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CAVING MINE ROCK MASS CLASSIFICATION AND SUPPORT ESTIMATION

VOLUME I – FINAL REPORT

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16. Abstract (Limit: 200 words)

A rock mass classification system has been developed for recommending drift support requirements in block caving mines. This development follows a two-year program incorporating field observations of support practice and performance in rocks of different classes, at several U. S. block caving mines. Popular rock mass classification systems presently in use for tunneling were applied at the various field sites and were evaluated for conformance with caving mine support requirements and for general utility to caving mine operations. The developed system (MBR System) accepts readily-obtainable engineering and geologic data to yield a rational estimate of support requirements. The background and development of the MBR system is described in this report (Volume I). The use of the system is given in the Manual, a separate volume (Volume II).

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FOREWORD

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This report is accompanied by a separate second volume (Volume II), entitled "Caving Mine Rock Mass Classification and Support Estimation--A Manual," giving instructions on the use of the rock mass classification system developed under Contract No. J0100103.

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TABLE OF CONTENTS

REPORT DOCUMENTATION PAGE 3
FOREWORD 4
TABLE OF CONTENTS 5
LIST OF FIGURES 8
LIST OF TABLES 11
LIST OF SYMBOLS 13
EXECUTIVE SUMMARY 15
1.0 INTRODUCTION 17
1.1 Scope of the Project 17
1.2 Methodology 18
1.3 Previous Work and Parallel Studies 20
1.4 Report Content 21
2.0 ROCK MASS CLASSIFICATIONS 23
2.1 Basic Concepts 23
2.2 RQI Systems Considered in This System 24
2.2.1 RSR Concept 24
2.2.2 Q-System 29
2.2.3 RMR (Geomechanics) System 32
2.2.4 Laubscher's Geomechanics System for Mining . 37
2.2.5 Abel's Tunnel Support Nomographs 44
2.2.6 McDonough System 46
2.2.7 Correlations Between Systems 47
2.2.8 Sensitivity Analysis 48
3.0 BLOCK AND PANEL CAVE MINING 51
3.1 Basic Description 51
3.2 Variations of the Method 53
4.0 KEY PARAMETERS AFFECTING DRIFT SUPPORT 58
4.1 Geologic Variables 58
4.1.1 Basic Rock Mass Class 58
4.1.2 Field Stresses 58
4.1.3 Structural Setting 59
4.2 Geometric Variables 59
4.2.1 Distance to Cave Line 59
4.2.2 Extraction Ratio 60
4.2.3 Panel Size 60
4.2.4 Depth 60
4.2.5 Location Within Panel 60
4.3 Development/Operational Variables 61
4.3.1 Excavation Practice 61
4.3.2 Necessary Standage Time 61
4.3.3 Degree of Acceptable Repair 61
4.3.4 Mining Stresses 62
4.4 Uncertainties in the Assessment of Variables . . . 62

TABLE OF CONTENTS
(Continued)

5.0 FIELD DATA 63
5.1 Mine A 63
5.1.1 Geologic Overview 63
5.1.2 Mining Practice 63
5.1.3 Ground Support Systems 65
5.1.4 Scope of Field Data 66
5.1.5 Rock Mass Conditions 67
5.1.6 Encountered Support Performance 67
5.1.7 Summary 70
5.2 Mine B 71
5.2.1 Geologic Overview 71
5.2.2 Mining Practice 72
5.2.3 Ground Support Systems 74
5.2.4 Scope of Field Data 75
5.2.5 Rock Mass Conditions 75
5.2.6 Support Performance 75
5.2.7 Summary 78
5.3 Mine C 78
5.3.1 Geologic Overview 78
5.3.2 Mining Practice 79
5.3.3 Ground Support Systems 82
5.3.4 Scope of Field Data 83
5.3.5 Rock Mass Conditions 83
5.4 Mine D 83
5.4.1 Geologic Overview 86
5.4.2 Mining Practice 87
5.4.3 Ground Support 90
5.4.4 Scope of Field Data 92
5.4.5 Rock Mass Conditions 93
5.4.6 Encountered Support Performance 95
5.4.7 Summary 97
5.5 Conclusions of the Field Work 97
5.5.1 Findings at Mine A 98
5.5.2 Findings at Mine B 98
5.5.3 Findings at Mine C 98
5.5.4 Findings at Mine D 98
5.5.5 Selection of RMR 103
6.0 CLASSIFICATION ADJUSTMENTS 105
6.1 Support Failure 105
6.2 Progression of Rock Mass Strains 106
6.2.1 Generalized Strain-Time/Distance Relation . 106
6.2.2 Dependence of Strain Zones on RMR 107
6.2.3 Effect of Strains on Lining Integrity . . . 116
6.3 Ratings and Adjustment Factors 126
6.3.1 First Step--MBR Rating 127
6.3.2 Second Step--Development Adjustments 134
6.3.3 Third Step--Permanent Support 144

TABLE OF CONTENTS
(Continued)

6.4 Support Recommendations at Studied Field Sites . . 149
6.4.1 Mine A 149
6.4.2 Mine B 160
6.4.3 Mine C 168
6.5 Evaluation of the Adjustment Process 178
6.5.1 Scope 178
6.5.2 Accuracy 178
7.0 ROCK MASS CLASSIFICATION FOR VEIN MINING 181
7.1 Nature of U. S. Vein Mining 181
7.2 Vein Mining Classification Requirements 182
7.3 Summary 185
8.0 AREAS FOR FURTHER INVESTIGATION 190
8.1 Cavability 190
8.2 Secondary Blasting 191
8.3 Draw Control 191
8.4 Drawpoint Spacing 192
8.5 Basic Research 192
9.0 REFERENCES 193
10.0 APPENDIX 198
Calculation of Extraction Ratio e_r

LIST OF FIGURES

Figure No.

1	Joint Spacing Ratings (Laubscher)	40
2	Typical Gravity Caving Scheme	54
3	Typical Slusher Drawing Scheme	55
4	Typical LHD Drawing Scheme	57
5	Main Effects of Undercutting	108
6	Field Observations in a Partially Undercut Block . . .	109
7	Generalized Rock Mass Behavior in the Production History of a Caving Area	110
8	Generalized Strain-Distance Relationships for Rock of Different Competence	117
9	RMR Values Leading to Concrete Lining Distress for Various Strain Zones	118
10	Dependence of Strain Zone Size on Rock Mass Competence	119
11	Predicted Lining and Rock Mass Behavior for Mines of Varying Competence	120
12	Panel Strains versus RMR	124
13	Support Prediction for Panels of Various Sizes	125
14	Organization of the MBR System	128
15	Rating for Intact Rock Strength	130
16	Theoretical Relationship between RQD and Discontinuity Spacing	131
17	Ratings for Discontinuity Density	132
18	Isolated Drift or Development Support Chart	135
19	Effective Extraction Ratios for 7-Foot-Wide Drifts for Various Degrees of Blasting Damage	137

LIST OF FIGURES
(Continued)

Figure No.

20 Effective Extraction Ratios for 10-ft-Wide Drifts for Various Degrees of Blasting Damage 138

21 Effective Extraction Ratios for 15-ft-Wide Drifts for Various Degrees of Blasting Damage 139

22 Adjustment A_S for Induced Stresses Due to Multiple Openings 141

23 Adjustment, DC, for Distance to Cave Line 147

24 Block/Panel Size Adjustment P_S 148

25 Permanent Support Chart 150

26 Geological Data, Mine A, Faulted Area 151

27 Geological Data, Mine A, Host Granite 152

28 Geological Data, Mine A, Intrusive Porphyry 153

29 Engineering Data, Mine A, Panel Drifts 155

30 Engineering Data, Mine A, Grizzly Drifts 156

31 Engineering Data, Mine A, Grizzly Level Fringe Drifts. 157

32 Engineering Data, Mine A, Haulage Drifts 158

33 Geological Data, Mine B, Very Weak Rock 161

34 Geological Data, Mine B, Competent Rock 162

35 Geological Data, Mine B, Very Competent Rock 163

36 Engineering Data, Mine B, Slusher Drifts 164

37 Engineering Data, Mine B, Fringe Drifts 165

38 Engineering Data, Mine B, Haulage Drifts 166

39 Geological Data, Mine C, AQM 169

LIST OF FIGURES
(Continued)

Figure No.

40	Geological Data, Mine C, HW	170
41	Geological Data, Mine C, QSF	171
42	Engineering Data, Mine C, Slusher Drifts	173
43	Engineering Data, Mine C, Grizzly Drifts	174
44	Engineering Data, Mine C, Haulage	175
45	Attempt to Apply the MBR Concept to Vein Mining . . .	183

LIST OF TABLES

Table No.		
1	Joint Condition Ratings	39
2	Ratings and Rock Mass Classes (Laubscher)	41
3	Total Possible Adjustments	42
4	Joint Orientation Adjustments (Laubscher)	43
5	Drift Support Requirements (Laubscher)	45
6	Geological Data for Mine A	68
7	RQI Ratings for Mine A	69
8	Geological Data for Mine B	76
9	RQI Ratings for Mine B	77
10	Geological Data for Mine C	84
11	RQI Ratings for Mine C	85
12	Geological Data for Mine D	94
13	RQI Ratings for Mine D	96
14	Predicted and Encountered Support for Mine A	99
15	Predicted and Encountered Support for Mine B	100
16	Predicted, Encountered, and Planned Support for Mine C	101
17	Predicted and Encountered Support for Mine D	102
18	Strain Data for Caving Mine Drift Strain Zones	111
19	Strain Zone Length Data for Caving Mine Drifts	114
20	Discontinuity Condition Ratings	133
21	Groundwater Condition Rating	133
22	Blasting Damage Adjustments A_B	140

LIST OF TABLES
(Continued)

Table No.

23 Fracture Orientation Rating A_0 Based on Direct Observation in Drift 143

24 Fracture Orientation Rating A_0 Based on Indirect Observation of Fracture Statistics 143

25 Major Structures Adjustment S 146

26 MBR and RMR Ratings for Mine A 154

27 Ratings and Adjustments for Mine A 159

28 MBR and RMR Ratings for Mine B . . . . 16[illegible]

29 Ratings and Adjustments at Mine B 167

30 MBR and RMR Ratings for Mine C 172

31 Ratings and Adjustments at Mine C 176

32 Geomechanics Classification in Coeur d'Alene District 186

33 Q-System Classification in Coeur d'Alene District . 187

LIST OF SYMBOLS

A_B	Blasting Damage Adjustment
A_O	Joint Orientation Adjustment
A_S	Induced Stresses Adjustment
AMBR	Adjusted Modified Basic RMR
AQM	Argillized Quartz Monzonite
B	Backfill
D	Tunnel Diameter
DC	Distance to Cave Line
e_r	Extraction Ratio
ESR	Equivalent Dimension
FMBR	Final Modified Basic RMR
G	Modulus of Rigidity
HCO	Haulage Cut-Outs
HW	Hanging Wall
IRS	Intact Rock Strength
J_a	Joint Alteration Number
J_n	Joint Set Number
J_r	Joint Roughness Number
J_w	Joint Water Reduction Factor
L	Length of Bolts
LHD	Load-Haul-Dump
MBR	Modified Basic RMR
PS	Block/Panel Size Adjustment
QSF	Quartz-Sericite Footwall
RQI	Rock Quality Indices
RQD	Rock Quality Designation
S	Major Discontinuities Adjustment
SRF	Stress Reduction Factor
t	Shotcrete Thickness
W	Rock Load
σ_1	Maximum Principal Stress
σ_3	Minimum Principal Stress
σ_v	Vertical Stress
σ_h	Horizontal Stress

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EXECUTIVE SUMMARY

A major part of this project used actual field data, as well as information in the literature, to assess the relationship between rock mass class and actual production support performance, for several U. S. block caving mines. Present rock classification systems were developed mostly from non-block-caving data and therefore are not appropriate for specifying drift support for typical caving rock masses. A classification system is proposed, based on observations of key determinants of drift behavior and collected data, that more usefully recommends drift support in block caving. The system, which allows manipulation of mine geometry and engineering factors, is chiefly intended for mine planning.

While an early, correct decision on drift support can be of great importance to mine operators, especially in view of the great capital (and pre-production) costs of the support systems, there are other concerns of critical importance to the economic viability and safety of a proposed caving mine.

One is cavability. Caving ease can make or break an operation, yet its pre-production assessment is one of the most difficult questions facing mine planners. Rock mass class has been suggested as a useful index of cavability, and some preliminary or site-specific relationships, based on elementary geomechanical data, have been proposed. Further research is needed to define caving mechanisms, to put together a useful classification system. The desirability of such a system extends beyond improvement in mine design for lowered development costs and improved productivity. Improved cavability assessments that result in smoother, continuous caving would mean less need for loosening hang-ups in drawpoints by prodding or secondary blasting, thus reducing miners' exposure to literally tons of potentially unstable rock overhead and reducing the hazards associated with additional blasting. Improvements in cavability assessments would result in designs that enhance the flow of rock through drawpoints, relieving much of the support system distress resulting from packing of draw raises, and would simplify undercut design, reducing the incidence of stubs that can have severe effects on drift support.

Similarly, classification systems could be used to specify drawpoint spacing, predict secondary blasting requirements, and improve draw control. In short, improved caving has far-reaching effects, for a simpler and safer, as well as cheaper, mining operation.

This research effort also identified that additional basic research in caving mine rock mass behavior is both necessary and desirable. Much data exist, but caving mine drift behavior is defined by

a number of factors that are not well understood by operators. At this time, thorough support design based on measurable geomechanical properties is simply not generally possible. Judgement is extensively relied upon. Computer models need to be developed that realistically model the large deformations and extremely strong support systems in use in caving mines. An understanding is needed of caving and rock fragmentation processes, and such an understanding would be useful for other caving mining methods as well, such as longwall and pillar removal as widely practiced in the coal industry.

1.0 INTRODUCTION

The support of production drifts remains a high-cost item facing the developer or operator of an underground caving mine. Support costs are incurred well in advance of ore production and there is a strong need to minimize these pre-production capital commitments. However, the extreme loads developed in production drift supports in caving mines, together with safety requirements and a still-incomplete understanding of rock mass behavior in caving areas, leads to a high degree of uncertainty in support design.

The research described here has led to the MBR (Modified Basic RMR) System, a rational means to predict drift support requirements for preliminary design, planning, and cost estimation, through the use of a rock quality index (RQI).

1.1 Scope of the Project

Presently, support design for caving production drifts is largely a trial-and-error process. Established mines have developed their support practices through years of experience in their respective orebodies. New or expanding mines commonly plan drift support by comparing rock conditions with other operations and adjusting support as appropriate.

Varying amounts of geotechnical data may be incorporated, but fully analytical designs still are not achieved because essential quantitative information is lacking on the behavior of the rock masses involved.

Similar problems have existed in civil tunneling because highly detailed support designs, while worthwhile, are expensive to achieve. However, there are numerous rock mass classification approaches in past or present use that generate a support recommendation, by comparing rock mass conditions from the planned tunnel with those of completed tunnels. These conditions are described by means of a rating or rock quality index (RQI) that is, in turn, based on evaluations of several diagnostic geotechnical characteristics. The differences between these classification systems relate to the types of data sought, the manner of combining these data, and the data base of completed tunnel projects.

Mine production drifts have certain characteristics that distinguish them from tunnels (for which most classification systems are intended). Unlike most tunnels, caving mine production drifts are not single, isolated openings. The imposed loads are not static but are both dynamic and variable in time and space. Support philosophies are less conservative in mining, and excavation techniques are less rigorously controlled. Opening shapes may be different.

The planned mine can seldom be restricted to the best ground conditions, while in tunneling, site investigations seek the best ground. In fact, weak ground is a requirement for easy caving and is therefore a fact of life in many mines.

Despite these differences with civil tunneling, an RQI approach has great potential for advance planning of caving mine drift supports. Such an approach would formalize the comparison approach to planning that is common in caving mining today.

The basic objective of the project described is therefore to apply presently used RQI methods to block caving drifts, and provide guidelines for ground support based on empirical data. For this, attention was focused on the Q system of Barton, Lien, and Lunde; the RMR or Geomechanics system of Bieniawski; and the RSR system of Wickham, Tiedemann, and Skinner. (These and other RQI systems are discussed in Chapter 2.) In addition, the applicability of the developed RQI approach to other mining methods is addressed.

1.2 Methodology

The number of variables that determines drift support and affects performance is a large one. These variables include overall rock competence, variation in rock competence, method and quality of excavation, method and quality of rock reinforcement, suitability and effectiveness of final support, undercutting method and practice, draw control, and repair techniques. Any one of these variables, if not properly accounted for, could lead to improper support or failed linings and, at worst, cessation of mining in some areas.

Furthermore, support practice, once established, is ordinarily not varied except under radically different conditions. This means that the installed support is not necessarily diagnostic of rock competence.

For these reasons, it was necessary to visit several active mines to classify the rock mass and account for variations in ground conditions, and to observe in detail the excavation and support practice. It was realized that this would be a greater effort at some mines than at others, so longer visits were anticipated to those mines not actively collecting or analyzing geotechnical data.

During the field visits, geologic data were obtained, where available, from company maps and files, including drill core logs. When necessary, detailed fracture surveys were conducted, and in one case, core was re-logged for required geotechnical data. Reconnaissance led to selection of specific areas for study, and RQI values were obtained for these areas by first-hand observation of rock conditions and incorporation of geotechnical data.

The data required by the three RQI systems being applied in this study (Q, RSR, and RMR) are similar, so that once the fracturing, rock type and strength, degree of alteration, and water conditions were assessed, ratings and associated support recommendations could be obtained for each system.

Concurrent with geologic data collection, the excavation and support practices were observed. Information was collected on drift dimensions and layouts, on blasting practice (drillhole pattern, powder requirements, delays, overbreak and so on); advance rate, development sequencing, and support sequencing; type, density, and effectiveness of rock reinforcement as installed; specifications and as-built characteristics of permanent steel support; type, placement, and strength of concrete as designed and as placed; and general observations relating to consistency and installation of support. These data were collected for the various types of drifts found in the production area - undercut, grizzly or slusher drifts, ventilation drifts, access or fringe drifts, panel drifts, and haulageways, and any "tailoring" of support practice to rock competence was also noted.

At each field site, production data were collected so that production-related factors affecting drift support performance could also be considered. Such data related to the method, progress, and direction of undercutting, pulling of swell muck, draw control, and amount of draw.

The field data were supplemented with published and unpublished information of utility to the overall project, which related to: geology and geometry of ore deposits that are amenable to caving, design and operation of present and inactive caving mines, behavior of and stresses developed in rock masses near caving areas, early classification systems, theoretical and finite element studies of rock mass stresses and displacements for various opening geometries and distributions, tunneling case histories, and vein-type orebodies and mining methods. This information was used throughout the study.

Following the field work, support recommendations were formulated according to the three systems employed, and comparisons were made with encountered support and its performance. Key variables affecting drift support integrity were selected and the methods for rating and combining these variables were formulated. This initial classification system was evaluated in light of the field and published data on support philosophy and performance, and revisions were made so as to make it better fit the data base. Finally, the system was evaluated for its potential for other mining methods.

1.3 Previous Work and Parallel Studies

Because of its multidisciplinary nature, this investigation made use of the efforts of workers in several areas: rock mass classification, theoretical and applied rock mechanics, mining engineering, and geology.

Laubscher (1977, 1981), Laubscher and Taylor (1976), and Taylor (1980) represent the only published material located during this investigation that develops an RQI approach specifically for drift support in block caving mines. McDonough (1976) discusses a subjective rating system for analyzing rock masses at Climax, and relates the ratings to support requirements. Other mines have ongoing efforts to develop predictive techniques for their specific circumstances; prominent among these techniques are RQI approaches.

Several mining operations have been studied in attempts to understand the nature of actual rock loads in block caving mines so that these loads may be predicted from measurable data. Such work continues at this writing. Recently, Maier and Brumleve (1981) discussed rock mechanics investigations at Henderson and their experience with RQI systems there. Panek and Tesch (1981) give the results of measurements of ground movements at San Manuel. Kendorski (1973) discusses rock mechanics at Climax. Thomas (1971) analyzed subsidence at San Manuel.

The development and expansion of the RMR or Geomechanics system can be traced through Bieniawski (1979(a), 1979(b), 1976, 1973). The Q system is described by Barton, Lien, and Lunde (1974, 1975). The development of the RSR concept is described by Wickham and Tiedemann (1974). Further description is given by Wickham, Tiedemann, and Skinner (1974).

Note should be given to the work of Deere (1964) in which RQD is introduced. RQD is widely accepted as a measure of core quality, depending in part on fracture spacing, and in fact is a key input parameter to a number of proposed RQI approaches. The use of RQD alone to recommend support is seldom practiced today, however.

Terzaghi (1946) proposed one of the earliest classification systems, and the method is in use today for estimating steel rib supports in horseshoe or arch tunnels.

Rutledge (1978), McCusker (1980), and Taylor (1980) review rock mass classification systems. Other prominent classification approaches were used by Ikeda (1970), Coates (1964), Abel (1967), Obert and Rich (1971), and Brekke and Howard (1972) under more specialized circumstances.

Block caving methods have been widely discussed in the literature. A summary of current caving practice was provided by Julin and Tobie (1973) and Tobie and Julin (1970). Earlier block caving practice is discussed by Bucky (1945). Merrill and Johnson (1964) considered the stress changes due to undercutting in block caving.

The geologic requirements for caving orebodies have been examined by Panek (1981), White (1979), Julin and Tobie (1973), McMahon and Kendrick (1969), Boshkov and Wright (1973) and McMahon (1967). Swaisgood, McMahon, and West (1972) discuss geologic investigations for block caving.

