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J. Calleja
SCT Operations Australia

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CMRR – PRACTICAL LIMITATIONS AND SOLUTIONS

Justine Calleja¹

ABSTRACT: The Coal Mine Roof Rating (CMRR) is a rock mass classification which was developed empirically, from a database of coal mines in the USA. The CMRR weighs some of the geotechnical factors which may effect the competence of mine roof and combines them into a single rating on a scale from 0 to 100.

The Australian underground coal industry has, in recent years, wholeheartedly embraced this system as a method of geotechnical characterisation. CMRR is a very simple system which is quick and easy for any engineer or geologist to learn and implement. It also provides a standard process and methodology and an output which can be compared between mine sites, for geotechnical characterisation and design which neatly fulfils the current requirements of the Occupational Health and Safety Act and the Coal Mine Health and Safety Regulation 2006.

However, the use of CMRR on its own will potentially lead to flawed geotechnical characterisation and design. The pitfalls of rock mass classification systems have long been known to respected geotechnical experts such as Brady and Brown (1985) who caution, "Although the use of this approach is superficially attractive, it has a number of serious shortcomings and must be used only with extreme care. The classification scheme approach does not always fully evaluate important aspects of a problem, so that if blindly applied without any supporting analysis of the mechanics of the problem, it can lead to disastrous results".

The objective of this paper is to explore the risks and practical limitations associated with the use of CMRR, and to consider strategies and guidelines for the use of CMRR in characterisation and design which will minimise the risks.

OVERVIEW

The CMRR is an extremely valuable geotechnical characterisation tool which can significantly simplify and enhance the identification and communication of different geotechnical regimes; however, the inappropriate or incorrect use of CMRR could potentially lead to severe consequences.

The pitfalls of rock mass classification systems have long been known to respected geotechnical experts:

Bieniawski (1997) stated, "Rock mass classifications on their own should only be used for preliminary, planning purposes and not for final tunnel support".

Hoek and Brown (1980) "recommend classification systems for general use in the preliminary design of underground excavations".

Brady and Brown (1985) caution, "Although the use of this approach is superficially attractive, it has a number of serious shortcomings and must be used only with extreme care. The classification scheme approach does not always fully evaluate important aspects of a problem, so that if blindly applied without any supporting analysis of the mechanics of the problem, it can lead to disastrous results".

Karl Terzhagi commented, "I am more and more amazed about the blind optimism with which the younger generation invades this field, without paying attention to the inevitable uncertainties in the data on which their theoretical reasoning is based and without making serious attempts to evaluate the resulting errors."

A person with limited geotechnical expertise could mistakenly believe that they can easily produce a safe, sound geotechnical roof support, mining method or pillar design by calculating CMRR and using it in conjunction with the readily available design tools (NIOSH and Colwell software and other case histories). The risk of this scenario occurring is exacerbated by both the recent changes in coal mining legislation, which has lead to the reduced involvement of the Inspectorate in reviewing geotechnical designs, and the lack of formal requirements and experience for a person to practice as a geotechnical engineer.

Incorrect or inappropriate CMRR results and designs can be calculated as a result of:

- human error, inexperience, or lack of competency;
- variation in data collection and calculation methodology;
- inaccuracies in the input data;

¹ Geotechnical Engineer, SCT Operations Pty Ltd

- limitations in the calculation process;
- limitation of the specific properties included in CMRR;
- limitation of the cases included in the original database;
- limitation of the empirical approach in not being targeted to identify potential failure mechanisms.

Incorrect CMRR results and designs can be calculated as a result of human error, inexperience or incompetency. The implementation of engineering design quality standards in geotechnical design in underground coal is an important measure to reduce the risk of human error. The risks associated with inexperience and incompetency can be mitigated by the implementation of confirmed competency and experience requirements for geotechnical engineers.

The CMRR result can vary by up to 41 points as a result of variability in methodology and random variation in the point load test data. This extreme variability can be reduced by clarifying the methodology (e.g. fracture logging must be done in the splits, only use diametral point load test data if more than 5 tests results are available for the unit). It can also vary by up to 10 points as a result of different observers. This variation alone can mean the difference between indicating a 4-6 bolt support pattern and the potential for extended cuts. As such the CMRR value needs to be considered to be a rough indication of roof strength. It is not a precise measurement suitable for precise design.

Limitations in the calculation method include inadequate de-rating of low UCS roofs and the lack of inclusion of the frequency of weak unfractured planes in the discontinuity rating.

As a geotechnical characterisation tool, CMRR is limited by some of the properties which are not included in the calculation. It is not suggested that these properties should be included in CMRR, but that they do need to be considered in combination with CMRR to facilitate a comprehensive understanding of conditions:

- faults, dykes, igneous structures;
- vertical and sub vertical structures such as joint sets and cleat which are under represented in vertical core;
- rock stiffness (eg. Young's Modulus);
- triaxial strength (angle of friction);
- pre-existing stress.

It is important to remember that CMRR is a rock mass strength indicator as opposed to a rock mass stability indicator. When using CMRR in determining mine or support design many other factors need to be considered in combination with CMRR to determine design specifications. In addition to the properties above, the CMRR does not take into account:

- mining induced stresses;
- mining geometry such as roadway span or orientation of workings;
- installed support;
- rib or floor conditions.

