DESIGN IN WEAK ROCK MASSES: NEVADA UNDERGROUND MINING OPERATIONS

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Abstract

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 data on weak rock masses into existing design relationships, with an emphasis on updating the span design curve for manned entries and the overbreak curve for longhole entries. Both curves were originally developed at the University of British Columbia, Vancouver, BC. The original database has been augmented by information from mines throughout the United States, Canada, Australia, and Europe. The common factor in all these mines is the presence of a weak back and/or walls. In most cases, the ore zone is the weakest rock unit and must be stabilized so that the mineral-bearing rock can be extracted safety. The current NIOSH research attempts to provide rock mechanics tools to assist a mine operator in making economic decisions that will also ensure a safe working environment. This paper documents the Nevada database with a special emphasis on Nevada underground gold mines.

Introduction

Many of the underground gold mines in Nevada are found in very weak ground that creates difficult and hazardous mining conditions. A comparative analysis by the Mine Safety and Health Administration (MSHA) for the years 1990 through 2004 (figure 1) shows that the number of injuries from roof falls in 13 Nevada underground gold mines has varied from a low of eight in 1990 to a high of 28 in both 1995 and 1997 (3). This high injury rate was the prime motivator for initiation of the present study by NIOSH. The goal is to address the extremely difficult ground conditions associated with mining in a weak rock mass and provide mine operators with a database that could lead to a better understanding of the mechanics of these conditions.

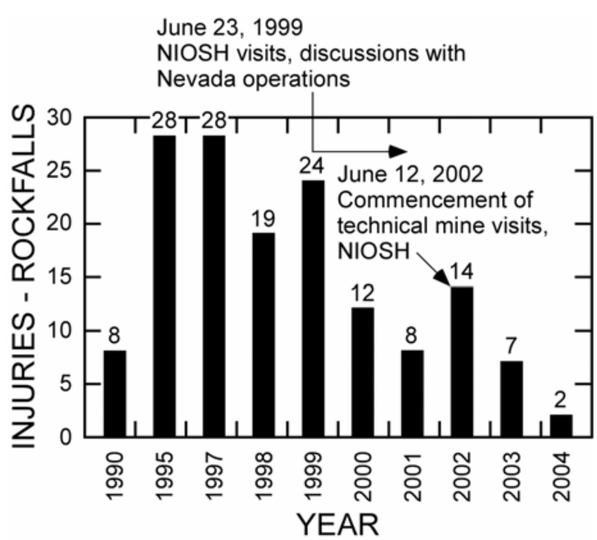

Figure 1.—Injuries from rockfalls in Nevada underground mining operations (after MSHA [3])

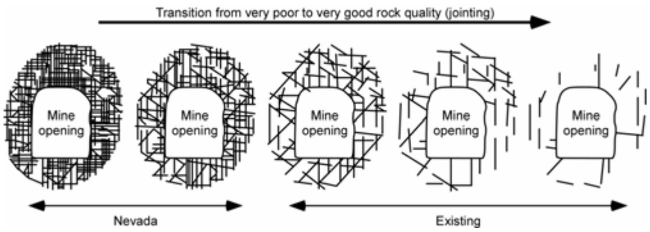

Figure 2.—Schematic showing jointed rock masses associated with Nevada

Figure 2 shows schematically that existing databases are based largely on stronger, less-fractured or altered rock masses than those found in Nevada mines. While design methods for excavating in a weak rock mass do exist (2), they are geared to caving operations.

The primary mining method used in Nevada is a form of mechanized cut-and-fill. This method is amenable to achieving a high rate of recovery when mining irregular ore geometries. Variations on this method are practiced at different mines and within different areas of the same operation, depending upon rock quality. Underhand cut-and-fill mining (3) is employed and larger spans are required under a weak back where the costs of installing roof support would be high. For purposes of this study, a rock mass is considered to be weak if it has a rock mass rating (RMR) (4) of less than 45% and/or a rock mass quality rating (Q) under 1.0.

In areas where rock quality allows, the mines employ a version of longhole open stoping (5). Where spans from the footwall to the hanging wall exceed 7.6 m, transverse open stoping is employed with the use of primaries and secondaries adjacent to previously cemented rockfill panels.

