilst cutting. It was already re-supported. Sharp corners formed in the fall. Minor potting along the sidewall, to the north of the fall.

The height of Fall D was 0.73 m at the western end and 0.24 m at the eastern end. It formed a pointed shape at the western side. The fall was re-supported. Water was running out of the roof at $\pm 0.2\text{ l/min}$.

Area E consist out of two areas where minor potting occurred along the sidewall parallel to the split and diagonally across the intersection.

Fall F was in the shape of a parallelogram with the western and the eastern side being 0.15m and 0.10m thick respectively.

Fall G with a height of $\pm 150\text{mm}$ had an elliptical shape diagonally across the intersection.

Fall H was a minor fall in the shape of a half circle across the roadway but it was out of or up to this that a cutter in the roof led, see Figure 9.



Figure 9. Cutter linking up with a roof fall.

Fall I was potting of the roof. On the other side of the pillar, towards the west, a minor piece of roof fell.

Falls J, K and M were all independent areas but linked to the major fall.

Fall J was $\pm 100\text{mm} - 200\text{mm}$ thick with the Mudstone clearly recognisable above the Gritstone.

K was Guttering up to 0.5m high with a small dome towards the eastern side. The height of fall at the dome was $\pm 0.75\text{m}$. This guttering extended around the corner up to fall H.

L was the major fall with a domed shaped top as indicated in Figure 6 by the black shaded area in Fall L. It extended around the corner and was $\pm 2\text{m}$ high. A general view of the fall is shown in Figure 10.



Figure 10. General view of the major fall, Fall L.

Fall M occurred a few days after the major fall and had a height of ± 1 m.

Area N was a single drum lift, ± 3.6 m wide, which was cut through for ventilation purposes. A fall, as indicated in Figures 5 & 6, occurred here as well. The height was ± 0.2 m.

Cause of collapse

The fact that this collapse followed a consistent direction across the panel, even working its way around corners in narrow channels, the deformed roof bolt hole and the floor lift all indicate that horizontal stress played a major role in its occurrence. The direction of the maximum horizontal stress was at right angles to the direction of the axis of the fall.

The fact that the support was installed late where the fall started, was possibly the trigger. After the first minor fall the roof was too high for the roofbolter to do proper installations. The bolts that were installed were consequently too short and could not be tensioned. This aggravated the situation and then the second fall occurred.

The horizontal stress had previously been measured at the mine in three different localities. The magnitudes of the major principal stress were respectively 6.4 MPa, 3.7 MPa and 5.7 MPa with inclinations of 18° , 0° and 5° . If one assumes that the horizontal stress at the fall was the maximum, i.e. 6.4 MPa, then the k-ratio (horizontal to vertical stress) was 5.7 which is greatly in excess of the ratio of 2 to 3 that is regarded as normal.

However, the absolute magnitude is still small in comparison to the compressive strength of the roof material that is usually in the range 50 MPa to 70 MPa. At a depth of 100 m below surface, the normal horizontal stress with a k-ratio of 2.5, is also 6.4 MPa and the effects observed in this investigation, are not common at a depth of 100 m. The horizontal stress in itself is thus not responsible for the fall, rather the ratio of stress to the local strength of the material.

The CMRR rating of the roof at the fall was 23, which is regarded as weak, indicating immediate collapse.

Conclusion

The fall was caused by a localised zone of weakness stretching diagonally across the panel. The driving force behind the fall was the horizontal stress. Once the roof rock was mobilised at the major fall, the effects continued in a direction perpendicular to the direction of the maximum horizontal stress. In the rest of the section, there are no visual indications of horizontal stress like guttering or floor heave. The effects are confined to the relatively narrow area of local weakness. The presence of water is a further indication of weaknesses in the rock mass in that area.

Prevention

This fall could have been prevented by a shorter cut-out distance, denser support pattern and quicker installation of support in the area of local weakness.

Case N Summary: undercutting unsupported sandstone, stooping, old mine

Mine locality: Vryheid Coalfield

Seam: Upper Dundas

Mining method: Drill and blast stooping

Mining depth: 130 m at site of investigation, rapid increase due to overlying mountain.

