

DEVELOPMENT AND APPLICATION OF THE COAL MINE ROOF RATING (CMRR)

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ABSTRACT

The Coal Mine Roof Rating (CMRR) was developed 10 years ago to fill the gap between geologic characterization and engineering design. It combines many years of geologic studies in underground coal mines with worldwide experience with rock mass classification systems. Like other classification systems, the CMRR begins with the premise that the structural competence of mine roof rock is determined mainly by the discontinuities that weaken the rock fabric. However, the CMRR is specifically designed for bedded coal measure rock. Since its introduction, the CMRR has been incorporated into many aspects of mine planning, including longwall pillar design, roof support selection, feasibility studies, extended-cut evaluation, and others. It has also become truly international, with involvement in mine designs and funded research projects in South Africa, Canada, and Australia. This paper discusses the sources used in developing the CMRR, describes the CMRR data collection and calculation procedures, and briefly presents a number of practical mining applications in which the CMRR has played a prominent role.

INTRODUCTION

Roof falls continue to be one of the greatest hazards faced by underground coal miners. In 2006, there were 7 fatalities from roof falls and nearly 500 rock fall injuries in the United States. In addition, more than 1,300 major roof collapses were reported to the Mine Safety and Health Administration. These roof falls can threaten miners, damage equipment, disrupt ventilation, and block critical emergency escape routes.

One reason roof falls have proven so difficult to eradicate is that mines are not built of manmade materials like steel or concrete, but rather of rock, just as nature made it. The structural integrity of a coal mine roof is greatly affected by natural weaknesses, including bedding planes, fractures, and small faults. The engineering properties of rock cannot be specified in advance with adequate precision and can vary widely from mine to mine and even within individual mines.

Engineers require quantitative data on the strength of rock masses for design. Traditional geologic reports contain valuable descriptive information, but few engineering properties. Laboratory tests, on the other hand, are inadequate because the strength of a small specimen is only indirectly related to the strength of the rock mass.

ROCK MASS CLASSIFICATION

Rock mass classification schemes were developed to address these concerns. The most widely known systems, including Deere's Rock Quality Designation (RQD), Bieniawski's Rock Mass Rating (RMR), and Barton's Q-system, have been used extensively throughout the world [Deere and Miller 1966; Bieniawski 1973; Barton et al. 1974]. Rock mass classifications have been successful [Bieniawski 1988] because they—

- Provide a methodology for characterizing rock mass strength using simple measurements;
- Allow geologic information to be converted into quantitative engineering data;
- Enable better communication between geologists and engineers; and
- Make it possible to compare ground control experiences between sites, even when the geologic conditions are very different.

This last point highlights an extremely powerful application of rock mass classification systems, which is their use in empirical design methods. Empirical designs are based on mine experience—on the real-world successes and failures of actual ground control designs. By collecting a large number of case histories into a single database and subjecting them to statistical analysis, reliable and robust guidelines for design can be developed. A key advantage of empirical techniques is that it is not necessary to obtain a complete understanding of the mechanics to arrive at a reasonable solution. Rock mass classifications play an essential role in empirical design because they allow the overwhelming variety of geologic variables to be reduced to a single, meaningful, and repeatable parameter.

Unfortunately, the standard rock mass classification systems are not readily applicable to coal mining because—

- They tend to focus on the properties of joints, whereas bedding is generally the most significant discontinuity affecting coal mine roof.

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- They rate just one rock unit at a time, whereas coal mine roof often consists of several layers bound together by roof bolts.

In addition, the dimensions and stability requirements of tunnels are often very different from those of mines.

COAL MINE GROUND CONTROL

The Coal Mine Roof Rating (CMRR) was developed more than 10 years ago to meet the needs of mine planners for a simple, repeatable, and meaningful classification system [Molinda and Mark 1994]. It employs the familiar format of Bieniawski's RMR, summing the individual ratings to obtain a final CMRR on a 0–100 scale. It is also designed so that the CMRR/unsupported span/standup time relationship is roughly comparable to the one determined for the RMR.

In determining the specific rock mass attributes and weightings to use, the CMRR built upon the rich vein of experience with coal mine ground control during the past 30 years. These sources can be divided into two groups. The first are papers describing specific geologic features, such as faults, clay veins, sandstone channels, kettle-bottoms, and others. A summary of this work was reported by Molinda [2003]. The second group, which includes efforts to generalize results for specific mines, regions, or countries, was more directly relevant to the development of the CMRR. In effect, these papers describe rock mass classification systems, although most are qualitative rather than quantitative. Table 1 provides a list of the coal mine roof classification systems consulted in the development of the CMRR, along with the significant geologic factors that they identified as being important to ground control. Following is a discussion of some of these factors and the issues involved with incorporating them into the CMRR.

Bedding: Bedding was the factor that was most consistently cited as causing roof problems in coal mines. The two most common examples were weak laminations in shale and thinly interbedded sandstone and shale (stackrock). In both examples, it is not just that the bedding planes are closely spaced, but also that the bedding surfaces are very weak. Indeed, several authors included "massive shale" as one of the more stable rock types [Moebs and Ferm 1982].

The issue of bedding (or grain alignment) is further complicated because some shales may appear massive, particularly to untrained eyes, but are actually highly laminated. The CMRR therefore emphasizes testing of the rock material to determine bedding plane strength even when the bedding is not visible. The approach is similar to that proposed by Buddery and Oldroyd [1992] and used successfully in South African coal mines.

Strong Bed: A problem unique to horizontally layered sedimentary rocks is that the roof structure often consists of several layers with different engineering characteristics. In developing the CMRR, two key questions had to be answered:

1. How far up into the roof should the evaluation extend?
2. How should the different layers be combined into a single rating? Should they be averaged together, or should the weakest or strongest layers be given precedence?