Underground mining methods as practiced in Nevada dictated which specific databases and stope design curves NIOSH would focus upon. Rock mass values were calculated during mine visits and varied from an RMR high of 70% and a low of 16% in gold-bearing fault gouge.

Several rock mass design curves developed by the rock mechanics group at the University of British Columbia (6) are available, but they were not thought to be relevant to the mining methods employed within the weak ground of Nevada gold mines and therefore were not updated for weak ground.

Research commenced with visits to Nevada operators in June 1999 to address concerns and determine where NIOSH would be able to assist. The first technical site visit was on June 12, 2002, and initial data were collected (figure 1). The major objectives were to obtain information on weak rock masses and incorporate this information into existing design curves (7) for back spans of manned entries and a stability graph (8) for longhole wall design.

The distribution of the original databases was based on Canadian mining data, as summarized in figure 3, and shows how few data exist for weaker rock masses.

Span Design, Man Entry

The initial span curve was developed by the geomechanics group at the University of British Columbia to evaluate back stability in cut-and-fill mines. It consists of two straight lines that divide a graph into three zones: stable, potentially unstable, and unstable. The database for this graph initially consisted of 172 data points from the Detour Lake Mine of Placer Dome, Inc., in Ontario, with most of the points having RMR values in excess of 60% (7). The database was expanded to 292 observations in the year 2000 with case histories from an additional six mines (9).

The successful use of empirical design techniques is based upon interpolation rather than extrapolation. Thus a decision was made to develop a database for a critical span curve in weak rock masses. The term “critical span” refers to

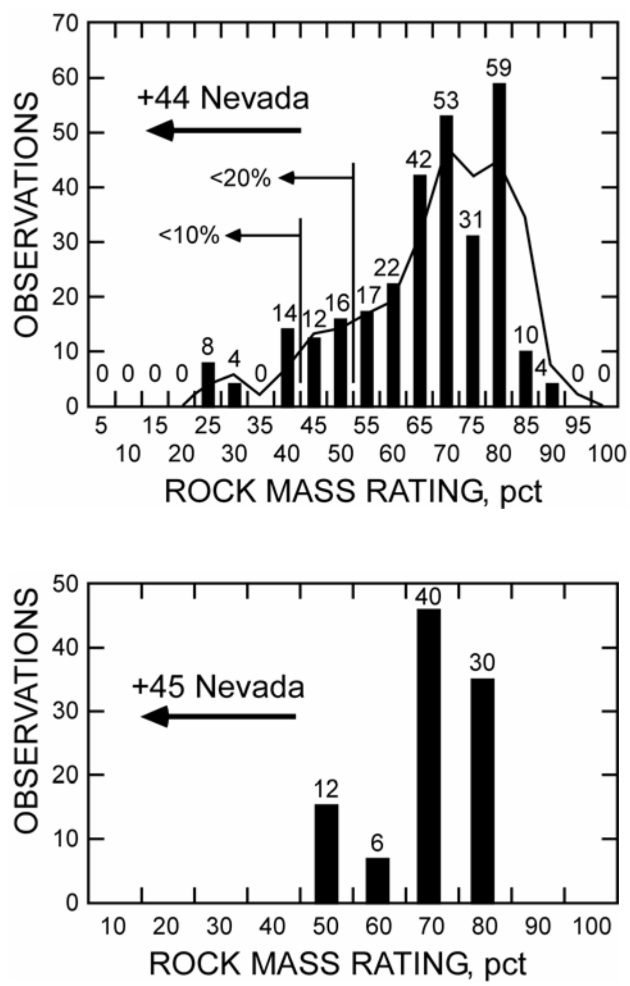

Figure 3.— Distribution of original database for back span design (top) (7) and stability graph (bottom) (8)