It is inappropriate to apply empirical systems such as CMRR to situations which lie outside the range of the original dataset. CMRR was developed for the immediate bolted horizon of underground coal mine roofs. As such, it is not likely to give a reasonable rating for shaft walls, or for rib or floor strength. It may not be a good indicator of roof strength for steeply dipping deposits, or for very deep or high stress conditions. Similarly the applicability of the design tools are limited by the databases they were built on. This is clearly illustrated by the differences in design outcomes between the American and Australian design tools for pillars and roof support density.

In many circumstances, CMRR and the associated design tools are not adequately able to indicate the potential for specific failure mechanisms. Roof lithologies which have failure mechanisms largely driven by horizontal discontinuity spacing are likely to have low CMRR results. However the failure mechanisms which are driven by other properties will not necessarily have low CMRR results.

HUMAN ERROR

The potential impact of human error in leading to an incorrect CMRR and inappropriate design is significant. The problems with human error are not specific to CMRR, and are relevant to all characterisation and design tools. However, the risk of human error leading to an inappropriate design may be higher than with other approaches because the person doing CMRR does not need to have geotechnical experience or expertise. With most of the other methods of characterisation and design if the person doing the work doesn't have experience and expertise they will be forced to obtain the assistance of someone who does. A person with experience and expertise is less likely to make a mistake because they understand the reasons for collecting the data and the end purpose as well as the normal range of input data values. They are also more likely to realise that they have made a mistake when they consider the results and compare them to what would be expected.

The risks of human error can be reduced by the proper implementation of a Quality Standard for engineering design. Specifically, the Australian Standard 3905.12/1999 which is the Quality System Guide to ISO 9001 for architectural and engineering design, describes many of the components and practices which should be second nature to any engineering graduate.

Some of the specific elements of quality design which are essential are:

- Performing alternative calculations (4.4.7 Design Verification) – this means doing any calculation which is going to be used for an important purpose, such as roof support design or pillar design twice using a different method to calculate the result. This also means using more than one design approach.
- Comparing the new design with a similar proven design (4.4.7 Design Verification).
- Undertaking tests and demonstrations of the proposed design (4.4.7 Design Verification).
- Having documented procedures to control and verify the design (4.4 Design Control).
- Conducting Internal Audits (4.17 Internal Quality Audits) – carried out by personnel independent of those having direct responsibility for the activity being conducted. It is really important that any design calculations and design process be checked by someone other than the designer to ensure that the designer understands and follows the design procedure correctly. This applies to a wide range of essential skills and competencies, not just design, but also, for example, geotechnical mapping, installation of geotechnical equipment and monitoring and analysis.
- Confirmed competency (4.18 Training) – training is required so that personnel are appropriately qualified and competent to perform the work. This can be demonstrated by formal training followed by the successful performance of work under supervision.

METHODOLOGY

The methods available for determining UCS (unconfined compressive strength) of core include laboratory testing, calculating inferred UCS from point load testing and calculating inferred UCS from the sonic log. Colwell (2003) states that “in practical terms it has been found (for Australian Collieries) that the UCS can be better quantified from the associated laboratory testing and/or correlated sonic logging” than from axial point load testing. This is consistent with the author's experience. The UCS rating typically makes up around 30% of the CMRR result and variation associated with errors resulting from the use of axial PLT inferred UCS can lead to a difference in the order of 6 CMRR points (e.g. an axial PLT UCS was 7 MPa = UCS rating of 7 vs. a Lab test UCS of 35 MPa = UCS rating of 13, see Figure 7 for another example).

The discontinuity rating is the most heavily weighted factor in the CMRR calculation and can make up around 70% of the final result. Colwell (2003) states that the core should be logged in the “splits” for RQD and Fracture Spacing. Mark and Molinda do not specify when the core should be logged. Depending on the timing of core logging the fracture spacing and RQD can be significantly different. The Figure 1 core photo below shows core which had the fractures marked up in the splits. The fractures which have occurred since are visible in the photo and not marked. It is evident that the fracture spacing in the splits was around 180 mm, but that the fracture spacing at the time of the photo, when the core had been put in the tray, was around 69 mm. This would lead to a difference of 5 CMRR points. In Figure 2 core photo it is apparent that the fracture spacing has reduced from 1000 mm to 140 mm which equates to a difference of 11 CMRR points.



Figure 1 - Core photograph



Figure 2 - Core photograph

The second major issue is that if fractures are logged in the core trays, rather than in the splits then the CMRR results are likely to be lower than the CMRR results Mark Colwell obtained and used in his database during the development of ALTS and as such a modification factor would need to be applied to be able to use ALTS. Similarly, if the data which Mark and Molinda used to calibrate their core rating and underground exposure rating was based on fractures logged in the core trays rather than in the splits, then a modification factor would be required to be able to use the NIOSH databases and to compare the Australian data with the US CMRR data.

The recommended method for calculating CMRR is to identify distinct geotechnical units and calculate unit ratings for each one. Then the bolt length to be used is entered and the ratings of the units which lie within the bolted horizon are combined and averaged (weighted by thickness). The adjustment factors are then applied for weak contacts, ground water, strong beds and weak overlying strata.

