Safety in Mines Research Advisory Committee

Rating system for coal mine roofs

Draft Final Report

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Executive summary

It is standard practice in the majority of rock engineering applications to use a rock mass classification method to evaluate the condition of rock, i.e. to quantify, in an objective manner, the characteristics of good or bad roof. These rockmass classification methods are routinely used in civil engineering and in tunnel excavations.

Although many rock engineering departments in South Africa use locally developed classification systems, these are sometimes descriptive in nature and undocumented and in most cases restricted in use to the mine on which they were developed. There is currently no standardization that will allow a locally developed method to be applied by a person employed by another company. The local systems cannot be compared with one another, nor can one set of results be converted to an equivalent rating in another system.

This project attempted to overcome the problem by evaluating and documenting all the existing systems that are used in South Africa and others that have been developed in other countries, and proposing the way forward for the development of a system that could be used universally on South African collieries (Phase2). Alternatively, the information presented in this report could be used to enhance local systems.

The review of all existing systems highlighted that in the mining industry the Q and RMR classifications form the basis of many empirical design methods as well as the basis of failure criteria used in many numerical modelling programs. Application of these systems in South African mines, specifically in coal, have been limited due to the lack of acceptance of the techniques from the industry, which can be attributed partially to the lack of personnel trained in the use of such systems and/or the generally good coal roof conditions in South Africa compared to the rest of the world. Where these systems have been used it has been mainly for portal design or feasibility studies, and there has been little documentation on their use in South Africa.

It is considered that the CMRR could overcome most problems associated with the application of rock mass classification systems in coal mining. However, due to the fact that the system is based on case histories from the United States, certain modifications need to be made to the system to cater for the different conditions in South African collieries. From initial research carried out so far in using the CMRR in the context of South African coal mining, it appears that appropriate modifications can be made.

The shortcomings of CMRR, which were identified during the application of CMRR are that:

- Exposure into the roof is required (underground CMRR only)
- Only the bolted height is rated. In South Africa, 2.0 m into the roof is the height that is usually rated.
- Although sets of joints have been considered in CMRR, single joints can have an influence and should thus also be included.
- Joint orientation is not taken into account (underground CMRR only).
- Stress adjustment is required in the rating system to account for the influence of high horizontal stress (underground CMRR only)
- No adjustment is made for the effects of blasting (underground CMRR only)
- The position of soft or hard layers into the roof is not taken into account (both underground and borehole core CMRR)
- Skilled personnel to carry out ratings are required (both underground and borehole core CMRR)

The investigation into the rating systems being used in South Africa highlighted that roof rating systems are applied mainly for planning purposes, and not for determining any change in conditions underground. However, rating systems have also been developed in South Africa by Ingwe Coal Rock Engineering (a division of BHP Billiton Energy Coal), in which support systems are changed based on the ratings obtained for underground conditions.

Impact Splitting Tests, Section Physical Risk Rating and the Section Performance Rating systems developed in South Africa are described as being most effective and appropriate for South African conditions. They can distinguish different roof conditions necessary for initial planning and support design. They can also be used for identifying changing conditions while mining and determining the best response to the different conditions. These systems are also found to be the best systems to address the risk (**Variable Step Model**) used by sectors of the coal mining industry.

It is finally concluded therefore that adoption of the CMRR system for South African conditions is neither appropriate nor necessary.

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Table of Contents

List of Figures

List of Tables

List of contracted Enabling Outputs

NO.	ENABLING OUTPUT	
	Phase1	
1.1	Literature survey on published coal mine roof rating systems	
1.2	In depth study of CMRR	
1.3	Collection and study of SA rating systems	
1.4	Parallel application of existing systems underground	
1.5	Comparison of results of existing systems	
1.6	Report	
	Phase2	
2.1	Combine best elements into new system	
2.2	Create Windows program, produce final report	
2.3	Launch at workshop	

1.0 Introduction

South Africa is the third largest coal exporting country after the United States of America and Australia. According to the Department of Minerals and Energy Affairs (DME), 2000, 223 Million tonnes of coal were extracted from South African mines in 1999. During the course of the daily underground operations, workers are regularly exposed to major safety hazards. Roof failures in underground South African coal mines are a major safety hazard as well as a substantial operational burden. From the South African Mines Reportable Accidents Statistics System (SAMRASS), 2000, a total of 231 accidents were reported from coal mines in South Africa. The number of underground accidents was 147, with 53 resulting from falls of ground. While these figures have been steadily decreasing over the last 5 years, the rates are still about 3 times greater than in the United States (Mark 1999).

Producing a mine plan with safe, stable roadways can be a complex exercise. The structural competency of the mine roof is directly related to geological conditions that can vary from mine to mine and also within the same mine. In a continuing effort to reduce the safety hazards in the South African coal mining industry, the Safety in Mines Research Advisory Committee (SIMRAC) has been funding research into determining the major causes of roof falls in South African collieries and providing solutions. A part of this research is the investigation of roof rating systems in coal mines using rock mass classification systems. These systems aim to guide the mine rock engineer in providing quick and easy field techniques that quantify geological characteristics of the roof in a standardized manner for engineering analysis.

Rock mass classification systems constitute an integral part of empirical mine design. The use of such systems can be either implicit or explicit. They are traditionally used to group areas of similar geotechnical characteristics, to provide guidelines of stability performance and to select appropriate support. In more recent years, classification systems have often been used in tandem with analytical and numerical tools. There has been an increase of work linking classification indexes to material properties such as modulus of elasticity, m and s for the Hoek & Brown failure criterion, etc. These values are then used as input parameters for the numerical models. Consequently the importance of rock mass characterization has increased over time. The primary objective of all classification systems is to quantify the intrinsic properties of the rock mass based on past experience. The second objective is to investigate how external loading conditions acting on a rock mass influence its behaviour. An understanding of these processes can lead to the successful prediction of rock mass behaviour for different conditions.

The earliest reference to the use of rock mass classification for the design of tunnel support is by Terzaghi (1946) in which the rock loads, carried by steel sets, are estimated on the basis of a descriptive classification. Since Terzaghi (1946), many rock mass classification systems have been proposed, the most important of which are as follows:

- Lauffer (1958)
- Deere (1970): Rock Quality Designation, RQD
- Wickham et al (1972): Rock structure Rating (RSR – Concept)
- Bieniawski (1973): Geomechanics Classification, Rock mass Rating
- Barton et al. (1974): Q- System

Most of the multi-parameter classification schemes by Wickham et al (1972), Bieniawski (1973, 1989) and Barton et al (1974) were developed from civil engineering case histories in which most of the components of the engineering geological character of the rock mass were included. Studies of these systems have shown that their main application is for hard and soft jointed rock masses. Several classification systems have been developed and modified for underground coal mining. Most rock engineers locally and abroad have been using locally developed classification systems that are in most cases not well documented and are restricted to the developer of such systems or the mine on which the system was developed. Furthermore, these systems cannot be compared with one another or results converted to an equivalent rating in another mine. In this project, the rock mass classification systems in mining are reviewed with an emphasis on the RMR by Bieniawski, Q-System by Barton, RQD by Deere and CMRR system that was developed in the United States by Mark and Molinda, (NIOSH), and Impact Splitting Test developed by Oldroyd and Buddery. This forms a part of the current COL812 project that aims at developing a custom made system for South African Coal Mines.

2.0 Literature review

An extensive literature review regarding rock mass classification systems was conducted and is summarised in Appendix A. It has been found that in mining engineering design, the Q and RMR classifications form the basis of many empirical design methods as well as the basis of failure criteria used in many numerical modelling programs. However, application of these systems in South African mining industry, specifically in coal, has been limited due to a lack of acceptance from the industry. Where these systems have been used it has been mainly for portal design or feasibility studies and in many cases, are not documented.

It was considered that the CMRR could overcome most problems associated with the application of rock mass classification systems in coal mining. However, due to the fact that the system is based on case histories from the United States, certain modifications would have to be made to the system to cater for the different conditions in South African collieries.

3.0 In depth study of Coal Mine Roof Rating (CMRR)

The United States Bureau of Mines (USMB) have developed the Coal Mine Roof Rating (CMRR) classification system to quantify descriptive geological information for use in coal mine design and roof support selection (Molinda *et al*, 1994). This system results from years of geologic ground control research in longwall mines in the United States. The CMRR weighs the geotechnical factors that determine roof competence, and combines them into a single rating on a scale from 0-100. The characteristics of the CMRR are that it:

- Focuses on the characteristics of bedding planes, slickensides, and other discontinuities that weaken the fabric of sedimentary coal measure rock.
- Applies to all U.S. coalfields, and allows a meaningful comparison of structural competence even where lithologies are quite different.
- Concentrates on the bolted interval and its ability to form a stable mine structure.
- Provides a methodology for geotechnical data collection.

The USBM is currently engaged in research to further develop the CMRR to be applicable to other coal mines around the world. The principle behind the original CMRR system (1994) is to evaluate the geotechnical characteristics of the mine roof instead of the geological description. CMRR emphasizes structurally weak or strong units instead of geologic divisions. The structure of the system is similar to Bieniawski's RMR system in that the important roof parameters are identified, their influence on roof strength is quantified and the final rating is calculated from the combination of all the parameters. Figure 1 shows the parameters that compose the CMRR system. The system is also designed such that the final rating/unsupported span/stand-up time relationship is comparable to that of the RMR. However, the CMRR is intended to be a universal system for coal mining and to initially exclude time-consuming and expensive laboratory analyses. Later, Molinda and Mark (1999) documented a revised approach that takes into consideration the Point Load Test.

An important attribute of the CMRR is its ability to rate the strength of bedded rocks in general, and of shales and other clay-rich rocks in particular. Layered rocks are generally much weaker when loaded parallel to bedding, and the CMRR addresses both the degree of layering and the strength of the bedding planes. Recent research has shown that most coal mine roofs are subjected to high horizontal stresses. The CMRR has been modified by Molinda and Mark (1999) to retain its ability to identify those rocks that are most susceptible to horizontal stresses.

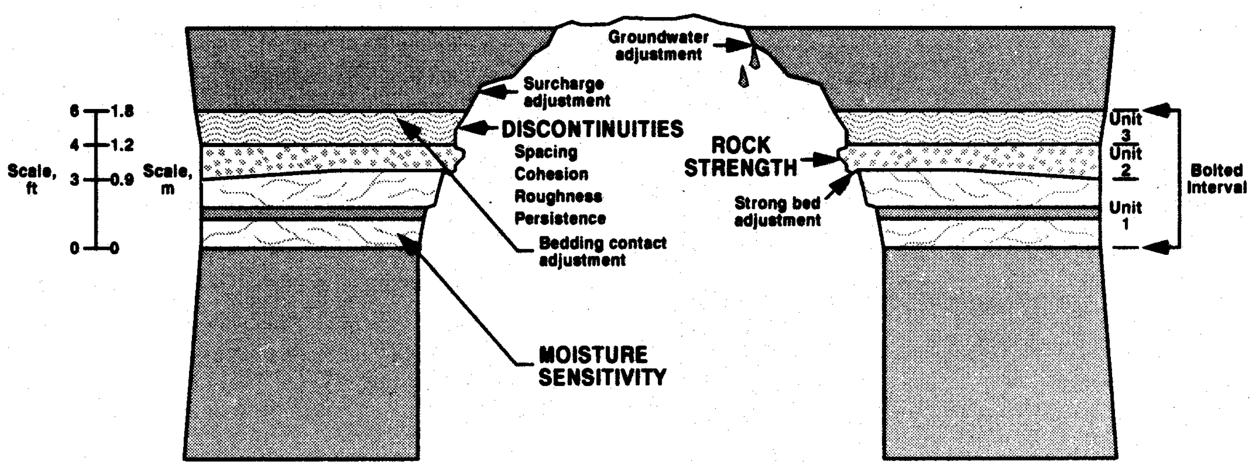

Figure 1 Components of the CMRR System

Data gathering for the system relies only on observation and simple contact tests using a ball peen hammer, a 9 cm mason chisel, a tape measure and sample bags. All the data is recorded in a designed data sheet that is used to calculate the final rating. A Lotus 1-2-3 spreadsheet program is available from the U.S. Bureau of Mines for use with the CMRR rock mass classification. The information that is recorded on the data sheets is entered into the spreadsheet program, creating a permanent computer record of the field notes. The calculation is based on rating the exposed roof that is divided into structural units. Each unit is rated individually mainly on an evaluation of the discontinuities and their characteristics. Next, the CMRR is determined for the mine roof as a whole. The ratings of the units within the bolted interval are first combined into a thickness-weighted average. Then a series of roof adjustment factors are applied with the most important being that of the strong bed. It has been found that the structural competence of a bolted mine roof is largely determined by its competent member. All the parameters are combined to calculate the final CMRR.

The following is a summary of the factors that contribute to the final unit rating value:

a) Compressive strength of intact rock: The ball peen hammer test is used to place rock into five classes, depending on the nature of the indentation.
b) Cohesion of discontinuities: The strength of the bond between the two faces of a discontinuity is estimated by observation of roof behaviour, assisted by the chisel test.
c) Roughness of discontinuity: The surface of the discontinuity is classified as “rough”, “wavy”, or “planar” by observation.
d) Intensity of discontinuities: The average observed distance between discontinuities within a unit.
e) Persistence of discontinuity: The observed areal extent of a discontinuity plane.

f) Moisture sensitivity: Estimated from an immersion test, and only considered if significant inflows of groundwater are anticipated or if the unit is exposed to humid mine air.

After the individual unit ratings have been determined, they are summed into a single rating for the entire mine roof and adjustments are applied from tables provided by the USBM by taking account of the following:

- Strong beds in the bolted interval
- Number of lithologic units contacts
- Groundwater and
- Surcharge

Individual ratings and adjustment factors are rated in Tables publish by USBM (1994).

Mark and Molinda (1999) modified the original CMRR described above owing to the lack of its application before any mining because it requires underground observations. An entirely new system was developed to determine the CMRR from exploratory drill core using the Point Load Tests (PLT) to determine the strength parameters that account for approximately 60% of the final rating. The new system uses both diametral (parallel to bedding) and axial (perpendicular to bedding) PLT's. The diametral tests allow the estimates of bedding plane cohesion and rock anistrophy, both of which are critical to estimating susceptibility to horizontal stresses. Traditional core logging procedures are used to determine discontinuity spacing and roughness. To ensure compatibility with the original CMRR (1994), the new rating scales were verified by comparing drill core results with nearby underground mining exposures.

A large database of strength ratings of rocks has been assembled through extensive point load testing and logging in the United States. Over 2000 PLT (both axial and diametrical) have been made on common coal measure rock types from mines representing most U.S. coal fields.

The CMRR has been determined for 97 roof exposures from 75 coal mines across the United States by Molinda and Mark (1994). All of the major U.S. coal basins are represented, with sizes ranging from small new mines to some of the largest longwall operations. The data has been partitioned to reflect the following three broad classes of roof based on a scale of 0-100: weak (0-45), moderate (45-65), and strong (65-100). Table 1 shows the CMRR classes with corresponding geological conditions.

CMRR has been integrated into support design programs like the USBM Analysis of Longwall Pillar Stability (ALPS) program in calculation of safety factor for given coal pillar sizes based on applied loads and strength of the pillar. A similar case study in Australia by Mark et al (1999) has used the CMRR to develop a new methodology for chain pillar design called the Analysis of Longwall Tailgate Serviceability (ALTS). In both cases, statistical analysis from case histories of CMRR values have been used in conjunction with existing pillar design formulae to develop a relationship between the Safety Factor and Roof Rating. The combination of CMRR with empirical formulas has improved the accuracy of design of gate entry systems in the U.S. by integrating case histories developed through in-mine data collection techniques with numerical modelling and empirical pillar design formulas.

Table 1 CMRR classes in the U.S., after Mark and Molinda (1994)

CMRR Class	CMRR Region	Geological Conditions
Weak	0-45	Claystones, Mudrocks, Shales
Moderate	45-65	Siltstones and Sandstones
Strong	65-100	Sandstones

Butcher (1999) has been documenting the application of the CMRR to South African strata conditions since it was first introduced to coal mining industry in 1998. Since that time, the system has been used on a limited basis owing to the fact that South African coal operations have generally been conducted in good geotechnical conditions compared to other parts of the world. Furthermore, rock classification systems have generally suffered due to the lack of trained Engineering Geologists or Rock Mechanics Engineers who can implement such systems.

