Empirical approaches for opening design in weak rock masses

Tom Brady, Lewis Martin and Rimas Pakalnis

A major focus of ground control research presently being conducted by the Spokane Research Laboratory of the National Institute for Occupational Safety and Health (NIOSH) is to incorporate weak rock masses (such as are associated with operations in the Carlin Trend in Nevada) into existing design relationships. The original database that led to most of the empirical design relationships presently employed in hard-rock mining was derived from fair-to-good-quality rock. In this study, the relationship between weak rock quality and opening design (non-entry/entry methods) is being investigated. The common factor in all mines is a weak back or wall. This work attempts to provide tools that will enable a mine operator to make economic decisions that will also ensure a safe working environment.

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INTRODUCTION

Researchers at the Spokane Research Laboratory of the National Institute for Occupational Safety and Health, Spokane, Washington, USA, and the University of British Columbia, Vancouver, Canada, have assembled a team to develop underground design guidelines for safe and cost-effective mining within a weak rock mass. Such work also includes developing novel support methods, such as the use of synthetic fibre reinforced shotcrete, ways to undermine under-hand-and-fill backfilled stopes, and assess supports presently in place in weak rock masses. In the present study, rock mass interaction with grouted bolt supports was investigated in three mines in Nevada and backfill, pillar, and bolt support were studied in one.

Many Nevada gold deposits are found in intensely fractured, faulted, and argillised host rock. As a result, underground mining is often difficult and hazardous, as indicated by the number of injuries and fatalities from uncontrolled falls of ground (Table 1).

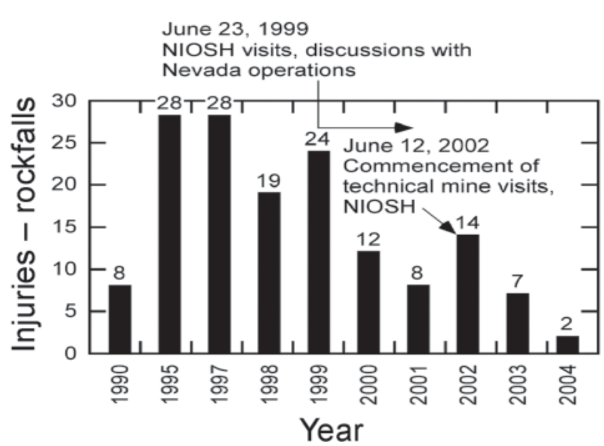
Table 1 Ground control injuries and fatalities in underground ground gold mines in Nevada, 1985–2000

Fatalities	7
Permanent disabilities	4
Lost-time injuries	49
Restricted-activity injuries	46
Other injuries	110
Total	**216**
Reported rock falls with no injuries*	69

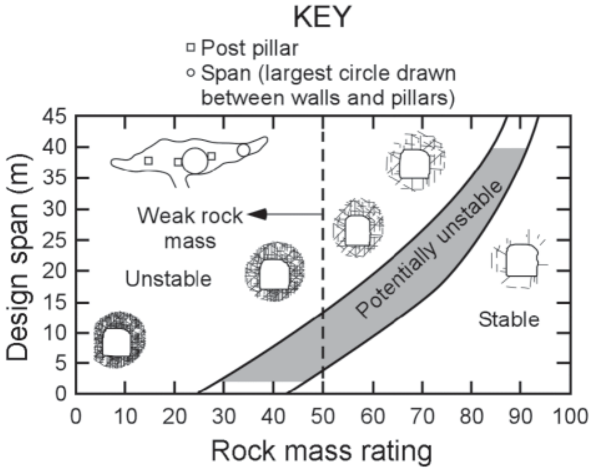
*Includes MSHA data for non-injury incidents where a reportable fall of ground occurred but did not cause injuries because the mining area was unoccupied.

A comparative analysis by the Mine Safety and Health Administration (MSHA)[9] for the years 1990–1999 indicated that the number of roof-fall injuries in Nevada has varied from a low of 8 in 1991 to a high of 28 in 1995 and 1997. As late as 1999, the number of injuries was still in double figures (Fig. 1). Analysis of the MSHA data shows 76·7% of the roof falls were from the back, 18·6% were rock falls from face or ribs with the remaining 4·7% of an unknown nature. The mines are required to report all injuries to MSHA by law.