Mining height: 1.1 m, some roads brushed to 1.7 to 2.1 m, see Figure 1.

General description of roof: The immediate roof consisted of 0.7 m cross-bedded sandstone (to brushed roof elevation) followed by 0.2 m bedded sandstone. During brushing the nether 0.7 m thick sandstone was removed. Where the roof was brushed, the bedded sandstone formed the roof. The roof that fell in this case was the nether 0.7 m. The contact between the two units is uncemented.

Discontinuities present: No major features, but the roof was bedded, jointed and randomly cross-bedded, shown in Figure 2.

Roof MRMR: 28 (poor)

Roof UCS: 12.7 MPa

CMRR: 38 (weak), reduced to 28 (weak) taking account of weathering by acid mine drainage, shown in Figure 3.

RMR: 55 (fair)

Angle of friction: 25° to 35°

Cohesion: 200 – 300 kPa

Time of fall: Between shifts

Supported/unsupported: Supported

Support: M16 by >1.2 m long mechanical anchors, 3 per row at 2 m spacings.

Cut-out distance: Not applicable.



Figure 1. *View of the brushed roadway and the undercut into the pillar during stooping.*



Figure 2. *Bedding and cross bedding of the nether sandstone and the weathered, uncemented contact between the bottom and top sandstones.*



Figure 3. *Discoloration caused by acid mine drainage from the overlying Gus seam workings.*

Support quality: Good

Monitoring: None

Locality: In pillar during stooping.

Horizontal stress indicators: None

General description of fall: There is goaf on either side of the panel being extracted now, which served as main development for several decades (estimated 100 years). The overlying Gus seam has also been stooped and the currently mined panel is approaching the edge of the overlying goaf. Subsidence cracks were found on surface, see example in Figure 4, indicating that the goafs failed through to surface.

The roadway was brushed during development several decades ago. During pillar extraction now, the coal only is extracted, undercutting the sandstone roof. The undercut roof fell, the dimensions being 3 m long by 1 m wide and 0.7 m high. Figure 5 shows the locality of the fall and Figure 6 is a view of it.



Figure 4. Example of a surface crack.

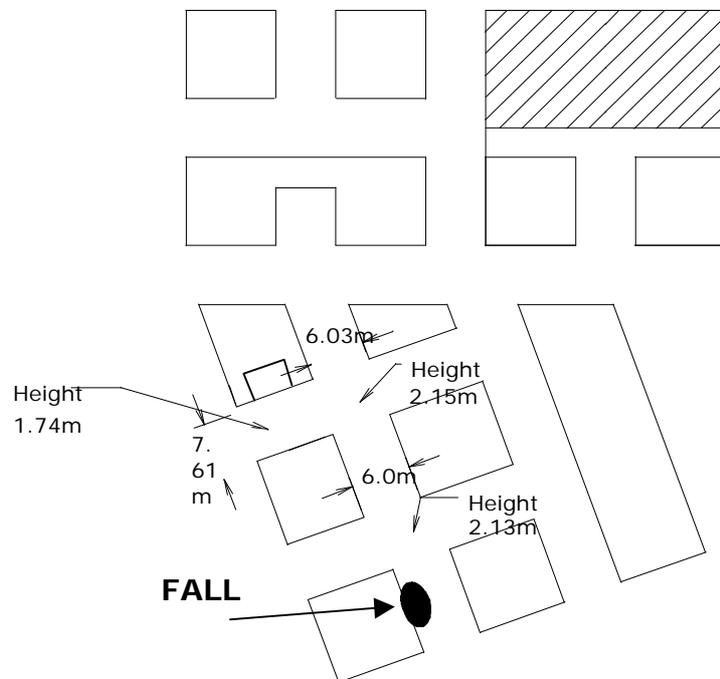


Figure 5. Plan view of the fall area.



Figure 6. View of the fall.

There was high vertical loading in the area, indicated by cracking noises in the pillars, bent roofbolt washers, ribside spitting, pillar compression - shown in Figure 7 - and even roofbolt washers flying off.

Cause of collapse

The high vertical loading caused by stooping and the presence of the overlying goaf edge, results in differential compression of the pillars, the pillar in the process of being mined compressing more than the others. This result in bending of the roof beam, which is a cantilever due to the one end having been removed by the brushing. This beam then fails. Failure is aided by the cross bedding of the sandstone beam and by the lateral movement of the coal as the pillar gets compressed.