Geotechnical site investigations were conducted (as part of SIMRAC COL613) from 20 fall of ground incident sites in South African coal mines. The CMRR classification system was used to classify the roof conditions at the fall sites. In addition to that, Bieniawski's Rock mass Rating and Laubscher's Mining Rock mass Rating were used as comparisons with the CMRR. A stress damage survey was also undertaken to relate rock mass damage to the horizontal stress regime. In addition a coal cleat damage was done to relate maximum horizontal stress direction to cleat orientation. All CMRR values obtain from the underground mapping sites fell in the weak class i.e. on a scale 0-100, between 0-45. Many observations from the fall of ground site mappings in South Africa were found to collate with experiences gained in the United States. However, a wide range of CMRR values were noted in some areas where roof conditions deteriorated in close proximity of major dykes or sills.

In another study by Butcher et al 2001, further CMRR classification studies were carried out as part of a SIMRAC project to create a geotechnical database of the South African coal fields for design input into open mining operations. The following conclusions with respect to CMRR values for South African coal mines were made:

- Roof shale's were generally below CMRR of 45 (weak)
- Sandstones were generally above CMRR of 45 (moderate to strong)
- Siltstones generally fell in the moderate CMRR range (45-65)

These observations correlate closely to Mark's (1994) work that siltstones and sandstones in the U.S. were moderate to strong. The CMRR has been found to be robust enough to classify and describe the roof conditions that are found in South Africa and that it was easy to learn the technique. Experiences by Butcher (1999) with the RMR and MRMR systems showed a limited application compared to the CMRR, as they tend to overate the ground conditions by at least one class. The RMR over rated roof conditions due to lack of sensitivity in the allocation of joint condition and fracture values.

However, despite these advantages in some cases the CMRR values gave a wide range in areas of high horizontal stresses and in proximity of major geological features. In one case the method over rated roof conditions (CMRR=55) in an area where orientation of major/minor geological features resulted in roof collapses due to its inability to cater for these in the unit contact adjustment.

3.1 Evaluation of CMRR

Both CMRR underground and drill core CMRR have been tested as part of this study. Mr. G. Molinda of NIOSH (one of two developers of CMRR) visited South Africa in October 2001, and these tests were conducted with his assistance.

During this study, the greatest difficulty experienced underground with the trials of CMRR was to find roof exposures with sufficient height. It was sometimes possible where there were air crossings, however, most of the time in most of the sections, CMRR could not be applied. Therefore, the underground visits suggested that for quick and comparative results, a detailed rating system which requires exposure of roof should only be used in the planning stage on borehole cores.

One other problem experienced underground was the effect of a single discontinuity which could cause significant damage to the roof. Because CMRR only takes sets of discontinuities into account, it was observed that the effect of single joints should also be included in a coal mine roof rating system. Also, the direction of joint or joints should be included. Although drill and blast method is not a common mining practice in South Africa, blasting damage should also be included in a rating system. A recent SIMRAC project (COL613) highlighted that approximately 10 per cent of all falls of ground in South African collieries, that were monitored during the course of the project, were caused by high horizontal stress. This highlights the importance of an adjustment factor in a rating system to account for high horizontal stress.

Other important shortcoming of CMRR was the rated height into the roof. As will be discussed in a later chapter of this report in South Africa usually 2.0 m into the roof is rated using rating systems. One advantage of this is that if the rating system is used for comparison purposes, it is important to rate the same height in each rating. Also, the effect of soft layers high into the roof, even if significantly thinner than those lower in the sequence, can affect the stability of roof. Therefore, it is strongly suggested that a rating system should always rate the same height into the roof.

During this task, there was difficulty in comparing underground CMRR with locally developed systems owing to the fact from the collieries visited, their systems were not developed for rating the roof, but for planning purposes. This part of the project therefore could not be carried out. However, impact splitting testing and CMRR were compared on surface using drill cores. This highlighted the major shortcoming of CMRR with respect to the relative positions of stiff and soft layers in the roof. Figure 2 shows three different 0.9 m long cores. Each core contains three different 0.3 m long layers, namely, sandstone, shale and siltstone, but set up in different sequences, e.g. sandstone is positioned at the top, middle or bottom of the different core runs respectively.

The results obtained from CMRR were exactly the same for all three cores. That means that CMRR does not consider the position of soft or stiff layers within the roof strata. However, impact splitting tests resulted in three different ratings based on the position of stiff sandstone layer into the roof that affects the stability of the roof. This indicates that the CMRR does not rate the stability of roof. Rather it rates the quality of roof as a whole without considering the positions of different layers in the roof. This has major implications in collieries, since in many cases the support design is based on the stiffness of the immediate roof layer. The last shortcoming of CMRR is requires skilled personnel and some degree of training.

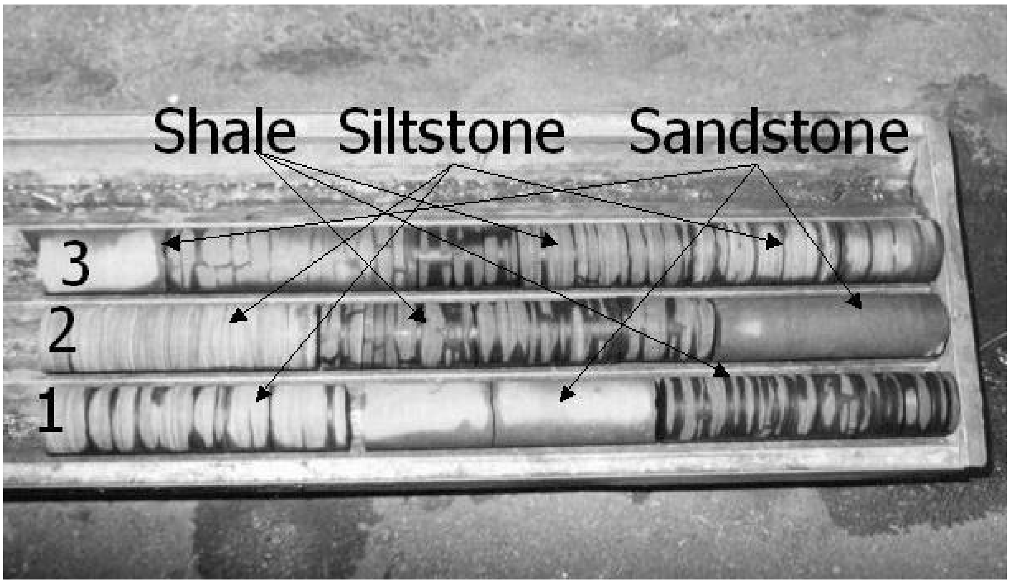

Figure 2 Cores used for CMRR and impact splitting testing

In summary, the shortcomings of CMRR, which were identified during the application of CMRR are summarized below:

- Exposure into the roof is required (underground CMRR only)
- Only the bolted height is rated. In South Africa, 2.0 m into the roof is the height that is usually rated.
- Although sets of joints have been considered in CMRR, single joints can have an influence and should thus also be included.
- Joint orientation is not taken into account (underground CMRR only).
- Stress adjustment is required in the rating system to account for the influence of high horizontal stress (underground CMRR only)
- No adjustment is made for the effects of blasting (underground CMRR only)
- The position of soft or hard layers into the roof is not taken into account (both underground and borehole core CMRR)
- Skilled personnel to carry out ratings are required (both underground and borehole core CMRR)
- Subjectivity rating is not entirely eliminated

4.0 Rating systems being used in South African collieries

4.1 Rating systems developed for planning purposes

Van der Merwe (2001) developed the first roof rating system in South Africa in 1980, using Rock Quality Designation (RQD). In this rating system the critical height into the roof was taken as 2.0 m. This height of the roof was initially rated with RQD. Following a splitting test conducted with a chisel at regular distances along the core, RQD was re-applied and final results were compared with the initial results. The final rating was then obtained based on the difference between the initial and final RQDs. Due to possible discrepancies resulting from the use of chisels with different geometries and forces, van der Merwe developed a standard chisel for all roof rating tests. However, this system has not been documented by van der Merwe. A summary of the rating systems that have been documented for use in coal mining in South African is given in Table 2.

Jermy and Ward, 1988 conducted an investigation into relating geotechnical properties of various sedimentary facies to their observed underground behaviour to quantify geological factors that affect roof stability in coal mines. Twenty-four distinct facies types were determined from borehole cores from a number of collieries throughout South Africa. A database of 10 000 tests from core samples was compiled from the Waterberg, Witbank, Highveld, Eastern Transvaal, Klip River, Utrecht and Vryheid Coalfields. The results from the tests have shown that those facies with lower direct tensile strengths generally gave rise to unstable roof conditions. Furthermore, the low direct tensile strengths of the argillaceous facies were found to be very important when considering the behaviour of these rocks underground. The arenaceous facies were found to have higher average direct tensile strengths. However, the authors found that this can be reduced dramatically by the presence of argillaceous or carbonaceous partings within the rock which can affect the roof stability. Other tests that were included in the assessment were the Brazilian Disc Strength and the Uniaxial Compressive Strength but these were found not to be of importance when considering the weakness of the rock in tension. Description of sedimentary facies and summary of their underground properties is given in Table 3.

Table 2 A summary of some classification systems used in South African coal mining and their main applications

Name of classification system	Form and Type**	Main Applications	Reference
Roof and floor classification for collieries	Descriptive form	For quantification of geological factors that affect roof stability	Jermy and Ward, 1988
Duncan Swell and Slake Durability tests	Numerical and behaviouristic form Functional type	Quantification of floor conditions	Buddery and Oldroyd, 1992
Impact Splitting Test	Descriptive and behaviouristic form Functional type	Coal roof characterization and support design	Buddery and Oldroyd, 1992
CMRR	Descriptive and behaviouristic form Functional type	Coal roof characterization and support design.	Molinda and Mark, 1994
Section physical risk and performance rating	Descriptive Functional type	Classification of adherence to mine standards and physical rating	Oldroyd and Latilla, 1999

**Definition of the Form and Type:

Descriptive form: the input to the system is mainly based on descriptions

Numerical form: the input parameters are given numerical ratings according to their character

Behaviouristic form: the input is based on the behaviour of the rock mass.

General type: the system is worked out to serve as a general characterization

Functional type: the system is structured for a special application (for example for rock support recommendation)

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Table 3 Description of sedimentary facies and summary of their underground properties

FACIES	DESCRIPTION	PROPERTIES OF ROCK STRATA UNDERGROUND
1	Massive dark grey to black carbonaceous siltstone.	Very poor roof and floor strata due to low tensile strength and deteriorates rapidly upon exposure. Roof falls common and floor heave occurs when depth of mining exceeds 150 m.
2	Lenticular-bedded siltstone with discontinuous ripple cross lamination. Resembles lenticular bedding of Reineck and Wunderlich (1986).	
3	Alteration of 1 cm thick layers of flat laminated siltstone and fine grained sandstone.	
4	Flaser bedded siltstone and fine grained sandstone as described by Reineck and Wunderlich (1968).	Reasonable roof strata which deteriorates upon exposure giving rise to spalling from the roof.
5	Ripple cross laminated fine-grained grey feldspathic sandstone.	Reasonable roof strata, although localised roof falls do occur due to parting along silt drapes. Durability good.
6	Ripple cross laminated fine-grained grey feldspathic sandstone with silt drapes and grit bands.	
7	Massive fine grained greyish white feldspathic sandstone.	Very competent floor and roof strata due to low porosity and high tensile strength.
8	Fine grained greyish white feldspathic sandstone with planar/trough crossbeds.	
9	Massive medium grained white feldspathic sandstone.	Good roof and floor strata with fairly high tensile strengths. Sometimes creates problems due to poor goafing ability in stoping areas.
10	Medium grained white feldspathic sandstone with planar/trough crossbeds	
11	Massive coarse grained white feldspathic sandstone.	Good roof and floor strata. Decomposes under prolonged saturation giving rise to stability problems.
12	Coarse grained white feldspathic sandstone with planar/trough crossbeds.	
13	Intensely bioturbated carbonaceous siltstone or fine-grained sandstone.	Deteriorates rapidly upon exposure and saturation to give roof and floor instability.
14	Medium to coarse-grained feldspathic sandstone with irregular carbonaceous drapes and slump structures.	No information available.
15	Highly carbonaceous silty sandstone.	No information available.
16	Whitish brown calcrete.	Not applicable.
17	Highly weathered creamy orange to grey Beaufort (?) mudstone.	
18	Unweathered grey Beaufort (?) mudstone.	
19	Massive khaki to grey mudstone associated with diamictite.	
20	Dark greyish black gritty diamictite with angular 0-4 mm matrix supported clasts	
21	Dark greyish black pebbly diamictite with , angular matrix supported clasts > 4 mm diameter.	
22	Coal mixed dull and bright.	More stable roof rock than facies 1-3.
23	Mixed coal and mudstone.	Not applicable.
24	Massive greyish black carbonaceous mudstone associated with coal seam middling.	

Buddery and Oldroyd (1992) developed a roof and floor classification system for collieries. The following philosophy was applied in devising a suitable classification system:

- The rock property tests should be related to the expected mode of failure of the strata.
- The whole spectrum of strata should be tested with particular emphasis being placed on obtaining the properties of the weakest material.
- Large numbers of tests should be able to be conducted simply, quickly, at low cost and in-house.

Roof failure in South African coal mines is strongly related to the frequency of laminations or bedding planes. In their roof classification, Buddery and Oldroyd (1992) considered a Coal Rock Structure Rating (CRSR) system to classify the roof condition. Tests to indicate the propensity of the laminations or bedding planes to open and separate will therefore be ideal for planning stages. The tests should indicate the mode of failure of the roof and it should be easy for a large number of the tests be conducted in-house. This was initially based on three parameters: RQD, the results of impact splitting tests, and a parameter related to joint condition and groundwater. Due to the impracticality of satisfactorily distinguishing between drilling-induced and natural fractures in the coal measures strata, the RQD parameter was discarded from the system. The third parameter proved to be difficult to determine irrespective of the roof type. It was, therefore, decided to confine the determination of roof ratings to the results of impact splitting tests.

The Impact Splitting Test involves imparting the same impact to the core every 20 mm intervals. The resulting fracture frequency is then used to determine a roof rating. The instrument shown in Figure 3 consists of an angle iron base which holds the core. Mounted on this is a tube containing a chisel with a mass of 1.5 kg and a blade width of 25 mm. The chisel is dropped onto the core from a constant height according to core size, 100 mm for a 60 mm diameter core and 64 mm for 48 mm diameter core. The impact splitter caused weak or poorly cemented bedding planes and laminations to open, thus giving an indication of the likely in situ behaviour when subjected to bending stresses.

Figure 3 The Impact Splitting Equipment

It is suggested that, when designing coal mine roof support, 2.0 m of strata above the immediate roof should be tested. If the roof horizon is in doubt, then all strata from the lowest likely roof horizon to 2.0 m above the highest likely roof horizon are tested so that all the potential horizons may be compared. In this classification system, the strata are divided into geotechnical units. The units are then tested and mean fracture spacing for each unit is obtained. An individual rating for each unit is determined by using one of the following equations:

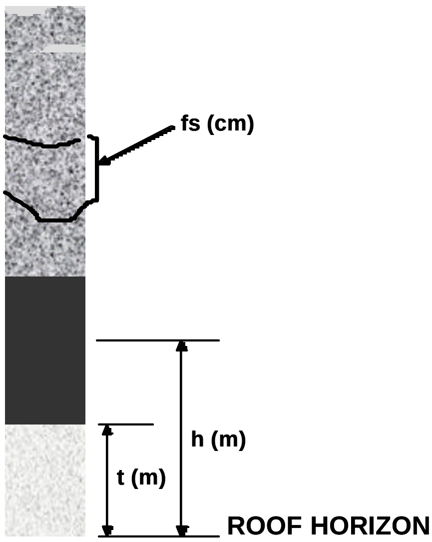

Figure 4 Impact splitting unit rating calculation

For $fs \leq 5$ rating = $4fs$

For $fs > 5$ rating = $2fs+10$

Where *fs* = fracture spacing is in *cm*

This value is then used to classify the individual strata units into rock quality categories as shown in Table 4. For coal mine roofs, the individual ratings are adjusted to obtain a roof rating for the first 2.0 m of roof. It was stated that the immediate roof unit will have a much greater influence on the roof and consequently the unit ratings are weighted according to their position in the roof by using the following equation:

Weighted rating = rating x 2(2-h) t

Where *h* = mean unit height above the roof in metres and *t* = thickness of unit in metres

The weighted ratings for all units are then totalled to give a final roof rating. Buddery and Oldroyd (1992) concluded that good agreement between expected and actual roof conditions has been found when using this rating system.