Mining is a dynamic process, and ground conditions can change over a short distance. A mine opening must perform in a predictable manner over its expected life.



1 Injuries in Nevada, 1990–2004



2 Design span curve

Empirical design methods have been used successfully over the past 30 years largely because they permit the overall behaviour of a rock mass to be predicted easily and accurately. The basis for the success of the empirical method is a strong foundation of field data coupled with on-going field observations that allow changing rock conditions to be evaluated as mining progresses. Two systems were initially developed from case studies and databases originally derived from civil engineering applications and augmented by mine studies – the rock mass rating (RMR) system and the Q system. As stated by the author, 'the Q system is specifically the permanent lining estimation system for tunnels and caverns in rock and mainly for civil engineering projects'.[1]

There are several advantages for utilising the RMR system over the Q system. The RMR system has a relatively straight-forward scoring system for each parameter on a 0–10 scale and consequently is easier to learn.[10,11]

One of the original extensions of the rock mass rating system from civil into mining was conducted under a US Bureau of Mines Spokane Research Laboratory research grant which developed empirical methods for block caving operations for US copper mines.[7]

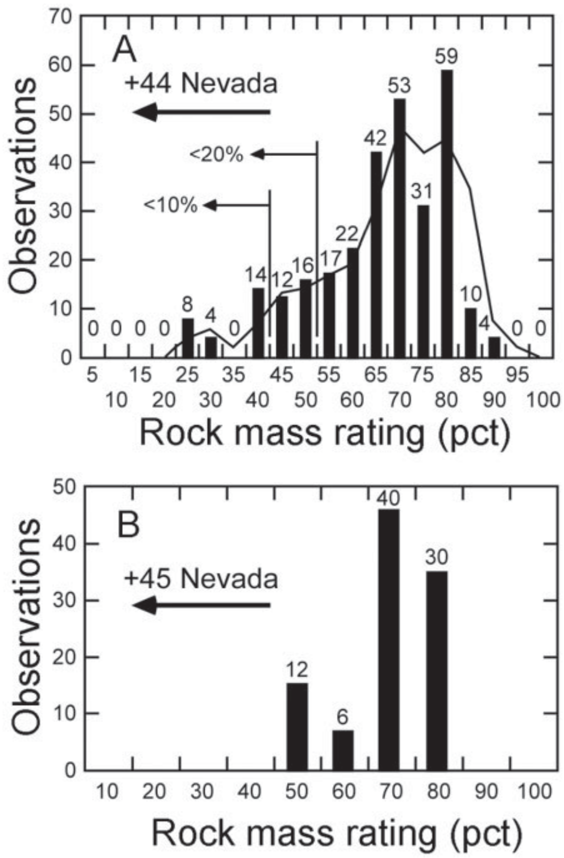
It is important to remember that any method of designing an opening must be easy to assess, understand, apply, modify if necessary, and reproduce for the next application if accepted as an on-going operational tool for design. A critical factor is that the design incorporates the degree of stability required for any mine entry.

SPAN DESIGN MAN-ENTRY METHODS

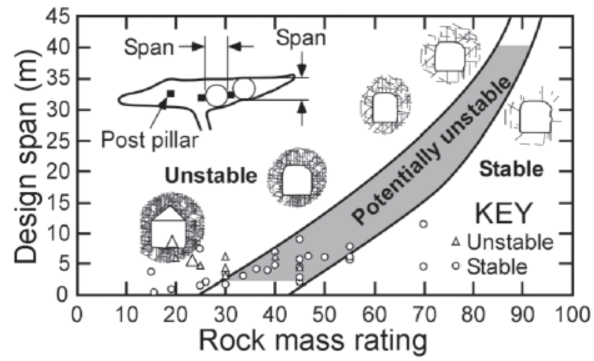
The 'critical span curve' was developed in 1994 to evaluate back stability in cut-and-fill mines.[13] In 2000, the span curve database of 172 observations developed by the University of British Columbia was expanded to include a total of 292 case histories from mines primarily in Canada.[21] The information from these case histories provides the basis for the span design curve shown in Figure 2.