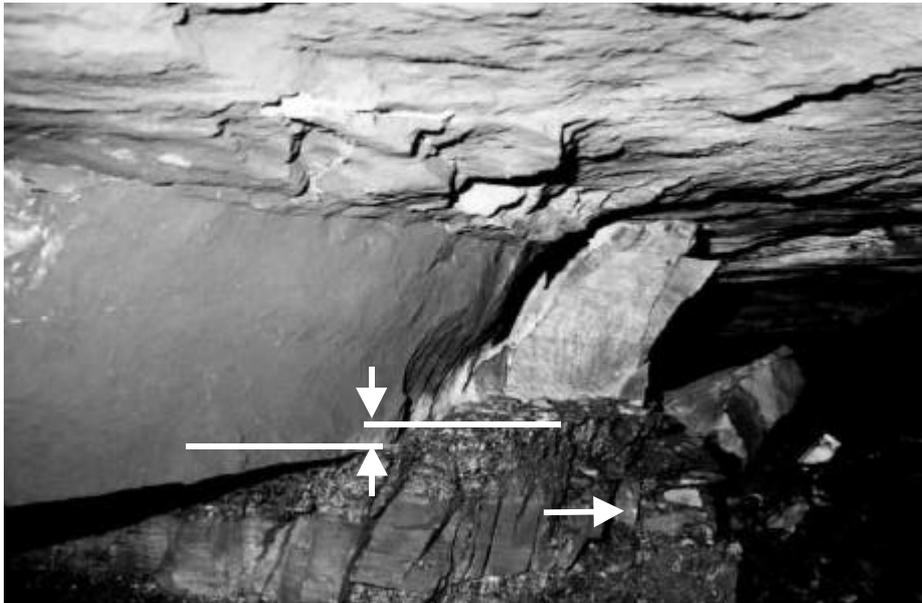


Figure 7. *Vertical compression and lateral expansion of the pillar during extraction.*

Conclusion

This fall was caused by increased vertical load due to stooping operations causing bending of the cross bedded and jointed nether sandstone roof beam, which was weakened by brushing.

The mining geometry was unfavourable, with goafs on both sides and a goaf edge on top. This highlights a unique problem with extracting the last coal from old mines – there is no control over the geometry that was created decades ago, at a time when no consideration was given to the later extraction of the last pillars.

Prevention

This fall could only have been prevented by adequate support of the sandstone brow. Taking into account the low mining height, roofbolting is not a viable option as the bolt length would have had to be greater than the mining height. The remaining options are to mine the brow, which would have an adverse effect on the economics of the operation, or to use temporary supports to protect drilling crews.

Case O Summary: Horizontal stress, Joints

Mine locality: Witbank coalfield

Seam: No 4

Mining method: Square bord and pillar. Pillar centres were 13,5 m, road widths nominally 6,5 m.

Mining depth: 61 m

Mining height: 2,9 to 3,2m

General description of roof: The succession from bottom to top at the falls was: 0,2 sandy laminated shale, 0,05 m fossil contact, 0,55 m laminated shale, 0,51 m sandstone with inter laminated siltstone, 0,98 m inter bedded sandstone/siltstone, 0,87 m sandstone and 1,62 m shale.

Discontinuities present: Prominent joint set running across panel, dip and dip direction approximately $85^{\circ}/319^{\circ}$, strike trace length 19m, dip trace length 0,7 m, large scale condition was planar. Fall was at the crest of a roll, ± 30 m from the limbs. On surface there was a pan ± 60 m away laterally. The panel, 87 m wide, was situated between two transgressional sills.

Roof CMRR: 31 (Weak)

Roof RMR: 49 (Fair)

Roof MRMR: 30 (Poor)

Supported/unsupported: Supported.

Support: 1,5m M16 full column resin bolts, 3 per row at 2m row spacing.

Support quality: Good

Monitoring: None

Locality: Roadway plus 2 intersections, one on either side.