Table 4 Unit and coal roof classification system (After Buddery and Oldroyd, 1989)

Unit Rating	Rock Class	Roof Rating
< 10	Very poor	< 39
11 – 17	Poor	40 – 69
18 – 27	Moderate	70 – 99
28 – 32	Good	100 – 129
> 32	Very good	> 130

This rating system has been recently modified by Ingwe Rock Engineering to take into account areas where the immediate roof is coal. The unit rating is multiplied by 1.56, which is the density of sandstone (2500 kg/m3) divided by coal density (1600 kg/m3).

Based on this rating system the following support patterns are adopted. It should be noted that the roof support patterns are only applicable where they have been exercised for many years. Also, as will be explained later in this chapter, this rating system is used together with a special current-with-mining assessment technique to adapt to changing roof conditions.

Table 5 Unit and coal roof classification system (After Buddery and Oldroyd, 1989)

Roof Rating	Estimated Support	*Comment*
Very Good	1.2m x 16mm point anchor, 5 bolts in intersections only	
Good	1.2m x 16mm point anchor, 5 bolts per intersection, 2 bolts per row with rows 2m apart	
Moderate	1.5m x 16mm full column, 9 bolts per intersection, 3 bolts per row with rows 1.5m apart	
Poor	2.0m x 20mm full column, 16 bolts per intersection, 4 bolts per row with rows 1m apart, possibly with W-straps	*Reduce road widths to less than 6m*
Very poor	Specialised support, e.g. a combination of cable anchors, trusses, shotcrete, W-straps, etc	*Very poor roofs are uneconomic and are usually only traversed to get to reserves.*

Sasol Coal has developed a roof rating system based on fall of ground accidents. Analyses of fall of ground (FOG) accidents in group collieries indicated that almost all such accidents occurred near dykes and underneath rivers. The collieries have been divided into four groups indicating the roof conditions based on these two criteria. These areas are marked on mine plans as “Normal”, Class “C”, Class “B” and Class “A”. The worst and the best ground conditions are expected in Class “A” and “Normal” respectively. Although there is no difference in specified mining parameters between the “Normal” and Class ”C”, Class ”C” gives the section a warning to be aware of possible changes in ground conditions thereby giving the section time to ensure their support systems are strictly adhered to before reaching Class “B” area. In Class "C" areas, a spare roofbolter and tell tales should be kept for possible roof deterioration.

On each special area plan, a borehole log is also attached to indicate to mining personnel the roof conditions in the area. This also assists mining personnel in determining what length of roofbolt to use in the area. The same mining group has also developed a rating system to be used on borehole cores in greenfield areas, called Percentage Lamination Plan. This plan assists mining personnel in determining;

- the thickness of laminated material,
- whether the laminated stratum is high or low in the roof,
- whether the lamination is such that intersection failure can occur,
- whether the section is approaching ground where drastic changes in roof conditions can occur.

This plan indicates the percentage laminated strata in the direct roof and is available in the following ranges: the first metre of roof, the second meter of roof and the first two metres of roof.

There are also rating systems used in South Africa that geological based. Since the geological characteristics of roof vary from mine to mine, such a rating system is only applicable for a particular mine and/or area. The geologists usually conduct observations to find a particular layer into the roof, during the logging of boreholes. This information is then marked on mine plans referred to as Roof Hazard Plans. In geology based rating systems, the thickness of particular layers is also found to be important. Therefore, for some mines, the roof rating is based on the thickness of particular layers, such as sandstone, shale or siltstone, and the roof support pattern is determined by the quality of the roof. It was also found that geological discontinuities are important and play a major role in the quality of roof, therefore, some mines adapted rating systems based on these features.

The investigation into the rating systems being used in South Africa highlighted that roof rating systems are being used mainly for planning purposes, and not to determine the changing conditions underground. However, rating systems have also been developed in South Africa by Ingwe Coal (Oldroyd and Latilla, 1999), in which support systems are changed based on evaluation of underground conditions.

4.2 Proactive rating systems developed for changing conditions

Mechanised mining allows sections to be developed at a rapid rate, typically more than 1000 m per month for most sections, this can result in a variety of conditions being encountered in a single section in a very short time. Ingwe Coal Rock Engineering (a division of BHP Billiton Energy Coal) has identified a number of accidents in their mines that are caused primarily by the inability to recognise changing conditions and therefore failing to apply necessary counter measures timeously. Furthermore, one of the Codes of Practice (CoP's) in terms of the South African Mine Health and Safety Act, 1996, requires mines to compile a mandatory Code of Practice to combat rock fall accidents.

In order to address the problems and requirements mentioned above, Ingwe Coal Rock Engineering has designed two underground forms: the "Section Physical Risk Rating" for measuring the physical conditions and the "Section Performance Rating" for determining how well the underground section personnel react to the conditions. Both forms are essentially risk matrices defining various scenarios, each with a certain weighting. The forms have been successfully applied for bord and pillar operations for the past 5 years in the Ingwe and Eyesizwe collieries.

The Section Physical Risk Rating form is a basic questionnaire requesting information regarding geological conditions relevant to roof and sidewall stability, the mining method, and the support system with geological information to determine a physical ranking ensures that the total system is examined. The Section Performance Rating form is designed to measure how conditions determined by the Section Physical Risk Rating are being addressed. Furthermore, the form also measures compliance with the Support Rules and Strata Control Standards. Both forms can be easily adapted for specific conditions. Should geological discontinuities, for example, represent a major problem in a particular area or for a specific mining method, then the importance of these features may be highlighted as a separate item with its own sub-divisions or by changing the weighting.

In summary, the following are some of the benefits of using the Ingwe Coal Rating forms:

- The rating forms enable quantification of previously subjective observations.
- Different auditors, i.e. shift supervisors, mine overseers and rock engineers, use the same format. This allows meaningful comparisons to be made in individual sections.

- A visit (audit) is structured such that people observe and record all potential hazards. It enables trends to be monitored and forms an integral part of the section management plan.

Ingwe describes the Impact Splitting Tests, Section Performance Rating and Physical Risk Ratings as a system that can be used during the planning stage and assigning appropriate support patterns; for identifying changing conditions while mining; determining the best reaction to those conditions. This system also ensures a better engineering design for roof support in collieries. As is well known the design of roof support for underground coal mines involves a large number of variables. It is, therefore, not possible to develop one design for a support system design, (as one might do in a civil or mechanical engineering environment). The need is to have a number of designs which are able to cope with both the variability and the unknowns associated with the mining environment and its inherent risk. It is, therefore, essential to adopt a risk-based management process in relation to ground control.

Three common risk management models are typically used to deal with a variable risk environment (Galvin 1995):

a) **Compromise Model:** A fixed compromise level of risk management is adopted. The setting of the absolute level is often based on historical evidence, experience and perception of acceptable/unacceptable risk levels. Compromise risk management accepts that an increase in risk will not be catered for in the management plan.

b) **Maximum Risk Model:** This model represents designing all systems to cater for the worst case or highest level of risk. Whilst it caters for all credible increase in risk, it is usually prohibitively complex and/or expensive for the majority of the time. It may be appropriate for a nuclear reactor but is likely to render most coal mines uneconomic.

c) **Variable Step Model:** This recognises that risks change with time, location and/or operation and the controls for the risks should change accordingly. The basis of this process working correctly is timely identification of change in the risk level, triggering a change in the control strategy.

It is not intended, in this report, to pursue the application of risk management techniques. However, it is important to understand their suitability and role in underground mining and ground control management. Of the models discussed above, the third, Variable Step Model, is considered most appropriate for ground control management (Galvin 1995), which can be adopted using Ingwe's systems.

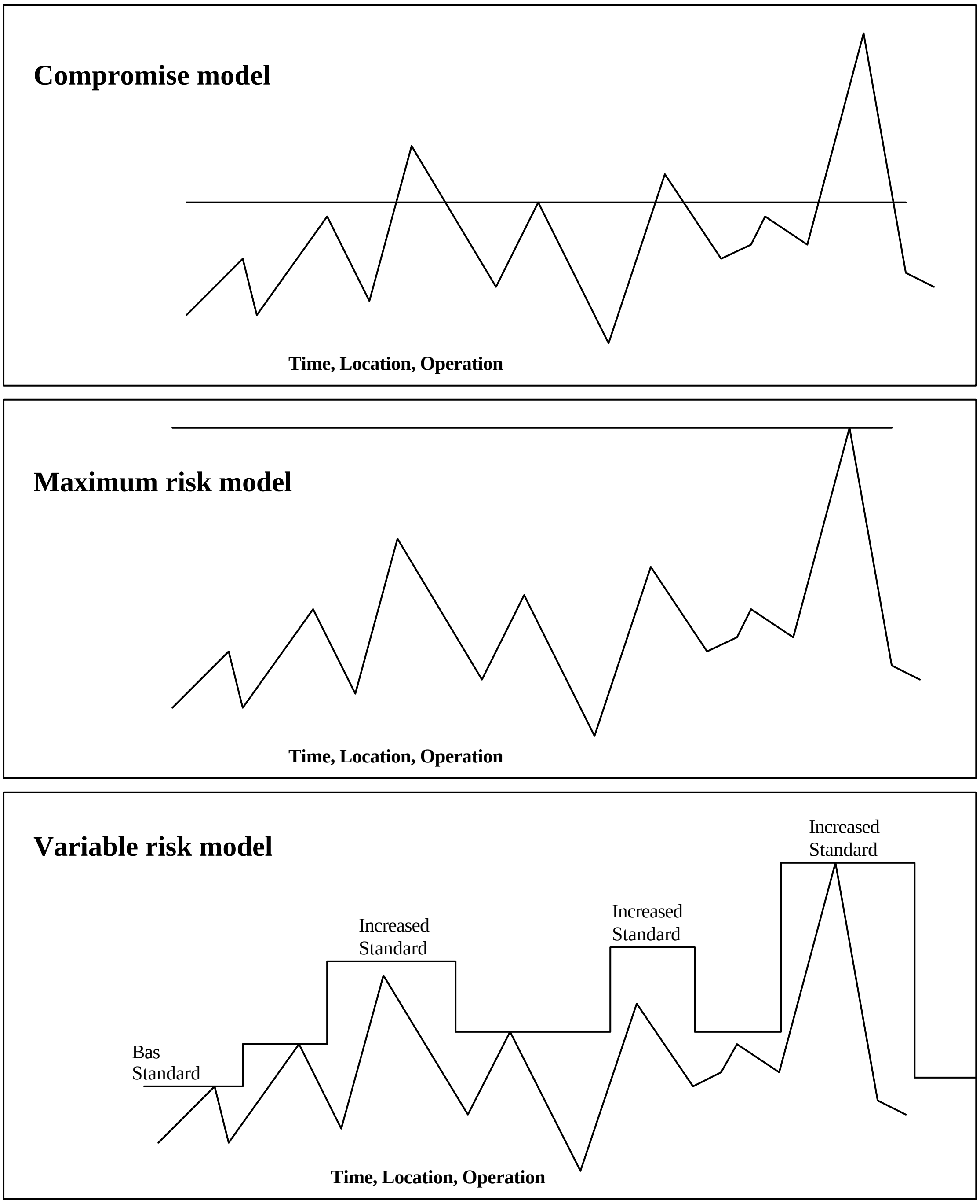

Figure 5 Risk management models (after Galvin, 1995)

5.0 Individual Colliery Systems

A number of hazard rating systems are used by the coal mines in South Africa. Some of these have already been documented but in most cases the systems are individually designed and implemented by the mines themselves. In light of this, it was necessary to investigate different hazard systems used at collieries by conducting visits to coal mines. It was decided that this task would be approached in three stages:

1. Documenting the colliery's hazard rating system;

2. Applying an existing system to test it against the colliery's system.

3. Comparison of results of the existing systems to the colliery's rating system.

One of the difficult parts before the start of this project task was to directly compare the different rating systems used in different collieries. The reason for this is that most of the systems are not documented and as already mentioned; differ from one mine to another. It is for this reason that impact splitting was considered as the most effective system to apply at each mine in order to test it against the mine's system and also to test one mine's results against another mine. Section Performance Rating and Physical Risk Ratings were also conducted underground to test their applicability at each colliery.

The research was conducted at eight collieries in the Witbank and Highveld Coalfields. The collieries that assisted in this investigation were Arnot, Bank, Kriel, New Denmark, Goedehoop, Greenside, Syferfontein and Twistdraai. This section of the report presents the results of the investigations at each colliery.

5.1 Arnot Colliery

At Arnot, a rating system is used to predict the anticipated underground conditions by the geology department during planning. The plan is based on the thickness of the gritstone (coarse grained sandstone), which is a strong stratum that can act as a self-supporting beam and is therefore referred to as the Roof Grit Plan. The grit is divided into 5 thickness categories and classified. Support recommendations are then made as shown in Table 6. The underlying principle in terms of support recommendations is: the thinner the grit, the longer the anchorage length will be. The geologist also makes use of a Point Load Tester, shown in Figure 6 to measure the strength of the rock types in the roof and the floor. This information is mainly used for contamination and floor cut ability purposes more than classification of the grit strength.

Table 6 Roof Grit hazard plan used at Arnot Colliery

Roof Grit	Classification	Typical Support
No Grit	Very Poor	W-straps with cable anchors
< 0.5 m Grit	Poor	1.8 m Full Column Resin, with W-straps for Slips
0.5 m - 1.0 m Grit	Moderate	1.2 m – 1.5 m Full Column Resin
1.0 m to 2.0 m Grit	Good	1.2 m Full Column Resin
> 2.0 m Grit	Very Good	0.9 m Full Column Resin

The roof grit plan is demarcated in different colours representing different roof grit thicknesses and the information is superimposed on to the underground mining plan. At each section, a separate Underground Section Plan is provided and incorporates the anticipated roof conditions from the Roof Grit Plan, geological structures, mining parameters, methane contents and horizontal stress mapping. The underground section plan is approved by the mine surveyor, mine geologist, assistant manager, planning officer and environmental officer to ensure that all parameters are correctly represented on the plan.

Figure 6 Arnot Colliery's Point Load Tester used to measure roof and floor cutability.

A comparative study was conducted on three borehole drill cores, about 100 m from each other on the No 2 Seam. This was done to compare the rating system of the immediate roof used by the mine and the results from impact slitting tests.

Table 7 to Table 9 show the results of impact splitting of the three borehole drill cores. The mine geologists classified borehole drill core ARN 4968 as Roof Grit of 2.19, i.e. “Very Good” roof. From, Table 7 the final rating of 145 from impact splitting also classifies the borehole drill core as “Very Good” roof. The lithological codes used in the table are described as follows:

SF : Shaley sandstone/siltstone
S/F : Interlaminated Sandstone/Shale
S : Sandstone

Figure 7 shows a unit in the roof before impact splitting. The initial fractures are counted before the impact splitting, i.e. one on this case. Figure 8 shows the same unit after Impact Splitting with 3 final fractures.

Figure 7 A fine to medium grained sandstone or “grit”unit beforeImpact Splitting, taken from borehole ARN 4968.

Figure 8 A fine to medium grained sandstone or "grit"unit after Impact Splitting, taken from borehole ARN 4968.