Table 2 Nevada mines database: back spans in weak rock

RMR (%)	Span (m)	Condition	Other
Mine 1			
45	5·5	S	Stable with support
45	9·0	S	Stable with support
40	6·0	S	Stable with support
Mine 2			
40	4·0	S	Stable with support
45	4·3	U	Caved with support
30	3·7	S	Stable with support
Mine 3			
40	7·0	S	Stable with support
45	2·1	S	Stable with support
26	2·1	S	Stable with support
25	4·6	U	Caved with support
55	7·6	S	Stable with support
45	3·0	S	Stable with support
Mine 4			
70	4·6	S	Stable with support
40	4·6	S	Stable with support
25	4·6	S	Stable with support
55	5·5	S	Stable with support
30	6·1	S	Caved upon longhole
30	6·1	S	Caved upon longhole
45	4·6	S	Stable with support
50	6·1	S	Stable with support
70	11·3	S	Stable with support
25	7·3	S	Stable with support
30	3·0	U	Prior to support placement
30	1·8	S	Prior to support placement
50	6·1	S	Stable with support
55	7·6	S	Stable with support
55	6·1	S	Stable with support
Mine 5			
30	3·0	S	Stable with support
30	4·3	U	Caved with support
20	5·8	U	Caved with support
15	3·7	S	Stable with support
Mine 6			
45	4·3	S	Stable with support
40	6·1	U	Caved with support
40	4·9	S	Stable with support
Mine 7			
40	4·6	S	Stable with support
35	4·6	S	Stable with support
Mine 8			
25	5·0	U	Caved; had to spile
20	1·2	S	No support; maximum round possible
25	2·4	S	No support; maximum round possible
35	3·1	S	No support; maximum round possible
55	3·7	S	No support; typical round; no problems
35	4·6	S	No support; typical round; no spile/shotcrete
20	7·6	U	Caved; had to spile
45	6·0	S	Stable with split-sets only



3 Distribution of RMR. (A) Span database;[13] (B) stability graph database (ELOS)[6]



4 Augmented span curve. Numbers in key correspond to mine numbers in Table 2. Letters indicate location on the span curve

A 'critical span' is defined as the diameter of the largest circle that can be drawn within the boundaries of the exposed back. The stability of this exposed span is related to the type of rock in the immediate back. The 'design span' refers to backs that have no support and/or spans that are supported with localised pattern bolting (1·8-m long mechanical bolts on 1·2 × 1·2-m spacings). Local support is deemed as support used to confine blocks that may be loose or that might open or fall because of subsequent mining in surrounding areas. Excavation stability is classified into three categories:

(i) **Stable excavation**: (a) no uncontrolled falls of ground; (b) no movement of the back is observed; and (c) no extraordinary support measures have been employed.

(ii) **Potentially unstable excavation**: (a) extra ground support has been installed to prevent potential falls of ground; (b) movement has occurred in the back; and (c) increased frequency of ground movement has been observed.

(iii) **Unstable excavation**: (a) area has collapsed; (b) depth of failure of the back is 0·5 times the span (in the absence of major structures); and (c) support was not effective in maintaining stability.

A –10 correction factor is applied to the final RMR_{76} value when evaluating rock with shallow-dipping or flat joints. However, the applicability of this factor is being re-assessed for weak ground because of its amorphous nature and because joint direction is expected to play a minor role. Where discrete ground wedges have been observed and identified, they must be supported prior to employing the critical span curve.