Few answers were available in the literature. Buddery and Oldroyd [1992] evaluated the first 2 m of roof, but weighted the layers nearest the roof line more heavily. Several authors seemed to imply that a weak layer can be very important by their emphasis on rider coal seams [Karmis and Kane 1984; Stingelin et al. 1979; Miller 1984].

The CMRR started with the insight that the *roof bolt length* largely determines the thickness of the mine roof structure. All coal mine roof in the United States must be bolted, and the bolts bind the different layers together. Generally, the layers above the bolts have much less influence than the units that are penetrated by the roof bolts.

Moreover, experience in many U.S. coalfields has clearly established that roof stability is greatly enhanced when the roof bolts anchor in a strong layer. This effect is most evident in the Illinois Basin, where roof falls are almost unknown when the bolts anchor in a limestone that is at least 0.6 m thick [Kester and Chugh 1980; Schaffer 1985; Damberger et al. 1980]. The strong bed effect has also been recognized in Alabama [Martin et al. 1988] and central Appalachia [Hylbert 1978]. Indeed, even the Code of Federal Regulations implies a strong bed effect when it states at 30 CFR 75.204(f)(1) that "roof bolts that provide support by suspending the roof from overlying stronger strata shall be long enough to anchor at least 12 inches into the stronger strata."

Moisture Sensitivity: Moisture sensitivity is another factor that has been ignored by traditional rock mass classification systems, but is extremely important to coal mine ground control. Two roof shales may initially have very similar properties, but one may be essentially impervious to moisture while the other completely disintegrates when exposed to groundwater or even humid mine air (Figure 1).

The presence of moisture-sensitive mudrocks may be just a nuisance, or it can severely damage the roof by reducing rock strength, generating swelling pressures, or compromising support effectiveness by causing sloughing around roof bolt plates. While the Slake Durability Test (SDT) has been widely used to evaluate moisture sensitivity [Hoek 1977], the CMRR employs a modified version of the simpler immersion test described by Sickler [1986].

Table 1.—Rock mass classification systems for coal mines

Author	Location	Rock strength	Bedding	Strong bed	Moisture sensitivity	Sandstone channels	Slickensides	Minor structures	No. of beds	Ground-water	Lineaments/faults	Rider coals	Joints
Buddery and Oldroyd [1992]; Lattila et al. [2002]	South Africa		x		x								
Damberger et al. [1980]	Illinois Basin		x	x				x					
Ealy et al. [1979]	S. West Virginia		x								x		
Hybert [1978]	E. Kentucky		x	x		x	x						
Karnis and Kane [1984]	Virginia		x			x		x	x			x	x
Kester and Chugh [1980]	Illinois Basin	x	x	x		x			x				
Martin et al. [1988]	Alabama	x		x				x			x		
Milici et al. [1982]	Virginia		x			x	x						
Miller [1984]	Central Appalachia	x					x	x		x	x		
Moebs and Ferm [1982]; Ferm et al. [1978]	Virginia, S. West Virginia	x	x			x		x					
Moebs and Stateham [1985]	United States	x	x		x	x	x	x					
Newman and Bieniawski [1986]	United States	x	x		x		x			x			x
Schaffer [1985]	Illinois Basin			x				x					
Sinha and Venkateswarlu [1986]; Venkateswarlu et al. [1989]	India	x	x		x	x	x	x	x	x			
Stingelin et al. [1979]	N. Appalachia	x							x			x	
Zhou et al. [1988]	United States		x	x	x					x			x
Coal Mine Roof Rating (CMRR)	United States	x	x	x	x		x		x	x		x	x



Figure 1.—Effect of water on a moisture-sensitive shale.

Slickensides and Other Discontinuities: While bedding is generally the most significant weakness in the fabric of coal measure rocks, often some other type of discontinuity is present. Slickensides, which are small-scale (<2-m) fault surfaces of highly aligned clay minerals distinguished by glassy, grooved surfaces, are frequently cited as greatly reducing the competence of coal measure mudrocks (for example, see Moebis and Stateham [1985]). Jointing is encountered in Virginia [Karmis and Kane 1984] and occasionally elsewhere. In sandstones, coal spars and crossbeds can be significant. The original RMR rates only the most significant discontinuity set and largely ignores the others. The CMRR contains a “multiple discontinuity adjustment” so that the weakening effects of slickensides and other discontinuities can be explicitly included.

Large-scale Features: Large-scale features include sandstone channel margins, lineaments, faults, and some medium-scale features such as seam rolls and clay veins. These types of features are not included directly in the CMRR, although in some cases one CMRR value can be determined for “typical conditions” and another for “fracture zones” or “sandstone channel margin areas,” and these can then be plotted on hazard maps. However, the CMRR is not designed to rate conditions impacted by a major throughgoing discontinuity such as a fault. Such features normally require specially designed support systems.

DATA COLLECTION AND CALCULATION OF THE CMRR

The data required for the CMRR can be determined either from underground exposures, such as roof falls and overcasts, or from exploratory drill core. In either case, the main parameters measured are the—

- *Uniaxial compressive strength* (UCS) of the intact rock;
- *Intensity (spacing and persistence)* of bedding and other discontinuities;
- *Shear strength (cohesion and roughness)* of bedding and other discontinuities;
- *Moisture sensitivity* of the rock; and
- Presence of a *strong bed* in the bolted interval.

Other secondary factors include the *number of layers*, the presence of *groundwater*, and *surcharge* from overlying weak beds.