Horizontal stress indicators: The Stress Damage Rating (SDR) varied from 6 (fall of ground) to 3,5 (deep cutter) in the vicinity of the fall to 1 (stable) elsewhere in the section. The direction of the maximum horizontal stress according to underground mapping was from 2° to 28° .

According to stress measurements done in the area, the direction of maximum horizontal stress was 23° and the magnitude 13,5 MPa.

General description of fall: A general view of the fall is shown in Figure 1. The fall height was 4,1 m to 4,5 m. In width it spanned across the roadway (planned width 6,5m – actual could not be measured) and had a length of more than 20 m. It appeared to have started at an intersection where pillar corners were cut away (according to the u/g manager).



Figure 1. General view of the fall.

A crank handled roofbolt, shown in Figure 2, was seen. The roof of the fall was flat, but had severe ripple marks, shown in Figure 3.

Steep dipping joints bound the fall on either side with discoloration caused by water, shown in Figure 4. The area was wet.



Figure 2. Crank handled roof bolt seen in the fall



Figure 3. Ripple marks on the sandstone roof of the fall. Note the discoloration caused by water.



Figure 4. Discoloured joint plane bounding the fall.

Cause of collapse

It is believed that this fall was by a combination of several factors, namely *weak roof strata, horizontal stress, jointing and water.*

The first factor was the weak roof. Three different rating systems, namely CMRR, RMR and MRMR all resulted in low ratings for the rock mass quality. The UCS, UTS, Shear strength and Elastic

modulus of the roof material were found to be low by laboratory tests. Stress mapping indicated the presence of horizontal stress. It was also measured in the area, and had a magnitude of 13,5 MPa. While this magnitude is not considered to be excessive, the weak roof allowed horizontal stress to result in failure.

The other aggravating combination of factors was the observed jointing and water. The maximum horizontal stress did not act either parallel or perpendicular to the joints, resulting in an estimated shear stress of 6,3 MPa to be present across the joints. Coupled to the presence of water, this was most probably the trigger that resulted in this major roof fall.

Appendix II – Summary of Mapped Falls

Before / After	Cause	Height	Width	Length	Int/RW	n/row	Spacing	Road width
n	w	0.2	6	12	i	3	2	6.7
n	w	0.14	6	12	i	3	2	6.7
n	j	0.4	6	12	r	0	0	6.7
n	j	1.9	6	12	r	3	2	6.7
n	w	0.27	6	12	i	3	2	6.7
n	w	0.15	6	12	i	3	2	6.7
n	w	0.18	6	12	i	3	2	6.7
n	h	0.5	6	12	r	0	0	6.7
n	j	0.3	6	12	l	4	2.5	6.9
n	b	0.4	6	12	l	4	2.5	6.9
n	b	0.3	6	12	r	4	2.5	6.9
n	b	0.5	6	12	l	4	2.5	6.9
n	m	0.3	6	12	l	4	2.5	6.9
n	tb	0.25	6	12	l	4	2.5	6.9
n	m	0.6	6	12	l	4	2.5	6.9
n	tb	0.1	6	12	l	4	2.5	6.9
n	m	0.3	6	12	l	4	2.5	6.9
n	m	0.3	6	12	l	4	2.5	6.9
n	h	0.2	6	12	l	4	2.5	6.9
n	h	0.25	6	12	l	4	2.5	6.9
n	w	0.15	6	12	l	4	2.5	6.9
n	j	0.5	6	12	l	4	2.5	6.9
n	m	0.2	6	12	l	4	2.5	6.9
n	j	0.25	6	12	l	4	2.5	6.9
n	tb	0.35	6	12	l	4	2.5	6.9
n	b	4	6	12	r	4	2.5	6.9
n	j	3	6	12	l	4	2.5	6.9
b	tb	0.3	1.5	2.4	l	3	2	6
b	tb	0.5	1	1	l	3	2	6
b	tb	0.06	1.7	3.4	l	3	2	6
b	tb	0.03	3	1.3	r	3	2	6
a	tb	0.025	3	4	l	3	2	6
a	tb	0.06	1.7	3	l	3	2	6
a	tb	0.03	3	1.3	l	3	2	6
a	tb	0.25	3	4	l	3	2	6
a	m	0.5	1.8	2	l	3	2	6
a	m	0.2	0.5	0.5	l	3	2	6
a	tb	0.02	1.2	2.1	l	3	2	6
a	d	0.023	5.7	3.9	l	3	2	6
a	h	0.3	4.7	8.2	l	3	2	6
a	tb	0.25	6.2	12	l	3	2	6
a	tb	0.2	6.2	25	l	3	2	6
a	h	0.15	2	12	l	3	2	6
a	tb	0.15	6	30	l	3	2	6
b	m	0.3	10	12	l	3	2	6
b	m	0.8	12	12	l	3	2	6
a	j	0.6	3	6	l	3	2	6