Stability is generally defined in terms of short-term stability because the database is based largely on stoping methods that, by their nature, are of short duration. Movement of the back greater than 1 mm within a 24-h period has been defined as a critical amount of movement for safe access.[18]

Some 44 case histories from five different mines with RMR_{76} values varying between 20 and 85 were added to the information base for the critical span curve (Table 2). Several values were less than 55% RMR_{76}; the lowest RMR_{76} value calculated for any location was 25%. This information was used to augment the original 'span design curve for man-entry mining[13] as shown in Figure 3A. The span curve enables an operator to assess back stability with respect to the rock mass. The information has been used successfully to predict the stability of weak rock masses and has provided operators with an additional design tool for making decisions concerning the stability of mine openings. The data are being coupled with depth of failure to define the amount of support required to arrive at a safe, cost-effective man-entry design. A brief description of the use of the critical span curve is presented; however, the reader is referred to the detailed reference as outlined by Pakalnis.[18]

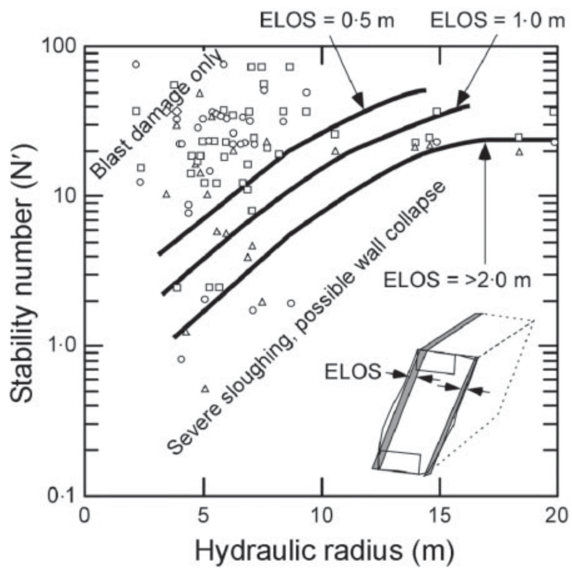
STABILITY GRAPH METHOD, NON-ENTRY MINING

The original stability method for open stope design was based largely on Canadian operations from 55 case histories which included 48 back studies and 7 wall data points.[15] The original database of case histories exhibited a rock mass rating (RMR_{76}) in excess of 50% or Q values of 2·0 or greater.[4] The method was extended and modified by Potvin[19] based upon data from 34 mines with 175 open stope case histories and 67 cases of supported stopes. Nickson[17] expanded the existing database of supported stopes by Potvin, by collecting 46 cases histories while visiting 13 mines. In

Table 3 Nevada mines database: stope spans and walls in weak rock (all values of *A* = 1)