The height of the top unit which is included in the calculation is not required to match the CMRR horizon. As such the top unit may be 1 m thick, but only have the lower 20 cm included in the calculation. The problem with this is that the lower 20 cm of that unit may have slightly different UCS, fracture spacing, RQD, Diametral strength and moisture sensitivity to the full unit.

The definition of a CMRR unit is that it has consistent geotechnical properties, however common sense has to be applied to avoid creating an excessive number of thin units, so it is likely to occur in practice that sections of a unit, if considered separately may vary by as many as 10 CMRR points. The difference between the section of top unit included in the CMRR and the full unit becomes quite important if sensitivity analyses are being done on various bolt lengths. The current method of calculating CMRR (using NIOSH software) makes it difficult to ensure that the height of the top unit is the same as the bolted horizon, unless the bolt length is well established at the site.

Rapid Rating (Calleja, 2006) is a system which allows the unit ratings and CMRR to be easily calculated for any horizon. This involves creating the following tables of data for the full section of core to be considered (6-8m) with values allocated for every possible depth value:

- fracture log data;
- UCS data table (based on sonic inferred UCS and/or Lab tests and/or Axial PLT inferred UCS);
- moisture sensitivity table;
- lithology table;
- diametral PLT data table.

The unit heights and bolted horizon are selected and the values for each of the unit rating inputs can be calculated by averaging the data in each of the tables between the selected unit heights, with the height of the last unit automatically set to be the height of the bolted horizon. Rapid Rating is a program which uses this methodology, but anyone can do it whether they use Rapid Rating, write their own code, or calculate it manually. The additional benefit of using this method, is that it is very easy to go back at a later date and recalculate for other horizons without having to re-log the core.

RANDOM VARIABILITY OF INPUT DATA

The discontinuity rating is the most heavily weighted factor in the CMRR calculation and can make up around 70% of the final result. Mark and Molinda (2005) state that the discontinuity rating is the lower of the Diametral PLT Rating or the Discontinuity Spacing Rating (determined from Fracture Spacing and RQD). The NIOSH software includes a table for up to 48 diametral point load test results which can be averaged for any unit. Colwell recommends doing as many diametral PLT tests as you can, whilst maintaining the core length to diameter ratio of more than 1 for each specimen. Unfortunately in practice it is often difficult to get more than 1 or 2 diametral point load tests on a particular unit. This becomes a problem because point load test results (diametral and axial) tend to be highly variable.

In the Figure 3 each value on the x axis represents a Unit number. Each unit represents a different ply within the coal seam. All of the units are coal (some more stoney than others) except for unit 3 which is a claystone. The graph shows all of the Is50 results for 32 holes at a mine site. There are some holes which have 2 or 3 tests on the same ply and the variability for these samples is similar to the overall variability when looking at all of the holes. Specifically, it is evident that the diametral data is almost a uniform distribution (i.e. there is an equal probability of obtaining any value between the minimum and maximum values), rather than a normal distribution where there is a higher probability that any single test value will be closer to the mean than to the minimum and maximum values. At this site, where most CMRR units which are usually individual plies and only have 1 or at best 2 or 3 diametral tests, the discontinuity rating associated with that unit could either be 25 or 41 or anywhere in between despite the fact that the physical properties of that unit are fairly consistent. The graph shows that the variation in Diametral Is50 is effectively random and if you use Is50 in the discontinuity rating and don't have the luxury of using the mean from a large number of tests e.g. 20 tests, the CMRR result could be randomly variable over a range of 16 points. Alternatively, if based on this data, you use an average Is50 for each ply for all holes then you will end up distinguishing CMRR by the holes which have very low RQD or Fracture Spacing and distinguishing them by variability in UCS. You would be capping the maximum possible CMRR value by the average diametral rating for that ply, which is probably not appropriate in reality, because some holes may have plies with higher average Is50 results (if it were possible to obtain 20 tests from that ply). It is generally not possible to obtain enough test data to

be able to get a reasonable average for an individual ply in a single hole, and as such it may be concluded that it is inappropriate to use diametral Is50 results in the CMRR calculation at this site.