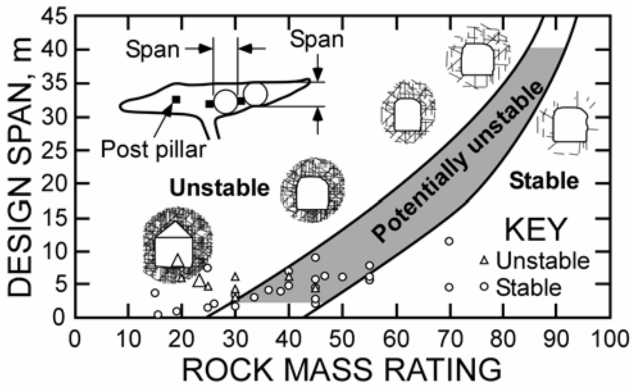

Figure 4.—Critical span curve adjusted to Nevada operations

the largest circle that can be drawn within the boundaries of the excavation when seen in plan view (figure 4). The term "design span" refers to spans that have no support and/or spans incorporating a limited amount of local support (for example, pattern bolting in which 1.8-m-long mechanical bolts are installed on a on a 1.2- by 1.2-m pattern). 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. The new database included RMR values from 24% to 87% with 63% of the cases over 60%. Less than 10% of the RMR values fell below 45%, and less than 20% fell below 55% (10). An additional 44 observations were added to the critical span curve from Nevada operations (figure 4; table 1), of which 35 had an RMR less than 45%.

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 (6).

Excavation stability is classified into three categories; each category is further divided into three subcategories.

1. Stable excavation (S)
   a. No uncontrolled falls of ground have occurred.
   b. No movement of the back has been observed.
   c. No extraordinary support measures have been employed.
2. Potentially unstable excavation.
   a. Extra ground support has been installed to prevent falls of ground.
   b. Movement has occurred in the back.
   c. Increased frequency of ground movement has been observed.
3. Unstable excavation (U)
   a. Area has collapsed.
   b. Depth of failure of the back is 0.5 times the span (in the absence of major structures). Within a weak rock mass, the depth of failure has been noted as one times the span and sometimes even greater.
   c. Limited local support was not effective in maintaining stability.

Table 1.—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	Stale 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

A minus-10 correction factor is applied to the final RMR when evaluating rock with shallow-dipping or flat joints. However, the applicability of this factor in weak ground is being reassessed because of its amorphous nature. Where discrete ground wedges have been 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-hour period has also been defined as a critical amount of movement for safe access (6). This value is also being addressed for weak rock masses as it applies to the initial database identified in figure 2. This critical value may be much greater than 1 mm.

Stability Graph Method–Nonentry

The original stability method for open stope design was based largely on Canadian operations and was proposed in 1981 by Mathews (11), modified in 1988 by Potvin (12), and updated in 1992 by Nickson (13). In all instances, stability was qualitatively assessed as either being stable, potentially unstable, or caved. Recent research at the University of British Columbia has augmented the stability graph by using stope surveys in which cavity monitoring systems were employed (8). This research has enabled the amount of dilution to be quantified. A parameter termed the "equivalent linear overbreak/slough" (ELOS) was introduced by Clark (8) and was used to express volumetric measurements of overbreak as an average depth over an entire stope surface. This has resulted in a design curve as shown in figure 5.

A limited number of observations existed for RMR values under 45% (figure 3-bottom). An additional 45 data points were added on the stability graph–nonentry from Nevada operations having an RMR under 45% (figure 6; table 2). 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 is defined as the surface area of an opening divided by perimeter of the exposed wall being analyzed.

The following equation was employed for calculation of parameters for the database shown in figure 5.

$$N' = Q' * A * B * C$$

where N' = modified stability number,

Q' = modified NGI rock quality index (14) where the stress reduction factor and joint water reduction factor are equal to 1, as they are accounted for separately within the analysis,

A = stress factor equal to 1.0 due to relaxed hanging wall,

B = rock defect factor. This value results from parallel jointing and the amorphous state of the weak rock mass being set to 0.2 and 0.3, respectively, as in table 3.

and C = stope orientation factor as defined in figure 5, that is, $C = 8 - 6 \times \cos \varphi$ (dip of hanging wall).