The geologic environments at major caving mines are discussed in numerous papers, each dealing with a specific orebody. Only selected, recent papers are mentioned here. The Urad and Henderson orebodies were studied geologically by Wallace, et.al. (1978). Thomas (1966) discusses the San Manuel orebody; a later paper (Thomas, 1971) summarizes the caving behavior of San Manuel ore. Wallace (1968) gives a geologic overview of the Climax orebody.

An extensive volume edited by Stewart (1981) provides a detailed overview of present domestic and foreign block caving practices. The volume includes papers describing the current status of many major caving operations.

1.4 Report Content

This report describes the research and development of the MBR (Modified Basic RMR) classification system. To this end, Section 2 also includes comparative, parametric, and sensitivity studies of the Q, RMR, and RSR systems that were undertaken in support of this development. The use of the MBR system the input variables, and the rationale behind the selection of variables, are principal topics.

To preserve confidentiality, the identities of the field sites are not disclosed but are referred to as Mine A, B, and so on. The field data are summarized mine-by-mine.

The applicability of the MBR system to domestic vein mining is discussed in Section 7. The Coeur d'Alene District is considered to be the major active vein mining area in the United States, and Section 7 therefore addresses itself to the rock conditions and mining methods of that area.

It is assumed that the reader has been involved with mine planning and rock mechanics to some degree. Therefore, descriptions of present RQI approaches and descriptions of block cave mining are not emphasized in this report. These subjects have received wide attention and the interested reader is referred to the references listed in Section 1.3 for more extensive discussions of these topics.

Special note should be given Section 6.5, which examines the utility of the MBR system, its weaknesses and limits of applicability.

Many of the concepts employed in the MBR system could be useful in other aspects of planning block caving mines. While beyond the scope of the present investigation, issues of secondary blasting and cavability have application in defining performance of drift support during production. These and other issues are discussed in Section 8.

Finally, it was thought that a manual for the MBR system would be most useful to industry if it were detachable from the Final Report. Accordingly, the Manual, Volume II, has been written as a separate volume. This report is devoted entirely to the process used in arriving at the MBR System. Specific instructions for the use of the System are given in the Manual, Volume II.

2.0 ROCK MASS CLASSIFICATIONS

This section is intended to convey a general familiarity with the purpose and use of rock quality index (RQI) systems. An exhaustive review of rock classification techniques is not undertaken; the interested reader is referred to the works mentioned in Section 1.3. The RQI systems employed in this study are outlined in the following paragraphs; references are given for more complete discussions.

2.1 Basic Concepts

The attractiveness of an empirical method of tunnel support design has been recognized for several decades. In fact, mine drift and tunnel design began as an empirical process, in which the support was specified by an experienced person based on his impression of ground competence.

Terzaghi (1946) was the first to formalize this process for general use. His method systematized the assessment of ground competence by providing definitions of observable conditions; these definitions are combined, and are linked to support requirements, through simple formulae that enable the weight of loosened rock over a tunnel to be computed.

The method is not wholly empirical, in that an intermediate value, corresponding to rock load, is computed, and certain assumptions of the source of support loads are inherent. Because of these assumptions, the method is closely tied to steel rib support in tunnels of arched cross-section. Rock reinforcement in flat-backed tunnels, and other options commonly used by present-day tunnel designers, have less applicability in the Terzaghi system.

In addition, a common problem with the Terzaghi system is that the definitions used are subjective, and the inexperienced user is likely to misapply them. Nonetheless, Terzaghi's system remains popular for its intended application.

The RQD concept was introduced by Deere (1964) as a description of rock core, and was only later proposed as a method of support design. This use of RQD has not been popular as a result of widespread awareness that RQD is essentially a measure of fracturing, and does not fully describe rock mass competence. Furthermore, RQD can be affected by factors such as drilling technique that are not directly related to rock competence. Today, however, RQD remains a valuable descriptive tool, and is an essential input parameter of many RQI systems presently in use.

More recent RQI approaches attempt to rate, individually, the most important factors that determine rock mass behavior. Although there is some variation, these factors reduce to the following:

intact rock strength, measures of fracturing intensity, measures of the shear strength of the fractures, the geometrical relationship between fracture patterns and the excavation, and groundwater. The chief differences center on the means of combining these ratings to achieve the rating for the rock mass, and the way in which the support recommendation is obtained.

In the development of a fully empirical RQI system, the method of combining and using the ratings is determined by studying as many excavations as possible, rating the rock masses present, and associating these ratings with the support that was installed, while maintaining an awareness of the effectiveness (overdesign as well as underdesign) of such supports.

It is therefore clear that the limit of applicability of any RQI system is defined by its base of data. Classification is essentially a means of accessing a data base; ideally, it allows a user to specify a support system that is the same as the most effective system used by others under identical geological conditions.

For the developer of an RQI system, there is always some uncertainty in associating each of the observed conditions in the data base with a rating. The user of an RQI system is again affected by uncertainty when he interprets the language describing the rating for his own rock mass conditions. Also, the support/rating associations in the data base, to be fully comparable, must all reflect the same degree of support effectiveness, and this may be difficult or impossible to accurately assess. Because of these situations, RQI approaches yield support recommendations that are approximations of the optimal design. There is still no substitute for detailed, analytical support design based on measurements of rock mass properties in the field and laboratory.

RQI systems are therefore not ordinarily used as a substitute for such analytical designs. However, they can afford significant savings when used for estimation, planning, and preliminary design.

2.2 RQI Systems Considered in This Study

Initially, the three most popular current systems (RSR, Q, RMR) were tried, to evaluate their applicability and utility for block caving drift support. In addition, the concepts formulated by Abel (1967) and Laubscher (1976) were used in the development of the MBR system. These five approaches are discussed in 2.2.1 through 2.2.5.

2.2.1 RSR Concept

The RSR Concept, a ground support prediction model, was developed in the United States in 1972 by Wickham, Tiedemann, and Skinner (1972, 1974). The concept presents a quantitative method for describing the quality of a rock mass for selecting the appropriate ground support. It was the first true rock mass classification system proposed since that introduced by Terzaghi in 1946.

The RSR Concept was a step forward in a number of respects: firstly, it is a quantitative classification unlike Terzaghi's qualitative one; secondly, it is a rock mass classification incorporating many parameters, unlike the RQD index that is limited to core quality; thirdly, it is a complete classification having an input and an output, unlike a classification such as proposed by Lauffer (1958) that relies on practical experience to decide on a rock mass class, which will then give an output in terms of the stand-up time and span.

The main contribution of the RSR Concept was that it introduced a rating system for rock masses. This is the sum of weighted values of the individual parameters considered in this classification system. In other words, the relative importance of the various classification parameters could be assessed. This rating system was determined on the basis of case histories as well as reviews of various books and technical papers dealing with different aspects of ground support in tunneling.

The RSR Concept considers two general categories of factors influencing rock mass behavior in tunneling: geologic parameters and construction parameters. The geologic parameters are: (a) rock type, (b) joint pattern (average spacing of joints), (c) joint orientations (dip and strike), (d) type of discontinuities, (e) major faults, shears, and folds, (f) rock material properties, and (g) weathering or alteration. Some of these factors are treated separately; others are considered collectively. The authors point out that in some instances it would be possible to accurately define the above factors, but in others, only general approximations can be made. The construction parameters are: (a) size of tunnel, (b) direction of drive, and (c) method of excavation.

All the above factors were grouped by Wickham, Tiedemann, and Skinner (1972) into three basic parameters, A, B, and C, which in themselves are evaluations as to the relative effect on the support requirements of various geological factors. These three parameters are as follows:

Parameter A. General appraisal of rock structure, on the basis of:

(1) Rock type (igneous, metamorphic, sedimentary).
(2) Rock hardness (hard, medium, soft, decomposed).
(3) Geologic structure (massive, slightly faulted/ folded, moderately faulted/folded, intensely faulted/folded).

Parameter B. Effect of discontinuity pattern with respect to the direction of tunnel drive, on the basis of:

(1) Joint spacing.
(2) Joint orientation (strike and dip).
(3) Direction of tunnel drive.

Parameter C. Effect of groundwater inflow, based on:

(1) Overall rock mass quality due to parameters A and B combined.
(2) Joint condition (good, fair, poor).
(3) Amount of water inflow (in gallons per minute per foot of the tunnel).

The RSR value of any tunnel section is obtained by summarizing the weighted numerical values determined for each parameter. This reflects the quality of the rock mass with respect to its need for support regardless of the size of the tunnel. The relation between RSR values and tunnel size is taken into consideration in the determination of respective rib ratios (RR), as discussed below. Since a lesser amount of support was expected for machine-bored tunnels than when excavated by drill and blast methods, it was suggested that RSR values be adjusted for machine-bored tunnels.

In order to correlate RSR values with actual support installations, a concept of the RR was introduced. The purpose was to have a common basis for correlating RSR determinations with actual or required installations. Since 90% of the case history tunnels were supported with steel ribs, the RR measure was chosen as the theoretical support (rib size and spacing). It was developed from Terzaghi's formula for determining roof loads in loose sand below the water table (datum condition). Using the tables provided in Rock Tunneling with Steel Supports, the theoretical spacing required for the same size rib as used in a given case study tunnel section was determined for the datum condition. The RR value is obtained by dividing this theoretical spacing by the actual spacing and multiplying the answer by 100. Thus, RR = 46 would mean that the section required only 46% of the support used for the datum condition. However, different size tunnels, although having the same RR wwwouldrequire different weight or size of ribs for equivalent support. The RR for an unsupported tunnel would be zero and would be 100 for a tunnel requiring the same support as the datum condition.

A total of 53 projects was evaluated, but since each tunnel was divided into typical geological sections, a total of 190 tunnel sections were analyzed. The RSR values were determined for each section, and actual support installations were obtained from as-built drawings. The support was distributed as follows:

Sections with steel ribs	147	(89.6%)
Sections with rockbolts	14	(8.6%)
Sections with shotcrete	3	(1.8%)
Total supported	164	(100.0%)
Total unsupported	26	
Total	190 sections	

An empirical relationship was developed between RSR and RR values, namely:

$$(RR + 80)\ (RSR + 30) = 8800, \quad \text{or}$$

$$(RR + 70)\ (RSR + 8) = 6000$$

It was concluded that rock structures with RSR values less than 19 would require heavy support while those with ratings of 80 and over would be unsupported.

Since the RR basically defined an anticipated rock load by considering the load-carrying capacity of different sizes of steel ribs, the RSR values were also expressed in terms of unit rock loads for various sized tunnels.

The RSR prediction model was developed primarily with respect to steel rib support. Insufficient data were available to correlate rock structures and rockbolt or shotcrete support. However, an appraisal of rockbolt requirements was made by considering rock loads with respect to the tensile strength of the bolt. The authors pointed out that this was a very general approach: it assumed that anchorage was adequate and that all bolts acted in tension only; it did not allow either for interaction between adjacent blocks or for an assumption of a compression arch formed by the bolts. In addition, the rock loads were developed for steel supported tunnels. Nevertheless, the following relation was given for 1-in.-diameter rockbolts with a working load of 24,000 lb:

$$\text{Spacing (ft)} = 24/W$$

where W is the rock load in 1,000 psf (ksf).

No correlation could be found between geologic prediction and shotcrete requirements, so that the following empirical relationship was suggested:

$$t = 1 + \frac{W}{1.25} \qquad \text{or} \qquad t = \frac{D}{150}(65 - RSR)$$

where

t = shotcrete thickness, in.

W = rock load, 1,000 psf (ksf)

D = tunnel diameter, ft.

Support requirement charts have been prepared that provide a means of determining typical ground support systems based on a RSR prediction as to the quality of rock structure through which the tunnel is to be driven. Charts for 10-, 20-, and 24-ft-diameter tunnels are available; similar charts could be used for other tunnel sizes. The charts are applicable to either circular or horseshoe-shaped tunnels of comparable widths.

The RSR Concept has been widely accepted as a very useful method for selecting steel rib support for rock tunnels. As with any empirical approach, one should not apply a concept beyond the range of sufficient and reliable data used for developing the concept. For this reason, the RSR Concept is not recommended for selection of rockbolt and shotcrete support. It should be noted that although the definitions of the classification parameters were not explicitly stated by the the proposers, most of the input data needed will be normally included in a standard joint survey; however, the lack of definitions (for example, slightly faulted or folded rock) may lead to some confusion.

To summarize, the following steps are required in applying the RSR concept: (The necessary charts and tables can be found in Wickham and Tiedemann, 1972).

Step 1 Divide rock mass into geological regions, such that each region would be geologically similar and would require one type of support; that is, it will not be economical to change tunnel support until rock mass conditions change distinctly (a new structural region can be distinguished).

Step 2 Obtain classification input data for each structural region.

Step 3 Determine the individual classification parameters A, B, and C and their sum, which gives the RSR = A + B + C.

Step 4 Adjust the RSR value if the tunnel is to be excavated by a tunnel boring machine.

Step 5 Select a support requirement chart appropriate for the tunnel size. These charts are applicable to both circular and horseshoe-shaped tunnels. From the selected chart, determine the rib type and spacing corresponding to the RSR value. Ignore curves for rockbolt and shotcrete support since they are not based on sufficient case history data.

Step 6 The rock load is then estimated from a table. The theoretical RR is given by:

$$(RR + 80)(RSR + 30) = 8{,}800$$

The values obtained are for comparison purposes between the structural regions.

2.2.2 Q-System

The Q-System of rock mass classification was developed in Norway in 1974 by Barton, Lien, and Lunde, all of the Norwegian Geotechnical Institute. Its development represented a major contribution to the subject of rock mass classifications for a number of reasons: the system was proposed on the basis of an analysis of some 200 tunnel case histories from Scandinavia, it is a quantitative classification system, and it is an engineering system enabling the design of tunnel supports.

The Q-System is based on a numerical assessment of the rock mass quality using six different parameters: (a) RQD, (b) number of joint sets, (c) roughness of the most unfavorable joint or discontinuity, (d) degree of alteration or filling along the weakest joint, (e) water inflow, and (f) stress condition.

The above six parameters are grouped into three quotients to give the overall rock mass quality Q as follows:

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

In this expression, J_n is the rating corresponding to the number of joint sets, J_r is the joint roughness rating, J_a is the joint alteration rating, J_w is the water inflow rating, and SRF is the Stress Reduction Factor, which is dependent on the rock stress condition and compressive strength.

Since it is assumed that the reader has some familiarity with RQI systems, the rating scales are omitted here for the sake of brevity. The reader should consult Barton, Lien, and Lunde (1974) for the ratings' descriptions, if desired.

The value of Q varies from 0.001 to 1,000, where the larger numbers denote better rock.

The proposers of the Q-System believed that the parameters, J_n, J_r, and J_a, played a more important role than joint orientation, and if joint orientation had been included, the classification would have been less general. However, the orientation is implicit in the parameters J_r and J_a, because they apply to the most unfavorable joints.

The Q is related to the tunnel support requirements by defining the equivalent dimensions of the excavation. This equivalent dimension, which is a function of both the size and the purpose of the excavation, is obtained by dividing the span, diameter, or the wall height of the excavation by a quantity called the excavation support ratio (ESR). Thus,

$$\text{Equivalent dimension} = \frac{\text{Excavation span, diameter, or height, meters.}}{\text{ESR}}$$

The ESR is related to the use for which the excavation is intended and the degree of safety demanded, as follows:

		ESR	No. of Cases
A.	Temporary mine openings	3-5	(2)
B.	Vertical shafts:		
	circular section	2.5	- -
	rectangular/square section	2.0	- -
C.	Permanent mine openings, water tunnels for hydropower (excluding high-pressure penstocks), pilot tunnels, drifts, and headings for large excavations	1.6	(83)
D.	Storage rooms, water treatment plants, minor highway and railroad tunnels, surge chambers, access tunnels	1.3	(25)
E.	Power stations, major highway or railroad tunnels, civil defense chambers, portals, intersections	1.0	(73)

		ESR	No. of Cases
F.	Underground nuclear power stations, railroad stations, factories	0.8	(2)

The relationship between the index Q and the equivalent dimension is illustrated by 38 support categories shown by box numbering. Support measures that are appropriate to each category are then listed. Since it was decided that bolting and shotcrete support deserves most attention, case histories featuring steel rib support, concrete arch roofs, and precast lining have been ignored.

The length of bolts L is determined from the equation:

$$L = 2 + 0.15\ B/ESR$$

where B is the excavation width. Dimensions are in meters.

The 38 support categories have been specified to give estimates of permanent roof support since they were based on roof support methods quoted in the case histories. For temporary support determination, either Q is increased to 5Q or ESR is increased to 1.5 ESR.

The maximum limit for permanent unsupported spans can be obtained as follows:

$$\text{Maximum span (unsupported)} = 2(ESR)Q^{0.4}.$$

The relationship between Q and permanent support pressure P_{roof} is plotted from the following equation:

$$P_{roof} = \frac{2.0}{J_r} Q^{-1/3}$$

If the number of joint sets is less than three, the equation is expressed as

$$P_{roof} = \frac{2}{3} (J_n^{1/2}) (J_r^{-1}) (Q^{-1/3}).$$

The proposers of the Q-System emphasized that while the support recommendations for the large-scale excavations would generally incorporate thicker shotcrete and longer bolts, the bolt spacing and theoretical support pressure would remain roughly the same.

When core is unavailable, the RQD is estimated from the number of joints per unit volume, in which the number of joints per meter for each joint set are added. The conversion for clay-free rock masses is

$$RQD = 115 - 3.3\, J_v$$

where J_v represents the total number of joints per cubic meter (RQD = 100% for $J_v < 4.5$).

The following steps are involved in applying the Q-System:

<u>Step 1</u> Divide the rock mass into structural regions, such that each region would be geologically similar and would require one type of support category.

<u>Step 2</u> Collect necessary input data for each structural region.

<u>Step 3</u> Determine the ratings of the six classification parameters and calculate the Q value.

<u>Step 4</u> Select the excavation category, and allocate the ESR.

<u>Step 5</u> Determine the support category for the Q value and the tunnel span/ESR ratio.

<u>Step 6</u> Select the support measures appropriate to the support category. Calculate the length of rockbolts as above.

<u>Step 7</u> The selected support measures are for the permanent support. Determination of the initial support measures is as described above.

<u>Step 8</u> For comparison purposes, determine the support pressure as above.

<u>Step 9</u> For record purposes, estimate the possible maximum unsupported span and the standup time.

2.2.3 RMR (Geomechanics) System

The Geomechanics Classification or the Rock Mass Rating (RMR) System was developed by Bieniawski in 1973. The system has been updated and the reader is referred to Bieniawski (1979(a)) for the current (1983) rating scheme. This engineering classification of rock masses, especially evolved for rock engineering applications, utilizes the following six parameters, all of which not only are measurable in the field but can also be obtained from borings:

<u>a</u>. Uniaxial compressive strength of intact rock material.
<u>b</u>. Rock quality designation (RQD).

c. Spacing of joints (discontinuities).
d. Orientation of joints (discontinuities).
e. Condition of joints (discontinuities).
f. Groundwater condition.

Initially, five of the above parameters (a, b, c, e, and f) are grouped into five ranges of values. Since the various parameters are not equally important for the overall classification of a rock mass, importance ratings are allocated to the different value ranges of the parameters, a higher rating indicating better rock mass conditions. These ratings were determined from 49 case histories investigated by the originator, while the initial ratings were based on the studies by Wickham, Tiedemann, and Skinner (1972).

To apply the Geomechanics Classification, the rock mass is divided into a number of structural regions, that is, zones in which certain geological features are more or less uniform within each region. The above six classification parameters are determined for each structural region from measurements in the field.

Next, the importance ratings are assigned to each parameter. In this respect, the typical rather than the worst conditions are evaluated since this classification, being based on case histories, has a built-in "safety factor". Furthermore, it should be noted that the importance ratings given for joint spacings apply to rock masses having three sets of joints. Thus, when only two sets of joints are present, a conservative assessment is obtained. Once the importance ratings of the classification parameters are established, the ratings for the five parameters are summed, to yield the basic overall rock mass rating for the structural region under consideration.

At this stage, the influence of the strike and dip of joints ("d" above) is included by adjusting the basic rock mass rating. This step is treated separately because the influence of joint orientation depends upon engineering application, for example, tunnel, slope, or foundation. The parameter "joint orientation" is not evaluated in quantitative terms but by qualitative descriptions such as "favorable." In the case of civil engineering projects, an adjustment for joint orientations will suffice. For mining applications, other adjustments may be called for such as the stress at depth or a change in stress.

After the adjustment for joint orientations, the rock mass is classified, according to the final (adjusted) rock mass rating (RMR), into one of five rock mass classes. The rock mass classes are in groups of twenty ratings each, and are denoted "very good rock" (Class I, RMR = 81 to 100), "good rock" (Class II, RMR = 61 to 80), "fair rock (Class III, RMR = 41 to 60), "poor rock" (Class IV, RMR = 21 to 40), and "very poor rock" (Class V, RMR = 0 to 20).

Next, the practical meaning of each rock mass class is related to specific engineering problems. For tunnels and chambers, the output from the Geomechanics Classification is the stand-up time of an unsupported rock span for a given rock mass rating. The stand-up time is compared with the design life of the tunnel.

Longer stand-up times can be achieved by selecting rock reinforcement measures. They depend on such factors as the depth below surface (in situ stress), tunnel size and shape, and the method of excavation. Rock reinforcement measures and other suport recommendations (Bieniawski, 1979(a)) are defined only for 10-meter- (33-ft-) diameter tunnels and the determination of support for tunnels of other dimensions is apparently left to judgement or other means.

The system also allows determination of in-situ modulus of deformation for foundation design, and rock mass cohesion for slope design.