The CMRR is calculated in a two-step process. First, the mine roof is divided into *structural units*, and Unit Ratings are determined for each. A structural unit generally contains one lithologic layer, but several rock layers may be lumped together if their engineering properties are similar. In the second step, the CMRR is determined by averaging all the Unit Ratings within the bolted interval (with the contribution of each unit weighted by its thickness) and applying appropriate adjustment factors. This second step is the same regardless of whether the Unit Ratings were from data collected underground or from core. Figure 2 illustrates the process.

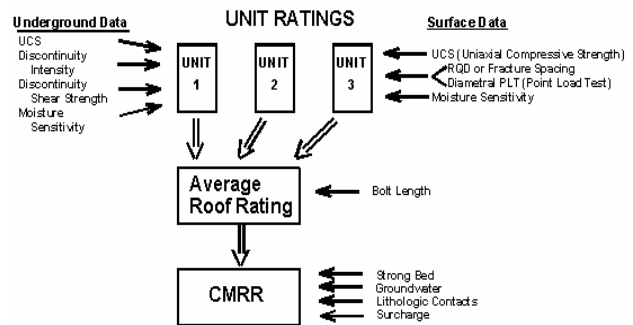


Figure 2.—Flowchart for the CMRR.

The procedures for gathering data and calculating the CMRR from underground exposures have remained essentially unchanged since they were first proposed in 1993. The underground data sheet is shown in Figure 3. Procedures to determine Unit Ratings from drill core have now been streamlined and updated based on new research [Mark et al. 2002]. Calculating the CMRR has been greatly simplified by the development of a CMRR computer program that can be obtained free of charge.

The sections below discuss each of the input parameters used in the CMRR.

Uniaxial Compressive Strength (UCS)

The UCS of the rock material influences roof strength in several ways. First, it determines the ease with which new fracturing (as opposed to movement along preexisting discontinuities) will take place. Second, the compressive strength of the rock is a factor in the shear strength of discontinuities. Approximately one-third of the CMRR is determined by the compressive strength rating, which is approximately twice the weight given to the UCS in the original RMR.

Laboratory testing is generally considered the standard method of determining the UCS. Unfortunately, laboratory tests are expensive because the samples must be carefully prepared. The variability in the results is also high, with the standard deviation typically about one-third of the mean for coal measure rocks [Rusnak and Mark 2000].

As an alternative, the CMRR recommends the point load test (PLT) for drill core. The PLT has been accepted in geotechnical practice for nearly 30 years [Hoek 1977]. An advantage of the PLT is that numerous tests can be performed because the procedures are simple and inexpensive. The apparatus is also inexpensive and portable. The International Society for Rock Mechanics has developed standard procedures for testing and data reduction [ISRM 1985].

Another advantage of the PLT is that both *diametral* and *axial* tests can be performed on core. In a diametral test, the load is applied parallel to bedding (Figure 4). The diametral test is therefore an indirect measure of the lateral strength, or bedding plane shear strength, and is further discussed later.

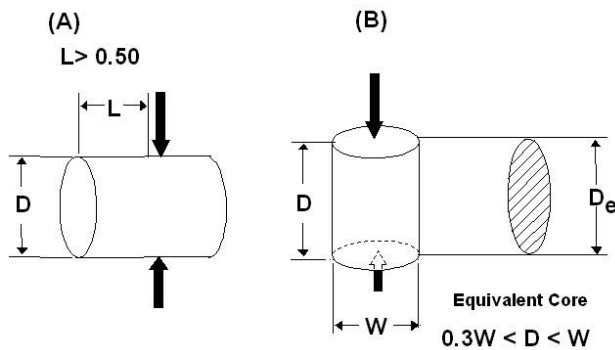


Figure 4.—Diametral and axial point load tests.

When the axial PLT is used to estimate the UCS, the Point Load Index (I_{s50}) is converted using the following equation:

$$UCS = K (I_{s50}) \quad (1)$$

where K is the conversion factor. A comprehensive study involving more than 10,000 tests of coal measure rocks from six states [Rusnak and Mark 2000] found that $K=21$

fit the data well for the entire range of rock types and geographic regions (Figure 5). The study also found that the variability of the PLT measurements, as measured by the standard deviation, was no greater than for UCS tests. The UCS rating scale used in the CMRR program is shown in Figure 6.

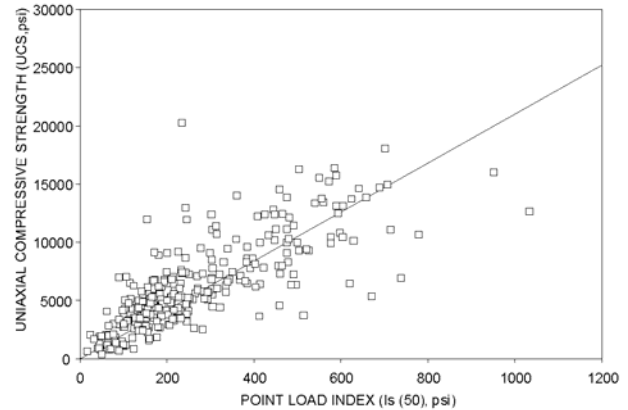


Figure 5.—Relationship between axial PLT and UCS test for shale (Rusnak and Mark [2000]).

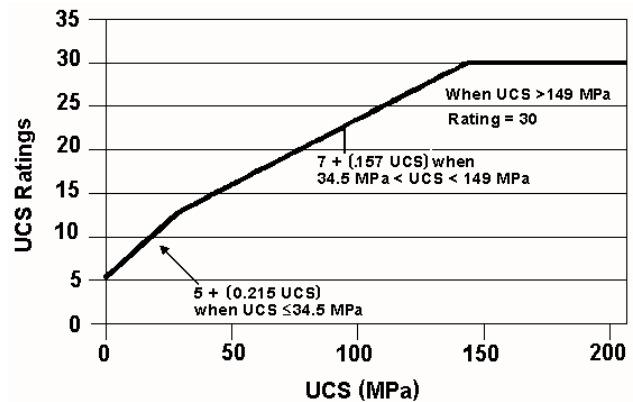


Figure 6.—CMRR rating scale for UCS tests.