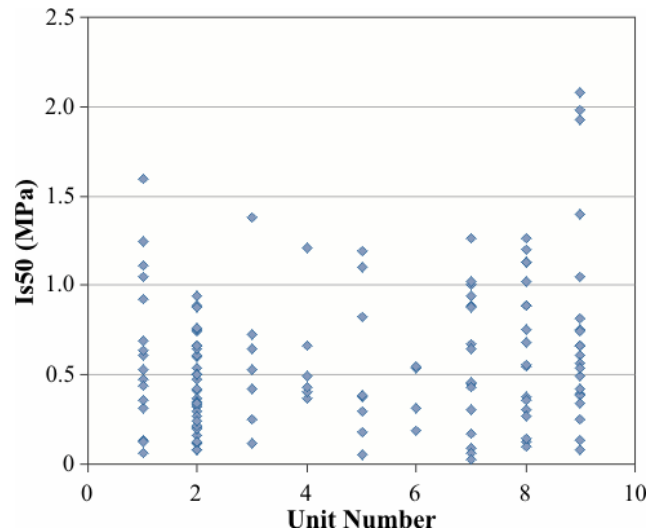


Figure 3 - Diametral pointload test Is50 by unit

The Figure 4 photographs below show diametral results from a different site, where the immediate roof unit is vertically extensive and has consistent properties. A large number of diametral tests were done on this unit and supported the evidence shown in the previous example. Figure 5 shows the Is50 results for this drill hole versus depth of the test. The tests in the graph start at the top of the seam, and are all in stone. The graph shows a general trend of increasing Is50 strength associated with increasing distance above the seam, however at any one point there is a large variation in diametral strengths in the order of 0.5-0.8 MPa, and in two locations the variability is 1.5-2 MPa. For this particular rock type the variability in the rating is typically around 10 CMRR points and as high as 35 CMRR points. This is an example where the rock has had almost the maximum possible number of diametral tests done. It could be argued that in this case a reasonable average for any location could be obtained, although there would have to be some concern about which diametral strength value would be appropriate to use for the locations where the PLT ratings ranged between 25 and 60. It is not very common for so much closely spaced diametral data to be available. At other sites it would be likely that only a quarter of the number of tests would be available which would mean that individual test values would have to be used rather than averages, and this will lead to high variability in the diametral ratings, and as a result in the CMRR ratings which may not be truly reflective of the properties of the units.

CALCULATION PROCESS

There are problems inherent in the method of calculation of the Discontinuity Rating. Fracture spacing and RQD are purely a measure of the frequency of discontinuities with less strength than the load applied in the drilling process (which is variable). On the other hand Diametral Point Load testing is purely a measure of the strength of a discontinuity and does not include any spacing/frequency component. As a result, when diametral point load testing is used to calculate the discontinuity rating, the same CMRR value could be obtained for strata with weak bedding planes at 300 mm spacing as strata with weak bedding planes at 50mm spacing.

The presence of weak but unfractured planes in the core is only taken into account by the unit contacts adjustment (unless diametral point load testing is used to calculate discontinuity spacing). The unit contacts adjustment is calculated by determining the number of contacts between CMRR units which are weak. The discontinuity rating can vary from 20 to 60 for a CMRR unit, however the maximum deduction for weak but unfractured planes (from the unit contacts adjustment) is 5.

The UCS of the rock is the second most important factor in CMRR with its rating ranging from 5 to 30 (compared with 20 to 60 for the discontinuity rating) (Figure 6). The author's experience with the CMRR results from the proportional weightings of UCS and discontinuities seems to give a reasonable result in most circumstances. However, at the lower end of the UCS scale the decrease in UCS rating is linearly proportional to the decrease in UCS from 34.48 MPa down to 0 MPa. Specifically, the difference in UCS rating (and CMRR value) between a 20 MPa and a 5 MPa core is 3. This would not even represent a different CMRR classification. Whilst 5 MPa core is not very common, it is essential to highlight areas where very weak rock occurs (whether it is fractured or not) because substantially different management approaches are required to accommodate the different risks which exist for 5 MPa rock as opposed to 20 MPa rock.