Table 7 Impact Splitting Results at Arnot, No 2 Seam, Borehole ARN 4968

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
46.5	13.2	S/F	1	6	2.2	8.8	4.5	Very Poor
46.7	20.0	S/F	1	2	10.0	30	21.2	Good
47.0	25.2	S	1	4	6.3	22.6	17.6	Moderate
47.3	34.5	S	1	2	17.3	44.5	38.2	Very Good
47.6	24.5	S	1	1	24.5	59	27.4	Very Good
47.8	24.0	S	1	2	12.0	34	11.5	Very Good
48.4	61.6	S	1	2	30.8	71.6	24.5	Very Good
							Final Rating	
							145	**Very Good**

Table 8 and Table 9 show the results from impact splitting of the other two borehole drill cores. The final ratings of borehole drill cores ARN 4974 and ARN 4975 are 155 - "Very Good" roof - and 172 - "Very Good" roof. The mine geologists classified the borehole drill cores as Roof Grit of 1.95 - "Good" roof -and 2.09 -"Very Good" roof respectively. These results show a good correlation between impact splitting tests and the roof grit plan classification. The exact correlation in this case is owed to the fact that the immediate roof was composed of the same rock type for all three borehole drill cores. The exact correlation of the gritstone rating is that the mine geologist was very experienced with the geology of the area being mined. The advantage of impact splitting is that it quantifies the roof condition as opposed to the mere description of the thickness of the gritstone. Moreover, where gritstone is not so obvious (i.e. 50 % sandstone and 50 % shale), then the mine's system may result in errors due to the subjectivity of the geologist.

Table 8 Impact Splitting Results at Arnot, No 2 Seam, Borehole ARN 4974

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
43.6	32.5	S/F	1	5	6.5	23	27.5	Moderate
43.9	36.0	S/F	1	5	7.2	24.4	26.3	Moderate
44.3	41.8	S	1	2	20.9	51.8	47.9	Very Good
44.8	45.3	S	1	2	22.7	55.3	33.6	Very Good
45.2	44.0	S	1	1	44.0	98	19.3	Very Good
							Final Rating	
							155	**Very Good**

Table 9 Impact Splitting Results at Arnot, No 2 Seam, Borehole ARN 4975

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
44.8	15.5	F/S	1	4	3.9	15.5	9.2	Poor
44.9	10.0	F/S	1	3	3.3	13.3	4.8	Poor
45.2	24.2	F/S	1	6	4.0	16.1	12.7	Poor
45.3	11.5	F/S	1	3	3.8	15.3	5.1	Poor
45.8	57.5	S	1	4	14.4	38.8	49.0	Very Good
46.1	21.0	S	1	2	10.5	31.0	9.2	Good
46.3	23.2	S	1	2	11.6	33.2	7.5	Very Good
46.7	37.0	S	1	1	37.0	84.0	74.8	Very Good
							Final Rating	
							172	**Very Good**

Underground visits were conducted to assess' adherence to the underground anticipated physical conditions and mine standards using Ingwe's Physical Rating System and Performance Rating System. These systems were successful in identifying possible hazards but because they originated from a different mine, some parameters could not be recorded owing to different specifications e.g. Arnot's standards of support spacing are not included in the rating systems.

5.2 Bank Colliery

At Bank Colliery, a roof hazard plan only exists for the No 5 Seam. The hazard plan is based mainly on geological structures, roof type above the coal seam (from boreholes), horizontal stresses, and surface structures e.g. pans. Geological structures include dykes and sills with associated burnt coal areas. The roof type above the coal seam is described from exploration boreholes and is classified from the lithological description of the borehole as shown in Table 10 .

Horizontal resistivity measurements are carried out on surface to determine the depth of weathering to assist in mine planning. Weathering allows increased water content which can affect the strength of the roof. Individual boreholes were analysed and the classification of normal, poor and bad roof is done according to the composition of the immediate roof and the overlying strata.

Table 10 Roof hazard classification at Bank Colliery

Classification	Roof type
Normal roof	Shale or siltstone of more than 30 cm thick overlain by sandstone
Poor	Interlaminated, laminated, fissile and micaceous sandstone, siltstone and shale less than 30 cm
Bad roof	Dolerite intrusions, deep weathering of the roof and faults

The hazards identified in the roof hazard plan are included in all section plans issued by the survey department. When mining towards an area that has been demarcated in the roof hazard plan, various procedures come into effect in terms of personnel awareness and roof support.

A comparative study was done on a total of five borehole drill cores, three from the No 5 Seam and two from the No 2 Seam. These borehole cores were mainly drilled for future planning and thus their numbers are the temporal numbers used by the drillers which may differ in future. The results of impact splitting of the five borehole drill cores from No 5 Seam and No 2 Seam are presented from Table 11 to Table 15. The lithological codes used in the table are described as follows:

S : Sandstone
S/F : Sandstone/Shale interlaminated
F : Shale

Figure 9 shows an example of the borehole drill core of the Sandstone/Shale interlaminated roof from the No 5 Seam. In Figure 10 the weaker roof composed mainly of shale is shown.

Figure 9 Borehole drill core from Bank, No 5 Seam

Figure 10 Borehole drill core from Bank, No 2 Seam

The final rating from impact splitting of borehole drill core H45S5 is 79 which is classified as “Moderate” roof. A similar classification of “Moderate” was obtained from the final ratings of drill cores H49S5 and H50S5 i.e. 80 and 76. These results from the three borehole drill cores could not be directly compared to the colliery's rating system due to the fact that the borehole drill cores were done for future planning purposes by a drilling contractor. Furthermore, due to staff changes, the new geologist had difficulty in learning the previous system which resulted in the system not being updated. However, impact splitting results show a good correlation between each other from all three tests, which were taken in close proximity i.e. distance between them is less than 500 m.

Table 11 Impact Splitting Results at Bank, No 5 Seam, Borehole H45S5

Depth (m)	**Thickness** (cm)	**Lithology**	**Initial Fractures**	**Final Fractures**	**Fracture Spacing** (cm)	**Unit Rating**	**Weighted Rating**	**Remarks**
38.0	14.5	S	1	2	7.3	24.5	13.7	Moderate
38.1	12.7	S/F	1	5	2.5	10.2	4.6	Poor
38.3	14.5	S/F	1	5	2.9	11.6	5.6	Poor
38.4	16.4	S/F	1	4	4.1	16.4	8.1	Poor
38.6	12.5	S	1	1	12.5	35.0	11.9	Very Good
38.7	14.2	S	1	1	14.2	38.4	13.3	Very Good
38.8	12.0	S/F	1	2	6.0	22.0	5.8	Moderate
39.0	14.5	S/F	1	2	7.3	24.5	6.8	Moderate
39.1	11.5	S/F	1	2	5.8	21.5	4.1	Moderate
39.2	11.0	S	1	1	11.0	32.0	5.0	Good
							Final Rating	
							79	**Moderate**

From Table 14 and Table 15, the final ratings obtained from the No 2 Seam are 20 and 13 which indicate "Very Poor" roof in each case. The weakness of the shale in this case made it difficult to rate up to 2 m into the roof due to the shale being easily broken by merely picking it up from the borehole drill core box. However, the results show the advantage of impact splitting over the colliery's system in its ability to readily quantify the roof instead of a mere description that can change from one person to another.

Table 12 Impact Splitting Results at Bank, No 5 Seam, Borehole H49S5

Depth (m)	**Thickness (cm)**	**Lithology**	**Initial Fractures**	**Final Fractures**	**Fracture Spacing (cm)**	**Unit Rating**	**Weighted Rating**	**Remarks**
37.9	14.5	S	1	1	14.5	39.0	21.8	Very Good
38.1	17.0	S/F	1	6	2.8	11.3	6.8	Poor
38.2	10.5	S/F	1	3	3.5	14.0	4.8	Poor
38.3	12.0	S/F	1	3	4.0	16.0	5.8	Poor
38.5	17.1	S/F	1	4	4.3	17.1	8.0	Moderate
38.6	11.5	S/F	1	4	2.9	11.5	3.3	Poor
38.8	18.0	S/F	1	4	4.5	18.0	7.0	Moderate
39.0	22.0	S	1	4	5.5	21.0	8.2	Moderate
39.4	35.2	S	1	4	8.8	27.6	11.6	Good
39.5	15.0	S	1	3	5.0	20.0	2.3	Moderate
							Final Rating	
							80	**Moderate**

Table 13 Impact Splitting Results at Bank, No 5 Seam, Borehole H50S5

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
37.6	15.0	S	1	2	7.5	25.0	14.4	Moderate
37.8	14.7	S/F	1	7	2.1	8.4	4.4	Very Poor
37.9	16.0	S/F	1	5	3.2	12.8	6.6	Poor
38.1	12.6	S/F	1	5	2.5	10.1	3.8	Poor
38.2	10.5	S/F	1	2	5.3	20.5	5.9	Moderate
38.3	12.4	S	1	1	12.4	34.8	10.8	Very Good
38.5	20.8	S/F	1	2	10.4	30.8	13.9	Good
38.7	19.5	S/F	1	4	4.9	19.5	6.7	Moderate
38.8	16.4	S/F	1	2	8.2	26.4	6.1	Moderate
39.0	15.4	S/F	1	2	7.7	25.4	3.0	Moderate
							Final Rating	
							76	**Moderate**

Table 14 Impact Splitting Results at Bank, No 2 Seam, Borehole P4S2

Depth	Thickness	Lithology	Initial	Final	Fracture	Unit	Weighted	Remarks

			Fractures	Fractures	Spacing	Rating	Rating	
(m)	(cm)				(cm)			
59.3	12.2	F	1	6	2.0	8.1	3.8	Very Poor
59.4	10.1	F	1	5	2.0	8.1	3.0	Very Poor
59.5	14.5	F	1	8	1.8	7.3	3.6	Very Poor
59.6	12.3	F	1	5	2.5	9.8	3.8	Very Poor
59.8	11.5	F	1	6	1.9	7.7	2.6	Very Poor
59.9	11.2	F	1	6	1.9	7.5	2.2	Very Poor
60.0	13.0	F	1	7	1.9	7.4	0.7	Very Poor
							Final Rating	
							20	**Very Poor**

Table 15 Impact Splitting Results at Bank, No 2 Seam, Borehole P3S2

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
54.6	12.5	F	1	6	2.1	8.3	4.0	Very Poor
54.8	13.0	F	1	6	2.2	8.7	4.1	Very Poor
54.9	10.5	F	1	4	2.6	10.5	3.7	Poor
55.0	12.5	F	1	6	2.1	8.3	0.8	Very Poor
							Final Rating	
							13	**Very Poor**

5.3 Twistdraai Colliery

At Twistdraai colliery, hazard plans are based on a description of the immediate roof, operational guidelines, support and additional support type for special areas. These are grouped into three different classes as shown in Table 16. A support recommendation is given for each class. All this information is transferred to the section plans issued by survey department. The parameters used in the colliery's guidelines are very important for roof control. However, there is no measure of the behaviour of the roof for planning that is based on immediate hangingwall strata composition.

Table 16 Guidelines used in hazard plan at Twistdraai

Guideline	Maximum Roadwidth	Maximum Cutting Distance
Class A	6.0m	9.0m
Class B	6.6m	18.0m
Class C	7.2m	24.0m

A comparative study was done on a total of four borehole drill cores from the No 4 Seam and the results are presented from Table 17 to Table 20. The lithological codes used in the table are described as follows:

S : Sandstone

S/s : Sandstone/Siltstone (Predominantly Sandstone)

The final rating of 159 from impact splitting classifies the borehole drill core as “Very Good” roof. Final ratings of 107 (“Good”), 221 (“Very Good”) and 195 (“Very Good”) were obtained from other three impact splitting tests. The results show a good correlation in quantifying the expected roof conditions. Even though the colliery’s system did not quantify the roof conditions, the geologist’s description of the expected conditions was also a “Good” roof.

Table 17 Impact Splitting Results at Twistdraai, No 4 Seam, Borehole G293584

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
153.0	23.0	S/s	1	8	2.9	11.5	10.0	Poor
153.2	22.0	S/s	1	5	4.4	17.6	12.9	Moderate
153.4	20.0	S/s	1	3	6.7	23.3	13.5	Moderate
154.1	70.0	S	1	2	35.0	80.0	112.0	Very Good
154.4	26.0	S	1	2	13.0	36.0	9.7	Very Good
154.7	36.0	S	1	4	9.0	28.0	4.2	Good
155.0	27.0	S	1	1	27.0	64.0	-3.6	Very Good
							Final Rating	
							159	**Very Good**

Table 18 Impact Splitting Results at Twistdraai, No 4 Seam, Borehole G293585

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
156.2	27.0	S/s	1	6.0	18.0	18.0	18.1	Moderate
156.4	26.0	S/s	1	8.0	13.0	13.0	10.8	Poor
157.3	85.0	S	1	6.0	38.3	38.3	68.1	Very Good
157.6	26.0	S	1	4.0	23.0	23.0	5.9	Moderate
157.7	11.0	S	1	1.0	32.0	32.0	2.1	Good
157.8	10.0	S	1	1.0	30.0	30.0	1.2	Good
158.0	24.0	S	1	3.0	26.0	26.0	0.4	Moderate
							Final Rating	
							107	**Good**

Table 19 Impact Splitting Results at Twistdraai, No 4 Seam, Borehole G293587

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
162.2	14.0	S/s	1	2	7.0	24.0	13.0	Moderate
163.0	79.0	S/s	1	3	26.3	62.7	145.1	Very Good
163.1	10.0	S	1	2	5.0	20.0	4.1	Moderate
163.9	80.0	S	1	3	26.7	63.3	57.8	Very Good
164.0	15.0	S	1	1	15.0	40.0	1.1	Very Good
							Final Rating	
							221	**Very Good**

Table 20 Impact Splitting Results at Twistdraai, No 4 Seam, Borehole G293588

Depth	Thickness	Lithology	Initial	Final	Fracture	Unit	Weighted	Remarks

			Fractures	Fractures	Spacing	Rating	Rating	
(m)	(cm)				(cm)			
163.6	0.1	S	1	1	10.0	30.0	11.7	Good
163.7	0.1	S	1	2	5.0	20.0	7.4	Moderate
163.9	0.1	S	1	2	6.5	23.0	10.4	Moderate
163.4	0.3	S	1	1	29.0	68.0	60.1	Very Good
163.7	0.3	S	1	1	25.0	60.0	37.7	Very Good
163.9	0.2	S	1	1	20.0	50.0	20.6	Very Good
164.1	0.3	S	1	1	27.0	64.0	27.5	Very Good
164.3	0.2	S	1	1	19.0	48.0	10.3	Very Good
164.9	0.6	S	1	3	18.3	46.7	10.0	Very Good
165.0	0.1	S	1	3	4.7	18.7	-0.8	Moderate
							Final Rating	
							195	**Very Good**

5.4 Kriel Colliery

At Kriel colliery, a roof hazard plan has been developed for the No 4 Seam by rating the roof lithology (e.g. Sandstone) and thickness of coal left in the roof. (shown in Figure 11) to form a Composite Roof Hazard Plan with the ratings shown in Table 21 . Due to changes of personnel, the new geologists could not describe how the scores, rating and ranking numbers were obtained. The classifications in Table 21 are coloured differently and demarcated in the composite roof hazard plan together with areas of floor roll and sill transgression.

Table 21 Composite Roof Hazard Plan classification at Kriel Colliery

Score	**Rating**	**Rank**
5	21 - 25	Strong
4	16 - 20	Moderate
3	11 - 15	Weak - Moderate
2	6 - 10	Weak
1	1 - 5	Very Weak

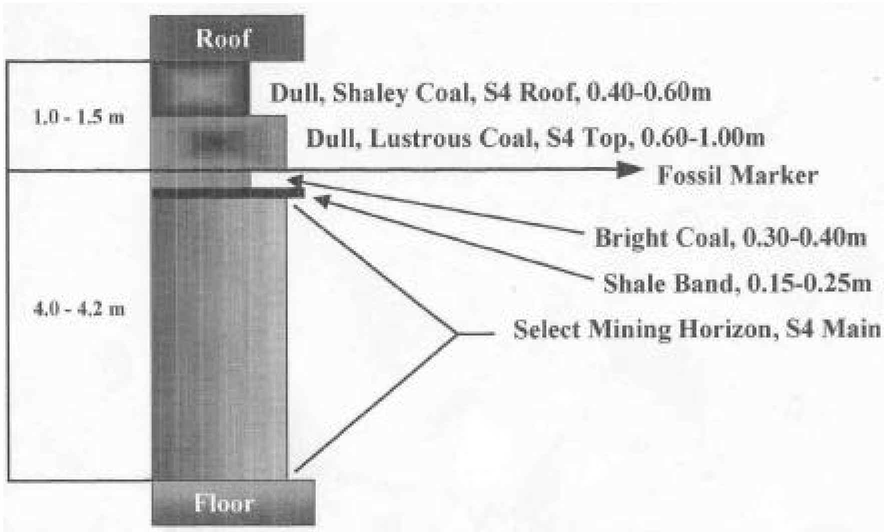

Figure 11 Typical Kriel colliery No 4 Seam roof lithology

During this investigation, there was no drilling taking place at Kriel colliery and thus our impact splitting tests were only limited to the borehole drill cores done by the mine from time to time for problem areas. Only one impact splitting test was conducted on borehole drill core KRL3811 and the results are presented in Table 22 and Table 23. The lithological codes used in the table are described as follows:

S/f : Sandstone with shale bands
C : Coal

When plotted on the composite roof hazard plan, the borehole drill core was on the border of the areas demarcated "Moderate" and "Very Weak". Based on the colliery's system, without underground observations, any of the rankings between Moderate to Very Weak could classify this borehole. However, the impact splitting tests rated it as moderate ("Final rating of 95"). The results presented in Table 22 are before a coal adjustment factor of 1.56 was applied as explained in the literature review.