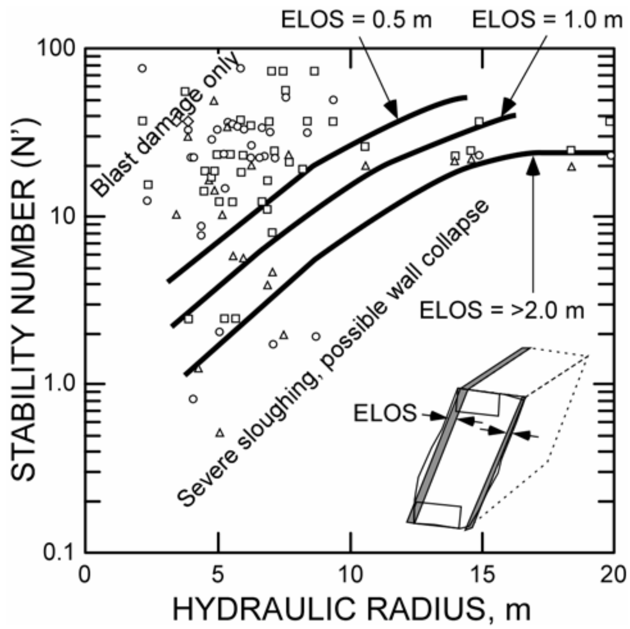

Figure 5.—Stability graph (after 8)

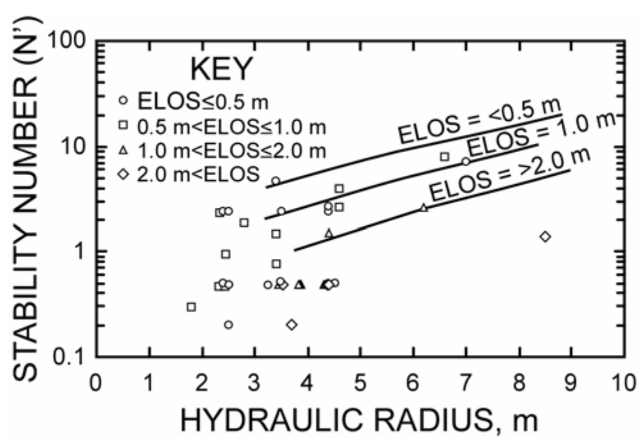

Figure 6.—Wall stability graph as developed for Nevada operations

Table 2.—Nevada mines database: Stope spans and walls in weak rock. All values of A = 1.