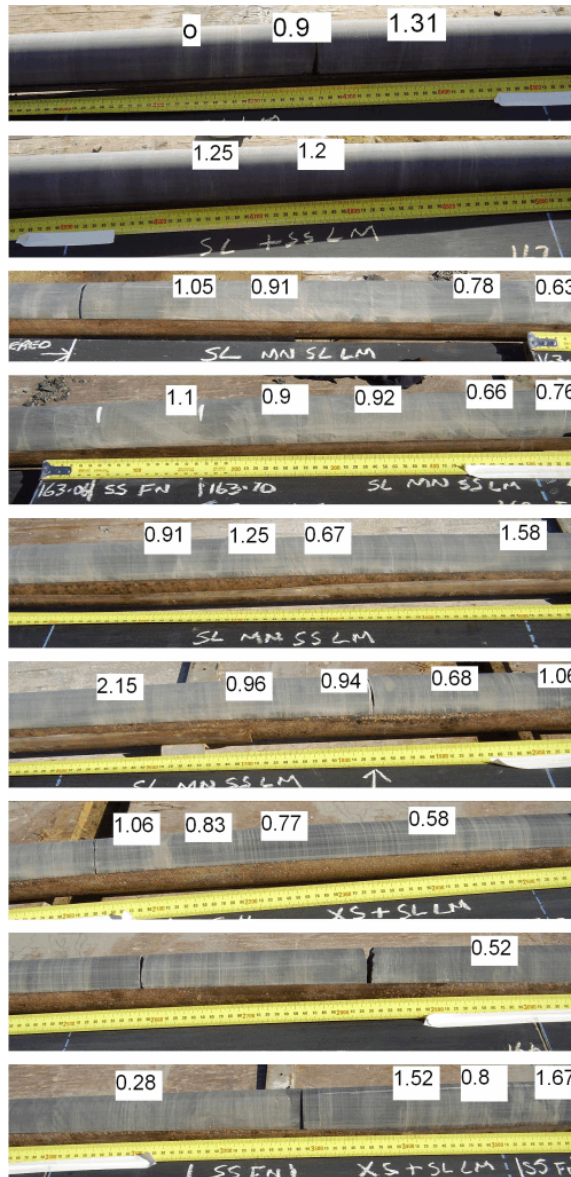


Figure 4 - Core photographs showing diametral PLT Is50 values

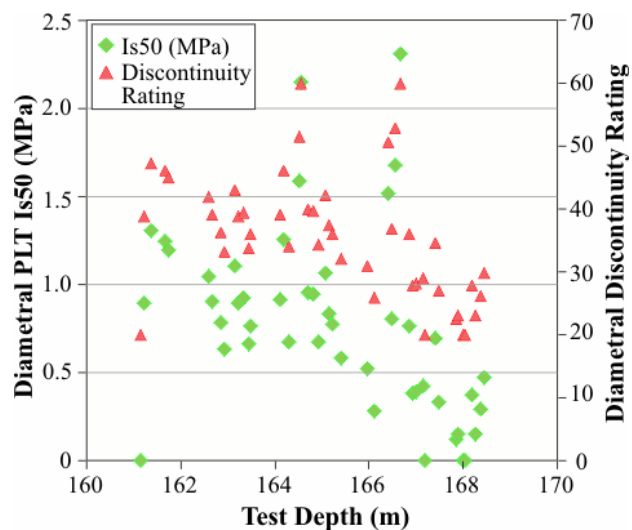


Figure 5 - Diametral PLT values and discontinuity ratings versus depth

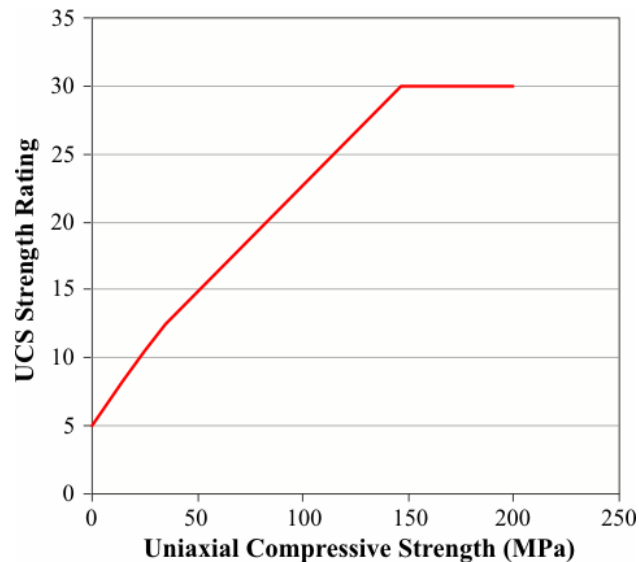


Figure 6 - UCS strength ratings

It can be argued that low UCS rock would end up with a low CMRR as a result of low diametral strength. However this assumption is not 100 % reliable, as shown in Figure 7. The laboratory tested UCS was 6 MPa, and this value is consistent with the lab test UCS results from this unit in other holes at the mine site. The photograph shows the Diametral Is50 and axial PLT inferred UCS results for this unit. For some sections of the core the axial PLT UCS results of 9 MPa and 5 MPa were appropriate, but other sections gave highly inaccurate UCS results of 25MPa. Similarly, some sections of the core gave appropriate diametral results of 0.05 and 0.14 but other sections gave results of 0.63 and 0.72. It is interesting to note that the low diametral results occurred with the high UCS results and vice versa, which indicates that the variability in results is not attributable to general variability in the core properties, but rather due to variability in small scale properties of the test specimens. A reasonable CMRR for this unit would be around 30 - 40. If this unit was less vertically extensive and only one set of PLT data was available, e.g. the first or last set then the CMRR obtained purely from point load testing would be 40 - 44 without groundwater. If the more appropriate diametral results of 0.05 and 0.14 were used the CMRR would be 28 without groundwater. This example illustrates that there are significant problems associated with:

- the use of axial PLT data for determining UCS;
- the use of diametral PLT data for determining bedding plane strength;
- the reliance on diametral strength to de-rate low UCS units (the UCS rating should do this not the diametral rating).



Figure 7 - Core photograph showing Axial PLT, inferred UCS and Diametral PLT Is50 values

LIMITATION OF ROCK PROPERTIES INCLUDED

CMRR is inadequate to be used on its own for the purposes of geotechnical characterisation because of the rock properties which can be very important to rock mass strength and are not included in core calculated CMRR:

- faults, dykes, igneous structures;
- vertical and sub vertical structures such as joint sets and cleat which are under represented, or over represented in vertical core;
- rock stiffness (e.g. Young's Modulus);
- triaxial strength (angle of friction).

It is possible to calculate CMRR in the vicinity of geological anomalies such as faults, dykes and igneous structures, however CMRR will not take into account potential geometric structural failure mechanisms, or the impact of such structures on the overall mine opening stability (the combined impact of the structure on roof, ribs and floor), or the impact of such structures on *in situ* and mining induced stress changes. It may be impossible to obtain a "representative" CMRR value in the vicinity of a geological anomaly because of large variations in the geotechnical conditions around the anomaly. In addition, CMRR is an empirical system, and any specific geological anomaly is not likely to be represented within the original dataset. As such, any conclusions which can be drawn about CMRR results based on the original database could not be applied to CMRR values for a specific geological anomaly.

Vertical fractures are typically not the cause of difficult conditions in underground coal mines due to gravitational interlocking or confinement provided by horizontal stress. When vertical fractures occur they are generally under represented in vertical core. However when vertical fractures are closely spaced (e.g. cleat in a coal roof) there may often be large sections of the coal core which is effected by a single vertical fracture and whilst a single vertical fracture will not lead to a very low fracture spacing on its own, it will often lead to many more bedding plane fractures than would otherwise occur, and thus lead to excessively low fracture spacing and an excessively low CMRR.