Underground, the CMRR employs an indentation test proposed by Williamson [1984] to estimate UCS. The exposed rock face is struck with the round end of a ball peen hammer, and the resulting characteristic impact reaction is compared to the drawings shown on the left side of Figure 3. It is the *nature* of the reaction (indentation), not its magnitude, that is important.

A study was conducted to compare the UCS ratings derived from the Ball Peen Test with the PLT. In 17 of the 23 sites studied (or 74% of the cases), the difference between the two measurements was 4 points or less (Figure 7). The analysis resulted in slight changes to the Williamson rock strength classes, as shown in Table 2.

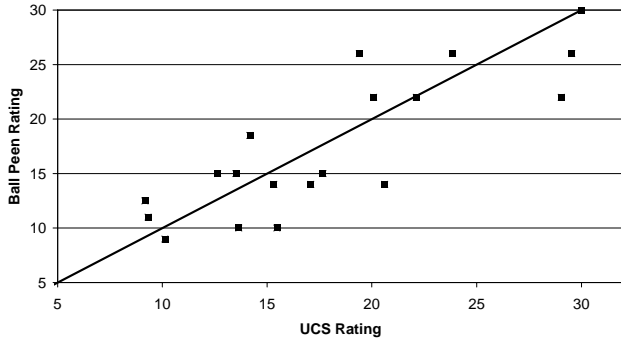


Figure 7.—Comparison between UCS and Ball Peen Tests.

Table 2.—Approximate UCS ranges from Ball Peen Hammer Tests

Ball peen hammer class	Williamson UCS range (MPa)	CMRR UCS range (MPa)	CMRR rating
Molds	<7	<14	5
Craters	7–21	14–35	10
Dents	21–56	35–70	15
Pits	56–105	70–120	22
Rebounds	>105	>120	30

Discontinuity Intensity

Intensity is determined by the *spacing* between bedding planes or other discontinuities and the *persistence*, or extent, of each individual discontinuity. The more closely spaced a set of discontinuities, the greater the weakening effect on the rock mass. Persistence is more important for discontinuities that are widely spaced. Like UCS, intensity accounts for about one-third of the total CMRR.

Underground, both spacing and persistence can be measured directly using the standard methods for rock mass characterization [ISRM 1982]. Table 3 shows the Bedding/Discontinuity Rating Scale for underground data. The matrix shows what point value is added for each combination of spacing and persistence of discontinuities.

Table 3.—Bedding/discontinuity intensity rating table for underground data

Persistence	Spacing				
	>1.8 m	0.6–1.8 m	0.2–0.6 m	60–200 mm	<60 mm
0–1 m	35	30	24	17	9
1–3 m	32	27	21	15	9
>1 m	30	25	20	13	9

Most standard geotechnical core logging procedures include some measure of the natural breaks in the core. The two most commonly employed are the *fracture spacing* and the *RQD*. Fracture spacing is easily determined by counting the core breaks in a particular unit and

then dividing by the thickness of the unit. The RQD is obtained by dividing combined length of core pieces that are greater than 4 in long by the full length of the core run.

Both measures have their advocates in the geotechnical community. Priest and Hudson [1976] suggested that the two can be related by the following formula:

$$RQD = 100 e^{-0.1L} (0.1L+1) \quad (2)$$

where L = number of discontinuities per meter.

As input, the CMRR uses either the RQD or the fracture spacing. When the fracture spacing is greater than about 1 ft, the RQD is not very sensitive, so the fracture spacing is used directly. At the other extreme, when the core is highly broken or lost, the RQD seems to be the better measure. Either measure may be used in the intermediate range.

The program uses the equations shown in Figure 8 to calculate the Discontinuity Spacing Rating (DSR) of core from RQD or the fracture spacing. The equations were derived from the original CMRR rating tables. The minimum value of the DSR is 20; the maximum is 48 (see Figure 8).

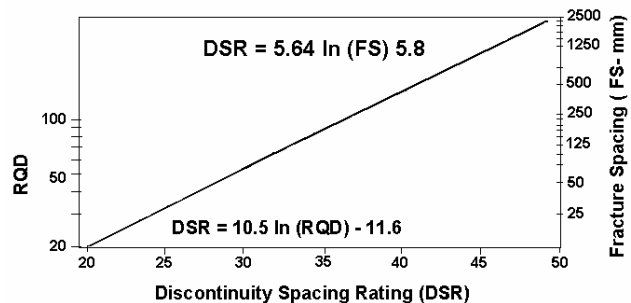


Figure 8.—CMRR rating scale for fracture spacing or RQD.

Shear Strength of Discontinuities

Bedding plane shear strength is a critical parameter for coal mine ground control because the most severe loading applied to coal mine roof is normally lateral, caused by horizontal stress [Mark and Barczak 2000]. Molinda and Mark [1996] found that the lateral strength of some shales are just one-sixth of their axial strength.

Underground, the cohesion of bedding surfaces is evaluated by using a 9-cm mason chisel and a hammer to split hand samples of rock. Weaker, less cohesive surfaces require fewer chisel blows to split (see Figure 3). Cohesion can also be estimated by observing the nature of the fractured wall of a roof fall. If the wall “stairsteps,” with most of the roof failure occurring along bedding, then the cohesion is probably low. On the other hand, if most of the

failure surfaces cut across bedding, then the strength of the bedding is most likely equal to or greater than that of the intact rock. Slickensided surfaces are already planes of failure and receive the minimum rating.