Table 22 Impact Splitting Results at Kriel, No 4 Seam, Borehole KRL3811

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
44.0	19.5	S/f	1	1	19.5	49.0	36.4	Very Good
44.3	25.5	S/f	1	2	12.8	35.5	30.4	Very Good
44.5	18.5	C	1	7	2.6	10.6	5.7	Poor
44.6	12.5	C	1	5	2.5	10.0	3.3	Very Poor
44.8	20.5	C	1	7	2.9	11.7	5.5	Poor
45.0	20.0	C	1	8	2.5	10.0	3.7	Very Poor
							Final Rating	
							85	**Moderate**

Table 23 Impact Splitting Results after coal adjustment factor, Borehole KRL3811

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
44.0	19.5	S/f	1	1	19.5	49.0	36.4	Very Good
44.3	25.5	S/f	1	2	12.8	35.5	30.4	Very Good
44.5	18.5	C	1	7	2.6	16.5	8.9	Poor
44.6	12.5	C	1	5	2.5	15.6	5.1	Poor
44.8	20.5	C	1	7	2.9	18.3	8.5	Moderate
45.0	20.0	C	1	8	2.5	15.6	5.8	Poor
							Final Rating	
							95	**Moderate**

5.5 New Denmark Colliery

The Roof Hazard Plan has been established to indicate potential hazards that may affect the safety of the employees. The hazards that are identified are:

- Dykes and sills with associated burnt coal areas
- Laminations, partings and shale from surface
- Sudden change in floor gradient
- All areas of poor roof identified from roof sounding
- Excessive bord widths
- All bord widths exceeding 9 m due to over cutting or scaling
- Horizontal stress concentrations and historical roof fall problems

This plan is constantly revised depending on the identification of new hazardous areas. A separate plan is included in all section plans issued by survey department. When mining towards an area that has been demarcated in the hazard plan, various procedures come into effect in terms of personnel awareness and roof support. A comparative study was done on Borehole 321, No 4 Seam to test the mines classification of the immediate roof against the results from Impact Slitting Tests. Table 24 presents the results of rating of the borehole drill core which has a final rating of 107 (i.e. “Good” roof). The geologists also classified the area as good roof on the Roof Hazard Plan. The lithological codes used in the table are described as follows:

S : Sandstone
SF : Shaley sandstone/siltstone

Table 24 Impact Splitting Results at New Denmark, No 4 Seam, Borehole 321

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
229.8	66.0	S	1	7	9.4	28.9	63.6	Good
230.2	36.0	SF	1	10	3.6	14.4	12.0	Poor
230.6	39.0	S	1	3	13.0	36.0	22.0	Very Good
230.8	19.0	S	1	2	9.5	29.0	5.5	Good
230.9	13.0	S	1	1	13.0	36.0	3.1	Very Good
231.0	10.0	S	1	1	10.0	30.0	0.8	Good
							Final Rating	
							107	**Good**

5.6 Syferfontein Colliery

The hazard plan used at this colliery is similar to that of Twistdraai colliery. The plan is also based on a description of the immediate roof, operational guidelines, support type and additional support for special areas. These are grouped into three different classes as shown in Table 25. A support recommendation is given for each class. All this information is transferred to the section plans issued by the survey department. The parameters used in the colliery’s guidelines are very important for roof control. As previously mentioned, there is no measure of the behaviour of the roof for planning that is based on immediate hangingwall strata composition.

Table 25 Guidelines used in hazard plan at Syferfontein colliery

Guideline	**Maximum Roadwidth**	**Maximum Cutting Distance**
Class A	6.0m	9.0m
Class B	6.6m	18.0m
Class C	7.2m	24.0m

A comparative study was done on borehole drill core V118043 from the No 4 Seam and the results are presented in Table 26 and Table 27. The lithological codes used in the table are described as follows:

SF : Shaley Sandstone/siltstone
SC : Sandstone/Coal (Predominantly Sandstone)
C : Coal

Table 26 Impact Splitting Results at Syferfontein, No 4 Seam, Borehole V118043

Depth (m)	Thickness (cm)	Lithology	Initial Fractures	Final Fractures	Fracture Spacing (cm)	Unit Rating	Weighted Rating	Remarks
84.1	15.0	C	1	5	3.0	12.0	6.9	Poor
84.2	10.0	C	1	2	5.0	20.0	7.2	Moderate
84.5	29.0	C	1	4	7.3	24.5	22.8	Moderate
84.8	26.5	C	1	4	6.6	23.3	16.4	Moderate
84.9	10.0	C	1	4	2.5	10.0	2.3	Very Poor
85.0	11.0	SC	1	2	5.5	21.0	4.8	Moderate
85.1	10.0	SF	1	3	3.3	13.3	2.5	Poor
85.3	19.0	SF	1	6	3.2	12.7	3.8	Poor
85.4	13.0	SF	1	2	6.5	23.0	3.8	Moderate
							Final Rating	
							70	**Moderate**

Table 27 Impact Splitting Results, Borehole V118043 after coal adjustment factor

Depth	Thickness	Lithology	Initial	Final	Fracture	Unit	Weighted	Remarks

(m)	(cm)		Fractures	Fractures	Spacing (cm)	Rating	Rating	
84.1	15.0	C	1	5	3.0	18.7	10.8	Moderate
84.2	10.0	C	1	2	5.0	31.2	11.2	Good
84.5	29.0	C	1	4	7.3	38.2	35.6	Very Good
84.8	26.5	C	1	4	6.6	36.3	25.5	Very Good
84.9	10.0	C	1	4	2.5	15.6	3.6	Poor
85.0	11.0	SC	1	2	5.5	21.0	4.8	Moderate
85.1	10.0	SF	1	3	3.3	13.3	2.5	Poor
85.3	19.0	SF	1	6	3.2	12.7	3.8	Poor
85.4	13.0	SF	1	2	6.5	23.0	3.8	Moderate
							Final Rating	
							102	**Good**

5.7 Results from Greenside and Goedehoop Collieries

At Greenside Colliery, Impact Splitting tests could not be done due to the size of the borehole drill cores that could not fit into the impact splitter (i.e. 75 mm). Although the mine had plans to change this in the near future to the 48 mm and 60 mm size, the plans were beyond the time limitations of this particular task. At Goedehoop colliery, the geology department is involved in a new hazard plan system that is expected to be implemented in 2003. It was therefore difficult to get borehole drill cores for impact splitting and to compare that to the new system. However, both collieries had some form of hazard rating which is based on immediate roof geology.

5.8 Application of pro active systems

During the underground visits, experience using the Section Performance Rating and Physical Risk Rating (Oldroyd and Latilla, 1999) showed the need to measure changing conditions and implement effective strategies quickly, in addition to ensuring compliance with the requirements of the code of practice. At each mine visited, both systems were used underground with mine personnel to check whether all the points that were rated included what the mine personnel perceived as important at that particular mine. In almost all the visits, the mine personnel requested a copy of the finished forms as they found them very useful. However, due to their originality, the forms need to be updated according to different mine standards (e.g. difference in systematic support types and spacing). Furthermore, it was experienced that the systems needed someone with a strata control background as most of the ratings are strata control related and constitute a big weighting in the final rating. The following is a summary of the points to note about the underground section rating systems:

- A structured check list ensured that the user observed and recorded all potential hazards. It also ensures that they are attended and re-evaluated again for improvement in the conditions.
- Different inspectors, i.e. shift supervisors, mine overseers and rock engineers, use the same format. This allows meaningful comparisons to be made in individual sections.
- Systems need to be applied by someone who has a strata control understanding.
- The systems could be used on any mine with small modifications to the control instructions (e.g. support types)

5.9 Windows Based Program

A Microsoft Windows based program has been developed in Visual Basic for Microsoft Excel. The program consists of the Impact Splitting calculations, the Section Risk and Section Performance rating forms.

6.0 Conclusions and Recommendations

The purpose of this project was to develop a roof rating system by evaluating and documenting all the existing systems that are used in South Africa and others that have been developed in other countries, and proposing the way forward for the development of a system that could be used universally on South African collieries. The results presented in the previous chapter have shown that, although many collieries have hazard plans, these plans do not readily quantify the mechanistic behaviour of the roof strata, they are mostly descriptive and are subject to different opinions. Furthermore, there is no uniform methodology behind the development of these plans, which makes it difficult for another person to apply them.

It was considered that the CMRR could overcome most problems associated with the application of rock mass classification systems in coal mining. However, due to the fact that the system is based on case histories from the United States, certain modifications would have to be made to the system to cater for the different conditions in South African collieries.

Impact Splitting Test has been found to be the most appropriate system to eliminate human error in core rating. The advantage of impact splitting over the individual colliery‘s geology based rating systems is its ability to readily quantify the roof instead of a mere description that can change from one person to another. Geology based systems have been developed based from experience by mine personnel that certain soft or hard layers in the roof were a major cause of instability. During this study, Impact Splitting has shown a very good correlation with the geology based rating systems. The system can therefore be used during planning for good prediction of conditions ahead of mining. Furthermore, the system requires minimal training time (about half-hour) and therefore does not require skilled personnel.

In conclusion, Impact Splitting Tests, Section Performance Rating and Physical Risk Ratings systems developed in South Africa are considered to be as effective and appropriate for South African conditions. They can distinguish different roof conditions necessary for initial planning and support design. They can also be used for identifying changing conditions while mining and determining the best response to the different conditions.

It must be noted that borehole core based systems like the Impact Splitting are dependent on the quality of the core. Layers that are very weak or have very low cohesion can easily break during the drilling process. Geophysical techniques may therefore be more accurate in such cases for prediction of these layers.

Appendix A: Literature Review

1.0 Literature review

1.1 Rock mass classification

A number of empirical methods have been developed to predict the stability of rock slopes and underground openings in rock and to determine the support requirements of such features. The various approaches used for coal mine roof stability assessment can be categorized as follows:

- Analytical methods
- Geological methods
- Observational methods
- Empirical methods

Analytical approaches generally make use of the fundamental concepts of strength of materials, solid-state mechanics, structural analysis and numerical modelling. Geological approaches basically try to quantify geological structures and other features affecting roof stability. There are various methods used to identify such features, such as core drilling, geological mapping and roof fall mapping. This type of approach has been applied in various coal mines around the world where geological features such as sandstone channels were the main cause of roof falls. Observational methods rely on instrumentation in an attempt to monitor movement and detect measurable instability in mines. Observational methods are the best for back analysis to check the results and predictions of the other methods. In underground mining, the instrumentation is installed to determine stresses, loads, strains and displacements to evaluate the stability of openings. The empirical methods of analysis employ the rock mass classification systems for assessing the stability of underground excavations.

An important issue in rock mass description and characterization is to select parameters of greatest significance for the actual type of design or construction. There is no single parameter or index, which can fully designate the properties of a jointed rock mass. Various parameters have different significance and only if combined can they describe a rock mass satisfactorily (Bieniawski, 1984). In situ testing of rock masses has brought out very clearly the enormous variations that exist in the mechanical behaviour of a rock mass from place to place. According to Lama and Vutukuri (1978) the engineering properties of a rock mass depend far more on the system of geological discontinuities within the rock mass than of the strength of the rock itself.

Further, the strength of a rock mass is often governed by the interlocking bonds of the unit "elements" forming the rock mass. Terzaghi (1946) also concludes that, from an engineering point of view, a knowledge of the type and frequency of the rock discontinuities may be much more important than the types of rock which will be encountered.

Similarly, Piteau (1970) has stressed the importance of distinguishing between the behaviour of the rock and the rock mass, especially for hard rocks. Thus, characterizing a discontinuity system in a way that describes the variability of its geometric parameters constitutes an essential step in dealing with stability problems in discontinuous rock masses (Tsoutrelis et al., 1990). This does not mean that the properties of the intact rock material should be disregarded in the characterization. After all, if discontinuities are widely spaced, or if the intact rock is weak, the properties of the intact rock may strongly influence the gross behaviour of the rock mass. The rock material is also important if the joints are discontinuous. In addition, the rock description will inform the reader about the geology and the type of material at the site.

Although the importance of rock properties in many cases are overridden by discontinuities, it should be remembered that the properties of the rocks very much determine the formation and development of discontinuities. Therefore, an adequate and reliable estimation of the nature of the rock is often a primary requirement. For some engineering or rock mechanics purposes the mechanical characterization of rock material alone can be used, namely for drillability, crushability, aggregates for concrete, asphalt etc. Also, in assessment for the use of fullface boring machines (TBM), rock properties such as compressive strength, hardness, anisotropy are among the more important parameters.

Kirkaldie (1988) mentions a total of 28 parameters present in rock masses which may influence the strength, deformability, permeability or stability behaviour of rock masses: 10 rock material properties, 10 properties of discontinuities and 8 hydro geological properties. Because it is often difficult or impossible in a general characterization to include the many variables in such a complex natural material, it is necessary to develop suitable systems or models in which the complicated reality of the rock mass can be simplified by selecting only a certain number of representative parameters.

Several rock mass classification systems have been developed and evolved over many years. Table 28 shows some major developments in classification systems over the years.

An important observation on the systems developed is that the following parameters are the most frequently by applied in design and classification systems:

- The rock material (rock type, geological name, weathering and strength)

- The degree of jointing
- In situ stresses

Table 28 Some of the classification systems and their main applications

Name of classification system	**Form and Type****	**Main Applications**	**Reference**
The Terzaghi rock load.	Descriptive and behaviouristic form Functional type	Design of steel Support in tunnels	Terzaghi, 1946
Lauffer's stand-up time.	Descriptive form General type	Input in tunnel design	Lauffer, 1958
The new Austrian tunnelling method (NATM)	Descriptive and behaviouristic form Functional type	Excavation and design in incompetent ground	Rabcewicz et al, 1954
The unified classification of soil and rocks	Descriptive form General type	Based on particles and blocks for communication	Deere et al., 1969
Rock classification for rock mechanical purposes	Descriptive form General type	Input in rock mechanics	Patching and Coates, 1968
The rock quality designation RQD	Numerical form General type	Based on core logging and used in other systems	Deere et al., 1967
The rock structure rating RSR	Numerical form Functional type	Design of steel Support in tunnels	Wickham et al.,1972
The Rock mass Rating RMR	Numerical form Functional type	Tunnel, mine and foundation design	Bieniawski, 1973
The Q-system	Numerical form Functional type	Design of support in underground excavations	Barton et al, 1974
The Modified Rock Mass Rating MRMR	Numerical form Functional type	Tunnel, mine and foundation design	Bieniawski, 1977
The unified rock classification system	Descriptive form General type	Based on blocks and used for communication	Williamson, 1980
Basic geotechnical classification (BGD)	Descriptive form General type	For general use	ISRM, 1981
The Geological Strength Index (GSI)	Numerical form Functional type	Design of support in underground excavations	Hoek, 1994
The CMRR	Descriptive and behaviouristic form Functional type	Coal Roof characterization and support design.	Molinda and Mark, 1994
The Rock mass Index (RMi)	Numerical form Functional type	General characterization, design of support.	Palmström, 1995

∗∗Definition of the Form and Type:

Descriptive form: the input to the system is mainly based on descriptions
Numerical form: the input parameters are given numerical ratings according to their character
Behaviouristic form: the input is based on the behaviour of the rock mass in a tunnel
General type: the system is worked out to serve as a general characterization
Functional type: the system is structured for a special application (for example for rock support recommendation)

Also such features as:

- Orientation of main discontinuities or joint set
- Joint condition
- Block shape or jointing pattern
- Faults and weakness zones and
- Excavation features (dimension, orientation, etc) have been considered as important parameters in rock masses.