RMR (%)	Dimensions, height by length (m)	Dip	B	C	N	HR (m)	ELOS (m)	Comments
Mine 1:								
45	20 by 17	90	0.3	8	2.7	4.6	<1.0	<m of ELOS
40	20 by 16	90	0.3	8	1.5	4.4	2.0	
55	49 by 18	90	0.3	8	8.1	6.6	1.0	
39	34 by 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 by 21	90	0.3	8	1.5	3.4	<1.0	<1 m of ELOS
50	11 by 21	90	0.3	8	4.7	3.4	<0.5	<0.5 m of ELOS
Mine 3:								
55	18 by 18	70	0.2	5.9	4.0	4.6	0.6	
26	12 by 18	70	0.3	5.9	0.2	3.7	>2.0	>2 m of ELOS
Mine 4:								
25	6 by 29	55	0.2	4.5	0.2	2.5	0.3	Cluster average, height/width
25	8 by 36	90	0.3	8	0.5	2.4	0.1	Cluster average, rib
25	17 by 12	90	0.3	8	0.5	3.5	0.1	Cluster average, rib
25	21 by 15	90	0.3	8	0.5	4.5	0.1	Cluster average, rib
55	30 by 26	90	0.3	8	7.2	7.0	0.1	Cluster average, rib
45	6 by 25	90	0.3	8	2.4	2.5	0.1	Cluster average, height/width
45	18 by 12	90	0.3	8	2.4	3.5	0.1	Cluster average, rib
45	19 by 16	90	0.3	8	2.4	4.4	0.1	Cluster average, rib
45	6 by 22	90	0.3	8	2.4	2.4	0.5	Moderate
25	16 by 5				0.5	3.2	0.5	Moderate
25	6 by 26				0.5	2.5	0.5	Moderate
45	6 by 22		0.3	8	2.4	2.4	0.5	Moderate
25	16 by 27	90	0.3	8	0.5	2.5	0.5	Moderate
25	6 by 20	90	0.3	8	0.5	2.3	0.6	Moderate
45	6 by 20	90	0.3	8	2.4	2.3	0.6	Moderate
25	6 by 20	90	0.3	8	0.5	2.3	0.6	Moderate
25	6 by 24	90	0.3	8	0.5	2.4	0.9	Moderate
35	6 by 25	90	0.2	4.8	2.4	2.4	1.0	Moderate
25	15 by 13	90	0.3	8	0.5	3.5	>2.0	Caved visually, estimated < 2 m
25	20 by 15	90	0.3	8	0.5	44	>2.0	Caved visually, estimated < 2 m
25	21 by 15	90	0.3	8	1.0	4.4	>2.0	Caved visually, estimated < 2 m
25	15 by 13	90	0.3	8	0.5	3.5	>2.0	Caved visually, estimated < 2 m
25	20 by 15	90	0.3	8	0.5	4.4	>2.0	Caved visually, estimated < 2 m
25	21 by 15	90	0.3	8	0.5	3.8	>2.0	Caved visually, estimated < 2 m
25	21 by 10	90	0.3	8	0.5	3.5	1.5	Caved visually, estimated < 2 m
25	22 by 14	90	0.3	8	0.5	4.4	1.5	Caved visually, estimated < 2 m
25	23 by 12	90	0.3	8	0.5	4.4	1.5	Caved visually, estimated < 2 m
25	21 by 10	90	0.3	8	0.5	3.5	1.5	Failed visually, estimated 1-2 m
25	22 by 14	90	0.3	8	0.5	4.3	1.5	Failed visually, estimated 1-2 m
25	23 by 12	90	0.3	8	0.5	3.8	1.5	Failed visually, estimated 1-2 m
25	19 by 13	90	0.3	8	0.5	3.	1.5	Failed visually, estimated 1-2 m
25	19 by 13		0.3	8	0.5	3.8	1.5	Failed visually, estimated 1-2 m
25	22 by 15		0.3	8	0.5	4.4	1.5	Failed visually, estimated 1-2 m
25	19 by 13		0.3	8	0.5	3.9	1.5	Failed visually, estimated 1-2 m
25	19 by 13		0.3	8	0.5	3.8	1.5	Failed visually, estimated 1-2 m
25	22 by 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

Table 3.—Nevada mines database: Underhand cut-and-fill

Mine	Amount of cement (%)	Span (m)	Sill thickness (m)	Unconfined compressive strength (MPa)	Comments
1	10.0	6.1	3	2.0	Paste
2a	6.5	7.6	4.6	5.5	Cemented rockfill
2b	8.0	9.1	4.6	6.9	Cemented rockfill design
2c	8.0	21.0	4.6	6.9	Mined remotely. No cave.
3	70	3.0	3.	4-12	Cemented rockfill
4a	9.0	13.7	4.0	8.3	Cemented rockfill test panel
4b	9.0	3.7	3.0	8.3	Cemented rockfill drift and fill
4c	9.0	7.3	3.0	8.3	Cemented rockfill panel
5	*7.0	2.7	3.0	3.4	Cemented rockfill
6	6.75	4.9	4.3	3.4	Cemented rockfill
7a	10.0	1.8	2.7	0.3	Paste (factor of safety = 1.5)
7b		2.4	2.7	0.5	
7c		3.0	2.7	0.7	
7d		3.7	2.7	1.0	
7e		4.3	2.7	1.4	7-day sample
7f		4.9	2.7	1.8	
7g		5.5	2.7	2.3	
7h		6.1	2.7	2.9	
8	7.0	4.6-6.1	4.6	5.5	Cemented rockfill
9	10.0	5.0	5.0	4.45	Cemented rockfill
10	12.8	6-9	6.0	2.0	High-density slurry (78% solids by weight)
11	10.0	3.0	3 (includes 0.9-m air gap)	2.5 (after 7 days)	10% cemented hydraulic fill (73%-75% solids by weight)
12	8.0	2.4-4.6	3 (includes 0.6-m air gap)	4.8	8% paste (no free water)

* Includes fly ash binder.