It should be noted that the support measures given represent the permanent and not the temporary support. Hence, additional concrete lining is not required for structural purposes.

However, to ensure full structural stability, it is recommended that tunnel monitoring during construction provide a check on stabilization of rock movements.

The Geomechanics Classification recognizes that no single parameter or index can fully and quantitatively describe a jointed rock mass for tunneling purposes. Each of the many factors has a different significance, and only if taken together can they describe a rock mass satisfactorily. Each of the six parameters employed in this classification is discussed below, since these concepts are drawn upon elsewhere in this report.

Strength of Intact Rock Material

There is a general agreement that knowledge of the uniaxial compressive strength of intact rock is necessary for classifying a rock mass. After all, if the discontinuities are widely spaced and the rock material is weak, the rock material properties will influence the behavior of the rock mass. Under the same confining pressure, the strength of the rock material constitutes the highest strength limit of the rock mass. The rock material strength is also important if the use of tunneling machines is contemplated. Finally, a sample of the rock material sometimes represents a small-scale model of the rock mass since they have been subjected to the same geological processes.

Rock Quality Designation (RQD)

RQD is used as a classification parameter, because although it is not sufficient on its own for a full description of a rock mass, the RQD index has been found most useful in tunneling applications as a guide for selection of tunnel support, has been employed extensively in the United States and in Europe, and is a simple, inexpensive, and reproducible way to assess the quality of rock core.

Spacing of Joints

The term "joint" emcompasses all discontinuities present in the rock mass that may be technically joints, bedding planes, minor faults, or other surfaces of weakness. The behavior of joints may govern the behavior of a rock mass as a whole. The presence of joints reduces the strength of a rock mass, and their spacing governs the degree of such reduction. For example, a rock material with a high strength will yield a weak rock mass a intensely jointed. Spacing of joints is a separate parameter, because the RQD index does not uniquely define the spacing of joints.

Studies by Wickham, Tiedemann, and Skinner (1972) have emphasized the effect of joint orientations on tunnel stability. A qualitative assessment of favorability is preferred to more elaborate systems for joint orientation and inclination effects.

Condition of Joints

This parameter includes roughness of the joint surfaces, their continuity, their opening or separation (distance between the surfaces), the infilling (gouge) material, and weathering of the wall rock.

Roughness, or the nature of the asperities in the discontinuity surfaces, is an important parameter characterizing the condition of discontinuities. Asperities that occur on joint surfaces interlock, if the surfaces are clean and closed, and inhibit shear movement along the joint surface. Roughness asperities usually have a base length and amplitude measured in terms of tenths of an inch and are readily apparent on a core-sized exposure of a discontinuity.

Continuity of joints influences the extent to which the rock material and the discontinuities separately affect the behavior of the rock mass. In the case of tunnels, a discontinuity is considered fully continuous if its length is greater than the width of the tunnel.

Separation, or the distance between the discontinuity surfaces, controls the extent to which the opposing surfaces can interlock as well as the amount of water that can flow through the discontinuity. In the absence of interlocking, the filling (gouge) controls entirely the shear strength of the discontinuity. As the separation decreases, the asperities of the rock wall tend to become more interlocked, and both the filling and the rock material contribute to the shear strength of joints. The shear strength along a joint is therefore dependent on the degree of separation, presence or absence of filling materials, roughness of the surface walls, and the nature of the filling material.

The infilling (gouge) has a two-fold influence:

a. Depending on the thickness, the filling prevents the interlocking of the fracture asperities.

b. It possesses its own characteristic properties, that is, shear strength, permeability, and deformational characteristics.

The type, thickness, continuity, and consistency of filling should be noted.

Weathering of the wall rock, that is, the rock constituting the joint walls, is classified in accordance with the recommendations of the Task committee of the American Society of Civil Engineers (197[illegible]):

a. Unweathered. No visible signs of weathering are noted; rock fresh; crystals bright.

b. Slightly weathered rock. Discontinuities are stained or discolored and may contain a thin filling of altered material. Discoloration may extend into the rock from the discontinuity surfaces to a distance of up to 20% of the discontinuity spacing.

c. Moderately weathered rock. Slight discoloration extends from discontinuity planes for greater than 20% of the discontinuity spacing. Discontinuities may contain filling of altered material. Partial opening of grain boundaries may be observed.

d. Highly weathered rock. Discoloration extends throughout the rock, and the rock material is partly friable. The original texture of the rock has mainly been preserved, but separation of the grains has occurred.

e. Completely weathered rock. The rock is totally discolored and decomposed and in a friable condition. The external appearance is that of soil. Internally, the rock texture is partly preserved, but grains have completely separated.

(It should be noted that the boundary between rock and soil is defined in terms of the uniaxial compressive strength and not in terms of weathering. A material with the strength equal to or above 150 psi is considered as rock.)

Groundwater Conditions

In the case of tunnels, the rate of inflow of groundwater in gallons per minute per 1,000 ft of the tunnel should be determined, or a general condition can be described as completely dry, damp, wet, dripping, and flowing. If actual water pressure data are available, these should be stated and expressed in terms of the ratio of the water pressure to the major principal stress. The latter can be either measured or determined from the depth below surface, that is, the vertical stress increases with depth at 1.1 psi per foot of the depth below surface.

2.2.4 Laubscher's Geomechanics System for Mining

The system discussed by Laubscher and Taylor (1976) was intended primarily for rock mass classifications in block-cave mining. This method was developed on the basis of Bieniawski's Geomechanics system and has been in use in African asbestos mines since 1973. It has since been extended and clarified (Laubscher, 1977, 1981). For purposes of discussion in this report, this system will be referred to as Laubscher's system.

The input data for Laubscher's system are very similar to those of the RMR, although the criteria for arriving at the values used are somewhat different in the two systems. The rock mass in question is assigned to one of 10 sub-classes (1A, 1B, 2A, 2B, etc.) by dividing each RMR class into two parts, so that each class has a value of 10 rating units. This in-situ rock mass rating is then adjusted for weathering, field and induced stresses, changes in stress, joint orientations, and blasting effects. The support requirements extend to massive concrete with steel reinforcement, and depend on the adjusted class and the degree of adjustment.

The ratings may also be related to cavability, stand-up time, cave angle and failure zone, and applied to slopes, pillars, and foundations. These applications are beyond the scope of this discussion. The reader is referred to Laubscher and Taylor (1976) and Laubscher (1981) for more on these topics.

Since this system is an important model for the MBR system, and since it is less likely that the reader will be familiar with it than with the three RQI systems just discussed, the following description includes considerable detail.

The five essential input parameters are (as in the RMR) the ROD, intact rock strength, joint condition, joint spacing, and groundwater condition. The descriptions of these parameters are the same as for the RMR, as described in Section 2.2.3 above, except in the following instances.

1. RQD - As conventionally determined.

2. Intact Rock Strength - The average uniaxial compressive strength between joints, and exclusive of alteration zones.

3. Joint Condition - The joint condition is assessed according to Table 1, which develops percentages to be applied to a maximum possible rating of 30. Note that separation and continuity are omitted. Using this scheme, the rating for a straight, polished joint would be 30 x (70% x 50%) = 10.5.

4. Joint Spacing - The ratings are obtained by means of the graph in Figure 1. It is necessary to distinguish the spacings associated with each recognized set. The intersection corresponding to the intermediate and minimum joint spacings (AB, for the example in Figure 1) is first found. This value is projected to the diagonal line as shown. This is projected to the right until the maximum spacing (C or D in the example in Figure 1) is intersected. The rating is read directly. For a 2-joint system such as AB, the rating used would be 15. If there are more than 3 sets (such as ABCD), the lowest 3-set rating would be used (in this instance, ABC, rated 6).

5. Groundwater - There are no differences with the RMR rating structure, except that it is recognized that much development work in mining is done in well-drained areas.

Having obtained ratings for these five basic parameters, the in-situ rock class is obtained from a chart (Table 2). As with the RMR, the five parameter ratings are summed to obtain the in-situ rating. This rating is a description of the rock mass competence for various engineering purposes.

The key feature of Laubscher's system is the application of adjustments to the in-situ ratings. Most of these adjustments are for conditions that significantly affect rock mass behavior in mining. The adjustment procedure is to apply sequential percentage multipliers (Table 4) to the in-situ ratings, rather than to subtract values.

The weathering adjustment recognizes that weathering may affect rocks in 3 areas: RQD, intact rock strength, and joint condition. An adjustment of 95% to the RQD rating is possible if weathering results in increased fracturing. An adjustment of 96% may be applied to the

TABLE 1 - Joint Condition Ratings
(after Laubscher and Taylor, 1976)

Parameter	Description	Percentage adjustment
A. Joint expression (large scale)	Wavy uni-directional	90-99
	Curved	80-89
	Straight	70-79
B. Joint expression (small scale)	Striated	85-99
	Smooth	60-84
	Polished	50-59
C. Alteration zone	Softer than wall rock	70-99
D. Joint filling	Coarse hard-sheared	90-99
	Fine hard-sheared	80-89
	Coarse soft-sheared	70-79
	Fine soft-sheared	50-69
	Gouge thickness < Irregularities	35-49
	Gouge thickness > Irregularities	12-23
	Flowing material > Irregularities	0-11

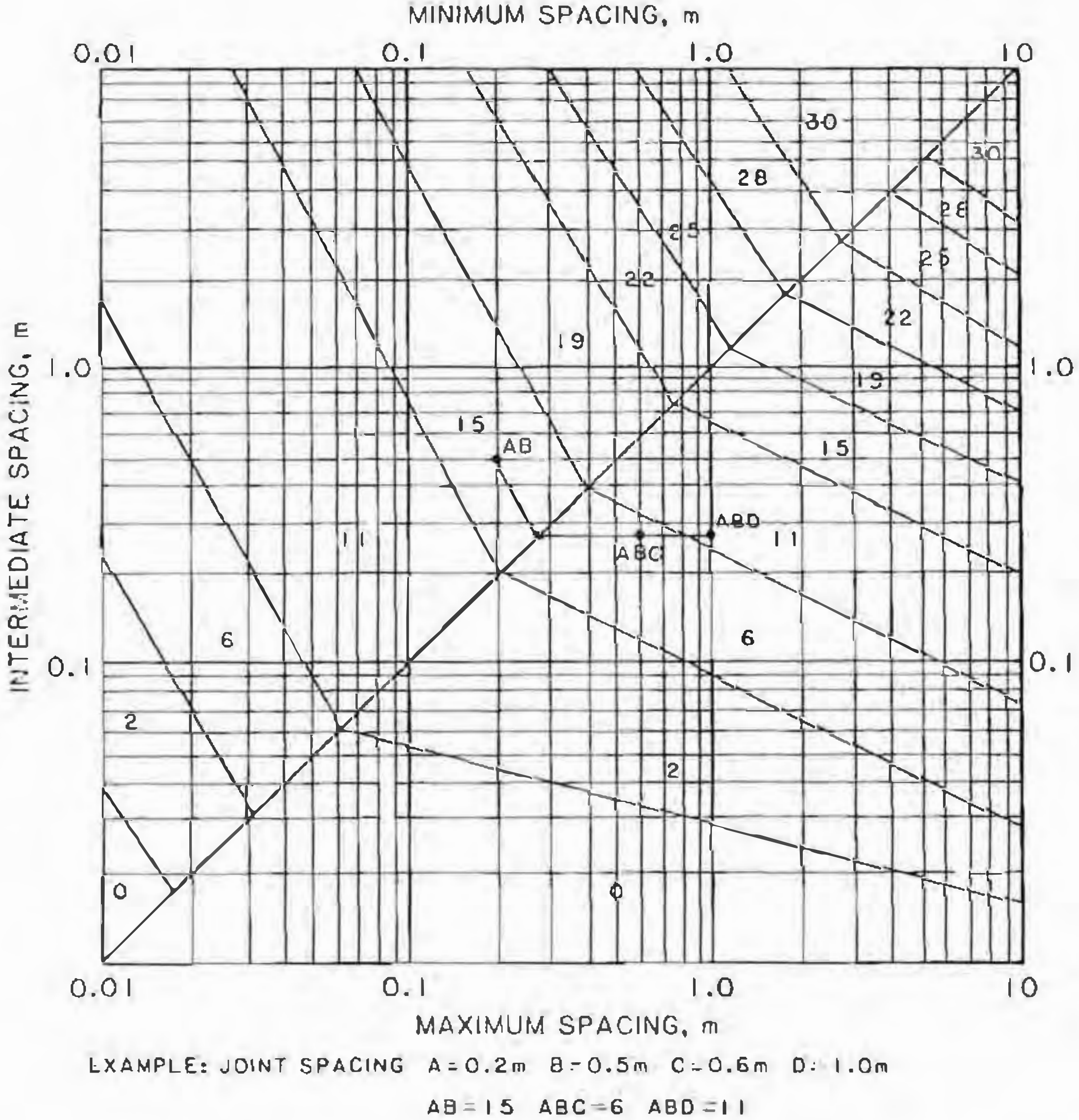

Figure 1 - Joint Spacing Ratings (Laubscher). This chart is used in connection with Laubscher's Geomechanics Classification for mining. Its use is explained in the text. Note that spacings are in metric units. Spacings measured in feet must be converted to metric units before using the chart. After Laubscher and Taylor (1976).

TABLE 2 - Ratings and Rock Mass Classes (Laubscher).
From Laubscher and Taylor (1976).

<table>
<tr><td colspan="2">Class</td><td colspan="2">1
A B</td><td colspan="2">2
A B</td><td colspan="3">3
A B</td><td colspan="2">4
A B</td><td colspan="2">5
A B</td></tr>
<tr><td colspan="2">Rating</td><td colspan="2">100 - 81</td><td colspan="2">80 - 61</td><td colspan="3">60 - 41</td><td colspan="2">40 - 21</td><td colspan="2">20 - 0</td></tr>
<tr><td colspan="2">Description</td><td colspan="2">Very good</td><td colspan="2">Good</td><td colspan="3">Fair</td><td colspan="2">Poor</td><td colspan="2">Very poor</td></tr>
<tr><td rowspan="2">1</td><td>R.Q.D.%</td><td>100- 91</td><td>90-76</td><td>75-66</td><td>65-56</td><td>55-46</td><td>45-36</td><td>35-26</td><td>25-16</td><td>15-6</td><td colspan="2">5 - 0</td></tr>
<tr><td>Rating</td><td>20</td><td>18</td><td>15</td><td>13</td><td>11</td><td>9</td><td>7</td><td>5</td><td>3</td><td colspan="2">0</td></tr>
<tr><td rowspan="2">2</td><td>I.R.S. (MPa)</td><td>141-136</td><td>135-126</td><td>125-111</td><td>110-96</td><td>95-81</td><td>80-66</td><td>65-51</td><td>50-36</td><td>35-21</td><td>20-6</td><td>5-0</td></tr>
<tr><td>Rating</td><td>10</td><td>9</td><td>8</td><td>7</td><td>6</td><td>5</td><td>4</td><td>3</td><td>2</td><td>1</td><td>0</td></tr>
<tr><td rowspan="2">3</td><td>Joint spacing</td><td colspan="11">Refer Figure 1</td></tr>
<tr><td>Rating</td><td colspan="11">30 <——————————————> 0</td></tr>
<tr><td rowspan="2">4</td><td>Condition of joints</td><td colspan="11">45° <—— Static angle of friction ——> 5°
Refer Table 1</td></tr>
<tr><td>Rating</td><td colspan="11">30 <——————————————> 0</td></tr>
<tr><td rowspan="2">5</td><td>Groundwater</td><td colspan="4">Inflow per 10 m length = 0
OR
Joint water pressure / Major principal stress = 0
OR
Completely dry</td><td colspan="3">= 25 litres/min
= 0,0 - 0,2
Moist only</td><td colspan="2">= 25 - 125 litres/min
= 0,2 - 0,5
Moderate pressure</td><td colspan="2">> 125 litres/min
> 0,5
Severe problems</td></tr>
<tr><td>Rating</td><td colspan="4">10</td><td colspan="3">7</td><td colspan="2">4</td><td colspan="2">0</td></tr>
</table>

TABLE 3 - Total Possible Adjustments
(after Laubscher and Taylor, 1976)

Parameter	R.Q.D.	I.R.S	Joint spacing	Condition of joints	Total
Weathering	95%	96%	-	82%	75%
Field and induced stresses	-	-	-	120% to 76%	120% to 76%
Change in stress	-	-	-	120% to 60%	120% to 60%
Strike and dip orientation	-	-	70%	-	70%
Blasting	93%	-	-	86%	80%
Maximum possible adjustment	-	-	-	-	50%

intact rock strength for strength reduction due to weathering. An adjustment to 82% is possible for the effect of weathering on joint conditions. The total adjustment possible is 75% but could be even lower for some rock types.

The adjustment for stress field and induced stresses depends on whether these stresses cause joints to open, to shear, or to remain in compression. The in-situ rating may be increased up to 120% if joints are kept in compression. An enhanced possibility of shear movement results in a 90% adjustment. Joints being caused to open behave as if they were filled with thin gouge, and the in-situ rating is decreased by up to 67%. This field and induced stress adjustment is mainly intended for large openings and applies to small tunnels only in exceptional cases.

Stress changes occur adjacent to caving areas, in crown pillars over extraction levels, in open pits, and due to abutment stresses and poor face shapes in sublevel mining. Using the same concepts as for field and induced stresses, possible adjustments range from 60% to 120%.

Laubscher's system considers a distinction between the pure weakening effects of discontinuities on a rock mass, and the special role that discontinuities (joints and shear zones) play in rock masses subject to large deformations.

Adjustments due to joint orientations are dependent on the freedom of rock blocks to move into the excavation. (Although this is not a condition unique to mining, it is accounted for as an adjustment rather than a basic rated item because the in-situ rating must be independent of the type of opening constructed in it.) The joint orientation adjustment is made as follows (Table 4).

TABLE 4 - Joint Orientation Adjustments (Laubscher)
From Laubscher and Taylor (1976)

No. of defining joints	No. of faces inclined away from vertical and adjustment percentage				
	70%	75%	80%	85%	90%
3	3		2		
4	4	3		2	
5	5	4	3	2	1
6	6		4	3	2, 1

The occurrences of shear zones also require adjustments to be made to the orientation term. These adjustments depend on the orientation of shear zones with respect to development: 0 to 15° = 76%, 15° to 45° = 84%, 45° to 75° = 92%. An additional 90% adjustment is applied when advancing against the dip.

Blasting may create new fractures and open or generate shear movement on old fractures and thus demands consideration. The adjustments to the in-situ rating are none for machine boring, 97% for smooth blasting, 94% for good conventional blasting, and 80% for poor conventional blasting. They are applied as shown on Table 3.

The adjustments are summarized in Table 3. Note that the maximum total adjustment is 50%.

The support requirements depend in part on the in-situ versus the adjusted values. A rock mass of in-situ class 5B (Table 2) probably cannot be reinforced, whereas a rock mass of in-situ class 3A and adjusted to 5B, could be rockbolted. In view of this, Laubscher and Taylor (1976) give the following support recommendations (Table 5). In using Table 5, it should be noted that bolts are ineffective in ground receiving a spacing rating less than 6. The support recommendations apply mainly to mines with field stresses less than (30 MPa) 4,350 psi.

2.2.5 Able's Tunnel Support Nomographs

Abel (1967), in a thorough regression analysis of rock mass behavior and set load response for the Straight Creek Tunnel (now Eisenhower Memorial Tunnel) pilot bore in Colorado, developed a series of nomographs for assessing the time to stabilization and set load response. While not exactly an RQI system, Abel's approach does permit the selection of steel rib supports based on a few easily-obtained geological parameters.

Abel (1967) performed computer analysis of set loads, rock strains, geologic conditions, and construction practices, incorporating regression analyses to determine the parameters most uniquely influencing inelastic rock mass behavior. This work resulted in a number of significant findings regarding tunnel mechanics. Only those of importance of block caving drift supports are mentioned here; the nomographs and other conclusions may be found in Abel (1967).

Stabilization time was most influenced by the thickness (not proximity) of the nearest fault zone and degree of rock alteration, and to a lesser extent by water condition, the degree of faulting or shearing, and the basic rock type.

TABLE 5 - Drift Support Requirements (Laubscher).
Taken From Laubscher and Taylor (1976).

Adjusted classes	In situ classes									
	1A	1B	2A	2B	3A	3B	4A	4B	5A	5B
1 and 2										
3A		a	a	a	a					
3B			b	b	b	b				
4A				c, d	c, d	c, d e	d, e			
4B					g	f, g	f, g j	h f, j		
5A						i	i	h i, j	h j	
5B							k	k	l	l

a = Generally no support but locally joint intersections might require bolting.
b = Patterned grouted bolts at 1 m collar spacing.
c = Patterned grouted bolts at 0.75 m collar spacing.
d = Patterned grouted bolts at 1 m collar spacing and shotcrete 50 mm thick.
e = Patterned grouted bolts at 1 m collar spacing and massive concrete 300 mm thick and only used if stress changes not excessive.
f = Patterned grouted bolts at 0.75 m collar spacing and shotcrete 100 mm thick.
g = Patterned grouted bolts at 0.75 m collar spacing with mesh reinforced shotcrete 100 mm thick.
h = Massive concrete 450 mm thick with patterned grouted bolts at 1 m spacing if stress changes are not excessive.
i = Grouted bolts at 0.75 m collar spacing if reinforcing potential is present, and 100 mm reinforced shotcrete, and then yielding steel arches as a repair technique if stress changes are excessive.
j = Stabilize with rope* cover support and massive concrete 450 mm thick if stress changes not excessive.
k = Stabilize with rope* cover support followed by shotcrete to and including face if necessary, and then closely spaced yielding arches as a repair technique where stress changes are excessive.
l = Avoid development in this ground, otherwise use support systems "j" or "k".