RMR (%)	Dimensions, height × length (m)	Dip	*B*	*C*	*N*	HR (m)	ELOS (m)	Comments
Mine 1								
45	20 × 17	90	0·3	8	2·7	4·6	<1·0	< 1 m of ELOS
40	20 × 16	90	0·3	8	1·5	4·4	2·0	
55	49 × 18	90	0·3	8	8·1	6·6	1·0	
39	34 × 34	90	0·3	8	1·4	8·5	4·6	
25		90	0·3	8	0·3	1·8	< 1·0	< 1 m of ELOS stable (estimated)
34		90	0·3	8	0·3	3·4	< 1·0	< 1 m of ELOS stable (estimated)
42		90	0·3	8	0·3	2·8	< 1·0	< 1 m of ELOS stable (estimated)
Mine 2								
40	11 × 21	90	0·3	8	1·5	3·4	< 1·0	< 1 m of ELOS
50	11 × 21	90	0·3	8	4·7	3·4	< 0·5	< 0·5 m of ELOS
Mine 3								
55	18 × 18	70	0·2	5·9	4·0	4·6	0·6	
26	12 × 18	70	0·3	5·9	0·2	3·7	> 2·0	> 2 m of ELOS
Mine 4								
25	6 × 29	55	0·2	4·5	0·2	2·5	0·3	Cluster average, height/width
25	8 × 36	90	0·3	8	0·5	2·4	0·1	Cluster average, rib
25	17 × 12	90	0·3	8	0·5	3·5	0·1	Cluster average, rib
25	21 × 15	90	0·3	8	0·5	4·5	0·1	Cluster average, rib
55	30 × 26	90	0·3	8	7·2	7·0	0·1	Cluster average, rib
45	6 × 25	90	0·3	8	2·4	2·5	0·1	Cluster average, height/width
45	18 × 12	90	0·3	8	2·4	3·5	0·1	Cluster average, rib
45	19 × 16	90	0·3	8	2·4	4·4	0·1	Cluster average, rib
45	6 × 22	90	0·3	8	2·4	2·4	0·5	Moderate
25	16 × 5				0·5	3·2	0·5	Moderate
25	6 × 26				0·5	2·5	0·5	Moderate
45	6 × 22		0·3	8	2·4	2·4	0·5	Moderate
25	16 × 27	90	0·3	8	0·5	2·5	0·5	Moderate
25	6 × 20	90	0·3	8	0·5	2·3	0·6	Moderate
45	6 × 20	90	0·3	8	2·4	2·3	0·6	Moderate
25	6 × 20	90	0·3	8	0·5	2·3	0·6	Moderate
25	6 × 24	90	0·3	8	0·5	2·4	0·9	Moderate
35	6 × 25	90	0·2	4·8	2·4	2·4	1·0	Moderate
25	15 × 13	90	0·3	8	0·5	3·5	> 2·0	Caved visually, estimated < 2 m
25	20 × 15	90	0·3	8	0·5	44	> 2·0	Caved visually, estimated < 2 m
25	21 × 15	90	0·3	8	1·0	4·4	> 2·0	Caved visually, estimated < 2 m
25	15 × 13	90	0·3	8	0·5	3·5	> 2·0	Caved visually, estimated < 2 m
25	20 × 15	90	0·3	8	0·5	4·4	> 2·0	Caved visually, estimated < 2 m
25	21 × 15	90	0·3	8	0·5	3·8	> 2·0	Caved visually, estimated < 2 m
25	21 × 10	90	0·3	8	0·5	3·5	1·5	Caved visually, estimated < 2 m
25	22 × 14	90	0·3	8	0·5	4·4	1·5	Caved visually, estimated < 2 m
25	23 × 12	90	0·3	8	0·5	4·4	1·5	Caved visually, estimated < 2 m
25	21 × 10	90	0·3	8	0·5	3·5	1·5	Failed visually, estimated 1–2 m
25	22 × 14	90	0·3	8	0·5	4·3	1·5	Failed visually, estimated 1–2 m
25	23 × 12	90	0·3	8	0·5	3·8	1·5	Failed visually, estimated 1–2 m
25	19 × 13	90	0·3	8	0·5	3·	1·5	Failed visually, estimated 1–2 m
25	19 × 13		0·3	8	0·5	3·8	1·5	Failed visually, estimated 1–2 m
25	22 × 15		0·3	8	0·5	4·4	1·5	Failed visually, estimated 1–2 m
25	19 × 13		0·3	8	0·5	3·9	1·5	Failed visually, estimated 1–2 m
25	19 × 13		0·3	8	0·5	3·8	1·5	Failed visually, estimated 1–2 m
25	22 × 15		0·3	8	0·5	4·4	1·5	Failed visually, estimated 1–2 m
Mine 6								
45					2·6	4·4	< 0·5	Typical stope
45					2·6	6·2	1·8	Caved stope

all instances, stability was qualitatively assessed as either being stable, potentially unstable, or caved. Research by Mah[14] and Clark and Pakalnis[6] at the University of British Columbia augmented the stability graph by using stope surveys in which cavity monitoring systems were employed. Mah's work added 96 data points onto Matthews's stability graph under mining conditions whereas Clark[5] added an additional 88 data points. This research has enabled quantification of the amount of wall slough. A parameter defined by Clark[5] as the equivalent linear overbreak/slough (ELOS; Fig. 5) was used to express volumetric measurements of overbreak as an average depth over the entire stope surface. ELOS is defined as



5 Stability graph

the volume of slough from the stope surface divided by the product of stope height times wall strike length known as the hydraulic radius (HR).

$$\text{ELOS} = \frac{\text{Volume of Slough}}{\text{HR}} \tag{1}$$

The stability graph relates hydraulic radius (HR) of the stope wall to empirical estimates of overbreak slough. Additionally, the database for the Matthews' method has been augmented by the addition of 400 case histories which includes open stoping experiences for a broad range of rock mass conditions in Australia specifically at the Mount Charlotte mine. The additional case histories include much larger stopes and extend the modified stability graph to a hydraulic radius of 55 m as compared to the previous maximum value of 20 m.[16] The drawback of this study was that it was largely qualitative because surveys were not available to measure the wall slough.