Sub vertical fractures are similarly under-represented in vertical core. Sub vertical fractures can lead to difficult conditions in underground coal mines, especially if there is more than one joint set. CMRR does not adequately indicate circumstances where stress or gravitational induced block failure can occur, and so where there is a potential for block failure to occur, analytical analysis is necessary to identify the potential failure mechanisms and determine appropriate management strategies.

Rock stiffness (e.g. Young's Modulus) is not included in the CMRR calculation. Rock stiffness is a measure of the amount of deformation (strain) which will occur under a certain amount of stress (load per unit area). If a roof has rocks which have different stiffnesses, then under uniform strain (a generally accepted condition for underground coal mine strata), the rocks with higher stiffness will be under higher stress and the rocks with lower stiffness will be under less stress.

A common assumption is that Young's Modulus is usually linearly related to UCS. The implication is that as UCS increases, so does Young's Modulus and the ratio of UCS to Young's Modulus remains fairly consistent. This means that although the rocks with higher stiffness would carry higher stress, they would also have higher strength and so would be equally as stable as the lower stiffness, lower stress, lower strength rocks. Therefore the variation in rock stiffness is sufficiently considered by analysing UCS. This is a reasonable argument, but only in situations where all of the strata has a consistent UCS to Young's Modulus ratio. Unfortunately, in reality there are many situations where this assumption is not true, and identifying those circumstances is essential to characterising strata behaviour and understanding the potential failure mechanisms.

For example, at one site the rock testing database of 205 samples (Figure 8) shows a range in the Young's Modulus to UCS ratio ($\times 0.001$) between 0.06 and 1.16. Coal measure rocks typically have E/UCS ratios ($\times 0.001$) between 0.2 and 0.3. However the samples in the dataset below demonstrated that some particular units had significantly different ratios. The weak sandstone unit shown in the previous photograph is one example with ratios between 0.4 and 0.7. Sixty three coal samples had ratios between 0.07 and 0.34 with an average of 0.14. The coal sample ratios were markedly lower than the non coal samples which had an average of 0.31.

The lower E/UCS ratio of coal samples has major practical significance. It may provide an explanation for why coal roofs are much more stable than one would expect after considering the typically highly fractured nature of coal and its generally low UCS (and resulting low CMRR values).

It is also very important to identify any units with much higher E/UCS ratios than the surrounding units as these may be the precipitators of progressive stress based roof failure. Units with high E/UCS ratios may be much less competent than their UCS alone would indicate because of the higher levels of stress which they carry. In Australia, where core drilling is standard practice, the collection and laboratory testing of core samples is a small proportion of the exploration cost. There is no reason why UCS cannot be determined from laboratory testing, and the determination of Young's Modulus is available as a standard component of a UCS test.

Uniaxial Compressive Strength (UCS) is the strength of a sample of rock when it is loaded uniaxially (only in one direction). It is a rough indicator of the strength properties of rock. Triaxial strength provides a more complete picture of the behaviour of rock *in situ*, as the strength of the rock in one direction is determined for varying conditions of confinement in the directions perpendicular to the primary loading direction. In virtually all circumstances the strength of a rock sample (intact or fractured) will increase associated with increasing confinement, and the rate of increase is consistent, whether the specimen is intact or already fractured. The ratio of increase in strength to increase in confinement is described by the friction angle. Some rocks have very low friction angles and gain very little strength when they are confined. In contrast other rocks have very high friction angles, and may for example have a low UCS but then high strength when under 4 MPa or 8 MPa of confining pressure (e.g. horizontal stress).

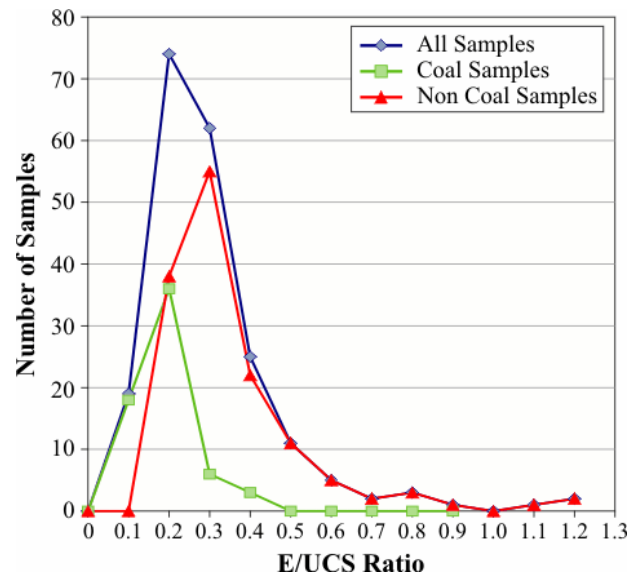


Figure 8 - E/UCS ratio for all samples.

For example, a claystone in the immediate roof with a UCS of 45 MPa and a friction angle of 44 degrees would have a strength of 100 MPa at 10 MPa horizontal stress. This unit would actually be stronger than a 59MPa UCS siltstone with a friction angle of 31 degrees which would have a strength of 90 MPa at 10 MPa horizontal stress. It is interesting to note that many of the coal samples in the previous graph which often had low E/UCS ratios also had high friction angles which may also contribute to the higher competency of coal roof.