*NOTE: "rope cover" refers to the use of old hoisting cable as spiling support in some African mines.

Set load response was observed as far as 2,500 ft away from the point of first penetration of a major shear zone. Set load response was primarily affected by construction variables (section modulus, number of effective blocking points). Geologically, set load response was found to depend chiefly on the water conditions, degree of faulting and shearing (if "moderate" or less), and the thickness of the nearest fault zone (not proximity).

While emphasizing that every tunnel is unique, Abel (1967) nonetheless develops concepts of use in rock classifications for mining. First, the very strong effect of water on tunnel stability that is reflected in most RQI approaches is reasonable. Second, the role of width of, as opposed to distance to, the nearest fault zone is a major one and must be accounted for. This is difficult in mining, where faults are ordinarily numerous and widespread (the MBR makes use of a width-distance relationship based on Abel's findings). Third, stabilization time should be considered when installing massive concrete linings, because it is undesirable to mobilize some portion of the lining capacity to stabilize the drift during development. Abel's absolute stabilization times and set loads are not necessarily applicable to mining, because of the geometric complications, but some ground intersected by the Straight Creek pilot bore is similar in structure to that of many caving mines.

2.2.6 McDonough System

McDonough (1976) reports on a system of ground classification in use at the Climax Mine. The system relies heavily on geologic judgement, and involves rating the rock mass competence on a scale of 1 to 9, based on rock type, core recovery, RQD, longest piece of core, number of pieces of core, number of joints per foot (of core), argillization (rated on a scale of 0 to 9), and distribution of faults and shears. Not all of these would always be used in assigning a competence rating, which is done both as part of the core logging procedure and during underground mapping.

The system requires considerable experience with the Climax geological conditions. Although it is not strictly limited to Climax, its use in other mines has not been reported. McDonough (1976) discusses the features attributable to rock of each class, ad provides photographs of representative core. The method is still relied upon with considerable success for projections of ground conditions and development planning. Maps of ground conditions showing the predicted classes aid in this regard.

Support for each type of ground varied depending on the type of opening under consideration, and different types of support may be installed in the same class of ground in the same type of opening, depending on conditions as encountered. A table is provided, giving such typical support practices (McDonough, 1976, p. 125). Support varied from none in the best ground to rigid steel sets with bolts and reinforced concrete linings in the worst ground, depending on the type of drifts or intersection.

2.2.7 Correlations Between Systems

Much effort has ben directed at correlating the ratings obtained when applying the various RQI systems to the same rock mass. Most of these consider Q, RSR, RMR, although there are doubtless many others that are not reported in the literature.

Rutledge and Preston (1978) developed the following correlations, based on New Zealand experience:

$$RMR = 13.5\ (\log_{10} Q) + 43$$

$$RSR = 0.77\ (RMR) + 12.4$$

$$RSR = 13.3\ (\log_{10} Q) + 46.5$$

Bieniawski (1976) suggested the relation

$$RMR = 9\ (\log_{10} Q) + 44$$

as describing the RMR and Q relationship.

Taylor (1980) observes that the results obtained by the RMR are somewhat higher than by Laubscher's system, and attributes this to the fact that the RMR uses a mean spacing criterion, while Laubscher's system uses a multi-joint rating scale. Taylor (1980, p. 37) provides a graph of ratings from Laubscher's system versus ratings from the Q system. It can be determined from this graph that the relationship can be described by (Laubscher rating) = 19.5 (log Q) + 43.

For conditions at the Henderson Mine, Brumleve and Maier (1981) found the following relations (assuming SRF in the Q system equal to 1.0).

$$RMR = 5.47\ (\ln Q) + 46.18$$

$$RSR = 1.119\ (RMR^{0.785}) + 8.191e^{0.0196\ RMR}$$

$$RSR = 11.43\ (\ln Q) + 38.62$$

They note that the relation for RMR and Q corresponds to that of Rutledge and Preston (1978).

At Climax, a number of relationships have been tried in relating the McDonough system to RMR, and correlations better than 0.90 have been found empirically and theoretically, for linear, power, and exponential relationships (White, oral communication, 1981). Presumably, this is due to the subjective nature of the McDonough system and inherent difficulty relating the McDonough and RMR values precisely. One preferred empirical relation is

$$M = 1.1397\ e^{0.0326RMR}$$

although others have been found useful as well.

During the investigations leading to the development of the MBR system, the relationships of Rutledge and Preston were tried using the field data. The following trends emerged:

1. Using the RMR, the predicted RSR values were almost always higher than the rated RSR. The difference is usually less than 15% of the rated RSR.

2. Using Q, the predicted RMR was always lower than the rated RMR. The difference was ordinarily at lease 30% of the rated RMR.

3. Using Q, the predicted RSR values were almost always lower than the rated RSR, between 2% and 42% of the RSR rating.

4. The agreement where Q is involved is less reliable in highly fractured rock because J_n is less reliable.

5. Because the Q system is designed to be applied to the most unfavorably oriented set, it is more sensitive to highly ordered systems than the RMR and RSR, where orientation is taken into account.

From this experience and that report by others, it is clear that the correlation between RQI systems may vary with different geologic settings and different levels of experience of the user.

2.2.8 Sensitivity Analysis

Some parameters are more difficult to evaluate than others, so RQI systems placing a heavy reliance on such parameters will yield less reliable results. In order to identify those geologic conditions most extensively influencing the final rating, the Q, RMR, and RSR were subjected to a sensitivity analysis.

This analysis was initially carried out by varying one parameter while holding the others constant in each system. Recognizing that the geological factors being evaluated are commonly associated (for example, weathering, RQD, and intact rock strength), the results of the initial study were re-examined based on field data. The following conclusions were drawn.

Q-System

1. RQD - The Q-system is uniformly sensitive to RQD. The principal difficulty is assigning an RQD value to the rock mass value considered.

2. J_n - The Q-system is very highly sensitive to J_n in rock with two or fewer joints sets. Fortunately, two-set systems are more readily recognized than systems with more diverse fracturing. However, recognition of the number of joint sets in more highly fractured rock is difficult, and even a slight misassessment of the number of sets from "two plus random," to "three plus random," coupled with a 10% error in RQD can throw Q off by a factor of approximately 2.5.

3. J_r - The range of values for J_r is small, and Q is uniformly sensitive to it, to about the same degree as RQD. This parameter presents little problem if the most unfavorable set can be readily identified.

4. J_a - Q is more sensitive where rock wall contact is observed, especially for the stronger joint conditions. However, this nonlinearity is less by an order of magnitude than that for J_n. This parameter is also difficult to rate, and serious miscalculations can result, due to the wide range of possible ratings.

5. J_w - Q is uniformly, but highly sensitive to J_w. However, this is one of the most straightforward parameters to rate, so miscalculations are less likely.

6. SRF - The range of values is large and a high degree of experience is required to properly evaluate this parameter. The sensitivity is very similar to that of J_a.

7. Quotients - In field data from this study, Q was found to be more sensitive to J_w/SRF than to J_r/J_a. Poor correlations were obtained for RQD/J_n, probably because of its high sensitivity (RQD/J_n may vary by 400%). Overall joint condition determined Q to a greater extent than RMR or RSR.

RMR System

1. The Geomechanics system is uniformly sensitive to all its input parameters, when they are considered individually. Thus, the degree of accuracy required for each is about the same.

2. Only slight variations in sensitivity were observed for the combinations of RMR input variables generated from the field study sites.

3. The "support intensity," as indicated by recommended internal support pressures based on the ratings, are most highly dependent on joint conditions and water conditions, are somewhat less dependent but about equally so on RQD and joint spacing, and depend the least on intact rock strength. This is in keeping with the intent of the RMR, which considers joint condition to be the most important factor for tunnel stability.

RSR Concept

1. Parameter A: Rock Type, Structure - The response of the RSR final rating to variations of the Parameter A components is fairly uniform throughout the ranges of these variables. Overall, the RSR is slightly more sensitive to structure than hardness or rock type, and errors can be significant.

2. Parameter B: Spacing, Orientation - The RSR responds fairly uniformly to individual variation in spacing and orientation; however, errors in assessing spacing can make a great difference in the RSR.

3. Parameter C: Groundwater, Joint Condition - The RSR is again uniformly responsive changes in both groundwater and joint conditions. Each affects RSR to about the same degree as hardness and rock type.

4. With regard to recommended internal support pressure: in our field data, spacing is by far the most important determinant, followed by groundwater, joint condition, and structure, hardness, and orientation, which have only a slight effect. The rock types in all our field data were igneous.

3.0 BLOCK AND PANEL CAVE MINING

It is assumed that the reader is familiar with block cave mining methods. Much has been written on these mining methods and a complete discussion of block caving would be lengthy and of doubtful utility to this report. This section will discuss those features of the caving method that are considered in the development and use of the MBR system.

For a general overview, the reader is referred to Julin and Tobie (1973) or Tobie and Julin (1982). Stewart (1981) edited an extensive volume on various aspects of the design and operation of caving mines, that includes many general discussions of the methods considered in the following sections.

3.1 Basic Description

Caving mines utilize the force of gravity to the fullest extent possible to fragment ore and move it to loading points. In order to do this, a large area of ore is undercut so that it caves, and in so doing, breaks itself up. The ore is then withdrawn in a systematic manner.

Caving orebodies must be weak enough to cave yet strong enough to be supported, must be of fairly homogenous grade, and must break into manageably small fragments. These characteristics generate a concept of a caving orebody such that certain simplifications and allowances are made when a RQI approach is to be applied to mining as opposed to tunneling. These are:

1. Weaker rock masses predominate.

2. The geological environment is one of structural discontinuity in the form of fractures and faults.

3. Altered and weathered rock is common if not predominant.

4. The complex structural fabric is overprinted onto broadly systematic variations in alteration and mineralization.

5. Extensive horizontal area.

Similarly, a concept of a block caving mine would have these components.

1. An undercut horizon where caving is initiated.

2. An extraction horizon where the caved ore is withdrawn, with or without intermediate levels to promote efficient ore handling.

3. A haulage level for transporting ore away from the production area and eventually to the surface.

4. Connection of undercut, extraction, and haulage horizons by raises for passage of men and ore.

5. Attendant service drifts and required ventilation drifts that may or may not be wholly within the extraction horizon.

Other general characteristics of importance to this investigation include the following.

1. Areas in production at any given time are extensive, and sites of ore withdrawal must be closely spaced, so that a high density of production workings is necessary.

2. A large amount of pre-production development is required, and production lead times are therefore long.

3. The mining method produces extensive changes in the initial stress field, due to the development and maintenance of a caved volume.

4. High transient stress levels (strains) may be achieved, and this in conjunction with necessarily weak rock masses dictates a tendency for heavy support.

5. Because of the capital-and time-intensive nature of the method, high production rates are emphasized, a highly systematic approach is maintained, and apparent minor adjustments to the mining system can have far-reaching effects.

Tobie and Julin (1982) classify the form of the cave into three distinct groups. In true "block" caving, the orebody is divided horizontally into blocks, most commonly square, and draw is evenly distributed over the entire area so that the ore/capping contact remains at its original angle. This method has been used more extensively in the past. More common in present-day mining is "panel" caving, where the orebody is divided horizontally into enlongate panels, and is undercut and caved by retreating from one end to the other (the panel may be divided into "blocks" for this), so that the ore/capping contact angle is changed. Pillars may or may not be left between panels. In "mass" caving, there is no division of the orebody into blocks or panels. Undercutting of manageable areas proceeds across the level from one end of the orebody to the other, changing the initial angle of the ore/capping contact. The active cave area is defined by the size of "block" that will not produce undue stresses, and by production requirements.

3.2 Variations of the Method

Aside from the common aspects listed in Section 3.1, there are almost as many variations of the block caving method as there are mines, because once it is begun, the method of a given mine inevitably evolves to fit the particular characteristics of the orebody. There are two major classes that can be distinguished, based on the type of ore transfer mechanism. Pillar (1981) discusses the advantages and disadvantages of each.

The full-gravity approach (Figure 2) uses only gravity to move the ore to the haulage. For this, a weak, flowing ore is required. The classic example is the San Manuel Mine; others include El Teniente and Rio Blanco in Chile and Lutopan in the Phillipines. There are three or more levels (the Inspiration Emma mine had 4 levels) per caving lift, in order to accommodate the transfer raises necessary to isolate the haulage level from the caving area in the weak ore. By inclining transfer raises, it is possible to have a wider spacing of haulage loading points than drawpoints. A level intermediate between the haulage and undercut known as the grizzly, permits control of the size of ore sent down the transfer raises.

The grizzly level is normally a short (less than 50 ft) distance beneath the undercut, and the haulage is much further (up to several hundred feet) below the grizzly.

The full gravity method uses less mechanized equipment and requires more development, and therefore higher ore columns, than the slusher/LHD system, on a per-ton basis.

The other broad class of caving mine layout uses some mechanical means of transporting ore to the loading point. Thus the grizzly level, and for some layouts, the transfer raises, are omitted. The means of ore transfer may be by slusher (Figure 3) or LHD (Figure 4), and these comprise two distinct subgroups. Although the design and operation of slusher systems is radically different from that of LHD systems, the basic development frameworks are similar.

Figure 3 shows a classic slusher system patterned after that currently in use at Climax. At other mines, different layouts and dimensions may be used. The slusher lanes at Lakeshore do not alternate and are longer, incorporating more drawpoints, than those in Figure 3; the ventilation drift is on the same level. At the King Beaver Mine, muck is slushed into chute raises to the haulage, rather than directly into muck trains. Mucking at several other mines is similarly done into transfer raises to the haulage. Also, in Figure 3, two sideline drifts are shown feeding each drawpoint. In other layouts, each drawpoint is served by a single finger raise that may be vertical or inclined.

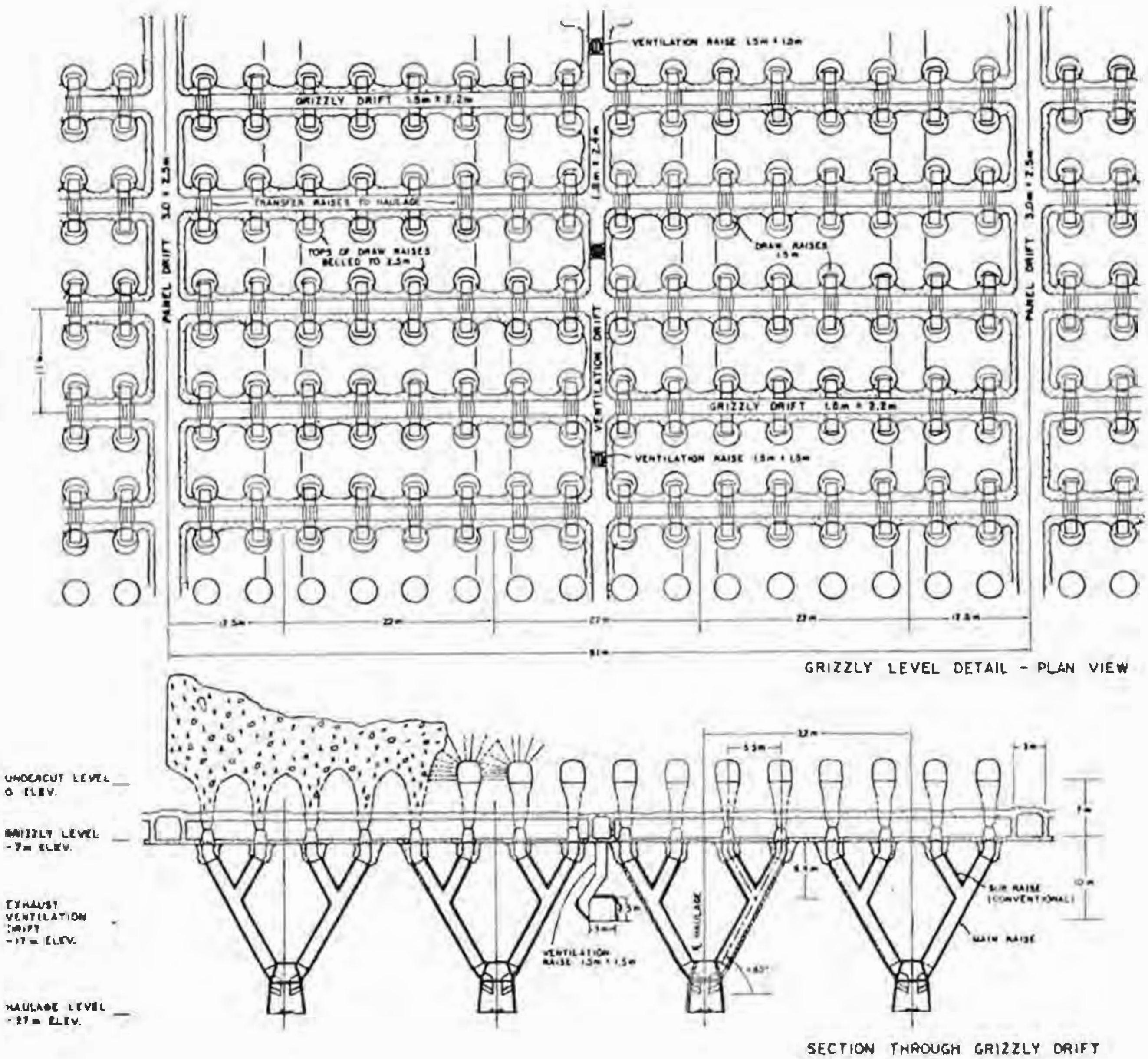

Figure 2 - Typical Gravity Caving Scheme (From Pillar, 1981, p. 90) Dimensions and layout details may vary from mine to mine (m = meters). Used with permission of the Society of Mining Engineers of AIME.

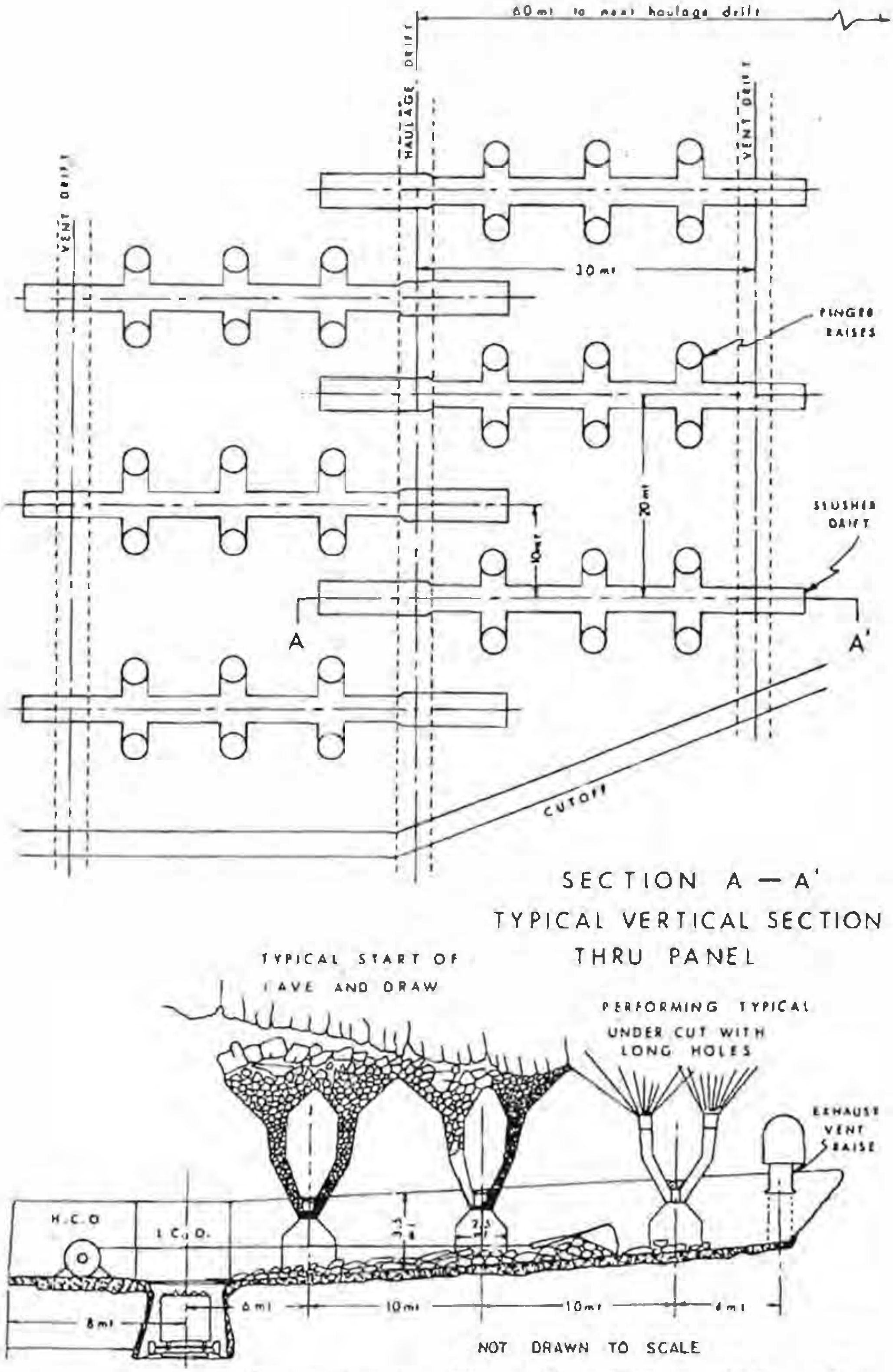

Figure 3 - Typical Slusher Drawing Scheme (from Pillar, 1981, p. 92). This is for a mass caving mine. In narrower deposits or where blockwise development is used, the alternating slusher drifts shown may be replaced by longer drifts running the same direction (mt = meters). See text. Use with permission of the Society of Mining Engineers of AIME.