An initial observation from figure 6 is that the classical design curves (ELOS) as shown in figure 5 are inaccurate at low N' and hydraulic radius values. If hydraulic radius is kept below 3.5 m in a weak rock mass, the ELOS value should remain under 1 m. It appears a hydraulic radius under 3 m would not result in ELOS values much greater than 1 m. This result is being further evaluated.

Underhand Mine Design

As noted earlier, most mines use a form of cut-and-fill mining for a major portion of their production. As ground conditions become weaker, the primary method is mechanized underhand cut-and-fill. These methods assure a high degree of gold recovery under an engineered back of cemented rockfill and/or cemented paste fill (3). To achieve the most production at the lowest cost while maintaining a safe work environment, mine personnel were interested in developing support methods for the back that would best utilize either cemented rockfill or cemented paste fill. The result was the development of a new empirical database of successful underhand mining scenarios for the weak ground of Nevada.

A major study is also underway at the University of British Columbia to develop design guidelines for mining under paste backfill. Figure 7 is adapted from Stone (15) and is largely based on fixed-beam bending as the critical failure mechanism. The guidelines relate material properties to the thickness of the beam, the exposed span, and the resultant unconfined compressive strength required. A factor of safety in excess of 2.0 is incorporated. The analytical relationship was augmented by underhand cut-and-fill and shows that actual mine design parameters generally exceed those required by simple beam theory. The database is shown in figure 7 and summarized in table 3.

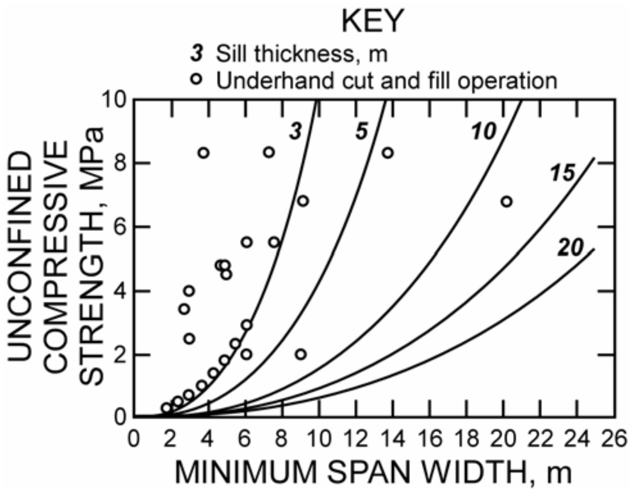

Figure 7.—Underhand mine design curve relating span, sill thickness, and fill strength

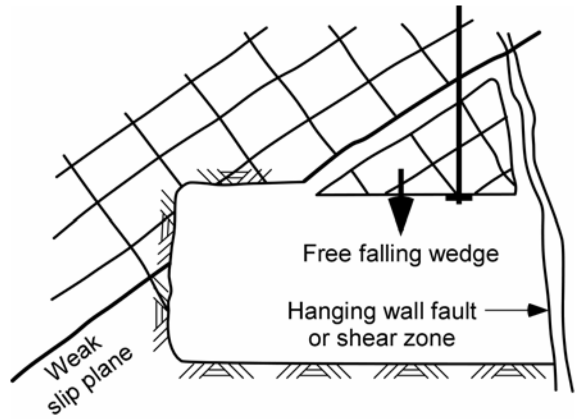

Figure 8.—Structurally controlled wedge

Support Capacity 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 figure 2). Ground support in weak rock presents special challenges. Underdesign can lead to costly failures, whereas overdesign can lead to high costs for unneeded ground support. Figure 8 depicts a classic wedge failure controlled by structure. 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 (10).

Over 400,000 Split-Set (10) friction bolts are used in Nevada mines as 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 broken rock 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); mines in Canada, however, use 33-mm Split-Set bolts. Canadian mines generally 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 9. 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 figure 10.

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 they can 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 enable where they lie on the design curve to be determined.

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 of 15% to 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 characterized and analyzed in terms of prevailing type of ground support, potential failure mechanisms, and rock behavior.

Variability in field conditions show 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 analyzed.