Triaxial strength testing is more expensive than UCS testing, and for this reason it is not the standard test performed on all core samples, however it is possible to pick a smaller proportion of representative samples out of a testing program and conduct triaxial testing without substantially impacting on the economics of the exploration program.

LIMITATIONS OF OTHER PROPERTIES INCLUDED

It is important to remember that CMRR is a Rock Mass Strength indicator as opposed to a Rock Mass Stability indicator. When using CMRR in determining mine or support design many other factors need to be considered in combination with CMRR to determine design specifications. The CMRR does not take into account:

- pre-existing or mining induced stresses (e.g. resulting from depth of cover, horizontal stress, longwall abutment and stress concentration, stress direction, faults);
- mining geometry such as roadway span or orientation of workings;
- installed support;
- rib or floor conditions.

The PSUP vs. CMRR graph (Figure 9) shows that for any CMRR value, the associated PSUP values can virtually range across the full spectrum for each country. This is due to the impact of all of the other factors, listed above, which combine with CMRR to lead to overall roof stability.

LIMITATION OF CASES IN THE DATASET

CMRR is an empirical system. This means that it has been developed based on a specific set of data (in this case a large number of underground coal mines in the USA). As with all empirical systems, it is generally useful and valid whilst used within the boundaries of the data from which it was developed, but it cannot be assumed to be applicable outside that dataset. For example, the dataset was developed based on mine roadways and would not be directly applicable to a drift which is oriented at an angle to bedding and would potentially have additional wedge failure risks. Following the same logic it may not be a good indicator of shaft wall properties. It is not necessarily a good indicator of floor conditions as poor floor conditions may be more influenced by slake durability and UCS rather than fracture spacing which is most heavily weighted in CMRR. As described previously, if the original core CMRR data was based on fracture logging in core trays rather than in the splits, then a modification factor would have to be applied to compare the original data with data calculated from fracture logging in the splits. Similarly, if any significant changes are made to the calculation methodology, then the modification factors will need to be applied to data calculated using the previous methodology.

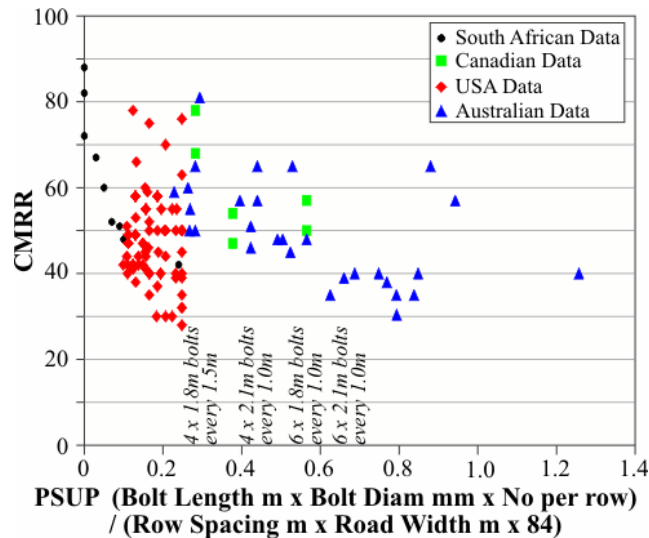


Figure 9 - PSUP (primary support density) versus CMRR.

The limitation of cases in the dataset is particularly relevant when considering the use of CMRR design tools. The PSUP vs. CMRR graph (Figure 9) clearly illustrates that the support densities used in the USA, Australia and South Africa are almost mutually exclusive. Whilst some of the difference can be attributed to lower average depth of cover in the USA and South Africa, and possibly other factors such as lower horizontal stress. However, the roof fall rates in South Africa and the USA were also higher than in Australia and it is likely that the lower support densities are directly related to higher roof fall rates, which are tolerated to different levels as a result of cultural differences.

CMRR AND FAILURE MECHANISMS

As an empirical system, CMRR is limited by the fact that it does not take into account different failure mechanisms associated with different geotechnical environments.

CMRR is primarily calculated from horizontal discontinuity spacing or bedding plane strength, UCS and moisture sensitivity. Roof lithologies which have failure mechanisms largely driven by horizontal discontinuity spacing (e.g. high angle shear failure of thinly weakly bedded roof) are likely to have low CMRR results. However the failure mechanisms which are driven by other properties (some in combination with horizontal discontinuities) will not necessarily have low CMRR results:

- block failure – determined by boulders or sub-vertical joints;
- skin slab failure – determined by properties of the first 0.2 m of roof rather than the full bolted horizon;
- overstressing failure – determined by *in situ* and mining induced stress, E/UCS, and triaxial strength properties;
- de-stressing failure – determined by sub vertical joints or mining induced fractures and mining induced reduction in stress;
- tension failure – determined by vertical or sub vertical joints in coal roof, or for bulking failure: the thickness of coal beam, the thickness of roof stone, the bedding plane properties, E/UCS properties, triaxial strength properties, tensile strength properties, and stress field;
- combined structure and stress failure - determined by *in situ* and mining induced stress, sub vertical discontinuities, E/UCS and triaxial strength properties.