Slusher draw methods require far less development than gravity methods and are better suited to more competent ores. Draw control may present problems since it is easy to overdraw from more freely-running drawpoints. Slushers are especially vulnerable to ground weight problems, and the maintenance of even draw to prevent this commonly results in reduced production, because freely-running drawpoints may frequently have to be put on hold.

The example of the LHD method is taken from the Henderson Mine (Figure 4). LHD drawing methods enable mining of coarsely fragmented ore in more competent massive deposits.

The LHD system affords the best draw control and flexibility, high production, and is preferred where a large degree of mechanization, high capital cost, long pre-production lead times, wide drawpoint spacings, high ventilation requirements, and large opening cross-sections can be tolerated.

Variations on this method include substitution of direct loading for transfer raises, the use of rock breakage equipment in the transfer raise area with inclusion of a chamber for the purpose, the configuration of the ventilation network, and a re-design of the production level layout to straighten the cross-cuts between haulage drifts.

In summary, it should be emphasized that present mining systems of the various types mentioned were developed in response to conditions prevailing at individual orebodies. Future mining systems will certainly incorporate innovations in layout and level design, to cope with changing and new conditions.

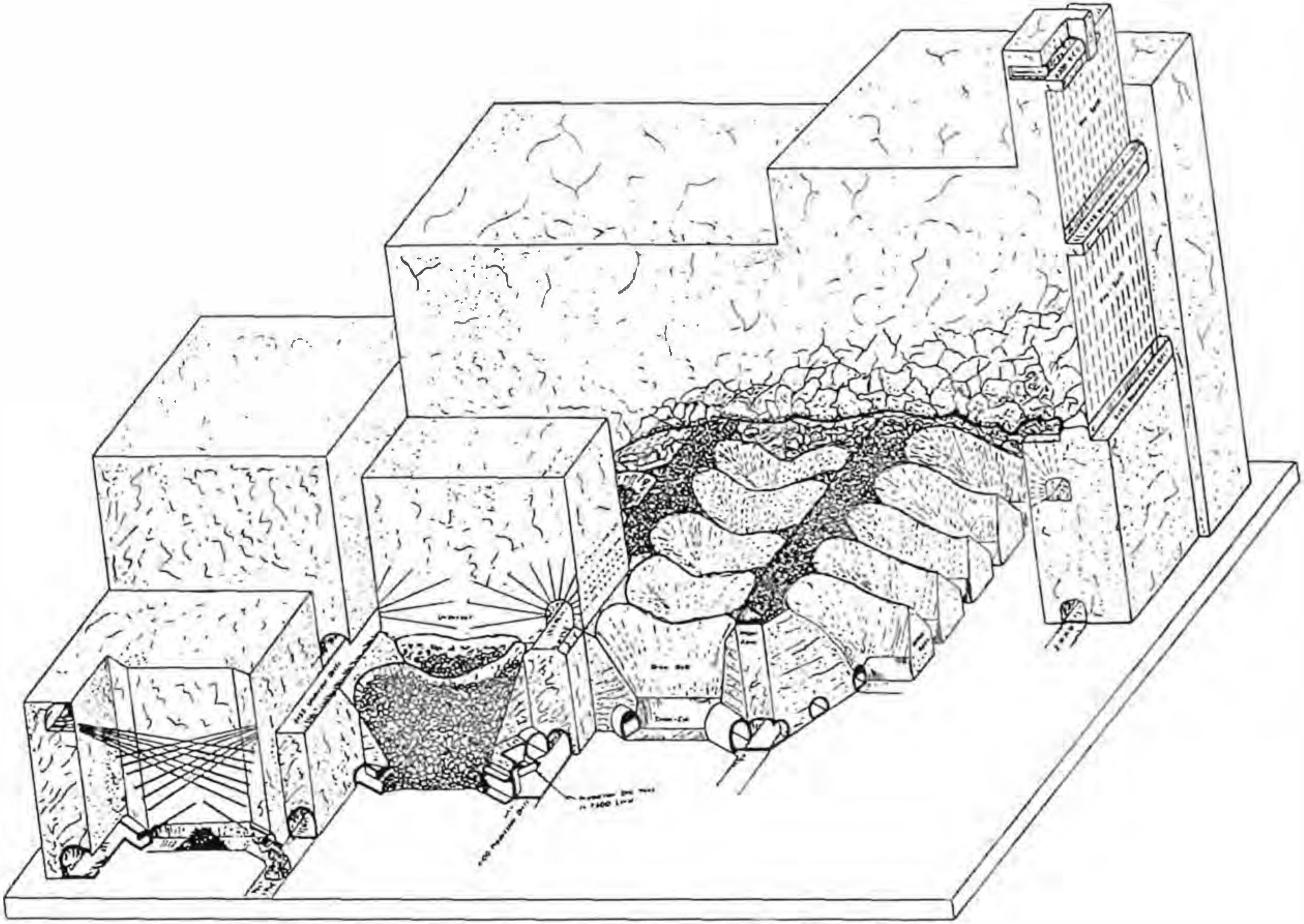

Figure 4 - Typical LHD Drawing Scheme (from Brumleve and Maier, 1981). The production level layout may differ from mine to mine. Not shown is a transfer raise and ore pass system that in this case transfers ore a vertical distance of 600 ft. At some mines, the production drifts are known as haulage drifts, which should not be confused with the haulage level. Used with permission of the Society of Mining Engineers of AIME.

4.0 KEY PARAMETERS AFFECTING DRIFT SUPPORT

This section addresses the initial work in the development of the MBR system, the identification and combination of those parameters most influential in drift support design.

For this selection, it is necessary to consider parameters active at each point in the caving cycle. During this cycle, the rock mass undergoes an initial stress change as the development workings are driven. When the undercut level is developed, the rock mass is partially unloaded, and severe stress changes occur as the undercut approaches (abutment stress). These loads then decrease when the undercut is directly overhead. During caving, erratic load changes occur due to point loadings and ore drawing.

At different times, the load on a drift support element is controlled by different factors. These factors may be summarized as geologic variables, geometric variables (related to mine plan), and development/production variables (related to the creation and use of the opening). Drift support design must then consider these factors throughout the mining sequence.

4.1 Geologic Variables

Geologic variables identified as most greatly affecting caving drift support requirements and behavior are rock competence, stress field, and structural setting. When taken together, these three elements evaluate basic rock strength, account for the distribution of various strengths and the interaction of these various strength zones, and provide a basis for assessing the modification and migration of whatever loads are produced by mining.

4.1.1 Basic Rock Mass Class

The overall rock mass competence is independent of the type of opening intended. A good measure of rock mass competence is rock classification, but the class must be determined in such a way that it is not unique to any particular type of drift.

As discussed in Section 2.1, popular current rock mass classifications all consider rock strength, fracturing, and water condition, in some way. The Q, RSR, and RMR were assessed for each case history used in developing the MBR.

4.1.2 Field Stresses

The field stress state is the initial condition from which the induced stresses develop. The interplay between mining and the field stresses determines the induced stress at a given point. Most tunneling classifications do not consider stress state because the tunnel geometries are simpler and the role of induced stress can be implicit in the classification.

4.1.3 Structural Setting

Structural setting as used here refers to the distribution and character of large-scale structural discontinuities. Small faults and the fracturing pattern are already accounted for in the basic rock mass class. Large faults are classified separately. However, structural setting is considered separately because the effects of the extensive rock mass movements involved in caving may move or change in the presence of major structural discontinuities. In rock classifications for tunneling, faults and other weakness zones are either considered in the aggregate, as weakening (producing a lower rating for) the rock mass in general, or are classified separately.

4.2 Geometric Variables

Geometric variables are non-geologic variables that determine the magnitude and distribution of stresses that can develop due to development and production. Geometric variables are related to the mine plan or layout and influence both the static loads experienced during development, and the dynamic and transient loadings experienced during production.

4.2.1 Distance to Cave Line

During undercutting and caving, the stresses generated are transmitted through rock to the drift support systems. Of particular importance are abutment stresses, which are concentrated at some distance around the margin of the caved volume. The emphasis is on abutment stresses because if caving proceeds smoothly and draw control is effective, abutment stresses pose the greatest potential for drift damage. The greater the volume of rock between the drift in question and the point of stress application, the less will be the stress realized.

The point of stress application is assumed to be at the floor of the undercut, beneath the leading edge of the cave line, when abutment stresses are being considered.

Other sources of caving stress (stubs, point loads, packed draw points) are operational in nature and cannot be implicit in an RQI approach without prior knowledge of caving behavior. However, the point of application is still near the floor of the undercut.

When considering this effect, the distance used is the shortest distance to the point of application of the stress. This may be a vertical or horizontal distance, depending on the type of drift in question.

4.2.2 Extraction Ratio

The drift spacing and size determines the volume of rock available to bear the load imposed on the level. The more rock is left, the less is the demand on the drift support system. A convenient measure of this is the extraction ratio.

The extraction ratio is commonly used in coal mining as an expression of the fraction of the resource removed. It is obtained by dividing the volume by the volume of the resource originally present. For a single level, it may be considered the fraction of area occupied by workings. See Appendix A for calculation of extraction ratios. For purposes of computing the extraction ratio in the MBR, it is assumed that the imposed loads act at the springline, where the opening is the widest and the extraction ratio is the highest.

4.2.3 Panel Size

Small panels or blocks develop less weight than large ones, because the volume of caved material is not as large and therefore the necessary redistribution of stresses is not as extensive. Also, it is less likely that caving stresses will be thrown outside the production area with large blocks than with small ones.

This effect becomes negligible once the block size exceeds a few hundred feet, although a slight dependence on ore competence is probable in this regard.

4.2.4 Depth

The role of depth in the generation of caving stresses is not well understood. It seems clear from operating experience that ores of a given competence generally cave more readily if deep than if shallow, and that shallow drifts are more readily supported than deep drifts in ores of similar competence. Depth would also affect the stress field and thereby the induced stresses. Depth would affect the magnitude of abutment and caving stresses if higher ore columns are used.

4.2.5 Location Within Panel

Caving stresses are higher towards the center of caving blocks than towards the boundaries. During undercutting, points near the start of the undercut will experience only partially-developed abutment stresses. These factors will affect the loads exerted on the drift support, but are most likely to be overshadowed by other influences.

4.3 Development/Operational Variables

Production and development variables are tied directly to the creation and use of the opening. In other disciplines, they might be described as construction and engineering variables. These affect rock mass competence and loading.

4.3.1 Excavation Practice

The creation of an underground opening unavoidably affects the rock surrounding the opening. Rock competence is reduced if blasting damage occurs. Even with controlled blasting or machine boring, simple elastic relaxation will occur due to excavation, and the resulting dilation will be beneficial for some rock types and will be detrimental to others.

Regardless of rock competence, poor blasting practice will create new fractures and widen and lengthen others, reducing the effective load-bearing area of pillars. Poor blasting practice is exhibited by excessive overbreak and barring-down requirements, loose, drummy, ravelling ribs and crown, and unshot blastholes.

Ideally, one should have prior excavation experience specific to the rock mass in question in order to assess blasting damage for classification purposes. Although this will not be the case in advance of mining, it is felt that the large base of experience with blasting that is available at most operations will enable good estimates.

4.3.2 Necessary Standage Time

Standage time is used here as an expression of the length of time the opening must remain stable before permanent support is installed. This includes standup time, which is related to unsupported span, and is used in tunneling to determine the necessary proximity of the support to the face.

Standage time therefore is a matter of the development-production sequencing. Openings requiring long standage times should be more conservatively supported than openings with short standage times.

4.3.3 Degree of Acceptable Repair

Some types of openings must be more durable than others. For example, failures of supports in fringe drifts, panel drifts, and haulageways are far more disruptive to mine operations than are failures in grizzly drifts or slusher lanes. Furthermore, deformations at the extraction level are more tolerable in some mining methods than others. For example, grizzly drift deformations in gravity caving are not nearly as serious as the same deformation in a slusher lane or LHD crosscut.

Each operation has its own requirements for drift support performance, but at every operation, the support philosophy applied will recognize the purpose for the drift in question. Those drifts not commanding as conservative a support philosophy will have a higher degree of acceptable repair. The assessment of repair frequency will then determine the drift support selected.

4.3.4 Mining Stresses

Mining stresses include initial induced stresses due to development, abutment stresses due to undercutting, and caving stresses due to production. The first two types have been accounted for by variables discussed previously. Caving stresses however, are not adequately described by mine layout and rock mass conditions. Human and operational factors essentially govern the magnitude that caving stresses will obtain, and these are only indirectly related to RQI. It is felt that caving stresses can be kept at levels lower than abutment stresses as long as careful draw control is practiced.

There is not yet sufficient information available to determine empirically a relationship between RQI and draw control. It is recognized that poor draw control can override all other mechanisms of drift support distress. Possible approaches for predicting such problems on the basis of RQI are mentioned in Section 8.

4.4 Uncertainties in the Assessment of Variables

Uncertainties in evaluating the variables listed in the preceding sections are found chiefly in the areas of geology and excavation practice. Even with considerable underground exposure, the assignment of typical values for the rock mass competence, structural setting, and stress field is not straightforward and requires considerable judgement and experience. Consequently, great effort has been devoted in the MBR system to clarifying the method of rating each parameter.

It is recognized that field stress determinations may not be available, even at mines that have been in operation for many years, so the impact of this parameter is minimized.

The method of combining the individual parameters, and indeed the choice of the parameters themselves, has been drawn from field observations by the project team and from published reports on caving rock mass behavior. Through continued and widespread use, it is expected that modifications will be suggested as a broader base of data becomes available.

5.0 FIELD DATA

In support of this investigation, field data were collected at four United States mines using block, panel, or mass caving methods. These data, as discussed in Section 1.2, include general geologic conditions, rock mass ratings for the Q, RMR, and RSR systems, excavation and support practice, production statistics, and support performance. Interviews with knowledgeable personnel were of great value. The mines were most cooperative in providing access to requested data.

These data are summarized in the following sections. To avoid disclosure of proprietary or otherwise sensitive information, the mines are not identified by name.

5.1 Mine A

5.1.1 Geologic Overview

Mine A is in a large, low-grade, disseminated porphyry deposit. Ore formation occurred upon intrusion of a porphyry into earlier granites, accompanied by pervasive alteration. The intrusion was followed by emplacement of unmineralized, acidic dikes and later, large-scale faulting that truncates the ore. Several major faults transect the orebody. In general, the ore is weak and caves readily.

The portion of this large orebody that was studied in detail contains one major fault zone trending northwest and dipping moderately to the southwest, accompanied by subsidiary east-west faults.

The geologic setting, for purposes of this analysis, was generalized into three structural domains, corresponding to the intrusive porphyry, the host granite, and a severely faulted area. The characteristics for each of these domains in the area of interest are discussed in Section 5.1.4.

The area considered comprises half a level and incorporates active, worked-out, and development panels. The faulted zone occurs within the western third of the study area and trends across the short dimension of the level.

5.1.2 Mining Practice

5.1.2.1 General

Mine A uses a full gravity caving system. Undercutting is by blocks of various sizes (at least 210 ft long), along panels 140 ft wide. Ore is caved in lifts of either 300 or 600 ft, depending on the attitude of the mineralized zone. Caving is presently being performed at depths from 2,000 to 3,000 ft.

Mining is accomplished through a pair of levels. The upper level is the mining or grizzly level and the lower level is the haulage level. Caving is initiated by completely undercutting the block above the grizzly level. Broken ore flows down through finger raises to the grizzly level. Here, the ore is manually reduced in size as necessary, so it passes the grizzly. It then flows down transfer raises to the haulage level by the action of gravity. Ore trains at the haulage level load and haul the muck to the production shafts where skips hoist the ore to the surface.

5.1.2.2 Development

In the primary development stage, peripheral drifts called fringe drifts are driven both at the grizzly and haulage levels along the orebody contacts. These are usually driven 12 ft wide by 12 ft high with conventional drilling and blasting.

Once the fringe drifts have sufficiently advanced, haulage panel drifts are started. These cross the orebody on 70 ft centers and are centered below each half of each panel. Panel drifts are connected with vent crossovers at various places to facilitate ventilation and emergency access. The haulage panel drifts, which have an arched section, are driven much the same way as the fringe drifts and have the same dimensions.

Concurrent with haulage panel development, grizzly panel drifts are driven on the grizzly level. These are driven across the orebody on 290-ft centers to serve pairs of panels. These have an arch-shaped cross-section, about 10 ft by 10 ft in size. Overbreak amounts to about 1 ft at the crown and ribs.

Secondary development, or block preparation, starts as soon as a sufficient amount of primary development is accomplished to allow access to the mining area.

At the haulage level, secondary development starts after the excavation of raise stations, which are spaced every 35 ft along the haulage panel drifts. Transfer raises are driven from each side of the raise station at an angle of 63°. They are normally 4-ft square and are driven blind with a bell at the top.

After the transfer raise reaches the grizzly level, the grizzly drifts are driven. These are on 35-ft centers, at right angles to the panel drifts, over the tops of the raises. The grizzly drifts are driven 9 ft high by 8 ft wide.

When several grizzly lines are complete, an access drift is driven parallel to the panel drift at the far end of the block to provide ventilation connections and emergency access.

Next, finger raises are driven from the grizzly drifts to the undercut elevation. These are roughly circular in cross-section with 5 ft minimum diameter.

5.1.2.3 Undercutting and Caving

The undercut drifts are driven 5 ft wide by 7 ft high on 17.5-ft centers over the tops of the draw raises, 15 ft above the grizzly drift floor, and at right angles to the grizzly drifts. Caving is initiated by drilling and blasting the pillar between drifts or at a boundary of the block. Before each pillar is blasted, the drift is widened on one side about 4 ft. The remaining pillar, about 8 ft thick, is shot to a height of 13 ft above the floor of the undercut by means of ring drilling. Generally, a 15 ft section along the drift length is taken with each blast and the swell muck is drawn off immediately thereafter.

The mining sequence has gone through a number of changes, finally envolving into diagonal retreat panel caving by blocks. With this system, undercutting is usually started against a caved block, retreating diagonally toward solid corners.

5.1.3 Ground Support Systems

5.1.3.1 Fringe Drifts

Support practices in the fringe drifts vary depending on the nature of ground encountered and the span to be supported. In areas of relatively stronger ground, support is afforded by 12 in. by 12 in. timber sets on about 5-ft centers, with 4 in. by 6 in. spreaders near the crown and the floor.

In areas of anticipated heavy ground and sections close to active mining blocks, 4 in. or 6 in. wide-flange arch steel sets are installed for initial ground support. There are usually 3 blocking points on the ribs and 4 at the crown. Poured monolithic concrete is then applied for permanent support up to a thickness of 18 to 24 in.

Variants of this support practice exist in some areas where steel sets may be used for permanent support, or concrete may be applied over timber sets.

5.1.3.2 Grizzly Panel Drifts

Pre-concrete support in grizzly panel drifts generally consists of 5/8 in. diameter ungrouted mechanical bolts with steel straps. Concrete is applied to a nominal thickness of 18 in. for permanent support. On deeper levels, however, most panel drifts are supported with 12 in. by 12 in. timber sets or 4 in. to 6 in. wide flange steel sets and concrete.

The average life of grizzly panel drifts is about 3 to 4 years.

5.1.3.3 Haulage Panel Drifts

Haulage drifts are supported similarly to the fringe drifts with either timber or steel sets for initial support. However, since these are wholly beneath the orebody limits, they are usually fully concreted.

5.1.3.4 Grizzly Drifts

Unless the ground dictates otherwise, all grizzly drifts are driven using 6 ft rock bolts and wire mesh for pre-concrete support. If necessary, 6 in. x 8 in. timber sets are used to supplement rock bolt support. Grizzly drifts are concreted within a few weeks of being driven, with a minimum of 18 in. of concrete on the side and 2 ft in the back. Finger raise windows are formed, and an 8 in. wide flange wear cap is placed across the top of the windows to protect the brow. In areas where heavy ground weight is anticipated, 5/8 in. reinforcing bars are placed on 6 in. by 12 in. centers at the finger raise openings. The reinforcing also tends to reduce the damage created by secondary blasting during draw operations. Grizzly drift life averages about 1½ years.

5.1.3.5 Undercut Drifts

Undercut drifts are timbered with 6 in. round posts and 6 in. by 8 in. caps. The timber is drilled with wood augers and shot with the undercut round. Due to the short life of these drifts and their position relative to the mining block, undercut drifts rarely experience ground control problems.

5.1.4 Scope of Field Data

The rock masses at Mine A were divided into three structural domains (intrusive porphyry, host granite, and faulted area) as mentioned above. Geotechnical information for each domain was assembled so that classifications, according to the Q, RMR, and RSR systems, could be made. Additional geotechnical information in the mine's files, pertaining to stress field, fracture strength, and support design, was also considered.

Joint orientation and spacing data were assembled from publications and were spot-checked in the field. Joint conditions and groundwater were determined underground. RQD, core recovery, and intact rock strength were obtained from core logs in the mine's files and from published data.

The general structural setting was derived from mine maps, interviews with geological staff, and first-hand observations.

Together with this, the excavation and support practices, including blasting, were observed, and historical information on this subject was obtained through interviews with knowledgeable mine personnel. Support performance was evaluated through first-hand observations in the area selected for study.

5.1.5 Rock Mass Conditions

Representative basic geotechnical data for Mine A are summarized in Table 6. Q, RMR, and RSR ratings for Mine A are given in Table 7. Data for the acid dikes were collected but were not used in MBR computations. Acid dikes are the most competent rocks in the ore zone, but are irregular in occurrence. Their competence is due to a combination of high compressive strength, weak or no alteration, and widely-spaced, rough fractures.