The roughness along a discontinuity surface is the other component of the surface's shear strength. In the CMRR, roughness of a surface is estimated visually and classified as *jagged*, *wavy*, or *planar*, using the system proposed by Barton et al. [1974]. This measure is to be applied on a scale that ranges from hand sample size to several feet across a fall exposure. The CMRR assumes that roughness significantly affects shear strength only when cohesion is in the middle range (see Table 4).

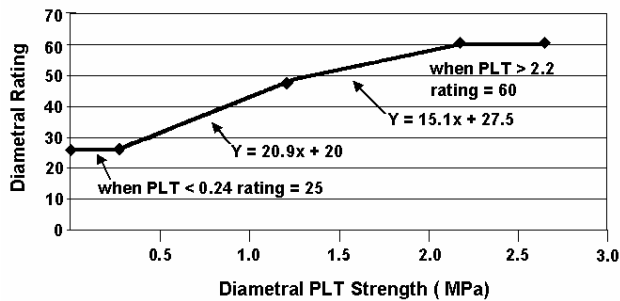


Figure 9.—CMRR rating scale for diametral point load tests.

Table 4.—Bedding/discontinuity shear strength rating table for underground data¹

Roughness	COHESION			
	Strong	Moderate	Weak	Slickensided
Jagged	35	29	24	10
Wavy	35	27	20	10
Planar.....	35	25	16	10

¹If unit has no bedding/discontinuities, then apply test to the intact rock.

When drill core is available, strength testing can be conducted. The diametral PLT is a convenient index test that provides a substitute for bedding plane shear testing. Because the precise relationship between bedding plane shear strength and the PLT is not known, and since it seems unlikely that the same K-factor used to convert the axial test to the UCS would apply, the CMRR uses the Point Load Index (IS_{50}) directly. The diametral PLT rating values were derived from the original CMRR tables and the data presented by Molinda and Mark [1996] and are shown in Figure 9.

If the diametral test results show that the rock fabric or laminations are low-strength, it would be illogical to give the rock high marks for discontinuity spacing. In fact, both the fracture spacing and the RQD also actually measure the strength of discontinuities as well as their spacing, because strong discontinuities might withstand the rigors of the

IMMERSION TEST			
Mine: _____		Date: _____	
Unit No.: _____		Tester: _____	
Sample Description (Lithology, bedding, etc.): _____			
Immersion		Breakability	
Observation	Rating	Observation	Rating
Appearance of Water			
Clear = 0	—	No Change 0	
Misty = -1		Small Change -2	
Cloudy = -3		Large Change -6	
Talus Formation		Breakability Index _____	
None = 0			
Minor = -1			
Major = -3	—		
Cracking of Sample			
None = 0			
Minor/Random = -1			
Major/Preferred Orientation = -3	—		
Specimen Breakdown = -9			
Total Immersion Index _____			
Procedure for Immersion Test			
<ol style="list-style-type: none"> 1. Select sample(s) - ~ hand sized 2. Test for hand breakability. 3. Rinse specimen (to remove surface dirt, dust, etc.) 4. Immerse in water for 1 hour 5. Observe and rate water appearance, talus formation, and cracking of sample 6. <i>Sum Rating for Immersion Index.</i> 7. Retest for hand breakability. 8. <i>Determine Breakability Index</i> 9. The final Immersion Test Index is the greater of the Breakability Index or the Immersion Index. 			

Figure 10.—Data sheet for the Immersion Test.

drilling process while weak ones break apart. Therefore, the *discontinuity rating* is the *lower* of the Diametral PLT Rating or the Discontinuity Spacing Rating.

Moisture Sensitivity Deduction

In the CMRR, the maximum deduction for moisture sensitivity is 15 points. The data sheet for the Immersion Test is shown in Figure 10. If Immersion Test results are not available, moisture sensitivity can sometimes be estimated visually in underground exposures.

Usually, some time is required for contact with humid mine air to affect rock strength. In short-term applications, therefore, it may not be appropriate to apply the moisture sensitivity deduction. The CMRR program reports both the Unit Rating and the CMRR *with* and *without* the moisture sensitivity deduction.

Research was conducted to explore the relationship between the Slake Durability Test (SDT) and the Immersion Test. In the SDT, 10 lumps of rock, each weighing about 0.1 lb, are oven-dried, weighed, and then rotated through a water bath for 10 min. The repeated wetting and drying, together with the mild abrasion that takes place during the test, causes moisture-sensitive rocks to break down. The Slake Durability Index is the final dry weight of the sample expressed as a percentage of the original dry weight [Hoek 1977].

To compare the two tests, rock samples were collected underground from a variety of mine settings, carefully wrapped to maintain in situ moisture content, and tested in the laboratory. A total of 96 tests were run on 16 distinct rock types from 9 mines. The results are shown in Figure 11. From the testing conducted to date, there is a good correlation between the two tests for the “not sensitive” and “slightly sensitive” classes. The correlation is less reliable for distinguishing “moderately sensitive” rocks from “severely sensitive” rocks. Table 5 indicates how the results from either test can be used for input to the CMRR.

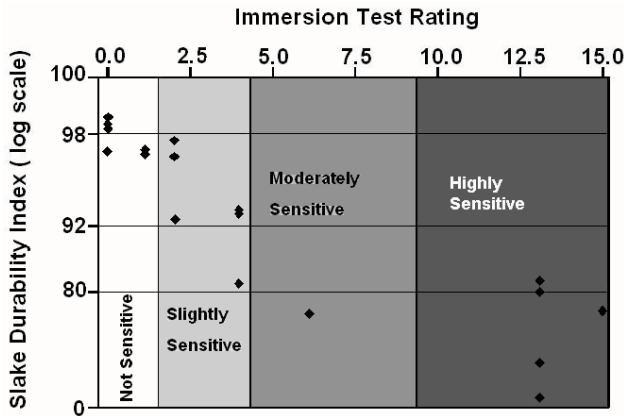


Figure 11.—Comparison of the Slake Durability and Immersion Tests.