For example, a 50 MPa massive sandstone roof with occasional weakly bonded boulders or with regular open joints at 5 m spacing, 60 degrees to horizontal and oriented parallel and at right angles to the roadway direction could have a CMRR of 60+ which would incorrectly indicate the potential for extended cuts and a low density support pattern. Alternatively a roof with 30cm of very thinly weakly bedded siltstone with 1.7 m of massive unfractured 80 MPa claystone could have a CMRR of 72 and have significant skin failure problems.

LIMITATIONS OF THE DESIGN TOOLS

There are various design tools and case histories which are available to use with CMRR: ALTSII (Australian Longwall Tailgate Design), ALPS (NIOSH Longwall Pillar Stability), ARBS (Roof Bolt Selection), extended cut stability, longwall mining through open entries and recovery rooms. The empirical design tools have similar limitations to CMRR. They are limited by:

- the use of CMRR (its variability and geotechnical factors not included);
- the range of the cases used to build the tool and;
- important factors effecting design which are not included in the design tool.

Some other factors which could potentially result in an unsafe pillar/tailgate design using ALTSII include the presence of very weak floor, the occurrence of faults or other geological anomalies, or the presence of a weak sliding plane such as a clay band which can prevent pillars from developing confinement. In addition to these factors, ALPS does not include consideration of horizontal stress. Problems with design outcomes due to the difference in the cases used are evident when using the same inputs for both programs. For example, ALTSII recommends a larger pillar width varying from 10 m to 25 m wider than the ALPS results for CMRR 45 and DOC 450 m.

ARBS is the "Analysis of Roof Bolts" program developed by NIOSH to provide roof bolt support design parameters based on CMRR, depth of cover and intersection span. It was developed by statistical analysis of a variety of roof bolt systems at 37 mines in the USA with the effectiveness of the systems determined by comparing the number of roof falls per 3000m driveage. ARBS does not include horizontal stress magnitude or the difference in reinforcement performance resulting from the use of point anchored or fully grouted bolts. The data used in the program did not include the effects of longwall loading, only development conditions. The program is based on USA support practices and there is a significant risk that higher roof fall rates than are generally tolerated in Australia, would occur if ARBS designs were applied in Australian mines. This is supported by the recommendation in the ARBS help file that "The field data also indicated that in very weak roof, it may be difficult to eliminate roof falls using typical U.S. roof bolt patterns. When the CMRR was less than 40 at shallow cover, and less than 45-50 at deeper cover, high roof fall rates could be encountered even with relatively high roof bolt densities. Faced with these conditions, special mining plans or routine supplemental support might have to be considered."

The extended cut stability and longwall mining through open entries and recovery rooms analysis are not design tools per se, but rather a compilation of case histories. The extended cut data was all taken from USA mines and represented in the form of a graph showing CMRR versus Depth of Cover with the points on the graph separated into "Always Stable", "Sometimes Stable" and "Never Stable". As described for the previous design tools, there are many other factors which effect extended cut stability and the risks associated with extended cuts than are included in the graph and in the data. The determination of stability was based on interviewing personnel at the specific mines, rather than a quantitative measure. The cultural safety differences between Australia and the USA may also be present in this dataset and it would be inappropriate for a geotechnical engineer to use this empirical database (on its own) to determine that extended cuts could be implemented in an Australian mine.

Oyler's paper on longwall mining through open entries and recovery rooms analyses factors (including CMRR) which effect whether these operations can be conducted without severe weighting or roof falls. It is a compilation of 130 case histories from USA, Australia and South Africa. The method of determining success or failure of the cases was more easily quantified and so more objective than the extended cut data. The potential cultural safety differences are less significant in this dataset because it includes Australian data, however there are still many other factors which would effect the stability of a recovery room which could not be included in the data and so the use of this data should be limited to a broad indication of the possibility for mining into a recovery room and any design should not be developed without additional extensive analysis using other appropriate design methods to confirm it.

CONCLUSIONS

The Coal Mine Roof Rating is a very valuable tool for geotechnical characterisation and empirical design, however it needs to be used by competent and experienced geotechnical engineers with careful consideration of its limitations.

Risks associated with human error, inexperience and incompetence occur with all characterisation and design methods but are more likely to occur with CMRR because less expertise and experience is needed for its use. These risks can be managed through the implementation of engineering design quality standards.

Variability in the CMRR results can be reduced by ensuring that fracture logging is done in the splits, diametral point load test results are only used where a large number of tests are conducted on each unit and UCS is calculated from lab test data, correlated sonic logging or high density correlated axial point load testing. Variability of up to 10 points is unavoidable due to the observer differences. Therefore CMRR should not be considered to be a precise value but rather a rough indicator of rock mass strength.

CMRR does not adequately de-rate low UCS lithologies and it does not include numerous important properties which are essential components of a thorough geotechnical characterisation. As such, CMRR should not be used in isolation for mine site geotechnical characterisation. It should be used as one component of a broader assessment.

The empirical design tools which can be used with CMRR are important and useful datasets, however, they are also limited by the range of the cases they were developed from, by important geotechnical properties which are not included and by the inherent variability in the CMRR values which are input. It would be unwise to implement an operational geotechnical design based on a CMRR design tool without considering all of the potential failure mechanisms and without employing alternative appropriate design methods to confirm any design outcomes.

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