Due to pervasive alteration and fracturing, the host granite and intrusive porphyry have similar ratings. The difference in Q value for host granite in Table 7 arises from a slight difference in joint roughness (J_r) and a difference in compressive strength, which affects SRF. The intrusive, possibly because of its greater strength, exhibits sharper, more angular fracturing than does the host granite. The two rock types respond similarly to mining stresses, although some feel that the intrusive porphyry is slightly more competent.

It is clear from Table 7 that all three RQI systems are sensitive to faulting.

5.1.6 Encountered Support Performance

Fringe drifts on the grizzly level evidence ground movement and excessive weight. Large-scale spalls in concrete at the crown and ribs are seen within about 70 ft of nearly all panel drift turnouts. Similarly, damage extends down panel drifts for about the same distance from the fringe drift junction, indicating that the wedge of rock between is highly stressed. Elsewhere, cracks and spalls do occur, but are not as severe or systematic. Timber supports are exposed near some turnouts and, although not failed, generally exhibit ground problems in the form of crushed wedges and blocking, split caps and posts, and deformed or broken lagging. The distance from the grizzly fringe drift to active production areas varies from 100 ft to over 300 ft, and damage, although present, does appear to be less severe at the greater distance. The most severe damage was found within 100 ft of the faulted zone.

Grizzly panel drifts all show damage due to ground movement. Spalling of the crown and shear at the back are widespread. This is ordinarily arrested through the application of jump sets and shotcrete.

TABLE 6 - Geological Data for Mine A

	Rock Mass Category		
	INTRUSIVE PORPHYRY	HOST GRANITE	FAULTED
Intact Rock Strength, psi (1)	19,300	13,500	8,000
RQD/Recovery, % (2)	30/50-80	40/60-100	15/30-80
Joint Spacing, ft (3)	0.40	0.45	0.2
Joint Condition (4)			
Roughness	rough	slightly rough	slickensided
Separation, in.	.01-.05	.01-0.1	0.1
Filling	clays, chlorite	clays, chlorite	clay, gouge
Weathering	softened	softened	very soft
Joint Orientations (5)			
Set 1	N60E, 80NW	N75E, 80NW	N30W, 80NE
Set 2	N30W, 75SW	N40W, 80NE	N30W, 80SW
Set 3	N10E, 15SE	N40W, 70SW	E-W, vertical
Set 4	-	N20E, 15SE	N-S, 15E
Groundwater (6)	dry	dry	wet

Stress Field: σ_1 vertical, 200 psi, measured by others
σ_3 horizontal NE/SW, 1,600 psi, measured by others

Structural Setting: Major fault zone N30W, 30-50W, width averages 75 ft, subsidiary E-W near-vertical faults on hangingwall.

(1) From existing mine data on unconfined compression tests.
(2) From mine core logs.
(3) From prior published studies and underground observations.
(4) From underground observations measured during this study.
(5) From prior published studies using detailed fracture data.
(6) From underground observations.

TABLE 7 - RQI Ratings for Mine A

	Q	RMR	RSR
Faulted, (all)	0.008	20	27
Intrusive Porphyry			
Area 1	1.25	51	57
Area 2	2.2	52	59
Host Granite			
Area 3	5.0	53	54
Area 4	0.83	44	49

Failure sufficient to prohibit passage is confined to the faulted area, where large slabs of concrete have separated from the back and ribs. In this area, mining could not be carried to completion. (Production records indicate fairly uniform draw throughout this area, although somewhat less from the interior of the block. Packing of draw raises became a problem within a month after production, presumably due to high clay content; the notes also mention an abundance of water.)

Problems with grizzly drifts that were available for inspection appear mainly to be related to secondary blasting. The sheer bulk of the concrete used apparently prevents closure in all but the most severe cases. Cracking and spalling are widespread; however, it should be appreciated that a considerable amount of damage can be tolerated in these drifts.

Unfortunately, the deteriorated conditions prevailing in the panel drifts in the faulted area precluded access to examine the grizzly drifts that were affected. Reportedly, most remained open.

Haulage panel drifts beneath production areas show cracks in the concrete, as evidence of ground weight, but nowhere was damage seen that approaches that of the production level. Apparently, the vertical separation of 60 ft is responsible for this. Some at the mine feel that nearly all cracking would be eliminated if this separation were increased to 100 ft, but the lack of many serious ground control problems at the haulage level suggests that this may not be necessary.

Haulage fringe drifts appear to be largely unaffected.

5.1.7 Summary

This investigation reached the following conclusions at Mine A.

1. For classification purposes, the intrusive porphyry and host granite behave similarly under mining stresses.

2. All three classification systems greatly overestimate the competence of the rock for block caving and drift support.

3. A pattern of lining damage was seen that corresponds to the mining sequence. First, springline cracking occurs during undercutting. Second, diagonal cracking, inclined away from the caving area, occurs in response to deformations towards the cave. Third, rib spalls may be developed in response to abutment pressures. Fourth, continued ground weight results in strong lateral pressures, producing crown spalls and shearing of the back.

4. Mine A provides an excellent illustration of the effects of level separation and lateral isolation of the most critical drifts.

5. The lining behavior is responsive to rock mass conditions as indicated by RQI. Planned development in the area of a second major fault is being critically reviewed in light of the experience in the fault zone area described above.

6. Extensive weight problems on the grizzly level are seen on the haulage, but it is unclear if this is due to the transmission of stress or the presence of common geologic influences. Limited (50 ft by 50 ft) weight problems on the grizzly level are ordinarily not reflected on the haulage.

Other points of interest include:

7. Experience has suggested that weight factors of two are found immediately beneath the undercut, and increase to three, parallel and within 35 ft of the undercut abutment.

8. Rock mass movements have been detected as far as 400 ft from the cave boundary.

9. Caving is certain in undercut widths of 70 ft or more, except in acid dikes, which are very difficult to cave.

5.2 Mine B

5.2.1 Geologic Overview

Mine B is in an extensive sequence of intrusive porphyries that invaded earlier granites. A characteristic feature of this orebody is a high degree of structural control, in the form of shattered zones and breccias, as well as a number of major faults that displace or truncate portions of the orebody. Rock strength and alteration patterns are such that this ore would be difficult to cave, were it not for this structural control.

This large orebody was studied in a general way. Because of the wide variation in rock competence that commonly occurs over limited distances, no single area or group of areas was selected for study. Rather, several levels of rock competence that are typical of the range of conditions were analyzed individually, and related to the support practice. At Mine B, support practice is necessarily responsive to these geologic conditions.

The weakest rock considered in this study corresponds to fault zone material, and the strongest type is exceedingly competent and difficult to cave, being essentially free of fractures.

5.2.2 Mining Practice

5.2.2.1 General

Mine B is a mass (as defined in Section 3.1) caving mine, locally referred to as continuous retreat panel caving. Draw is from draw-points using slushers. Muck is slushed directly into muck trains in the haulage. Haulageways are on 200-ft centers and are driven such that the back of the haulage drift is at the same elevation as the floor of the slusher drift. An exhaust ventilation drift lies halfway between and parallel to each haulage drift and is connected to the end of the slusher drift by a small raise. Each successive slusher drift is driven in an opposite direction from the last, and the spacing between loading points along the haulage is 34 ft.

Mine B uses multiple levels 300 ft apart. Present mining depth is variable due to irregular surface topography, but may be taken as 700 ft on the average.

5.2.2.2 Development

Initial level development consists of fringe drifts, which are driven peripheral to the orebody, about 300 ft from the actual caving area, to isolate them from caving stresses. Fringe drifts are driven 14 ft wide by 12 ft high.

Haulageways and vent drifts are turned off the fringe drifts and alternate at 100-ft center-to-center spacings, as indicated above. Haulageways are driven 14 ft wide and 12 ft high. Vent drifts are smaller than haulage drifts, 8 ft wide by 8½ ft high.

Slusher lanes are driven directly over the haulageways and at right angles to them, with the drop points over the haulage center-line. Successive slusher drifts are driven in opposite directions, such that the center-to-center spacing of similarly-oriented slusher drifts is 68 ft. Slusher drifts are 12 ft high by 9 ft wide and are driven on a +5 percent grade. These drifts are wider (11 ft) near the loading points and slusher cutouts. To allow for the slushers, the end of each slusher drift opposite the haulage is extended past the haulage drift rib. This extension is known as the haulage cut-out. Slusher drifts are normally 112 ft long, including the haulage.

In developing slusher drifts, access is gained from the haulage either through the drawhole or up a ramp to a cross-cut drift (8½ by 8½ ft) that is parallel to the haulage and above it. This cross-cut drift is driven such that it intersects every other slusher drift at the first set of finger raises. The slusher drifts driven the other direction are therefore intercepted at the end of their haulage cut-outs (HCOs). There is one cross-cut drift per haulage drift, alternately intercepting finger raise line, haulage cut-out, finger raise line, haulage cut-out, and so on.

As slusher drifts are driven, LHDs tram the muck to the drop point for haulage. Ramps are driven from selected slusher drifts to provide access and muck removal during ventilation drift development.

The cross-cut drift, where used, is backfilled with muck and bulkheaded with concrete where it intercepts either finger raises or HCOs. This is done after the last two sets of finger raises have been driven.

Three pairs of finger raises are driven at right angles to the slusher drift, at an upwards angle of 45°. The first pair of fingers is 21 ft center-to-center from the drop point; the second pair is 31 ft center-to-center from the first; and the third pair is 32 ft center-to-center from the second. Finger raises are 9 ft high by 11 ft wide in cross-section, and are 21 ft long.

The 9½ ft by 9 ft undercut drifts are driven over the top of the fingers, parallel to the slusher drifts. The undercut drifts are connected by similar crosscut drifts driven parallel to and over the haulage.

5.2.2.3 Undercutting and Caving

The first step in undercutting is fan drilling from the undercut drifts. The lowermost holes are inclined upwards at 45° and are of sufficient length to overlap with holes drilled from the adjacent undercut drift. After blasting, this forms an apex of rock over the crown of the slusher drift.

The 3-inch fan drill holes are pneumatically-loaded with ANFO and are blasted 3 to 6 at a time.

After blasting, swell muck is drawn through the fingers to be sure no stubs are left.

Caving can proceed in any direction, since there are no designated blocks for undercutting. At Mine B, block concepts are used only to keep track of reserves and grade; blocks measure 97.5 ft by 100 ft. The width of the cave is approximately 300 ft, and it advances uniformly across the level.

Boundary cut-off work may be used along the hanging wall and footwall of the ore. A raise and sublevel system develops a fracture called a "presplit" with 3-in. holes drilled between sublevels. Sublevel interval and other dimensions vary, depending on the height of the presplit required.

5.2.3 Ground Support Systems

Mine B uses a variety of ground support systems for each type of drift, depending on rock conditions.

Fringe drifts are supported with rockbolts, shotcrete, timber, concrete, or combinations as dictated by rock conditions.

Haulage drifts are initially supported with rockbolts, or rock-bolts and mesh, in better ground. Rigid steel sets or 12 in. by 12 in. timber may be used when required in poorer ground. Haulageways are normally concreted prior to caving, except in the best ground. Intersections and loading (drop point) sections are concreted, with rigid steel sets and/or rebar reinforcement in poorer ground. If haulageways are to be concreted, this is done concurrently with drop-point lining.

Ventilation laterals are supported in a similar way, except that Bernold linings may be used with or without concrete in poor ground.

Initial slusher drift section support is most commonly rockbolts, mesh, or shotcrete, depending on ground conditions. Combinations may be employed, as required, and steel sets are used in poorer ground.

Over the haulage cut-outs and drop points, steel sets are placed during development, in poor to average ground. Where rock conditions are more favorable, bolts, mesh, and possibly shotcrete are used for initial support.

Permanent slusher drift support is afforded by massive formed concrete, with or without rebar reinforcement, placed over the initial support. Concrete provides a wear surface in addition to support. For a time, shotcrete permanent support was tried, but was not wholly satisfactory. The finished dimension of the concreted slusher drifts is 9 ft high by 7 ft wide. The finished linings have flat backs with angled corners.

Finger raises are concreted for the first 15 ft to a 4½ ft high by 8 ft wide cross-section. To combat the typically extensive brow wear, finger raise linings are reinforced at the cutout in the slusher drift.

Undercut drifts are supported with bolts and mesh, and shotcrete or timber may be used in the poorest ground. Steel sets are avoided because of the problems they create in ore drawing.

5.2.4 Scope of Field Data

Mine B employs a full-time geological engineering staff with responsibility for evaluation of ground conditions. Because of this available expertise and highly variable rock conditions, a general approach was preferred to the detailed study of a few specific areas. This approach incorporated first-hand inspections of typical ground conditions and support performance, extensive interviews with the engineering staff, and review of published, unpublished, and mine file geotechnical information. A characterization of ground support procedure, based on RQI, was then developed, and the results used in the support prediction model for the MBR.

5.2.5 Rock Mass Conditions

Rock mass conditions were generalized into categories of very weak rock, weak rock, competent rock, and highly competent rock, for the purposes of this analysis. The various geotechnical properties and parameters applied to these categories are given in Table 8.

Rock conditions in Table 8 are generalized and simplified somewhat to provide a spectrum of RQIs. The different types of rock alteration (which range at Mine B from strong argillization, to strong silicification and healing of fractures with quartz) are represented in Table 8 by variations due to weathering of a fresh granite. Joint fillings are assumed to be varying proportions of clay and crushed rock, the effect of which is modified by the separation and by intact rock strength.

The resulting classifications are given in Table 9.

5.2.6 Support Performance

Cracking of the concrete linings in the haulage cut-out and drop point sections of the slusher drifts is common and is severe in very weak and weak ground. Where this occurs, damage is ordinarily reflected in the haulage as well. Normally these problems are remedied with repair shotcrete or jump sets.

In general, support performance is satisfactory (mining operations are not significantly disrupted) except in very weak ground, where repair is common. However, severe problems and even drift closure extending to haulage drifts can occur if stubs are left. Although great care is taken to be sure that complete undercutting has occurred, problems still arise. During the field visit, one area of near-total closure of a haulage drift was observed. Caving all around this area had left a large stub and resulted in excessive stress on the haulage support.

TABLE 8 - Geological Data for Mine B

	Rock Mass Category			
	Very Weak	Weak	Competent	Very Competent
Intact Rock Strength, psi[1]	1,000	5,000	16,000	25,000
RQD/Recovery, %[2]	0-20/<80	35-50/<90	50-70/80-100	80-100/100
Joint Spacing, ft[3]	<0.2	0.2-0.5	0.5-1.0	>1.0
Joint Condition[4]	poor	fair-poor	fair	good
Roughness	sl. rough	sl. rough	sl. rough	rough
Separation, in.	>0.1	0.1	.05-0.1	<0.05
Filling	gouge, clay	clay, crushed rock	clay, crushed rock	minor clay
Weathering	strly. altered	mod. altered	wk. altered	v. wk. altered
Joint Orientation[5]				
Favorability	poor-fair	fair	fair	fair-good
Number of sets	4	3+	3+	3
No. vertical sets	1	1	1	1
Groundwater[6]	wet	damp	damp/dry	dry

Stress field: Assumed lithostatic stress field, no excess horizontal stress. σ_1 vertical, 800 psi; σ_3 horizontal, 350 psi.

Structural setting: Variable. Assumed structural setting is equivalent to a steeply dipping 2-ft wide fault every 40 ft with no other significant structures. See Section 6.3.2 and 5.2.7.

(1) Generalized from published and unpublished mine data.
(2) Unpublished mine data.
(3) Unpublished mine data.
(4) Published and unpublished data and mine observations.
(5) Generalized from published and unpublished reports.
(6) Generalized from published reports and mine observations.

TABLE 9 - RQI Ratings for Mine B

	Q	RMR	RSR
Very Weak Rock	0.023	21	24
Weak Rock	0.21	34	39
Competent Rock	1.14	49	56
Very Competent Rock	5.0	70	84

5.2.7 Summary

A continuing effort at Mine B is the projection of zones of weak rock. The distribution of these zones is highly complex and does not lend itself to simple characterization.

A further complication is the interaction of weak and competent rock. Competent rock, that would pose no particular support problem by itself, may deform excessively under caving stresses if it is surrounded by very weak or weak rock. This is a prevalent condition at Mine B, and to incorporate it into the MBR support scheme, it was necessary to develop a means to account for this interaction. This was done by equating this effect of very weak rock zones with that of regularly distributed (40-ft spacing), unfavorably oriented 2-ft-wide fault zones that bound and isolate areas of uniformly competent rock. The aggregate effect of such hypothetical faults on an area of more competent ground is thought to be similar to general mine conditions. Where the interactive effect is especially severe, the width of these hypothetical faults could be increased and their spacing decreased; the above spacing and width approximates typical mine geologic conditions.

5.3 Mine C

The study of Mine C was undertaken in this investigation because the depth, geological environment, and slusher-gravity layout make it distinct from the other field sites. Caving at Mine C during the study period was to have been an experiment to determine the feasibility of the method, and the single planned level was therefore of limited extent.

Considerable detailed data had been obtained on rock conditions from the underground workings, in a succession of mine visits, when a decision was made by the mine operator to indefinitely suspend the trial caving project. Consequently, production was never attained and support performance could not be evaluated. Nonetheless, the case history has some aspects of interest that will be discussed here.

5.3.1 Geologic Overview

Mine C is in a large mining district that has seen continuous production for many decades. Ore mineralization is known to extend to at least 1 mile in depth. The present caving project was directed towards mass mining of veins, stringers, and old stopes backfilled with low-grade ore. The caving blocks were centered about one major, steeply dipping, mined-out vein, and several smaller, partially-mined veins.

The mineralization occurs in altered quartz monzonite. The type and intensity of alteration at any given point has to do with proximity to one of the many mineralized zones. At Mine C, argillization and sericitization bracket the important veins. These zones are overprinted on one another, presenting a complex geological picture.

Major prevailing structural and mineral trends in the district are east-west, east-northeast, northwest, and northeast, occurring in that approximate sequence. Dips range from moderate to vertical; most are steep.

The orebodies of present interest at Mine C have moderate to steep dips and are bracketed by intense quartz-sericite-pyrite alteration, grading to argillization with increasing distance away. This imparts a marked contrast in rock strength and overall competence. The weakest material, however, is the backfill from early stoping operations. Much of this contains sufficient mineralization to be regarded as ore under present economic conditions. The backfill is consolidated but still weak, and includes old timber, junk iron, and lagging.

These various alteration and structural zones provide a clear basis for grouping the ground conditions into structural domains, reflecting the relationship of these domains to the main mineral zone to be recovered by the trial caving project. These domains are the Argillized Quartz Monzonite (AQM), Quartz-Sericite Footwall (QSF), strongly altered Hangingwall (HW), and Backfill (B).

5.3.2 Mining Practice

5.3.2.1 General

Block caving had been successfully carried out at shallower levels (600 ft depth) some three decades before this investigation. At that time, full-gravity and slusher gravity systems were in use. The ore caved well but had a tendency to stick, resulting in frequent packing of draw raises.

At the time of the field work, the mine operator was evaluating the feasibility of caving at greater (2,000 ft) depth. Key concerns were for cavability, run-of-mine ore grade (conventional sampling methods were not wholly reliable in this heterogeneous ore), drift support in the argillaceous and backfill material, and the identification of operational problems. Both gravity-slusher and LHD blocks were planned, beginning with the former. At the time of cancellation, development was almost complete on the slusher and haulage levels and was under way on the grizzly level. No development was ever begun on the LHD blocks.

There were to be three blocks at the outset; this was subsequently reduced to two. Blocks are 70 ft wide and 150 ft long.

Each block is served by one slusher drift and two grizzly drifts. One block is located so that the slusher drift would be driven almost entirely in backfill, to assess its behavior.

5.3.2.2 Development

Initial development consists of a single haulage cross-cut, 13 ft by 13 ft, of horseshoe cross-section. This drift trends N34°W and was driven using conventional drill and blast methods.

Some 200 ft outside the block area the slusher level access drift departs from the haulage cross-cut and ramps up to the slusher level, just above the haulage crown turning back parallel to, and about 175 ft center-to-center from the haulage in the block area. Several hundred feet downdrift, the access turns back towards the haulage and is connected to it at the far end by a vertical raise.

Slusher drifts, driven off the access drift, are 165 ft long on 70-ft centers and are opened to an 11-ft wide by 13-ft high horseshoe cross-section. At transfer raise windows the design width is 12 ft. Slusher drifts are laid out perpendicular to the haulage drift and the slusher level is immediately above the haulage back.

Transfer raises are angled upwards from the slusher drifts at 60° from the horizontal and are about 30 ft long, intersecting the floor of the grizzly drifts. They are 5 ft square in cross-section and are on 17.5 ft spacings; there are 8 pairs of transfer raises, directly across from each other, in each slusher drift. Transfer raises are perpendicular to slusher drift axes.

The grizzly level floor is 29 ft above the elevation of the slusher drift inverts. Grizzly drifts are parallel to slusher drifts and are spaced uniformly 35 ft apart.

There are two grizzly drifts for each slusher drift. They are driven 9 ft wide by 10 ft 9 in. high with a horseshoe cross-section. Near each transfer raise collar location, finger raises are driven vertically upwards to the undercut level (planned 19.5 ft above). The finger raises are 6 ft square at the midpoint but flare slightly towards the top.

Access to the grizzly level is via a trackless ramp that leaves the No. 1 haulage cross-cut about 300 ft northwest of the slusher access drift intersection, or about 500 ft from the edge of the first block. The access drift has a 12 ft by 12 ft horseshoe cross-section, and is quite sinuous in plan.