1.2 Rock mass classification systems in mining

The Geomechanics classifications Rock mass Rating, RMR and the Norwegian Geotechnical Institute (NGI) Q-system are the most commonly used. Both of these systems incorporate Rock Quality Designation RQD and are based on actual case histories. Because of the dynamic nature of the two systems, they have been modified and employed in a study of coal mine roof conditions around the world. The RMR and Q systems have evolved over time to better reflect the perceived influence of various rock mass factors on excavation stability. The introduced modifications have arguably enhanced the applicability of these classification systems, but there are still areas of potential errors and confusion. This section of the literature discusses the evolution of these systems as well as problems associated with estimating the Q, RMR and RQD indexes. Changes associated with the classification systems are of two forms. The first one lies with the actual properties of the systems, the way these are determined on site and the associated weight assigned to each parameter. The second form is the evolution of support recommendations as new methods of reinforcement such as cable bolting and reinforced shotcrete gained acceptance.

1.2.1 Rock quality designation index *(RQD)*

The Rock Quality Designation index *(RQD)* was developed by Deere et al (1967) to provide a quantitative estimate of rock mass quality from drilling core logs. RQD is a modified core recovery index defined as the total length of intact core greater than 100mm in length, divided by the total length of the core run. The resulting value is presented in the form of a percentage as shown in Figure 12. RQD should only be calculated over individual core runs, usually 1.5 metres long.

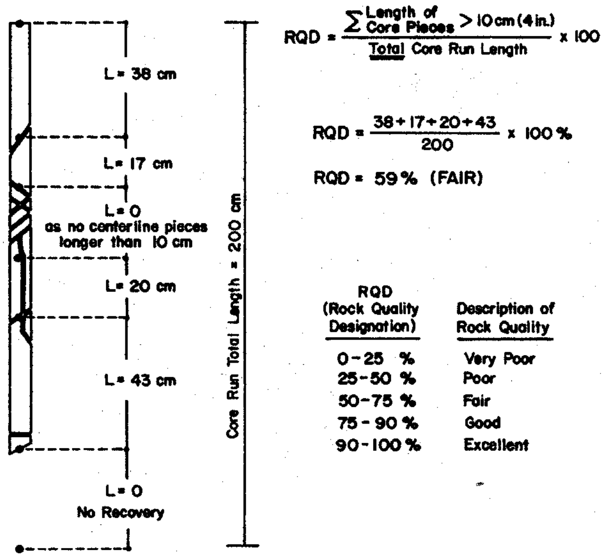

Figure 12 Procedure for determining RQD, after Deere etc (1988)

Intact lengths of core only consider core broken by joints or other naturally occurring discontinuities so drill breaks must be ignored, otherwise the resulting RQD will underestimate the rock mass quality.

In practice, a high RQD value does not always translate to high quality rock. It is possible to log 1.5 metres of intact clay gouge and describe it as having 100% RQD. This may be true based on the original definition of RQD, but is very misleading and gives the impression of competent rock. To avoid this problem, a parameter called 'Handled' RQD (HRQD) was introduced, Robertson (1988). The HRQD is measured in the same way as the RQD, after the core has been firmly handled in an attempt to break the core into smaller fragments. During handling, the core is firmly twisted and bent, but without substantial force or the use of any tools. An estimate of RQD is often needed in areas where line mapping or area mapping has been conducted. In these areas it is not necessary to use core since a better picture of the rock mass can be obtained from line or area mapping. Two methods for estimating RQD are recommended:

(a) For line mapping data, an average joint spacing can be obtained (number of features divided by traverse length). Bieniawski (1989) relying on previous work by Priest and Hudson (1976) has linked average joint spacing to RQD, Figure 12. The ratings in the figure refer to RMR_{89}. It should be noted that the maximum possible RQD based on joint spacing given by Bieniawski actually corresponds to the best-fit relationship proposed by Priest and Hudson. The RQD can be estimated from average joint spacing based on the following equation by Priest and Hudson (1976):

$RQD = 100 \cdot e^{-0.1l}(0.1l + 1)$

where λ = 1/(joint frequency)

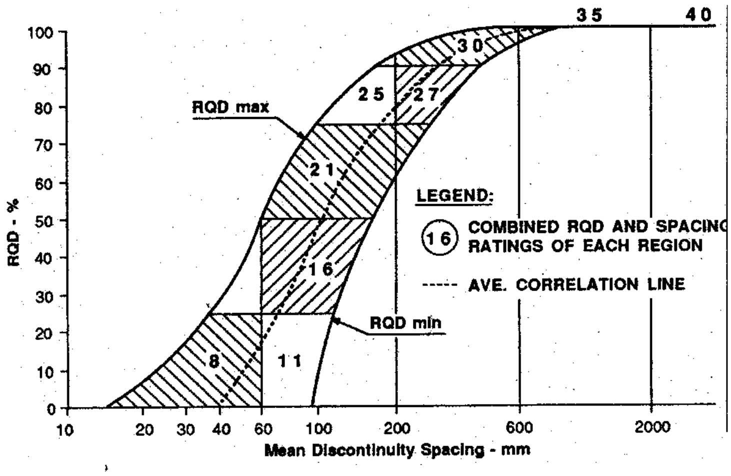

Figure 13 Relationship between discontinuity spacing and RQD after Bieniawski (1989)

Relating joint spacing to average RQD using Figure 13 will likely lead to conservative estimates. Consequently the use of the RQD equation above is probably more appropriate. It should be noted, however, that this relationship is also dependent on the direction of the traverse. For a given average joint spacing there is a significant range in possible RQD values. RQD should not be calculated from line mapping based on the same approach used for core (sum of un-jointed mapped distances greater than 100mm). Line mapping distances are seldom accurate enough to warrant this approach.

(b) For area mapping, a more three-dimensional picture of joint spacing is often available. Palmström (1982) suggested that, when no core is available but discontinuity traces are visible in surface exposures, the *RQD* may be estimated from the number of discontinuities per unit volume. The suggested relationship for clay-free rock masses is:

$RQD = 115 - 3.3J_v$

Where J_v is the sum of the number of joints per unit length for all joint sets known as per volumetric joint count. *RQD* is intended to represent the rock mass quality in situ and is a directional dependent parameter whose values may change significantly depending upon the borehole orientation. *RQD* has been widely used in rock mechanics around the world and has been related to Terzaghi's rock load factors and to rockbolt requirements in tunnels. In the context of rock mass classification systems, the most important use of *RQD* has been in the RMR and Q rock mass classification systems covered later in this literature review. The main drawbacks to RQD are that it is sensitive to the direction of measurement, and it is insensitive to changes of joint spacing, if the spacing is over 1m. The main use of RQD is to provide a warning that the rock mass is probably of low quality.

1.2.2 Rock Structure Rating (RSR)

Rock Structure Rating (RSR) classification is a quantitative method developed by Wickham et al (1972) for describing the quality of a rock mass and selecting appropriate support. Despite this system being the first one to make reference to shotcrete support, most of the case histories used in the development of this system were relatively small tunnels supported by means of steel sets. The significance of the RSR system in the context of rock mass classification is the introduction of the concept of rating each component to arrive at a final rating value of:

$RSR = A + B + C$

where parameter A relates to general assessment of geological structures; parameter B relates to effect of discontinuity patterns and parameter C relates to groundwater inflow and joint condition. Although the RSR classification system is not widely used today, it has played a significant role in the development of the classification schemes that will be discussed in this literature review.

1.2.3 Geomechanics Classification System (RMR)

Bieniawski in (1976) published a rock mass classification called the Geomechanics Classification System or the Rock Mass Rating (RMR) system. Since then, this system has been refined, as more case records have been examined and Bieniawski has made significant changes in the ratings assigned to different parameters. Table 29 summarizes the evolution of the RMR ratings until 1989 as well as the modifications to the weights assigned to each factor.

Table 29 RMR Ratings of Bieniawski over the years

	1973	1974	1975	1976	1989
Rock Strength	10	10	15	15	15
RQD	16	20	20	20	20
Discontinuity Spacing	5				
Separation of joints	5				
Continuity of joints	10	10	10	10	15
Ground Water	10	10	10	15	15
Weathering		15	30	25	30
Condition of joints		15	30	25	30
Strike and Dip orientation		15			
Dip orientation for tunnels	3-15		0-12	0-12	0-12

The components of the system have remained the same and are as follows:

- Uniaxial Compressive Strength (UCS) of rock material
- Rock Quality Designation, RQD
- Spacing of discontinuities
- Condition of discontinuities
- Groundwater conditions
- Orientation of discontinuities

In the application of this system, the rock mass is divided into a number of structural regions and each region is classified separately. The final RMR value is the sum of the ratings of each of the above parameters. A set of guidelines has been published by the author for selection of support in tunnels in rock based on the final value of the RMR. Figure 14 shows the relationship between the RMR value, stand-up times and maximum unsupportable spans. The main factors that have been changed with the RMR system are the weightings given to joint spacing, joint condition and ground water. In assessing both RQD and joint spacing, the frequency of jointing is included twice. In the 1989 version of RMR, the weighting factor for the spacing term was reduced and the influence of both water and joint condition was increased. A further important modification to the RMR was in the definition of different rock mass classes (i.e. very good, good rock, etc.).

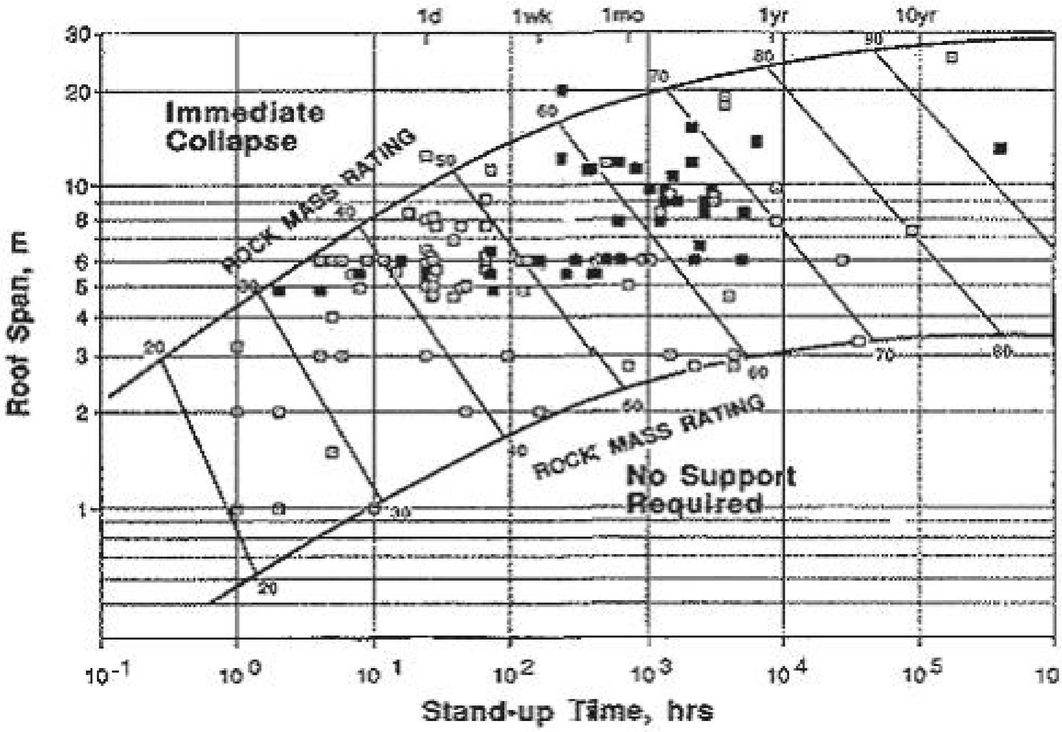

Figure 14 Relationship between Stand-up time, span and RMR classification, after Bieniawski (1989)

In the latest version of the RMR system, the condition of discontinuities was further quantified to produce a less subjective appraisal of discontinuity condition. This brings RMR closer to the Q-system that will be described later which allows the assessment of discontinuity condition by two independent terms, J_r and J_a. Despite efforts to specifically modify the RMR system for mining (Laubscher (1976), Kendorski et al. (1983) etc.) most mines use one of the versions of RMR given in Table 29. Depending on the required sensitivity and the design method used, this might lead to discrepancies. The main advantage of the RMR system is that it is easy to use. Common criticisms are that the system is relatively insensitive to minor variations in rock quality and that the support recommendations appear conservative and have not been revised to reflect new types of reinforcement.

1.2.4 Rock Tunnelling Quality Index, Q

On the basis of an evaluation of a large number of case histories of underground excavations, Barton et al (1974) of the Norwegian Geotechnical Institute proposed a Tunnelling Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirements. The numerical value of the index Q varies on a logarithmic scale from 0.001 to a maximum of 1,000 and is defined by:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

where:

- *RQD* is the Rock Quality Designation
- J_n is the joint set number
- J_r is the joint roughness number
- J_a is the joint alteration number
- J_w is the joint water reduction factor
- *SRF* is the stress reduction factor

It has been suggested that RQD/Jn reflects block size, J_r/J_a reflects friction angle and Jw/SRF reflects effective stress conditions. The main advantage to the Q classification system is that it is relatively sensitive to minor variations in rock properties. Except for a modification to the Stress Reduction Factor (SRF) in 1994, the Q system has remained constant. The descriptions used to assess joint conditions are relatively rigorous and leave less room for subjectivity, compared to other classification systems. One disadvantage of the Q system is that it is relatively difficult for inexperienced users to apply. The J_n term, based on the number of joint sets present in a rock mass, can cause difficulty. Inexperienced users often rely on extensive line mapping to assess the number of joint sets present and can end up finding 4 or more joint sets in an area where jointing is widely spaced. This results in a low estimate of Q. An important asset of the Q system is that the case studies employed for its initial development have been very well documented. The use of the Q system for the design of support has also evolved over time. In particular Barton has introduced a design chart that accounts for the use of fibre-reinforced shotcrete. This has been based on increased experience in tunnelling. For most mining applications, however it is common to rely on the design chart shown in Figure 15.

In mining, the use of the ratio of Excavation Span/Equivalent Support Ratio (ESR) is limited. In open stope design this term is replaced altogether by the hydraulic radius. Alternatively one can assign different ESR values dependent on the type of opening (e.g. 5 for non-entry stopes, 1 for Shaft etc.). As there are limited documented case studies this involves considerable judgment. The next section looks at the weightings given to the different parameters used in the Q and RMR classification systems, and how the two systems are related.