Table 5.—Moisture sensitivity classes and ratings from both Immersion and Slake Durability Tests¹

Moisture sensitivity class	Rating adjustment	Immersion Index	Slake Durability Index
Not sensitive.....	0	0–1	100–98
Slightly sensitive.....	–3	2–4	98–92
Moderately sensitive..	–7	5–9	92–80
Severely sensitive.....	–15	>9	<80

¹Apply rating adjustment to Unit Rating only when unit forms the immediate roof or if water is leaking through the bolted interval.

Calculation of the Unit Rating

When using underground data, the equation for calculating the Unit Rating is—

$$\begin{aligned} \text{Unit Rating} = & \text{UCS Rating} + \text{Discontinuity Intensity} \\ & \text{Rating} + \text{Discontinuity Shear Strength Rating} \\ & + \text{Multiple Discontinuity Adjustment} \\ & + \text{Moisture Sensitivity Deduction} \end{aligned}$$

For drill core data, the equation is even simpler:

$$\begin{aligned} \text{Unit Rating} = & \text{UCS Rating} + \text{Discontinuity Rating} \\ & + \text{Moisture Sensitivity Deduction} \end{aligned}$$

Thickness-weighted Average Roof Rating

The next step in calculating the CMRR is to determine the thickness-weighted average of the Unit Ratings of all the units within the bolted interval. For example, assume that the roof consists of three units (from top down):

- 2-m sandstone, Unit Rating = 60
- 0.8-m siltstone, Unit Rating = 50
- 0.4-m shale, Unit Rating = 40

If the length of the roof bolts is 1.8 m, then the thickness-weighted average (RRW) is:

$$\text{RRW} = \frac{[(0.4 * 40) + (0.8 * 50) + (0.6 * 60)]}{1.8} = 51.1 \quad (3)$$

Note that even though the uppermost layer was 2 m thick, only the lowest 0.4 m (the distance to the top of the bolts) was used in the calculation.

The CMRR is now determined by applying several adjustment factors to the RRW.

Strong Bed Adjustment (SBADJ)

One of the most important concepts in the CMRR is that the strongest bed within the bolted interval often determines the performance of mine roof. The strong bed’s effect on the CMRR depends first upon how much stronger it is than the other units. Second, the strong bed must be at least 0.3 m thick before it can provide any additional support, and the amount of the adjustment is maximum when the bed is at least 1.2 m thick. Third, the roof bolts must obtain at least 0.3 m of anchorage in the strong bed for the adjustment to be considered. Finally, the higher into the roof that the strong bed is located, the less its positive effect will be.

In the original CMRR, the SBADJ was determined using a table. For improved accuracy and to facilitate implementation of the table in the computer program, Equation 4 was derived using multiple regression:

$$\begin{aligned} \text{SBADJ} = & [(0.72 \text{ SBD} * \text{THSB}) - 2.5] \\ & * [1 - (0.33 (\text{THWR} - 0.5))] \end{aligned} \quad (4)$$

where:

- SBD is the strong bed difference—the difference between the strong bed’s Unit Rating and the thickness-weighted average of all the Unit Ratings within the bolted interval;
- THSB is the thickness of the strong bed (m); and

- THWR is the thickness of the weak rock suspended from the strong bed (m).

Note that if the strong bed is at the top of the bolted interval, its *full thickness* is used in the calculation of the SBADJ (up to a maximum of 1.2 m).

Other Adjustments

Number of Units: Many workers have indicated that mine roof that contains numerous lithologic contacts is less competent than roof that consists of a single rock type [Karmis and Kane 1984; Kester and Chugh 1980]. When depositional processes change and deposit distinctly different material, there is generally, but not always, a sharp contact between units. Since gradational contacts do not weaken the roof, the characteristics of major bedding contact surfaces (cohesion and roughness) should be noted. The maximum deduction from the CMRR is 5 points when more than four weak contacts are present.

Groundwater Adjustment: Groundwater is most prevalent in shallow mines, particularly beneath stream valleys, but it can also be introduced by leakage from pooled water in abandoned mines or fracturing of overlying aquifers during high-extraction mining. The CMRR maintains the RMR system's rating scale, with a maximum deduction for flowing groundwater of 10 points.

Surcharge: The strength of rocks overlying the bolted interval is considered only when they are significantly weaker than the rocks within it. An example is a western mine where 1.2 m of relatively strong top coal was overlain by 6 m of weak, rooted claystone. Because the roof beam needed to carry some of the surcharge (extra weight) of the incompetent claystone, stability was reduced. The CMRR accounts for the surcharge with a 3-point deduction.

THE CMRR COMPUTER PROGRAM

The CMRR program is designed to facilitate the entry, storage, and processing of field data. Either core or underground data can be entered, and calculations are updated instantly when a change is made. This allows the user to vary parameters, such as the bolt length, to see their effect on the final CMRR.

The underground data entry screen contains drop-down menus that are used to enter the data for each of the parameters. In the core data screen, the user has the option of entering PLT test data and having the program automatically determine the mean UCS and diametral $I_{s(50)}$. Otherwise, the user can enter the mean strength values directly.

An important feature of the new program is a built-in interface with AutoCAD. Data from up to 200 locations can be entered and saved in a single file, along with their geographic location coordinates. The program can create a

file for export that includes both the calculated CMRR values and the locations. A CMRR layer can then be created in AutoCAD for use in mine planning.