All development was by conventional drill-blast methods, using jacklegs.

In the haulage, each 1-3/8 in. by 6 ft hole was loaded with Ireco slurry and an advance rate of 6-1/2 to 7 ft per round was achieved. The powder factors and blasthole patterns were flexible. Overbreak reached 1-1/2 ft on sidewalls and 2 to 3 ft or more at the crown. This high level of overbreak may be attributed to a combination of unfavorable flat fracturing, and poor control of perimeter holes, which were drilled with a jackleg and commonly diverged from parallelism with the driveage. Extensive scaling was commonly required, especially within the Argillized Quartz Monzonite (AQM) domain (Section 5.3.4), where loose rocks due to blasting damage were a subject of concern. In this domain, ribs are drummy through about half the drift length, wherever joint spacings are close. More widely spaced joints and less intense mineralization result in markedly less drummy rock.

Blasting and excavation practice in the slusher access drift was observed at a point 100 ft or so past the block area. Excavation practice was essentially the same as for the two slusher drifts in AQM. Overall, overbreak is somewhat less than in the slusher drifts, averaging a foot in ribs and back and 1 to 2 ft in the area where the backfill is crossed. Most of the rib rock seemed tight, except where the HW domain is crossed, where the ribs are drummy throughout.

As indicated above, the three slusher drifts were excavated by drill-and-blast methods. Although blasthole pattern and loading were flexible, each shot was in general composed of 20 (twenty) 1-3/8 in. by 6 ft holes plus a 6-hole burn cut. Powder factors are said to have been 150 lbs per shot, and used water gel in 1 x 16 sticks packed 6 ft in each hole for the lifters, and bulk ANFO packed 5 ft for the rest of the shot. Most fragments were fine to 0.5 ft or so; about 20% were 1 ft or more. Overbreak was up to 1 ft or so on the ribs but up to 3 or 4 ft in the back in the AQM domain. No hole casts were noted. The adverse over-break is due to divergent perimeter holes and is of particular concern in areas of flat jointing. Barring requirements were minimal to moderate and drummy areas are limited to 30% or less of the drift length. There is a strong tendency for the blast breakage to occur along joints rather than through intact rock blocks.

The slusher drift in backfill posed special problems. Initially, the same blasthole pattern and loading practice was used in backfill as the other slusher drifts, with similar results in a 30-ft section of the QSF domain. Conditions began to worsen as the backfill boundary was approached. At about 80 ft downdrift, with bad overbreak in the backfill a chronic problem, blasting practice was changed to a drastically reduced powder factor. Although practice was very erratic (some rounds consisted of only a few holes and a few sticks of Iremite 40), results were markedly improved.

The backfill was found to be a heterogeneous mixture of old timber square sets, rock fill that has hardened and packed over the years, and fine muck. Intact chutes and lagging were found. The backfill is dry and most of the timber is intact but greatly weakened by age.

Overbreak ranged from the usual 1 ft or so on the ribs and 2 to 3 ft in the back for the first 35 ft, to 4 or 5 ft in the back in the backfill portion. After changing the blasting practice, overbreak was reduced to 1 to 2 ft, on the ribs and crown.

Barring down requirements were extensive. In many instances, oversize excavation that might be attributed to overbreak was actually produced by severe ravelling after the shot. In the later excavation practice, the drift was enlarged to its finished dimensions by barring and scaling.

The undercut level, 17.5 ft above the grizzly level, was planned to consist of 5 ft by 7 ft subdrifts on 17.5-ft-centers, driven parallel to the grizzly and slusher drifts, over the finger raises. Similar cross-cuts were laid out on 17.5-ft-centers. The undercut level was not developed.

The grizzly level was begun prior to suspension of the project, but these workings were not available for our inspection.

5.3.3 Ground Support Systems

5.3.3.1 Initial Support

At the time of the field visits, only initial support was in use. All drifts under development were either unsupported or used 5-ft Split Set bolts with wire mesh for the more competent AQM and QSF domains. The bolt spacing varied from a 3-ft circumferential spacing, due to additional spot bolting and requirements for holding the mesh in close contact with the irregular rock surface.

The less competent HW domain is intercepted by the slusher level access and haulage drifts. Steel strapping supplements bolts in this weaker rock.

Initial support where the haulage drift intercepts backfill, and for the first 30 ft of the slusher drift in backfill, is afforded by 8-in. wide-flange steel sets. These sets, on 5-ft spacings, were not blocked in at the time of the field visits. Elsewhere in the backfill slusher drift, Split Sets on 2-ft spacings and mesh are used.

Permanent support throughout the area was planned to be massive, formed concrete. This support was never completed. The planned final size for the grizzly drifts was 5 ft by 7 ft, for the slusher drifts, 7 ft wide by 9 ft high, and for the haulage, 11 ft by 11 ft. In addition to the concrete, it was planned to extend the use of steel sets prior to concreting over them, to a length of 270 ft, centered around the 90-ft-wide backfill area.

Since the development and caving were not completed, final support details and performance could not be addressed.

5.3.4 Scope of Field Data

The highly detailed geologic maps kept by the operator provided an excellent baseline of information, as well as numerous details. Structural orientations were scaled off these maps and were computer analyzed for the generation of fracture statistics. RQD data were obtained from core logs.

Detailed fracture information, relating to spacing and condition, were collected in the field. Samples were taken for point load compressive strength determinations. Excavation and support practice were observed during the field visits. During later visits when fracture orientation data for the identified domains were known, classifications were performed while underground.

5.3.5 Rock Mass Conditions

The relative rock mass conditions for the naturally occurring rock types at the site are typified in Tables 10 and 11. The backfill, not a naturally occurring material, is not amenable to classification.

5.4 Mine D

Mine D, a smaller block caving operation, was intensively studied because the development and production scheduling allowed first-hand observations of geological conditions and mining progress, from a very early stage in block development. Because of this, accurate knowledge was possible of variables influential in the analysis. In addition, ground conditions vary from good to very poor over a few hundred feet, extensive past production experience exists under similar geological conditions, and strong structural influences are present. These factors combine to extend the data base.

The mine was visited repeatedly, to monitor development, cave initiation, and production experience, as well as to enable further refinements of geological projections.

Despite being a small mine and thus not employing a full-time rock mechanics staff, Mine D does include rock mass classification as part of new development and exploration. This work is handled by the

TABLE 10 - Geological Data for Mine C

	Rock Mass Category			
	AQM	QSF	HW	B
Intact Rock Strength, psi(1)	1,400	16,500	700	3,000(est.)
RQD/Recovery, %(2)	59/80-100	61/60-80	15/20-50	-
Joint Spacing, ft.(3)	0.15-1.5, avg. 0.8	0.4	0.2	No Joints
Joint Condition(4)				
Roughness	rough	very rough	rough	-
Separation, in.	0.1	0.2	0.1	-
Filling	minor clay	qtz., ser.	clay	-
Weathering	sl. to mod.	slight	v. str. alt.	-
Joint Orientations(5)				
Set 1		ENE steep SE		-
Set 2		NNW steep SW		-
Set 3		NE steep NW		-
Set 4		WNW steep SW		-
Groundwater(6)	dry	dry	damp/dripping	

Stress Field: σ_1 vertical = 2,200 psi, assumed
σ_3 horizontal = 700 psi, assumed

Structural Setting: E-W trending major structures dipping steeply south, one within block (backfill) and one over 300 ft to the northeast.

(1) From point-load tests.
(2) Scaled from mine graphic core logs.
(3) Measured underground, this study.
(4) Underground observations, this study.
(5) Scaled from detailed mine geologic maps at 1:120.
(6) Observations underground, this study.

TABLE 11 - RQI Ratings for Mine C

	Q	RMR	RSR
AQM			
2000 access	3.9	71	71
2000 haulage	3.9	66	76
slusher drift 5	3.9	71	59
slusher drift 4	3.9	73	62
QSF			
2000 access	22.6	76	71
2000 haulage	-	76	-
HW			
2000 access	0.02	31	29
2000 haulage	0.02	31	29
B	NOT CLASSIFIED		

geological staff. In the past, difficult ground conditions have disrupted operations, and Mine D is taking steps to plan for and cope with these problems.

Mine D is the example mine in Section 5.0 of the Manual, appended to this report.

5.4.1 Geologic Overview

Mine D is in the western United States. Prior to its discovery, the surrounding district was relatively obscure by national or world standards. The main ore zones at Mine D, of which there are several, were discovered recently in comparison to the mines discussed earlier. Ore grades are slightly higher than the average for porphyry ore-bodies.

The ore occurrence is believed related to the intrusion of a granodiorite porphyry, which invaded a thick sequence of earlier volcanic and sedimentary rocks. Stock emplacement occurred along with strong faulting and folding, and was followed by deposition of great thicknesses of volcanic and sedimentary rocks. Faulting continued, truncating and burying some portions of the orebody, while exposing other portions to erosion. This faulting, and the supergene processes resulting from it, acted to form the ore zone that is presently being mined. It also presents a complex fracture pattern. Deeper ore is slated for future production.

Mine D refers to its levels with respect to sea level, so that higher-numbered levels are higher in the mine. At the time the study was begun, mining was in progress on the 1,100 level, with extensive past production; slusher dashes* were being developed on the 900 level below it, affording excellent rock exposures. For this reason, attention was focused on the 900 level, where the whole development cycle could be observed.

The ore zone dips moderately, so that the 900 level, being down the dip of the ore from the 1,100 level, is offset from the 1,100 level in plan.

Three basic rock types are found in this part of the mine, including intrusive (granodiorite) porphyry, fine-grained volcanic flow rocks (mostly andesite), and a fanglomerate that caps the ore zone and was faulted downwords along with it. The intrusive porphyry was initially altered hydrothermally, and was later weathered. The volcanics were similarly altered and weathered, but owing to their finer grain size, do not exhibit the disaggregation that can be quite strong in the porphyry.

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* "dash" is a local term for "drift."

The lower fanglomerate surface dips moderately away from the active mining area, striking roughly parallel to the long axis of the level. Although the fanglomerate does contain scattered mineralized fragments, it is not ore, and its presence above limits the ore column height. On the 900 level, as one moves away from the lower contact down the slusher dashes, weathering decreases in intensity, supergene enrichment increases in intensity, and a highly argillized zone is reached. Varying proportions of porphyry and volcanics in complex association, are found in this interval in the slusher dashes.

This ore zone is truncated at the far end of the slusher dashes on the 900 level by a highly argillized, damp, strongly slickensided, incompetent zone corresponding to a major fault. This fault, known as the "footwall crud" because it defines the footwall of the ore zone, strikes generally parallel to the fanglomerate contact, dips moderately to steeply towards the mining blocks, and may reach 50 ft or more in thickness. The horizontal distance at the 900 level between the fanglomerate contact and the edge of the "crud" is of the order of 200 ft, so that the mining layout is basically a chain of single blocks, each about 200 ft on a side, that follows the trend of the "crud." Subsidiary faults, not separately classifiable, cross the blocks at shallow angles to the "crud."

Three basic structural domains were recognized, corresponding to the weathered porphyry, a structural transition between the fanglomerate and the "crud," and the "crud" itself.

Despite the intense weathering and alteration present, it was felt that fracturing plays a significant role in rock mass behavior at Mine D.

5.4.2 Mining Practice

5.4.2.1 General

Overall, Mine D uses a blockwise panel cave method with diagonal retreat undercutting. Slushers are used to move ore from the fingers to the drop points, with the haulage directly beneath. The blocks of each production level, the 1,100 and the 900, are arranged such that each level is essentially a single panel. On the 1,100 level, nine blocks had been mined with one in production and one at an advanced stage of development at the time of this study. On the 900 level, the first block, slusher level only, was nearing completion. Production from the 1,100 level had been intermittent, having just resumed after several idle years.

Mining levels consist of undercut, slusher, and haulage levels, in much the same fashion as Mine B. The chief difference is that at Mine D, slusher lanes are longer and all run the same direction.

Past experience had been to lay the panel out using a fringe drift driven in fanglomerate, with crosscuts into the blocks constituting, alternately, haulage and vent drifts. This meant that slusher dashes were sub-parallel to the long axis of the ore zone, and therefore sub-parallel also to the "footwall crud." Severe ground conditions were experienced in the two or three slusher dashes nearest the "crud" which had a tendency to roll over in response to caving pressure. Noticing that fewer problems were experienced in the haulage and vent access drifts (although some total failures did occur at the ends, near the "crud" zone), the mine decided to change the layout so that on the 900 level, the slusher dashes are perpendicular to the structure. The original layout, on the 1,100 level, was to enable dash-by-dash draw control, seen as advantageous because of the moderate dip of the ore zone. There were several instances where entire slusher dashes on the 1,100 level were in severe distress due to ground weight from the "crud", so it was decided on the 900 level to turn them by 90° so that the "crud" would be in a more favorable orientation with respect to the slusher drifts. This would also confine most distress to areas near the ends of the drifts.

With the 900 level layout, however, it was necessary to put the ventilation drift almost entirely within the "crud" zone. Recognizing the experience on the 1,100 level with rigid (massive concrete) support in slusher dashes of similar orientation, it was decided to support the vent drift with steel sets and considerable blocking that could be removed in response to ground pressure.

With these general comments made, the following sections address in greater detail the development and support practice at Mine D.

5.4.2.2 Development

On the 1,100 level, the 12 ft wide by 10 ft high fringe drift parallels the ore zone, with vent access and haulage drifts turned off at 90° angles to it, alternating on 200-ft-centers. The horseshoe-shaped vent access drifts are driven 10 ft wide by 9 ft high, and ramp up from the fringe drift to reach the level of the slusher dashes. Each vent drift serves two blocks, and vent drifts are the only openings at the 1,100 level slusher elevation that are perpendicular to the trend of the ore zone. Haulage or "panel" drifts are kept at the elevation of the fringe drift and extend directly beneath the line of drop points in the slusher dashes. Each drop point serves two blocks in succession. When one block is mined out, the slusher drums are moved to the other side of the drop points and turned around. The sheave blocks are moved to the opposite end of the slusher dash under the uncaved block, the cables and scrapers are installed, and the new block is then ready to be pulled.

Slusher dashes were driven 11 ft wide by 10 ft high on 35-ft centers, and continue from one block to the next. They are sub-parallel to the overall trend of the ore zone on the 1,100 level. Access to the slushers thenselves are through manway raises from the haulage.

On the 900 level, an access drift ramps up from the fringe drift to the slusher elevation and intersects the slusher level access drift at approximately a right angle. The access drift essentially parallels the ore zone, the vent drift, and the "footwall crud." The access drift is 12 ft wide by 11 ft high and is extended the length of the level, with slight bends to better follow the ore zone.

At right angles to the access drift are the slusher lanes, which are nominally 200 ft in length but in practice vary, according to encountered ground conditions in the vicinity of the "footwall crud." Slusher lanes are driven 10 ft by 10 ft, on 35-ft-centers.

Turning off the fringe drift, and remaining at the same elevation, is the haulage, which is parallel to the slusher access drift, to the block side and beneath it. The haulage, of approximately the same dimensions as the fringe drift and access drift, is centered beneath the line of drop points.

Opposite from the access drift and at the same elevation is the vent drift. The vent drift joins up with the far ends of the slusher lanes. To maximize slusher lane length, yet avoid irregular zones of incompetent ground associated with the "crud," the vent drift direction is irregular. It is designed 10 ft by 10 ft and was driven meticulously, keeping steel sets as close to the face as possible. This made for slow development in this ravelling and squeezing ground.

All development is by conventional drill and blast methods using jacklegs. Present practice in average ground is to pull 6 to 8 ft per round, using a 5-hole burn cut and 30 to 60 holes packed with water gel explosive. On the 1,100 level, overbreak of 20% was not uncommon, and was somewhat worse in blocky, weathered intrusive porphyry. Overbreak on the 900 level averages 10% in average ground, with some areas slightly higher, and blasting practice continues to improve. In strongly altered or ravelling ground, 5 ft of overbreak, even with very light charges, is not uncommon. Overbreak is generally greatest in the crown. Powder factors of 0.8 lb/ton of 45% gelatine dynamite in the slusher drifts yield fine fragmentation.

5.4.2.3 Undercutting and Caving

Finger raises have a 5-ft-square cross-section and are driven vertically from the slusher drifts in pairs, with 17.5-ft center-to-center spacings down the drift.

The undercut level is driven 15 ft above the slusher level, using 7 ft wide by 5 ft high drifts driven on 17.5-ft-centers over the tops of the finger raises. Undercutting is accomplished by drift widening and ring drilling and blasting of pillars. Pillar blasting is advanced in sections, maintaining a diagonal cave line. Swell muck is drawn off immediately, through the fingers.

Caving proceeds block-by-block across the level, with each block completed before caving is begun in the next. Blocks are about 200 ft on a side.

5.4.3 Ground Support

Temporary support on the older 1,100 level consisted of 1/2-in.-diameter, 6-ft-long point-anchor mechanical bolts, on 3- to 4-ft-centers, ordinarily with 12-gauge steel strapping. Where necessary, steel support was installed. In the access drifts, the bolts seem to be accepting load effectively. The access drifts and haulage drifts are unlined except within the block.

Slusher drifts in an abandoned block were observed. Support practices in other blocks are expected to be similar.

Temporary support consisted of mechanical expansion-shell type rock bolts, 1/2 in. diameter by 6 ft long, spaced 2 to 3 ft apart at the crown. Both steel matting and 12 in. wide 12 gage steel straps were installed to avoid falls of loose rock. A close inspection of some bolts from an incomplete finger raise revealed that about 50% had little or no anchorage.

Monolithic concrete is used for permanent support throughout the block, to a uniform design thickness of 3 ft over the ribs and crown. The floor was concreted to a design depth of 10- to 12-in. The concrete used an average aggregate size of 3/4 in. and a 4 in. slump. Although the design strength of the concrete mix was 4,000 psi, a number of Schmidt hammer tests on the linings during this investigation yielded an average value of 1,600 psi.

Considerable segregation of aggregate was observed in many areas, indicating field difficulties in concreting practices. Also 1 to 1.5 ft voids were observed between the rock surface and the concrete layer at the back of some of the drifts. At one locality, concrete thickness in the crown was less than 2 in.

The haulageways are supported with 10-in. wide-flange steel sets with cast concrete to cover. Set spacing is in general about 5 ft.

Undercut drifts on the 1,100 level had been supported with timber posts, caps, and girts, lagged as necessary. Lately, Split Sets have found increasing use.

On the 900 level, support practices and workmanship were found to be much improved. Support systems were found to be better adapted to ground conditions.

In the access drift, 1-1/2-in.-diameter, 6-ft-long steel Split Sets with 12-gage steel straps comprise the only support outside the block area. Most of this is in fanglomerate, which is essentially unfractured and stands well. Within the block area, 10-in. wide-flange steel sets are set on 5-ft centers, with lagging of the crown and of the lower part of the ribs.

In all slusher drifts, 1-1/2-in.-diameter 6-ft-long steel Split Sets are used for temporary support. They are installed soon after the previous round is mucked. Steel strapping is placed during bolting to provide safety from falls of loose rock. A detailed survey of Split Set spacings indicated that the average longitudinal spacing is about 4 ft while the circumferencial spacing varied between 3 to 5-1/2 ft. Although the spacings of the bolts are fairly consistent in all the slusher drifts, the percentage of the area bolted varies at different sections of the drifts depending on the nature of the exposed ground. Three bolts are used to hold up each steel strap and the long dimension of the straps are parallel to the drive. In the fairly competent intrusive porphyry, five such straps are used starting from about three quarters of the way up the ribs and over the crown. In the intermediate zone, about six or seven straps span the crown and the upper two-thirds of the ribs. In the "crud" zone, some 8 to 10 straps are installed covering the entire drift, floor to floor.

In general the Split Sets appeared to have been installed properly but due to the nature of their design it is difficult to determine how well they are taking load. They appear to be effective, although evidence of corrosion was found. There is considerable ravelling of loose rock from the ribs and crown, in the "crud" zone and transition zone. Close to the vent/access drift, where the rock is intrusive porphyry these effects are greatly diminished or absent.

Permanent support in 900 level slusher drifts incorporates rigid square steel sets of W6x20 sections. These are installed at several areas within the slusher drifts soon after excavation. Their purpose is to provide some additional support at the intersections of the slusher drifts with the access and ventilation drift, and are used in drift sections in weaker ground. Blocking and lagging consists of 4 in. by 6 in. and 3 in. by 12 in. timber. Usually there are two blocking points at the crown and two in each rib.

Haphazard blocking and lagging practices were found in some areas, especially in one slusher drift, having about 60 ft of steel set support. No appreciable deformation of timber wedges was noted at this location showing little indication of ground movement. Elsewhere, such as in the vent drift where steel support may be regarded as more critical, blocking practice is improved.

The slusher drifts are ordinarily fully concreted a few weeks to a month after they are driven. Monolithic concrete of very high 9-in. slump containing 3/4-in. rock aggregate, and having a 28-day design strength of 4,000 psi, is poured uniformly to a thickness of 3 ft. Collapsible steel arch forms are used for placing concrete. A wheel-mounted, hydraulically-powered jumbo is used to collapse and expand the forms into position. The forms are hinged and are collapsed by jack-screws connecting the sides of the form to the jumbo. During concreting, styrofoam blocks are placed at the intended locations of the finger raise portals, to avoid blasting through concrete when starting the finger raise. A 5-in. metal angle and a 10-ft-long 90-lb rail are placed over the Styrofoam and embedded in the concrete to reinforce the finger raise brow.

A few weeks after concreting, circumferential cracks were noted which typically originate from the downslope side of the Styrofoam blocks in the inclined (4°) slusher drifts. The displacement of the cracks is very small and the reason for cracking has not been conclusively determined, but may well be due to shrinkage upon curing.