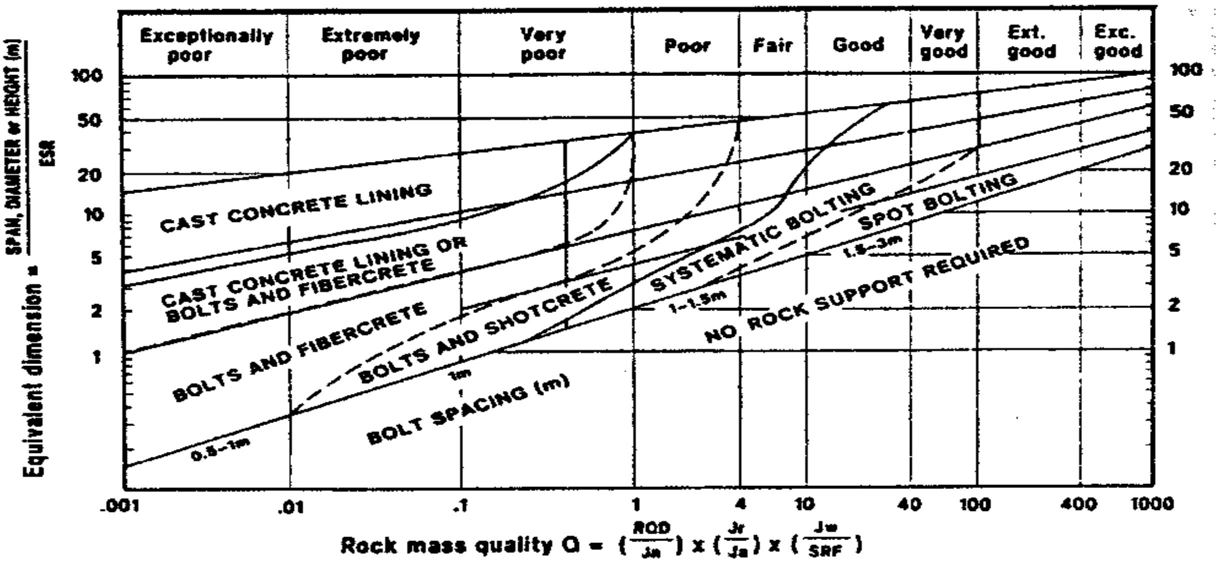

Figure 15 Design and Excavations based on the Q-System, after Barton & Grimstad (1994)

1.2.5 Comparative Rock mass Property Weightings

Both the Q and RMR classification systems are based on a rating of three principal properties of a rock mass. These are the intact rock strength, the frictional properties of discontinuities and the geometry of intact blocks of rock defined by the discontinuities. For the Q system, the intact rock strength is only a factor in the context of the induced stress in the rock as defined by the SRF term. In order to investigate the influence of these parameters, the approximate total range in values for RMR and Q are used as a basis of comparison. Table 30 shows the degree by which the three principal rock mass properties influence the values of the Q and RMR classification.

Table 30 Influence of Basic Rock Mass Properties on Classification, after Milne (1988)

	Q	**RMR_{76}**
Basic Range in Values	0.001 to 1000	8 to 100
Strength as % of the Total Range	19%	16%
Block Size as a % of the Total Range	44%	54%
Discontinuity Friction as a % of the Total Range	39%	27%

Table 30 shows the surprising similarity between the weightings given to the three basic rock mass properties considered. Despite this it should be noted that there is no basis for assuming the two systems should be directly related. The assessment for intact rock strength and stress is significantly different in the two systems. Despite these important differences between the two systems, it is common practice to use the rating from one system to estimate the rating value of the other. The following equation proposed by Bieniawski (1976) is the most popular, linking Q and RMR:

$$RMR = 9\ln Q + 44$$

Referring to Table 31, it is evident that the equation above does not provide a unique correlation between RMR and Q. Depending on the overall intact rock and discontinuity properties and spacing, different relationships between Q and RMR can be expected. Another difference between RMR and Q is evident in the assessment of joint spacing. If three or more joint sets are present and the joints are widely spaced, it is difficult to get the Q system to reflect the competent nature of a rock mass. For widely spaced jointing, the joint set parameter J_n in the Q system appears to unduly reduce the resulting Q value.

Table 31 Correlation between RMR and Q, after Choquet and Hadjigeorgiou (1993)

Correlation	Source	Comments
RMR = 13.5 log Q + 43	New Zealand	Tunnels
RMR = 9 ln Q + 44	Diverse origin	Tunnels
RMR = 12.5 log Q + 55.2	Spain	Tunnels
RMR = 5 ln Q + 60.8	South Africa	Tunnels
RMR = 43.89 - 9.19 ln Q	Spain	Mining Soft rock
RMR = 10.5 ln Q + 41.8	Spain	Mining Soft rock
RMR = 12.11 log Q + 50.81	Canada	Mining Hard rock
RMR = 8.7 ln Q + 38	Canada	Tunnels
RMR = 10 ln Q + 39	Canada	Mining Hard rock

1.2.6 The Mining Rock Mass Rating (MRMR) system

The geomechanics classification (RMR) has been extended for different mining environments. Initially, Laubscher and Taylor (1976) applied this technique in asbestos mines in Africa while Ferguson (1979) used this classification for mining tunnels and haulages.

Laubscher and Taylor (1976) made some essential additional adjustments to the RMR system to cater for diverse mining situations. The fundamental difference was the recognition that in situ rock mass ratings (RMR) had to be adjusted according to the mining environment so that the final ratings (MRMR) could be used for mine design. The adjustment parameters are given as weathering, mining induced stresses, joint orientation and blasting effects.

Adjusted RMR parameters are as follows:

- Blasting damage adjustment (A_B), (0,8-1,0),
- In situ stress and change in stress adjustment (A_S), (0,6-1,2), and,
- Major faults and fractures (S), (0,7-1,0).

Adjusted RMR=RMR*A_B*A_S*S where: maximum value of A_B*A_S*S is 0,5. In recent years, there have been some modifications and improvements to the system. Laubscher (1984) has emphasised a comprehensive system based on a comparison between the in situ rock mass strength and the mining induced stresses. The important key point here is that the application of this system should be applied to an intact rock mass rather than to a broken rock mass.

Laubscher (1990) revealed that it is possible to use the ratings to determine an empirical rock mass strength (RMS) which is adjusted as above to give a design rock mass strength (DRMS). This figure is extremely useful when related to the stress environment and has been used for numerical modelling. Also, these ratings provide good guidelines for mine design purposes.

In addition, Laubscher (1990) indicated in a separate study that the average numbers can be misleading and the weakest zones may determine the response of the whole rock mass. It is, therefore, necessary to identify narrow and weak geological features that are continuous within and beyond the stope or pillar, and rate them separately.

The Intact Rock Strength (IRS), joint/fracture spacing, and joint condition/water must be included into the assessment of geological parameters. The analysis of these is as follows:

Intact rock strength (IRS)

The definition of IRS is a function of uniaxial compressive strength of the rock between fractures and joints. As mentioned before, the results of laboratory testing carried out is usually not representative of the average rockmass values because the samples are invariably the strongest pieces. Undoubtedly, the presence of weak and strong intact rock and deposits of varying mineralisation affects the value of IRS for a defined zone. An average value is assigned to the zone with the knowledge that the weaker rock will have a greater influence on the average value. In such cases, the IRS for the weak and strong zones is determined separately, and expressed as a ratio. Knowing the percentage of weak rock and selecting the appropriate curve which defines the relationship of weak rock IRS expressed as a percentage of strong rock IRS, the average IRS can be estimated as a percentage of strong rock IRS.

A detailed empirical chart to determine an average IRS, where the rock mass contains weak and strong zones, was presented by Laubscher (1990).

Spacing of fractures and joints

Spacing is the distance between all the discontinuities and partings, and does not include cemented features. Two techniques have been developed for the assessment of this parameter (Laubscher, 1990).

a) measuring the rock quality designation (RQD) and joint spacing (JS) separately. The detailed analysis of RQD is presented in the previous sections. In the assessment of joint spacing rating developed by Taylor (1980), the three closest-spaced joints are used to read off the rating. The equations for all lines defined in the chart of assessment of joint-space rating (R), for the different number of joints, are as follows:

one joint set;

R=25*((26,4*log10*x)+45)/100

two joint sets;

R=25*((25,9*log10*xmin)+38)/100*((30*log10*xmax)+28)/100

three joint sets;

R=25*((25,9*log10*xmin)+30)/100*((29,6*log10*xint)+20)/100*((33,3*log10*xmax)+10)/100

where x=spacing*100 measured in metres

b) measuring all the discontinuities and recording these as the fracture frequency per metre. It is possible to have a rating value for a rock mass either from a measurement of all the discontinuities that are intersected by the sampling line or from a borehole log sheet. In comparing these two techniques, it is concluded that the fracture frequency per metre technique is more sensitive than the RQD for a wide range of joint spacings.

Joint condition and water

Joint condition is an assessment of the frictional properties of the joint (not fractures) and is based on expression, surface properties, alteration zones, filling, and water. Originally, the effect of water was catered for in a separate section; however, it was decided that the assessment of joint condition allowing for water inflows would have greater sensitivity.

Adjustments

In order to estimate the value of MRMR, the rock mass value derived by the RMR system is multiplied by an adjustment percentage as already defined in the earlier part of this section.

Weathering

Weathering must be taken into consideration in decisions on the size of an opening and the support design. Its effect is time dependent, and influences the timing of support installation and the rate of mining. The basic three parameters, IRS, RQD or fracture frequency per metre, and joint condition, are affected by weathering. The relation between these parameters could be summarised as below:

- An increasing number of fractures will result in a decreased value of RQD
- Chemical composition changes taking place have a significant effect on the IRS
- Alteration of the wall rock and the joint filling will affect the joint condition

Laubscher (1990) published a table delineating adjustment percentages related to degree of weathering, after a period of exposure of various years.

Joint orientation

Size, shape and orientation of an excavation play a significant role in rock mass behaviour. The attitude of the joints, and whether or not the bases of blocks are exposed, has a significant bearing on the stability of the excavation, and the ratings must be adjusted accordingly. In his adjustment procedure, the attitude of the joints with respect to the vertical axis of the block has the most important role. As gravity is the most significant force to be considered (in shallow mines), the instability of the block depends on the number of joints that dip away from the vertical axis. A modified orientation adjustment applies to the design of pillars or stope sidewalls. The applicability of this rating has been described in detail by Laubscher (1990).

Mining-induced stresses

Laubscher (1990) indicated that the major influences on mining induced stresses, arising as a result of the redistribution of field (regional) stresses, are the geometry and orientation of the excavations. The redistributed stresses that are of interest are major, minor and differences; hence it is essential that the magnitude and ratio of these stresses are known.

a) Major Stress

The maximum principal stress can cause spalling of the wall parallel to its orientation, crushing of pillars, and the deformation and plastic flow of soft zones. The deformation of soft intercalates leads to failure of hard zones at relatively low stress levels. A compressive stress close to perpendicular joints increases the stability of the rock mass and inhibits caving.

b) Minor Stress

The minimum principal stress plays a significant role in the stability of the sides and back of large excavations., the sides of openings, and the major and minor apexes that protect extraction horizons. The removal of a high horizontal stress on a large stope sidewall will result in relaxation of the ground towards the opening.

c) Stress differences

A large difference between major and minor stresses has a significant effect on jointed rock masses resulting in shearing along the joints. The effect increases as the joint density increases (since more joints will be unfavourably orientated) and also as the joint condition ratings decrease.

The factors, which should be considered in the assessment of mining-induced stresses, have been listed by Laubscher (1990).

Blasting effects

Some adjustment would be required since the blasting operation forms new fractures and loosens the rock mass resulting in some movement on joints. These factors vary the rating between 80 % and 100 % depending on the technique of opening, i.e. boring, conventional blasting, etc.

Strength of the rock mass

Laubscher (1990) emphasised that the strength of the rock mass cannot be higher than the corrected average IRS of a zone and large specimens, i.e. the “rock mass”, will be equal to 80 % of the value obtained from laboratory tests on small specimens, if there is no joint.

The following empirical formula is adopted to calculate the RMS.

$$RMS = \frac{(A-B)}{80} xCx \frac{80}{100}$$

where:

A= total rating of rock mass,

B= IRS rating,

C= IRS.

Design strength of the rock mass

The definition of the design rock mass strength (DRMS) is given as the strength of the unconfined rock mass in a specific mining environment. Laubscher (1990) explained that the size of the excavation will affect the zone surrounding the excavation in terms of instability conditions. Adjustments, which relate to that mining environment, are applied to the RMS to give the DRMS. As the DRMS is in MPa, it can be related to the mining-induced stresses. Therefore, the adjustments are those for weathering, orientation, and blasting. It is concluded that the value of DRMS, which can be related to the total stresses, is the unconfined compressive strength of the rock mass.

1.2.7 The Modified Basic RMR (MBR) system

The MBR system was developed by the US Bureau of Mines to examine how a ground classification approach could be used fruitfully in planning support for drifts in caving mines. It follows closely the RMR system and incorporates some ideas of Laubscher. Bieniawski (1984) explained that key differences lie in the arrangement of the initial rating terms and the adjustment rating. It is still possible to use very preliminary geotechnical information from drill holes. The MBR is also a multi-stage adjustment and its rating is the result of the initial stage and is the simple sum of the raw ratings.

Bieniawski (1984) pointed out that the MBR is an indicator of rock mass competence without regard to the type of opening constructed in it. There are three stages to the determination of the MBR value:

The first step in using the MBR system is the collection of representative data on intact rock strength, discontinuity density and conditions, and groundwater conditions as defined in the

RMR. The only difference in the application of RMR is the importance ratio of each parameter to be considered in the evaluation. Tables and figures for ratings and adjustments are presented by Kendorski *et al.,* (1983).

The second stage is to consider the "development adjustments". The objective in the development adjustment is to initially stabilise the opening during development so that permanent support may use its full capacity to resist the abutment loading increment. The third stage deals with the additional deformations due to abutment loadings. After the extraction ratio is computed, the blasting damage, termed as severe, moderate, slight, or none, is assessed, and the induced stress adjustment is determined. The horizontal (Φ_h) and vertical (Φ_v) components of the stress field must be computed or estimated and the adjustment can then be done for the appropriate effective extraction ratio, depth and stress field. The next adjustment is for fracture orientations. The third stages are development and production adjustments. The multiplication of these three adjustments, having a value between 0,45 and 1,0, with the MBR will give the AMBR value.

The final stage is to consider the role of structural geology and mining geometry as defined as "production adjustments". The basic parameters to be considered in the development adjustments are as follows:

Development adjustments

- blasting (AB, 0,8-1,0),
- induced stresses (AS, 0,8-1,2), and
- fracture orientation (AO, 0,7-1,0).

Production adjustments

- major structures (S, 0,7-1,1),
- distance to cave line (DC, 0,8-1,2), and
- block/panel size (PS, 1,0-1,3).

Adjusted MBR (AMBR) = $MBR*A_B*A_S*A_O$

Final mining MBR (FMBR) = AMBR*DC*PS*S

Disadvantages

The MBR is an adapted version of prior work of Bieniawski and Laubscher with a modification for caving, which is radically different from driving a tunnel. In developing the modifications and adaptations for the MBR system, all data were collected for horizontal drifts in mines. Thus, the MBR system is not necessarily valid for non-horizontal workings (inclines, raises, and shafts) or for other mining methods.

1.2.8 The Rock Mass Index (RMi) system

The RMi system proposed by Palmstrom (1996) is based on defined inherent parameters of the rock mass and is obtained by combining the compressive strength of intact rock and a jointing parameter. The jointing parameter represents the main jointing features, namely block volume (or density of joints), joint roughness, joint alteration, and joint size. Quantitatively, the RMi can be expressed as:

$$RMi = s_c\ JP.$$

where:

σ_c= the uniaxial compressive strength of intact rock measured on 50 mm samples;

JP= the jointing parameter, which is a reduction factor representing the block size and the condition of its faces as represented by their friction properties and the size of the joints. The influence of JP has been found by using calibrations from test results. Because of problems in obtaining compression test results on rock masses at a scale similar to that of typical rock works, it was possible to find appropriate data from only eight large-scale tests and one back-analysis. These have been used to arrive at the following mathematical expression as:

$$JP = 0.2\sqrt{jC}\,VB^{D}$$

where VB is given in m^3, and $D = 0\ 37\ jC^{0.2}$

The joint condition factor is expressed as:

$$JC = jL(\,jR\,/\,jA)$$

where:

jL = factors for joint length,

jR = joint wall roughness and,

jA = continuity and joint surface alteration

The factors jR and jA are similar to the joint roughness number (Jr) and the joint alteration number (Ja) respectively in the Q-system. The joint size and continuity factor (jL) has been introduced in the RMi system to represent the scale effect of the joints.

The RMi is numerical and therefore differs from earlier general classifications of rock masses, which are mainly descriptive or qualitative. Palmstrom (1996) discusses three applications of the RMi. These include;

- determination of the constants in the Hoek-Brown failure criterion for rock masses,
- assessment of stability and rock support in underground excavations, and
- quantification of the classification applied in the New Austrian Tunnelling Method (NATM).

Some of the benefits and limitations of the RMi system are explained by Palmstrom (1996) as follows:

- The RMi will significantly improve the use of geological input data, mainly through its systematic use of well-defined parameters in which the three-dimensional character of rock masses is represented by the block volume.

- The RMi can easily be used for rough estimates when only limited information on the ground conditions is available, for example, in the early stages of a project, where rough estimates are sufficient.