A limited number of observations existed for RMR_{76} values under 45% (Fig. 3B). An additional 45 data points were added on the stability graph–non-entry from Nevada operations having an RMR_{76} under 45% (Fig. 3B and Table 3). In addition, mine 4 reflects over 338 observations that have been averaged to reflect the design points shown in Table 2. The stability graph relates hydraulic radius of the stope wall to empirical estimates of overbreak slough. Hydraulic radius (HR) is defined as the surface area of an opening divided by the perimeter of the exposed wall, analysed as shown in Figure 5.

$$N' = Q' \times A \times B \times C \tag{2}$$

where N′ is the modified stability number; Q′ is the Norwegian Geological Institute (NGI) rock quality index (after Barton *et al.*[2]), with stress reduction factor (SRF) and Jw = 1·0); A is high stress reduction; B is orientation of discontinuities; and C is the orientation of surface.

The stress reduction factor and joint water reduction factor are equal to 1, as they are accounted for separately within the analysis. For the ELOS graph points, the database for the stability graph was derived from mining operations that are generally dry.

The following relationship was used to convert RMR to Q′ (from Bieniawski[3]):

$$\text{RMR} = 9\text{Ln}Q' + 44 \tag{3}$$

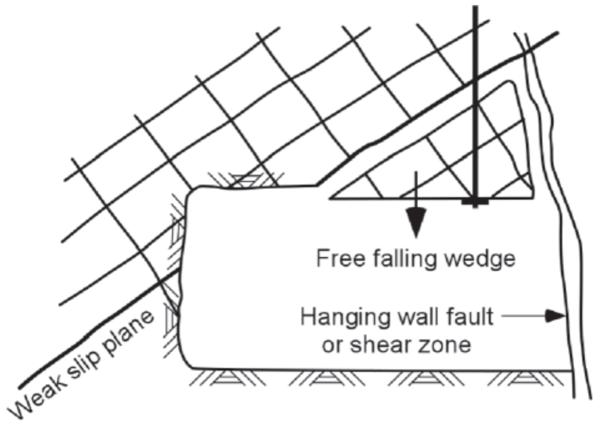
The A factor accounts for the influence of high stresses that reduce rock mass stability and is determined by the ratio of unconfined compressive strength of intact rock to maximum induced stress parallel to the opening surface. It is set to 1·0 if intact rock strength is 10 or more times induced stress, which indicates that high stress is not a problem. It is set to 0·1 if rock strength is two times induced stress or less, which indicates that high stress significantly reduces opening stability. In the mines visited for this study, the value of A was equal to 1·0 because the hanging wall was largely in a relaxed state.

The B factor looks at the influence of the orientation of discontinuities with respect to the surface analysed and states that joints oriented 90° to a surface do not create stability problems and the B factor is assessed a value of 1·0. Discontinuities dipping up to 20° to the surface are the least stable and represent geological structures that can topple. In this case, the B factor is equal to 0·2, which was the value used for the Nevada database. In extremely weak rock masses (RMR_{76} = 25%), the material largely resembles an amorphous mass with geological structures throughout; therefore, reduction due to jointing is suspect. The authors are presently analysing the data to augment this factor.