Before / After	Cause	Height	Width	Length	Int/RW	n/row	Spacing	Road width
a	j	0.5	10	11	l	3	2	6
n	w	1.2	4	15	r	0	0	7
n	w	1	3	18	r	0	0	7.5
a	w	1.3	3	12	r	0	0	5.5
n	w	1	3	16	r	0	0	6
a	j	0.5	11	9	i	3	2	7.5
a	w	1.1	5	10	i	3	2	6.7
a	j	0.1	1	1	i	3	2	6.7
n	w	0.5	4	25	i	3	2	6.7
a	j	1.3	6	45	r	0	0	6.7
a	j	0.7	4	12	i	3	2	6.7
a	w	0.9	5	22	i	3	2	6.7
a	j	0.1	1	2	i	3	2	6.7
a	j	0.7	5	13	i	3	2	6
a	w	0.7	4	12	r	0	0	6
a	w	0.9	8	8	i	3	2	6
n	w	1	5	18	i	3	2	6.7
a	j	1.5	6	60	i	3	2	6
a	w	1.2	3	15	r	0	0	6
a	j	0.9	4	21	r	0	0	6
a	j	1.4	5	7	i	3	2	6
a	j	1.5	4	10	r	0	0	6
a	j	0.2	1	7	i	3	2	7
b	m	0.3	3	3	i	3	2	7
a	b	0.5	3	3	i	4	2	5.5
a	h	2	5	5	i	5	1	5.5
a	tb	0.2	6	12	i	4	2	7
a	tb	0.05	3	5	i	3	2.5	7
a	j	0.05	3	6	i	4	3	7
a	tb	0.1	4	15	i	4	3	7
a	w	0.07	1	1	i	4	3	7
a	j	0.3	1.5	3	i	4	1.5	6.3
a	b	4	6.5	6.5	i	4	1	7
a	m	0.07	0.5	0.5	i	4	2	7
a	m	0.1	1	3	i	4	2	7
b	j	0.3	3	7	l	4	2	7
b	j	0.7	4	13	r	4	2	7
a	m	0.15	3	6	i	4	3	7
a	m	0.1	0.6	2	i	4	3	7
a	tb	0.06	0.7	3.9	i	3	1.5	6
a	j	0.7	1.2	1.8	r	4	1	5.8
a	tb	0.05	2.5	5	r	3	2	6
a	j	0.05	3	5	r	3	2	5.9
a	tb	0.05	2	6	i	3	2	6
n	tb	0.1	3	6	i	0	0	6
a	j	0.3	1	3	i	3	2	6
n	w	0.04	6	6	i	0	0	6.2
a	j	0.07	1	2.5	i	3	1	6
a	j	0.15	1	2	i	0	0	6
a	d	0.04	4	5	r	3	1.5	6
a	d	0.15	2	3	r	2	2	6