- The RMi is well suited for comparisons and exchange of knowledge between different locations. In this way it can improve communication between those involved in rock engineering and design.

- The RMi offers a platform suitable for engineering judgement. RMi is a general parameter that characterises the inherent strength of rock masses, and may be applied in engineering design. Because the RMi is composed of real block volumes and common joint parameters for rock masses, it is easy to relate it to field conditions. This is important in applying engineering judgement.

- The RMi system covers a wide spectrum of rock mass variation, and therefore has possibilities for wider applications than other rock mass classification and characterization systems used today.

Any attempt to mathematically express the variable structures and properties of jointed rock masses in a general failure criterion may result in complex expressions. By restricting the RMi to uniaxial compressive strength alone, it has been possible to arrive at the relatively simple expressions in the above equations. Because simplicity has been preferred in the structure as well as in the selection of parameters in RMi, such an index may result in inaccuracy and limitations. The main limitations relate to:

- The range and types of rock masses covered by the RMi. Both the intact rock material and the joints exhibit great directional variations in composition and structure, resulting in a large range in compositions and properties of rock masses. It is not possible to characterise all of these combinations in one single number. Nevertheless, the RMi probably characterises a wider range of materials than most classification systems.

- The accuracy in the expression of RMi. The value of the jointing parameter (JP) is calibrated from a few large-scale compression tests. Both the evaluation of the various factors (jR, jA and Vb) used in obtaining JP and the size of the samples tested, which in some of the cases had a small number of blocks, may be sources of error in the expression for JP. Therefore, the value of RMi found may be approximate. In some cases, however, errors in the various parameters may partly neutralise each other.

- The effect of combining parameters that vary in range. The parameters used to calculate the RMi generally will express a certain range of values. As with any classification system, combining such variables may cause errors. In some cases, the result is that the RMi may be inaccurate in its characterisation of the strength of the complex and varied assemblage of materials and defects that constitute a particular rock mass. For these reasons, the RMi may best be considered as a relative index in its characterisation of the rock mass strength.

1.3 Rock mass classification for coal mining

The original RMR classification system was based on case histories drawn from civil engineering. As a result, the system was regarded by the mining industry as conservative and several changes were made to make the system more applicable to mining applications. A Modified Rock Mass Rating (MRMR) was presented by Laubscher (1977, 1984), Laubscher and Taylor (1976) and Laubscher and Page (1990).

Rock mass classification systems, specifically Q-system and RMR have been widely applied for tunnels in civil engineering and in mining. The first attempt to apply these systems in coal mining was made by Djahanguiri (1978) in rock mechanics study of underground mining of a thick coal seam in Wyoming. Subsequent to that, Bieniawski, Rafia and Newman (1979) used the Rock Structure Rating (RSR), Q-system and RMR to study the coal mine roof conditions affected by mining with an automated extraction system (AES) developed by the National Mine Service Co. with the U.S. Bureau of Mine (1979).

The authors concluded that the Geomechanics classification systems were useful in the assessment of roof conditions but needed to be modified for application in coal mine roof rating as follows:

- Reduce the number of classification parameters
- Inclusion of effects of field stresses and
- Correlating roof support requirements with rock mass classes.

The three problems experienced by Bieniawski et al (1979) that for evaluating coal mine roof conditions using the geomechanics classification systems were:

a) The lack of geotechnical data from in-mine engineering, geological mapping and core logging needed as input parameters for the classification systems.
b) The need for more information on the relationship between the stand-up time and the span of coal mine roof, so that the chart in Figure 14 could be applied with greater confidence to U.S. coal mining.
c) The need for simplification of classification systems for coal mining by modifying them as already discussed above.

Several other coal mine roof classification systems have been proposed. Hylbert (1978) devised a roof characterization scheme based entirely on geology, i.e., lithologic and structural features. Four roof categories were defined based on whether discontinuities were present or a sandstone channel was in contact with a shale zone. Instances or roof falls were then related to each category. In many instances, stastical approaches have been used in coal roof classification to link geological parameters with categories of mine roof such as good, moderate, bad and fallen. Support recommendations were then made for each category of roof. Most of these methods have not been well documented and are limited to application to areas where they originated.

The geomechanics classification systems have also been applied in Indian Coal Mines for support selection. Sinha & Venkateswarlu (1985) modified the Bieniawski's RMR system to a new CMRS classification applicable to Indian mines. The non-applicability of the Q-system and RMR approaches for tabular mining is related to the parameters used in the evaluation and to those excluded. The Q-system gives more importance to joint attributes whereas in coal measure strata bedding planes play a greater role. The Stress Reduction Factor, SRF values were also found not relevant to Indian coal mining stress fields.

The CMRS was developed by taking into consideration the mining conditions in Indian coal mines. Statistical analyses of geological parameters of the roofs were done to determine their relative importance. The five parameters selected were the RDQ, rock strength, groundwater seepage, rock weatherability and structural features. Whilst the other parameters are similar to that of Bieniawski's RMR, emphasis was given on weatherability and structural features which were the main factors contributing in roof problems in India. The ratings were given separately for each rock type in the immediate roof bed. The RMR for the whole roof was obtained by weighting the RMR of each bed with its thickness to get the combined RMR. This system was applied to 47 coal mines in India and the support recommendations have been documented to be successful.

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Appendix B: Section Performance Rating And Section Physical Rating Forms

SECTION PERFORMANCE RATING FORM **FORM : MF02 Rev 1.4 (16/03/2001)**

Mine : ____________________ Seam : ______________ Section : ___________________ Date : ________________________ Panel : ____________________

Visited by : __________________ Bolt type : ___________________ Mining method : _________________________ Roofbolter Type:_____________________

FOG Stats (Previous 6 months)	Fatal	-10	Lost time Injury	-7	High Potential Incident (eg. Major fall of supported ground)	-5	Property Damage	-3	None	5	
Temporary support (includes onboard temporary support)	Not being used	-5	Not used due to incorrect length/breakdown	-3	Not observed / no clear evidence of incorrect use	3	Not used but system such that no exposure to unsupported roof	4	Correctly spaced and properly set against roof (on board)	5	
Slip Support	Very poor (more than 2 slips not detected or bolted)	-8	Poor – 1 or 2 slips not detected or bolted	-5	All slips identified but bolt length incorrect / slips not plotted on miner's plan	-3	No slips or slips well bolted but not marked	4	Good. All slips correctly supported	5	
Horizon control	Roof control poor at pillar corners, small brows not trimmed down or bolted	-5	Very poor – numerous (>4) brows created	-3	Poor – 4 or less brows created Frozen coal left against roof	-1	No brows created but thin shale band left in immediate roof	1	Horizon control good – no brows created	3	
Brow Support	Very poor – 3 or more brows not supported	-5	Poor – 1 or 2 brows not supported	-3	Low step down middle of roadway / split due to dipping seam – bolts on wrong side.	-1	Brows supported but spacing or bolt length incorrect	1	Good- brows well supported with correct length bolts	2	
Cutting direction (% cut off-line)	> 12 %	-5	9.1 – 12 %	-2	6.1 – 9 %	0	3 – 6 %	4	< 3 %	5	
Road width control	Average > 1m	-5	Average 0.5 to 1m	-3	Average 0.2 to 0.5m	-1	Averaged < 0.2m	0	On design width	5	
Intersection cutting *	Diagonal >3m deviation	-5	Diagonal 2 to 3m deviation	-2	Diagonal 1 to 2m deviation	0	Diagonal <1m deviation	2	Correct length	5	
*** Intersection corner to corner distances for various bord widths**			7 = 9.8m		6.5 = 9.1m		6.0 = 8.4m		5.5 = 7.7m		
Cutting beyond unsupported slips	CM seen cutting beyond slips	-5	CM inferred to have cut beyond slip	-3	Not observed / No clear evidence of CM cutting beyond slips			4	CM clearly not cutting beyond slips	5	
Support of intersections	Seen to be cutting into unsupported intersections	-5	Pre-supported intersection not holed properly	0	All intersections supported before being cut	1	All intersections supported before being cut and extra bolts in enlarged intersections / over-cut areas			3	
Bolt installation quality	Crimps not broken/plates loose/broken bolts not replaced on 10% or more of installations	-10	Crimps not broken/plates loose/broken bolts not replaced < 10% of installations	-2	Protruding thread length variable	0	Most defective or damaged bolts replaced and 95% or more of installations to standard			5	
Roofbolting Controls (may be added)	Torque correctly set on machine	2	Hole length control correct	2	Adequate drill bit size control	2	Correct resin and not time expired			2	
Support - General	Incorrect support type or length being used	-8	Incorrect colour coding	-3	Extra bolts in hollow roof areas	2	Additional support always installed (e.g. "W" straps on slip/timber in sliped areas)			3	
Maximum cutting distance ahead of support	> 2m more than stipulated maximum	-5	< 2m more than stipulated maximum	0	Not observed / No clear evidence of cutting too far	4	Clearly not exceeding stipulated maximum			5	
People working under unsupported roof	People seen to be working under unsupported roof/clear evidence of people working under unsupported roof	-10	People inferred to be working under unsupported roof	-5	Not observed / no clear evidence that people are working under unsupported roof	4	Clear evidence that people are not working under unsupported roof			5	
General (these ratings may be added)	Non adherence to approved mining layout	-10	Tell tales correctly installed and monitored	5	Sufficient sounding sticks and pinch bars of correct length	2	Lines and bolt positions definitely not marked ahead of support			2	
Roof inspections and sounding	Inspections and sounding clearly not done to standard	-10	Inspection and sounding suspected to be sub-standard	-5	Inspection and sounding done throughout face areas only	3	Roof and sidewalls inspected and sounded well according to standard			5	
Barricades and last row of bolt indicators	Not installed	-5	> 2 Missing	-3	2 or less missing, rest well installed	2	Unsafe / unsupported roof well demarcated			5	

Mine : ________________________ Seam : ____________ Section : ________________ Date : ______________________________

Bolt spacing (systematic)	Average spacing more (>0.1m) than stipulated	-5	Average spacing within 0.1m of stipulated	5	Average spacing less (< 0.1m) than stipulated	6			
Bolt spacing (discontinuities)	Average spacing more (>0.1m) than stipulated	-5	Average spacing within 0.1m of stipulated	5	Average spacing less (< 0.1m) than stipulated	6			
Barring	Numerous loose pieces	-2	Occasional loose pieces	-1	Conditions such that no barring needed	1	General standard of barring very good	3	
Mining height control	Incorrect roof horizon being mined. Coal beam thickness not being monitored	-5	Not applicable, mining full seam to sandstone roof or coal roof in excess of 1m thick.	4	Correct roof horizon being maintained and where applicable, coal beam thickness properly recorded	5			
Training	No essential personnel have attended SC course	-5	Few essential personnel have attended SC course	-3	Most essential personnel have attended SC course	3	All essential personnel have attended SC course	5	
Waiting place (These ratings may be added)	Safely sited	2	Recommended height report available	2	Correct support rule on notice board	2	Section management plan up to date	2	
POINTS IN TABLE ARE A GUIDE ONLY **ADJUST AS REQUIRED, e.g., IF NOT SATISFIED WITH PERFORMANCE REDUCE POINT ACCORDINGLY** **ABBREVIATIONS : N/A = NOT APPLICABLE N/M = NOT MEASURED N/C = NOT CHECKED**			**PREVIOUS RATING : ____________**		**Total (max 100)**				

Section Performance Rating : Very poor <30% Poor 30-45% Moderate 46-60% Good 61-75% Very good 76-90% Excellent.>90%

Recommendations / Action Plan : ____

Recommendations made by:____________________________ **Designation:**______________________ **Date:**______________

Received by:_____________________________________ **Designation:**______________________ **Date:**______________

Mine : ______________________________ Section : ________________________ Date : _____________

COMMENTS :

1. Spot check/s on spacing between rows of bolts (systematic support): -____

   ____Avg:__________

2. Spot check/s on spacing between bolts along slips :____Avg:__________

3. Spot check/s on slip frequency : ____Avg:__________

4. Spot check/s on intersection diagonal (split turn off) : ____

5. Bord width spot check/s : ____

6. Tell – Tale Readings (If applicable): ____

7. Spot checks on Sounding Stick and Pinch Bars:____

8. Spot checks on bolt installation quality: ____

9. Spot check on pillar corner brows:____

10. Check on SC course attendance : S/Boss ☐ Miner ☐ R/Bolt 1 ☐

    R/Bolt 2 ☐ CM 1 ☐ CM 2 ☐

11. Main recommendations from previous visit : ---____

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<u>BORD AND PILLAR</u> <u>SECTION PHYSICAL RISK RATING FORM</u> **FORM : MF01 Rev1.4 (16/03/2001)**

Mine : ______________________ Seam : ______________________________ Section : ___________________________ Panel : _____________________________

Date : _________________________ Completed By ___________________________ Signed: _____________________________

Suggested risk categories : **<30 Special Area** **31 to 60 Moderate Area** **>60 Good Area**

													Points
Section Performance Rating	< 30%	-5	30 – 45%	-2	46 – 60%	0	61 – 75%	1	76 – 90%	3	> 90%	5	
Mining Method	Stone Development / B10	-5	Top Coaling	-2	Bottom Coaling	3	Longwall Development	5	CM bord and pillar	8	Conv. Bord & pillar	10	
Roof Lithology	Shale/Mudstone	0	Shale / Sandstone interlaminated	2	Coal and shale	4	< 0.5m Coal	6	> 0.5m Coal	8	Massive sandstone	10	
Roof Conditions	Very poor (falls frequently some time after being exposed)	-5	Poor (falls occasionally some time after being exposed)	-2	Variable (conditions change unpredictably from area to area)	0	Moderate (occasional slabbing after being exposed)	3	Fairly Good (occasional hollow areas identified)	8	Good (no hollow or false roof areas detected)	10	
Pillar Conditions	Highly jointed <2m spacing / burnt coal	-5	Moderately jointed >2m spacing	-2	Frequent spalling	0	Occasional spalling	1	Occasional pillar corner deterioration	3	Good pillar conditions	5	
Discontinuities	Within 10m of a dyke or fault	-5	Slips less than 5m apart	-2	Slips 5 to 10m apart or within 50m of a dyke or fault	0	Slips 10 to 20m apart	3	Slips more than 20m apart	8	No slips	10	
Influence of Discontinuities	Severe (falls > 1m common some time after being exposed)	-10	Very strong (falls > 1m high common immediately after being exposed)	-7	Strong (falls < 1m high some time after being exposed)	0	Moderate (falls <1m high immediately after being exposed)	3	Slight (small falls < 0.2m immediately after being exposed)	7	Negligible (no discernable influence on roof stability) (no slips =10)	9	
Other Geological Conditions	Severe weathering	0	Slight weathering or false roof	1	Wet roof or coal floor	2	Floor / roof rolls or dipping seam	3	Floating stone / soft floor	2	Damp roof No geological problems (5)	3	
Mining Height	> 5m	-5	4.5 to 5m	-2	4.0 to 4.5m	2	3.0 to 4m	3	2.0m to 3m	4	< 2.m	5	
Systematic Support	None	0	Intersection only	2	2.5m grid	6	2m grid	8	1.5m grid	10	1m grid	12	
Systematic Support Type	Point Anchor 0.6m	-4	Point Anchor 0.9m	-2	Point Anchor 1.2m	2	Point Anchor ≥ 1.5m	5	M16 Full Column Resin ≥ 1.5m	6	M20 Full Column Resin ≥ 1.5m	7	
General (Must be added)	Signs of horizontal stress roof failures e.g. guttering / large falls without slips	-10	Shallow workings <40m Experimental or trial roof support / panel layout	-5	Within 50m of surface water course or dam/pan or above low SF workings	0	General back-bye conditions (Good 5 / Fair 0 / poor -5).	5 0 -5	Tell tales or other roof performance monitoring method not required	4	Tell tales being systematically installed	5	
Advance per month (m)	> 1800	2	1600 to 1800	4	1400 to 1600	6	1000 to 1400	8	600 to 1000	10	< 600	12	
					PREVIOUS RATING :		__________________				**TOTAL**		

Received by : ____ **Designation** : _______________________ **Date** : __________________________

Summary of changed conditions / Remarks : ____

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Remedial Action/s to be taken : ____