The C factor considers orientation of the surface being analysed and is assigned a value of 8 for the design of vertical walls and a value of 2 for horizontal backs. The C factor reflects the inherently more stable nature of vertical walls compared to a horizontal back. In this paper, the ELOS curves employ a value for C of 8·0 for the footwalls. For a more complete explanation of factors A, B, and C refer to papers by Potvin[19] and Clark and Pakalnis.[6]

SUPPORT GUIDELINES

The development of support capacity guidelines is critical to the overall success of the mining method selected in terms of ensuring a safe work place (see Fig. 2). Ground support in weak rock presents special challenges. Under-design can lead to costly failures, whereas over-design can lead to high costs for unnecessary ground support. Figure 6 depicts a classic wedge failure controlled by structure if the ground support has been under-designed. It is critical to design for the dead weight of the wedge in terms of the breaking load of the support, as well as the bond strength associated with embedment length.[8] Split-Set and Swellex have been defined as continuous friction coupled (CFC) non-grouted bolts by CSIRO.[24] The load transfer mechanism between the rock and borehole depends on the borehole characteristics (hole diameter and



6 Classic wedge failures

roughness) compared to the outside diameter of the bolt element. Independent tests conducted by NIOSH,[12] Tomory *et al*.,[20] and Villaescusa *et al*.[25] confirm that relationship for all diameter of bolts – 33-mm, 39-mm and 46-mm Split-Sets.

Over 400 000 Split-Set[8] friction bolts are used in Nevada mines for primary support. Friction bolts are particularly useful in fissile, buckling, or sheared ground where it is difficult to secure a point anchor. Caution must be used when using this method of primary support because of the low bond strength between the weak rock mass and the bolt and because of the susceptibility of the bolt to corrosion. In mine 4, Split-Set bolts had a life of 6 months because of corrosion resulting from acidic ground conditions. An analysis of the performance of friction bolts in mines with weak rock (as determined by RMR) needed to be addressed. With one exception, Nevada mines use 39-mm Split-Set bolts (the exception uses 46-mm Split-Set bolts); however, mines in Canada, generally use 33-mm Split-Set bolts. Canadian mines normally use these bolts only in the walls and not in the back. Table 4 shows an updated support capacity chart as augmented by this study.

Data points gathered from several pull tests in weak rock were plotted as shown in Figure 7. A neural

Table 4 Nevada mines database: back spans in weak rock

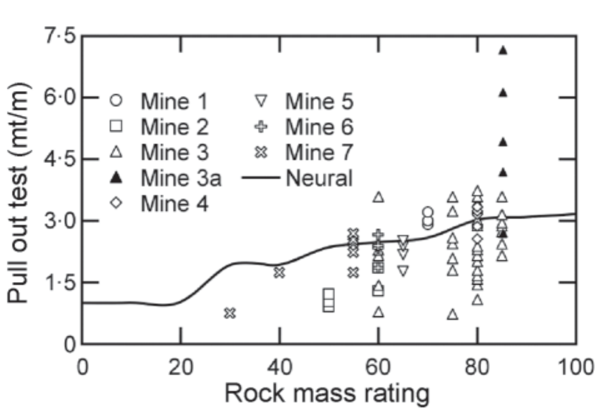
	Rock properties (t)	
Bolt strength	Yield strength	Breaking strength
5/8-inch mechanical	6·1	10·2
Split-Set (SS 33)	8·5	10·6
Split Set (SS 39)	12·7	14·0
Standard Swellex	NA	11·0
Yielding Swellex	NA	9·5
Super Swellex	NA	22·0
*20-mm rebar, No. 6	12·4	18·5
*22-mm rebar, No. 7	16·0	23
*25-mm rebar, No. 8	20·5	30·8
No. 6 Dywidag	11·9	18·0
No. 7 Dywidag	16·3	24·5
No. 8 Dywidag	21·5	32·3
No. 9 Dywidag	27·2	40·9
No. 10 Dywidag	34·6	52·0
1/2-inch cable bolt	15·9	18·8
5/8-inch cable bolt	21·6	25·5
1/4 × 4-inch strap	25·0	39·0

Note: No. 6 gauge = 6/8-inch diameter; No. 7 gauge = 7/8-inch diameter; No. 8 gauge = 1-inch diameter; NA = Not applicable.