APPLICATIONS OF THE CMRR

During the past 10 years, the CMRR has been used extensively in the United States. Figure 12 shows the current database, containing 264 observations from more than 200 mines. The figure reveals some very important regional trends. Weak roof predominates in the northern Appalachian and Illinois Basin coalfields, which are also areas where roof falls tend to occur more frequently [Pappas and Mark 2003]. Central Appalachian mines have a wide range of CMRR values, but the typical roof is of moderate strength. Utah mines tend to have the most competent roof in the United States.

A number of mine planning design tools based on the CMRR are discussed below.

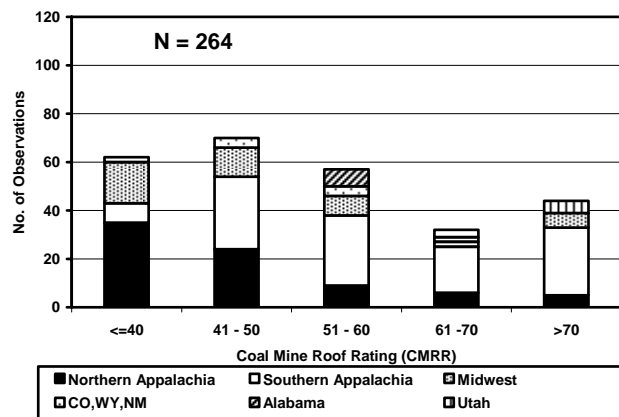


Figure 12.—Current CMRR database.

Analysis of Longwall Pillar Stability (ALPS)

The first, and perhaps the best known, application of the CMRR is the ALPS pillar design method [Mark et al. 1994]. A large database of longwall case histories was collected from throughout the United States and subjected to statistical analysis. The results showed that when the roof was strong (CMRR>65), longwall chain pillars with an ALPS stability factor (SF) as low as 0.7 could provide satisfactory tailgate conditions (Figure 13). On the other hand, when the roof was weak (CMRR<45), the ALPS SF might need to be as high as 1.3. ALPS has been the standard technique employed to size pillars for most U.S. longwalls for many years.

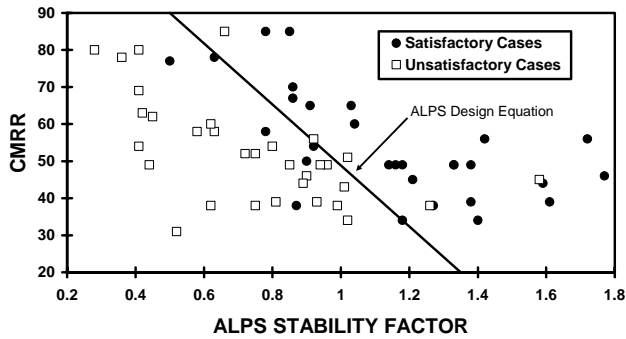


Figure 13.—Relationship between the CMRR and the ALPS SF.

Longwall Tailgate Design (Australia)

ALPS was the starting point for a project under the Australian Coal Association Research Program (ACARP) to develop an Australian chain pillar design methodology [Colwell et al. 1999]. The project aimed to calibrate ALPS for the different geotechnical and mine layouts used in Australia. Ultimately, case history data were collected from 60% of Australian longwall mines.

The study found strong relationships between the CMRR, the tailgate SF, and the installed level of primary support. Design equations were developed that reflected these trends. The final product, called the Analysis of Longwall Tailgate Serviceability (ALTS), was implemented in a computer program and has become widely used in Australia. Most recently, an expanded study resulted in an updated version called ALTS II [Colwell et al. 2003].

Stability of Extended Cuts

Place change mining, in which mining equipment moves from entry to entry as the section is advanced, is the standard development method in the United States. The traditional 6-m cut length was determined by the distance from the cutting head to the operator's compartment. With the advent of remote-control continuous miners, extended cuts up to 12 m long have become common. However, many mines with extended-cut permits only take them when conditions allow. Where the roof is competent, extended cuts are routine. At the other extreme, when the roof is very poor, miners may not be able to complete a traditional 6-m cut before the roof collapses.

To help predict when conditions might be suitable for extended cuts, a study was conducted at 36 mines throughout the United States. The study found that when the CMRR was greater than 55, extended cuts were nearly always routine, but when the CMRR was less than 37, they were almost never taken [Mark 1999a]. The data also

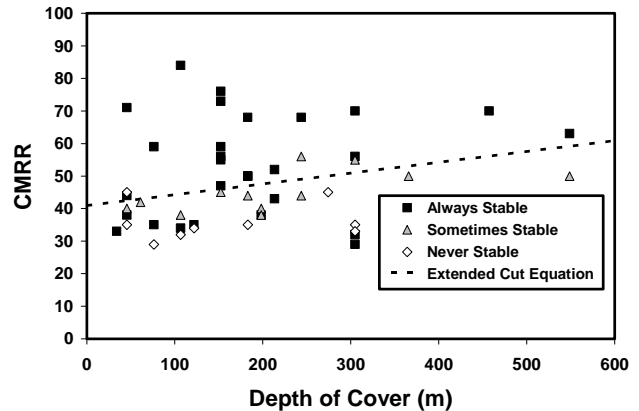


Figure 14.—Relationship between the CMRR and the feasibility of extended cuts.

showed that extended cuts were less likely to be feasible as the roof span or the depth of cover increased (Figure 14).

Roof Bolt Selection

To help develop scientific guidelines for selecting roof bolt systems, the National Institute for Occupational Safety and Health conducted a study of roof fall rates at 37 U.S. mines [Mark et al. 2001; Molinda et al. 2000]. The study evaluated five different roof bolt variables, including length, tension, grout length, capacity, and pattern. Roof spans and the CMRR were also measured. Performance was measured in terms of the number of roof falls that occurred per 3 km of drivage.