Before / After	Cause	Height	Width	Length	Int/RW	n/row	Spacing	Road width
a	d	0.2	0.3	1.7	r	2	2	6
a	tb	0.15	1	4	i	2	2	6
a	tb	0.12	2	6	i	2	2	6
a	j	0.05	1	1	r	3	2	6
a	h	0.08	1	2	r	3	2	6
a	j	0.1	5	12	i	4	2.5	6
a	tb	0.2	5	5	i	4	2.5	6
a	tb	0.05	0.5	0.5	r	3	2	6
a	j	0.1	1	2	i	3	2	6
n	tb	0.3	3	5	i	0	0	6
a	j	0.3	3	6	i	3	2	6
a	j	0.2	3	4	r	0	0	6
a	w	0.8	9.1	9.7	i	3	2	6.8
n	j	2	4	15	r	0	0	6.8
a	w	0.7	9	9	i	3	2	6.8
a	w	0.5	9	9	i	3	2	6
a	j	0.3	8	8	i	3	2	6.5
a	j	2.2	8	8	i	3	2	6
a	j	1.5	8	8	i	3	2	6
n	j	0.2	5	25	r	0	0	6
a	j	0.8	4	6	i	3	2	6
a	j	0.2	1	3	i	3	2	7
a	j	0.2	1	1	i	3	2	7
a	j	0.1	1	4	i	3	2	7
a	tb	0.1	1	1	i	3	2	7
a	tb	0.2	2	2	r	3	2	7
a	tb	0.3	1	1	r	3	2	7
b	h	0.3	2	4	r	3	2	7
b	j	0.3	1	1	r	3	2	7
b	h	3	5	10	i	4	1.5	6
a	tb	0.1	1	1	r	3	2	7
a	tb	0.3	3	5	r	3	2	7
a	tb	0.05	1.5	1.5	i	3	2	7
a	j	0.5	0.5	1	r	3	2	7
b	h	0.1	5	5	r	3	2	7
a	h	0.2	1	2	r	3	2	7
a	j	0.2	0.5	1	i	3	2	7
a	tb	0.1	1.5	1.5	i	3	2	7
b	tb	0.2	1	1.5	i	3	2	7
b	h	0.2	2	7	i	3	2	6
a	j	0.1	1	2	r	3	2	7
a	j	0.5	5	15	r	3	2	7
a	j	0.3	1	1.5	r	3	2	7
a	j	0.2	2	5	r	3	2	7
a	j	0.2	2	2	r	3	2	7
b	h	1	5	6	r	3	2	7
a	b	0.3	5	7	r	3	2	7
a	j	0.2	0.5	2	i	3	2	6
a	j	0.2	2	5	r	3	2	6
b	tb	0.3	2	4	r	3	2	6
b	tb	0.1	1	1.5	r	3	2	6

Before / After	Cause	Height	Width	Length	Int/RW	n/row	Spacing	Road width
a	tb	0.05	6	6	r	3	2	6
a	m	0.2	1	1	i	3	2	6
a	j	0.2	2.6	4	r	3	2	6
a	tb	0.05	2	3	i	3	2	6
a	tb	0.05	6	6	i	3	2	6
b	tb	0.1	2	4	i	3	2	6
a	tb	0.1	4	8	i	3	2	6
b	j	0.3	3	4	r	3	2	6
a	j	0.2	2	4	i	3	2	6
a	h	1	6	30	r	3	2	6
a	tb	0.1	1.5	2	i	3	2	6
a	tb	0.2	5	5	r	2	2	6
a	j	0.3	3	6	i	3	2	6
a	tb	0.5	3	6	r	2	2	6
a	tb	0.3	6	10	r	2	2	6
a	tb	0.4	7	7	i	3	2	6
a	j	0.1	3	6	i	2	2	6
a	tb	1.6	13.6	14.8	i	4	2	7.2
a	j	0.5	4	12	r	3	2	6.6
a	tb	1.4	8.5	10.5	i	3	2	6.1
a	j	0.5	2	6	i	3	2	6.2
a	tb	1.5	6	15	r	3	2	6.2
a	h	0.7	5	12	i	3	2	6.9
a	j	1.5	2	12	i	3	2	6.6
a	j	1.5	6	15	i	3	2	6.5
b	j	0.5	4	7	r	4	1.5	7.2
a	j	6.5	5	5	i	4	1.5	5.5
a	j	2	6	8	r	4	1.5	6
a	j	1.5	6	8	i	4	1.5	4.3
a	b	4	6	20	r	4	1.5	6.4
a	h	2	6	15	r	4	1.5	5.8
a	m	0.7	1	3	r	2	3	6
a	h	4.5	6	30	r	2	3	6

Notes

1. i = intersection
r = roadway
2. n = no support
b = before support
a = after support
3. j = joints
m = bad mining
tb = excessive spacing
d = dyke
b = burnt coal
h = horizontal stress
w = weathering