Screen	Bag strength (t)
4 × 4-inch welded mesh, 4 gauge	3·6
4 × 4-inch welded mesh, 6 gauge	3·3
4 × 4-inch welded mesh, 9 gauge	1·9
4 × 2-inch welded mesh, 12 gauge	1·4
2-inch chain link, 11 gauge, bare metal	2·9
2-inch chain link, 11 gauge, galvanised	1·7
2-inch chain link, 9 gauge, bare metal	3·7
2-inch chain link, 9 gauge, galvanised	3·2

Note: 4 gauge = 0·23-inch diameter; 6 gauge = 0·20-inch diameter; 9 gauge = 0·16-inch diameter; 11 gauge = 0·125-inch diameter; 12 gauge = 0·11-inch diameter.
Shotcrete shear strength = 2 MPa (200 t m^{-2}).

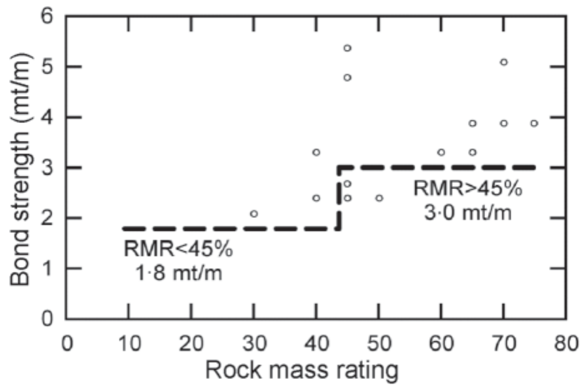
	Bond strength
Split-Set, hard rock	0·75–1·5 Mt per 0·3 m
Split-Set, weak ground	0·25–1·2 Mt per 0·3 m
Swellex, hard rock	2·70–4·6 Mt per 0·3 m
Swellex, weak rock	3-3·5 Mt per 0·3 m
Super Swellex, weak rock	> 4 Mt per 0·3 m
5/8-inch cable bolt, hard rock	26 Mt per 1 m
No. 6 rebar, hard rock	18 Mt per 0·3 m, ~12-inch granite



7 Rock mass rating versus pull-out strength. A neural trend line is superimposed

net[23] was superimposed on the mine data to determine if trends or predictions could be made. The neural net methodology has been used in establishing design curves for span and stope design by Wang *et al*.[22] The graph shows a strong trend between RMR and bond strength; this relationship is being assessed as part of on-going research. Preliminary results are shown in Figures 7 and 8.

Variability in test results shows the difficulty in assessing overall support for a given heading. Thus, it is important that mines develop a database with respect to the support used so as to design for variable ground conditions. Factors critical to design, such as bond strength, hole size, support type, bond length, and RMR, should be recorded so as to determine their influence on the design curve to be determined.



8 Pull-out load versus RMR at mine 4

CONCLUSIONS

The Spokane Research Laboratory and the University of British Columbia Geomechanics group are focusing on the development of safe and cost-effective underground design guidelines for weak rock masses having an RMR in the range 15–45%. Weak ground conditions, ground support, and mining methods used in several Nevada underground mines were observed. The RMR values were calculated to update both span design calculations and stability graphs, and a database on underhand mining methods was developed to reflect existing Nevada mining conditions. The immediate rock mass was also characterised and analysed in terms of prevailing type of ground support, potential failure mechanisms, and rock behaviour.

Variability in field conditions shows the difficulty in assessing overall support for a given heading. It is imperative that mines develop their own databases based on the type of support used in their mines so unexpected ground conditions can be analysed. The results from augmented design curves and pull-out tests are presented in the hope that they will aid mine professionals in the task of designing a safe workplace. A systematic approach allows an operator to understand overall failure mechanisms and resultant loads that could affect the system. This approach would allow an engineer to develop an optimal support strategy for the mining method employed.

The work would not have been possible without the partnership between NIOSH, the University of British Columbia Geomechanics Group and the Nevada gold mining company personnel. This continued partnership is critical to the development of safe and cost effective mine strategies. Figure 2 shows that since the inception of the ‘team’ approach and resultant collaboration that injury statistics have declined dramatically. The cause and effect may be a result of many factors; however, it is clear that this approach is important and relevant to mine operations.

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