The study found that the depth of cover (which correlates with stress) and the roof quality (measured by the CMRR) were the most important parameters in determining roof bolting requirements. Intersection span was also critical. The study's findings led to guidelines that can be used to select the proper span, bolt lengths, and bolt capacity based on the CMRR. The results have been implemented into a computer program called Analysis of Roof Bolt Systems (ARBS).

Multiple-seam Mining

Interactions with previous mining in underlying or overlying seams are a major cause of ground instability in the United States. A statistical analysis of a database of more than 360 case histories found that the CMRR was highly significant in predicting the outcome of a multiseam interaction. Other significant variables include the pillar SF, the total pillar stress, whether the previous seam was above or below, and what type of pillar structure is present in the previous seam.

The statistical analysis became the foundation for the Analysis of Multiple-seam Stability (AMSS) software package. The output from AMSS is the critical interburden thickness that is necessary to avoid interactions. AMSS

indicates that, all else being equal, a CMRR=45 roof requires approximately 15 m more interburden than a CMRR=65 roof.

Longwall Mining Through Open Entries and Recovery Rooms

Unusual circumstances may require that a longwall retreat into or through a previously driven room. The operation is usually completed successfully, but there have been a number of spectacular failures. To help determine which factors contribute to such failures, an international database of 131 case histories was compiled [Oyler et al. 1998]. The study found that the CMRR and the density of standing support were the two most important parameters in predicting severe weighting-type failures. These failures occurred only when the CMRR was less than 55 and when the support density was less than 0.5 MPa. When the CMRR was 40 or less, all of the successful cases employed a standing support density of at least 1.0 MPa.

Roof Fall Evaluations (South Africa)

The CMRR featured prominently in an important research project sponsored by the Safety in Mines Research Advisory Committee (SIMRAC) and other leading industry, labor, and government organizations in South Africa. The goal of the project was to investigate the causes of fatal roof failures in South African coal mines. A team of recognized experts visited a broad spectrum of mines and collected data at 182 roof fall sites. The study found that roof falls were more likely where the roof was less competent in terms of the CMRR. Another finding was that the CMRR correlated well with roadway widths. Based on data presented by Mark [1999b] (see Figure 15), the study also concluded that “in South African coal mines, less support is used for comparable roof conditions than either the USA or Australia. This supports previous conclusions that in South African coal mines, the density of supports needs to be increased.” [van der Merwe 2001].

Another SIMRAC study found the CMRR easy to use and robust enough to adequately describe the roof conditions at most South African collieries [Butcher 2001]. It took less than 4 hr for a trained geologist to become competent with the method. The results seemed more reasonable than those obtained from the RMR, which tended to overrate ground conditions by at least one class (20 points) due to its lack of sensitivity to the characteristics of bedded strata. Some improvements were suggested for the CMRR, including adjustments for joint orientation, blasting, and horizontal stress.

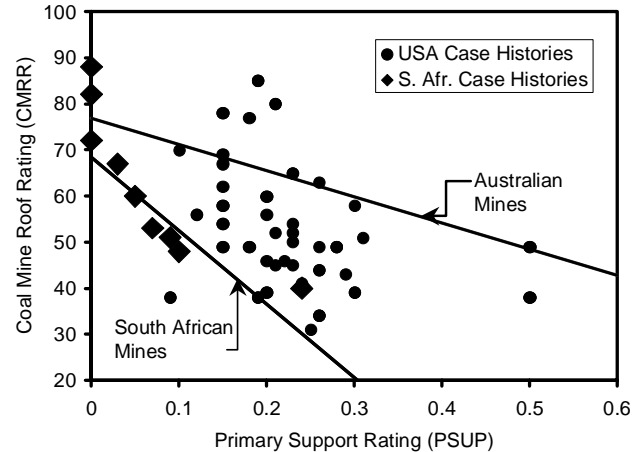


Figure 15.—Relationship between the CMRR and roof bolt density in the United States, Australia, and South Africa.

Baseline Comparison of Ground Conditions (Canada)

The Canadian underground coal industry is small and geographically dispersed. To assist the mines in maintaining world-class safety standards, the Canada Centre for Mineral and Energy Technology (CANMET) established the Underground Coal Mine Safety Research Consortium. One of the consortium’s first projects was aimed at establishing a “best practice” baseline for conducting geological and geomechanical assessments and applying the findings to geotechnical design.

The CMRR was found to be particularly valuable in the assessment [Forgeron et al. 2001]. It allowed the Canadian underground mines to be compared with each other and with international benchmarks. Based on the CMRR, many ground control safety technologies developed in the United States were found to have direct application to Canadian mines.

Other Applications

- *Highwall mining* can become uneconomic if the roof is so weak that it collapses before the miner has been withdrawn from the hole. The CMRR has been used to evaluate potential highwall mining reserves and to identify potentially unsuitable areas [Hoelle 2003].
- *Tailgate support guidelines* incorporating the CMRR have been included in the Support Technology Optimization Program (STOP) [Barczak 2000].
- *Input for numerical models* have been derived from the CMRR [Karabin and Evanto 1999].

CONCLUSIONS

Roof geology is central to almost every aspect of ground control. The CMRR makes it possible to quantify roof geology so that it can be included in mine planning decisions. Worldwide experience has shown that the CMRR is a reliable, meaningful, and repeatable measure of roof quality.

A wide variety of design tools based on the CMRR have now been developed. They address a broad range of ground control issues and rely upon large databases of actual mining case histories. Without the CMRR, it would not have been possible to capture this invaluable experience base.

The new core procedures and computer program further expand the potential of the CMRR. It is now possible to routinely collect CMRR data during geologic exploration or from underground mapping, complete the calculations, and integrate the results into mine mapping software. Foreknowledge of conditions means better mine planning and fewer unexpected hazards underground.

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