Table 4.—Nevada mines database: Back spans in weak rock

Rock properties, tonnes		
Bolt strength	Yield strength	Breaking strength
5/8-in 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.o
*20-mm rebar, No. 6	12.4	18.5
*22-mm rebar, No. 7	16.o	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-in cable bolt	15.9	18.8
5/8-in cable bolt	21.6	25.5
1/4 by 4-in strap	25.o	39.0

Note: No. 6 gauge = 6/8-in diameter.; No. 7 gauge = 7/8-in diameter.; No. 8 gauge = 1-in diameter.

NA = Not applicable.

Screen	Bag strength, tonnes
4- by 4-in welded mesh, 4 gauge	3.6
4- by 4-in welded mesh, 6 gauge	3.3
4- by 4-in welded mesh, 9 gauge	1.9
4- by 2-in welded mesh, 12 gauge	1.4
2-in chain link, 11 gauge, bare metal	2.9
2-in chain link, 11 gauge, galvanized	1.7
2-in chain link, 9 gauge, bare metal	3.7
2-in chain link, 9 gauge, galvanized	3.2

Note: 4 gauge = 0.23-in diameter; 6 gauge = 0.20-in diameter; 9 gauge = 0.16-in diameter; 11 gauge = 0.125-in diameter; 12 gauge = 0.11-in 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-in cable bolt, hard rock	26 mt per 1 m
No. 6 rebar, hard rock	18 mt per 0.3 m, ~12-in granite

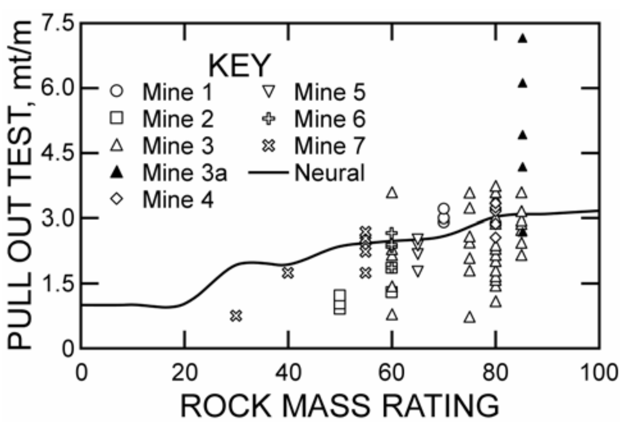

Figure 9.— Pull-out load versus RMR for SS39.

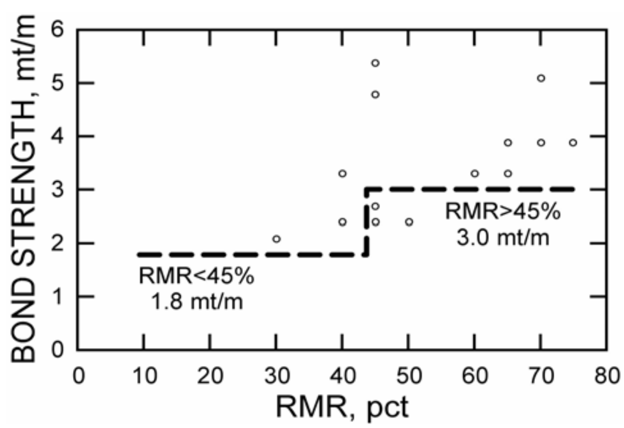

Figure 10.— Pull-out load versus RMR at mine 4

The results from augmented design curves and pull-out tests are presented in the hope that they will aid mine professionals in their 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 Nevada gold mining company personnel. This continued partnership is critical to the development of safe and cost-effective mine strategies. Figure 1 shows that since the inception of the team approach and resultant collaboration, injury statistics have declined dramatically. This decline may be a result of many factors; however, it is clear that this approach is important and relevant to mine operations.

Acknowledgments

Specific thanks go to Patrick Carroll, chief engineer, Newmont Midas Mine; J.J. Hunter, chief rock mechanics engineer, Newmont East Carlin and Deep Post Mine; Monica Dodd, chief engineer, Newmont Deep Post Mine; Simon Jackson, chief engineer, Placer Dome, Turquoise Ridge Joint Venture; Shawn Stickler, chief ground control engineer, Queenstake Murry Mine; and Rad Langston, senior rock mechanics engineer, Stillwater Mining Company.

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