It was realized from the outset that the vent drift would be driven in troublesome ground. Accordingly, permanent square steel set support is installed as soon as possible after mucking. Heavy duty steel of W10x45 section make up the sets. Usually, a 3-ft spacing is maintained between adjacent sets with smaller spacings near intersections and areas of extremely poor ground. Two 4 in. by 6 in. timber blocks are used as spreaders between adjacent posts along with 1/2-in.-diameter tie rods. Blocking on the crown of the sets is generally at two points at end of the cap, to retard bending. Blocking at the ribs is irregular, owing to the use of considerable timber which was to be removable if substantial ground movement occurred.

It should be noted that the vent drift was necessarily driven with several re-entrants and sharp corners, to avoid the worst ground. Throughout development, the "crud" zone in the vent drift and at the nearby ends of some slusher dashes could be heard "working" and ravelling was almost continuous.

Undercut drifts on the 900 level were supported with Split Sets and, where necessary, steel straps. No timber was in use at the time of the study.

5.4.4 Scope of Field Data

Repeated visits were made to observe the development of the first block on the 900 level and later the initiation of caving in a new block on the 1,100 level. In the latter instance, however, detailed geologic data could not be obtained because of lack of rock exposures; early geologic maps and some observations in the undercut level were relied upon for geologic control.

At Mine D, it was necessary to develop a data base for fracturing and projection of structural domains. Core from the area of the first block on the 900 level was thoroughly logged for geotechnical information. Five detail line surveys on the 900 level yield information on the nature and distribution of fracturing. The lines had various orientations, to remove directional bias. These data were analyzed statistically and Schmidt plots of fracturing were computer-generated. Rock classifications were carried out underground throughout the then-existing workings on the 900 level, and at several locations on the 1,100 level where rock was exposed. Information in the mine's files, including field stress measurements and point-load compressive strength of core, was intensively reviewed and analyzed. Detailed and reconnaissance geotechnical maps and cross-sections were prepared.

In addition to the geological data, information on the development layout, blasting procedure, support practice, and draw control history was collected, through first-hand observations (900 level) and review of company maps and files (1,100 level). Support performance was assessed by examining present caving experience on both levels, as well as through visits to worked-out areas on the 1,100 level. Support effectiveness and placement was documented through Schmidt hammer (concrete test hammer) tests, torque tests on mechanical bolts, and measurement of set spacings and blocking practice. The foregoing discussion is a summary of this detailed data.

5.4.5 Rock Mass Conditions

Basic geologic conditions at Mine D were found to fall into a spectrum, from the weathered but hard intrusive porphyry to the strongly altered and sheared "footwall crud." The transition zone is of intermediate but variable rock class. The basic geological data are summarized in Table 12. Ranges of data are given where descriptive of the variation within a single structural domain. The given intact rock strength is biased on the high side in the "crud" zone, due to sampling effects, and on the low side in the transition and altered porphyry zones, due to the role of hairline fractures in controlling breakage during point load tests. The values used in obtaining the RQI ratings were adjusted somewhat according to the range given in the table.

Similarly, the other measured variables were somewhat different from level to level and within a level. This is to be expected in this complex geological environment, so RQI values from various areas are reported.

The "altered porphyry" group occurs near the fanglomerate contact and consists of highly to moderately weathered, altered granodiorite porphyry. Fracturing tends to be moderate in spacing, and the ground is commonly blocky. On the 1,100 level, weathering is more severe and

TABLE 12 - Geological Data for Mine D

	Rock Mass Category		
	ALTERED PORPHYRY	TRANSITION	FOOTWALL CRUD
Intact Rock Strength, psi [1]	6,900-15,000	3,700-5,900	6,100-9,900
RQD Recovery, % [2]	39/83	41/80	17/43
Joint Spacing, ft [3]	0.6	0.4	0.2-0.5
Joint Condition [4]			
Roughness	rough	sl. rough to smooth	smooth or slickensided
Separation, in.	0.05	0.05	0.1-0.05
Filling	clay, FeOx	FeOx	clay, gouge
Weathering	slight	strong	very strong
Joint Orientation [5]			
Set 1	N40E, 85SE	N84E, 83SE	N52E, 75NW
Set 2	N77W, 63NE	N07E, 71NW	N65W, 78SW
Set 3	N31E, 23SE	N29W, 42NE	N43W, 20SW
Set 4	N58W, 74SW	N52W, 72SW	N23W, 68NE
Set 5	-	-	N17W, 75SW
Groundwater [6]	dry	dry	damp

(1) From mine data--point load and unconfined compression tests on core.
(2) Obtained from mine core logs.
(3) Measured underground during this study by detailed fracture surveys.
(4) Obtained from detailed fracture surveys and core.
(5) From computer analyses of detailed fracture surveys.
(6) Observations underground.

joint surfaces are slickensided and coated with iron oxides. The rock commonly has the appearance of a leached capping. Small faults occur, chiefly as single shears, but are few.

The transition zone shows the influence of the nearby "crud" zone faulting. The transition zone contains volcanics and altered porphyry. The altered porphyry is ordinarily not as weathered as it is bleached, and fracture spacings in the transition are somewhat lower than in the "altered porphyry" group. The volcanics are shattered, with thin coatings of carbonates or sulfates on the fracture surfaces, and the groundmass is moderately altered. Areas of blocky ground extend for only 20 ft or so, and are bounded by sheeted or shattered zones. The transition zone is commonly bounded by faults.

The "crud" zone may contain either or both rock types; they are difficult to distinguish due to intense shearing and bleaching. The material is damp, with many gouge and clay seams up to several inches in thickness. Fractures commonly are noticeably open. Blocks of intact rock do exist but are scarce and of limited extent, and are always bounded by slickensided zones with central gouge.

The RQI ratings for each of these groups, at various places in Mine D, are given in Table 13. All the ratings are sensitive to the faulting, and the contrast between zones, while not of large magnitude, is distinct and consistent.

5.4.6 Encountered Support Performance

As mentioned earlier, some slusher drifts on the 1,100 level near (or in) the "crud" zone are in severe distress; a few had to be abandoned. It is difficult to pull swell muck from the "crud" zone due to packing in the fingers; if this situation is not alleviated within a few days, spalling and cracking (with offsets up to 1 in.) begin in the concrete lining. In one case, an attempt had been made to keep the drift open by installing W6x20 steel sets spaced 3 ft apart and spraying 1 to 3 in. thick shotcrete near the damaged area. Several of these steel sets had bent over and buckled and the drift was abandoned. Ground support problems in this area are thought to be one reason why numerous finger raises were never completed and full production was never obtained.

The haulageways and access drifts in this type of ground are also distressed. In the above instance, concrete in the haulage had spalled badly from the ribs and crown, exposing damp, gougy, squeezing rock. The steel support prevented severe closure, however, and the drift remained in service. A nearby vent access drift had almost closed completely, despite extensive remedial timbering.

Far better results were seen on the 1,100 level in the more competent transition and altered porphyry domains. A half-caved block

TABLE 13 - RQI Ratings for Mine D

	Q	RMR	RSR
Altered Porphyry			
900 level, 48 S.D.	0.8	68	68
900 level, 51 S.D.	0.6	63	53
1,'00 level, 5/6 V.A.	1.3	62	47
Transition			
900 level, 48 S.D.[1]	0.3	57	47
900 level, 51 S.D.[2]	0.09	52	36
900 level, 51 S.D.[3]	0.4	54	47
1,100 level, 16 S.D.[3]	0.2	55	52
Footwall Crud	0.03	37	41

---

[1]Mixed porphyry and volcanics
[2]Altered porphyry, strongly bleached, weak, but not faulted.
[3]Mostly volcanics

S.D. = slusher drift
V.A. = vent access

on the 1,100 level showed moderate spalling of the ribs in the concreted slusher dashes, immediately beneath the cave line. Crown spalling was limited to the area in production at the time. Diagonal cracks with apertures of 0.2 in. and displacements of up to 0.3 in. were found in advance of the cave line in this block. The surrounding rock, although not directly visible, was projected to belong to transition and altered porphyry groups.

On the 900 level, the slusher dashes and access drift were reportedly functioning well (the area was not caved until after the field investigations had been completed). However, severe problems were experienced in the vent crosscut. The removable wood blocking was quickly smashed flat soon after caving was begun, and severe distortion of the steel supports ensued. Despite continuous maintenance, the drift closed such as to severely restrict airflow. Several remedies were considered, such as remining, stress relief, shotcrete, and others. These deformations affected only the ends of the concreted slusher drifts and their intersections with the vent drift. As would be expected, severe problems developed at sharp corners and in re-entrants.

5.4.7 Summary

Mine D is an excellent example of control and localization of caving stresses by major structural zones. In addition to being weak in themselves, such zones can cause very high deformations nearby. Although the variation in rock class is somewhat more subtle than might be expected, the difference in behavior is great.

Mine D also affords examples of reliance on permanent concrete support to help stabilize the ground prior to caving. At the Mine D, 1,100 level far better results would have been obtained if the ground has been stabilized prior to concreting.

5.5 Conclusions of the Field Work

In formulating a classification system that would be suitable for drift support recommendations in block caving mines, it was thought that the best procedure would be to adapt an existing, proven RQI scheme. This could be done through adjustment (change of rating structure for existing parameters) or modifications (addition of new parameters) or a combination of adjustments and modifications. It was decided to select, as the starting point, the most suitable RQI system from among Q, RMR, and RSR. These systems were evaluated from the standpoints of agreement with successful mining practice, adaptability to block caving, and general utility.

The range of ground competence found at the four field sites covers a spectrum from very weak, swelling, ravelling, and gougy ground, to hard, strong rock with moderately spaced, tight fractures.

Typical ratings and associated support recommendations for various types of ground are given in the following subsections. These are support recommendations derived directly from the classification systems, for temporary openings with 10 to 15 ft spans. They are highly generalized and are meant to provide broad comparisons between the "basic" (Q, RMR, and RSR) systems. Laubscher's adjustments were also applied, and summary statements regarding the outcome of that work are given.

5.5.1 Findings at Mine A

Table 14 compares predicted and encountered support from Mine A. The supports are generalized slightly to permit ready comparison between systems.

None of the basic systems predicts the heavy timber/steel and massive concrete support actually installed in the production openings. Laubscher's classification system gives varying results depending on the type of drift, the orientation, and other details. In general, Laubscher's predictions do recommend heavier support than Q, RMR, or RSR, in many instances extending to massive concrete. None of the systems recommends an approach that might have prevented the failures that occurred in the faulted areas.

5.5.2 Findings at Mine B

At Mine B, none of the basic classification systems predicts the heavy types of concrete support that are actually used (Table 15). In many instances, the systems are insensitive to factors that have precipited total failures at Mine B. The information in Table 15 is generalized over the mine area and there are numerous exceptions. Laubscher's system recommends various thicknesses of massive concrete for all but "very competent" rock.

5.5.3 Findings at Mine C

Predicted and encountered support at Mine C is shown in Table 16. In general, encountered support is initial support, (except where steel ribs had been installed early, near backfill areas) and permanent support was only planned. No support performance history was developed, but it can be seen that the RQI-recommended supports probably would not have been adequate, had mining occurred.

5.5.4 Findings at Mine D

Predicted and encountered supports at Mine D are compared in Table 17. The information contained in Table 17 is quite generalized, since Mine D has changed its overall support practice, and has also varied the support depending on type of drift. Massive concrete support, for example, was foregone in the 900 level vent drift in

TABLE 14 - Predicted and Encountered Support for Mine A

	Faulted	Intrusive Porphyry	Host Granite
Q	8 in. mesh-reinf. shortcrete ± grouted 3/4 in. 7-ft-long bolts, tensioned, @ 1.5-3 ft. spcg	1-2 in. plain shotcrete ± 3/4 in. bolts, grouted, @ 3 ft. spcg, 7-ft-long	Prob. 1-2 in. plain shotcrete ± grouted 3/4 in., 7-ft-long bolts @ 3 ft. spcg (Q varies widely)
RMR	3/4 in bolts, 15-ft-long @ 3-5 ft. spcg + mesh or 6 in. shotcrete or med. ribs @ 3-4 ft. spcg	3/4 in. bolts, 12-ft-long @ 4-6 ft. spcg + mesh or 1 in. shotcrete no rib support	12-ft-long @ 4-6 ft. spcg + mesh or 3 in. shotcrete no rib support
RSR	3/4 in. bolts @ 2 ft. spcg or 3 in. shotcrete or 6H13 ribs @ 2.5 ft. spcg	3/4 in. bolts @ 6.5 ft. spcg or 1½ in. shotcrete or 4I7.7 ribs @ 5.5 ft. spcg	3/4 in. bolts @ 4 ft. spcg or 2 in. shotcrete or 4I7.7 ribs @ 4 ft. spcg
Encountered	18-24 in. locally reinforced concrete plus 4-in. to 6-in. WF sets @ 5 ft. spcg or 6 by 8 in. to 12 by 12 in. timber sets	18-24 in massive concrete plus 5/8-in.-dia., 6 ft-long bolts plus straps or mesh or (deeper levels only) 12 in. by 12 in. timber sets or 4 in. to 6 in. WF sets	18-24 in massive concrete plus 5/8-in/-dia., 6 ft-long bolts plus straps or mesh or (deeper levels only) 12 in. by 12 in. timber sets or 4 in. to 6 in. WF sets

Support Performance

Grizzly fringe drift crowns and ribs spall within 70 ft of panel drifts turnouts.
Grizzly panel drift crowns and ribs spalled and cracked near turnouts.
Timber supports stressed--cracked posts, crushed caps, deformed lagging, where exposed on grizzly level.
Severe distress on grizzly level in faulted area. Mining ceased, many panel drifts abandoned.
Damage on haulage level generally restricted to cracking, generally not serious.
Grizzly drifts tolerate extensive damage.

TABLE 15 - Predicted and Encountered Support for Mine B

	Very Weak	Weak	Competent	Very Competent
Q	6 in shotcrete plus mesh reinforcing	2 in. shotcrete plus mesh reinforcing plus 3/4 in., 9-ft-long, grouted bolts @ 3 ft. spcg	1-2 in. shotcrete	3/4 in. bolts, 7-ft-long, grouted, @ 3 ft. to 4 ft. spcg
RMR	Bolts, 6 to 7 ft long @ 2.5 ft. spcg, + mesh or 6 in shotcrete or med. ribs @ 3 ft. spcg	Bolts, 6 to 7 ft long 4 ft. spcg, + mesh or 4 in shotcrete or light ribs @ 5 ft. spcg	Bolts, 4 to 5 ft. long 3 to 5 ft spcg, + mesh or 3 in shotcrete	Bolts, 4 to 5 ft long 6 ft. spcg or 2 in shotcrete
RSR	3/4 in bolts @ 2.5 ft. spcg or 3 in. shotcrete or 4H13 sets @ 2 ft. spcg	3/4 in. bolts @ 3 ft. spcg or 2 in shotcrete or 4H13 sets @ 4 ft. spcg	3/4 in. bolts @ 4 ft. spcg or 2 in. shotcrete or 4I7.7 sets @ 4 ft. spcg	no support
Encountered	7-ft-long bolts plus mesh and shotcrete plus reinforced concrete ± yield steel sets	7-ft-long bolts plus mesh and shotcrete plus massive concrete	Massive concrete ± bolts ± mesh and shotcrete	Massive concrete spot bolting local shotcrete

Support Performance

Lining cracking common over haulage cut-outs; severe in weaker ground.
Damage in weak ground extends to haulage.
Repair common in weaker ground.
Problems can be severe in isolated competent areas and very severe beneath stubs.
Generally good support performance in average to competent ground.

TABLE 16 - Predicted Encountered, and Planned Support for Mine C.

	Altered Porphyry	Transition	Footwall Crud
Q	Bolts, 3/4 in., 7-ft-long, grouted, @ 3 ft. spcg plus 2 in shotcrete, mesh-reinf.	Bolts, 3/4 in., 9-ft-long, grouted, tensioned, @ 3 ft. spcg plus 2 in. shotcrete	Bolts, 3/4 in., 7-ft-long, grouted, tensioned, @ 3 ft. spcg plus concrete arches
RMR	Bolts, 3/4 in., 10-ft-long, 5 ft. to 7 ft. spcg plus 2 in. shotcrete as req'd.	Bolts, 3/4 in., 12-ft-long, 4 ft. to 5 ft. spcg with mesh plus 2 in. to 4 in shotcrete	Bolts, 3/4 in., 14-ft-long, 3 ft. spcg., with mesh plus 6 in. shotcrete plus light-med ribs @ 4 ft. spcg., as req'd.
RSR	Bolts, 3/4 in. @ 4 ft. spcg or 4I7.7 sets @ 4 ft. spcg or 2 in. shotcrete (varies)	Bolts, 3/4 in. @ 4 ft. spcg or 4I7.7 sets @ 3.5 ft. spcg or 2 in shotcrete	Bolts, 3/4 in. @ 2.5 ft. spcg or 4H13 sets @ 4 ft spcg or 2.5 in. shotcrete
Encountered	3 ft. massive concrete plus bolts @ 3 ft to 5 ft. with straps plus w6 x 20 sets @ 5 ft spcg (intersections, access)	3 ft. massive concrete plus bolts @ 3 to 5 ft. spcg with straps plus w6 x 20 sets in weaker ground	3 ft. massive concrete plus timber or steel sets (heavy steel sets in vent drift)

Support Performance

No production history.

TABLE 17 - Predicted and Encountered Support at Mine D

	[illegible]	OSP	HW
Q	Shotcrete, 1 to 2 in.	No support	Shotcrete, mesh-reinf., 4 in. to 10 in.
RMR	Bolts 3/4 in., 10-ft-long, @ 7 ft. spcg plus occa. wire mesh plus occa. shotcrete, 2 in.	Bolts, 3/4 in., 10-ft-long, @ 7 ft. spcg as req'd. plus occa. shotcrete	Bolts, 3/4 in., 11-ft-long, @ 3 to 5 ft. spcg with mesh or 4 to 6 in. shotcrete or light steel sets @ 4 ft. spcg
RSR	5/8-in.-dia. bolts @ 5 ft. spcg	5/8-in.-dia. bolts @ 5 ft. spcg	1-in-dia. bolts @ 3 ft spcg or 1 in. shotcrete or 4H13 sets @ 3 ft. spcg
*Encountered (temporary)		Split set bolts + mesh, 2 ft-5ft spcg	Plus steel straps
*Planned (permanent)	massive concrete	massive concrete	massive concrete plus steel sets

*8-inch wide-flange steel sets in (unclassified) backfill.

Support Performance

Slusher drifts parallel to "crud" and close to slusher drifts perpendicular to "crud" affected by squeeze at ends.

Vent drift on 900 level parallel to "crud" and partly within it nearly closed during production.

Generally adequate performance elsewhere, where construction practice in accordance to design.

favor of steel sets. Also there are rock competence variations within the three broad groupings given and the installed support has, in many cases, been sensitive to these. Support recommendations by Laubscher's scheme vary from concrete in the "crud" zone to bolts and 2 in. of shotcrete in the transition, to no support in altered porphyry.

5.5.5 Selection of RMR

Since none of the three basic RQI systems tried could recommend adequate support for block caving drifts, it was necessary to select a system that could be made effective through modification or adjustment.

From Table 14 through 17, it can be readily seen that the RSR has limited usefulness for predicting rock bolt and shotcrete temporary support for block caving mining. The RSR does not extend recommendations to heavily reinforced concrete support. By design, the RSR is general in its approach. In the complex geological setting of most caving orebodies, the ratings tend to be too subjective. Finally, the initial ratings are not independent of type and orientation of drift. Independence of orientation was a prime consideration, because in most caving mine layouts, there are numerous orientations that will occur within a given geological regime, and it would be better to rate the geology, in such cases, only once, than to have to do it for each orientation of interest.

The Q-system, while giving excellent recommendations for initial support in many instances, is too complex to be applied to the widely-ranging geologic conditions within most caving orebodies. The basic ratings are highly sensitive to the most difficult parameters to evaluate (Section 2.2.8) and it was thought that to do an adequate job of this, more time would be required than would be available at many mining operations. Furthermore, the data base from which the Q-system is drawn makes it better suited to large-scale underground structures that are amenable to more detailed study. Openings of the size and purpose of caving mine production drifts are not well-represented in the data base. Finally, the highly detailed and meticulous support recommendations depend greatly on such concepts as squeezing or swelling potential, rock reinforcement potential, and excavation support ratio, which are not defined by the basic input data. While the application of these concepts is certainly valid and useful, their misinterpretation within the mining community is a distinct possibility with significant consequences.

The RMR system, while requiring some judgement, is neither as subjective as the RSR nor as detailed as Q. The support recommendations, although alone not adequate to predict drift support requirements, are not so rigorous that they exceed the precision of the input data. The RMR is uniformly sensitive to all input parameters, and the rating is fully defined by the input data. Thus the data are amenable to routine collection by geologic technicians who may be employed by a mining company for such a purpose. The problems with the support recommendations can be traced to conditions external to the classification process, such as geometric and engineering variables. The RMR can also readily be made independent of drift azimuth, since fracture orientation is an adjustment rather than an initial input factor. Finally, an adjustment scheme (Laubscher's) has been devised that yields results in conformance with U. S. mining practice. Although this adjustment scheme also could be improved in some ways, it is a valuable starting point.

For these reasons, it was decided to base the modified and adjusted classification scheme on an RMR-type approach. The derivation of the adjustments is summarized in Section 6.0.

6.0 CLASSIFICATION ADJUSTMENTS

This section describes the process used in arriving at the adjustments and modifications to the RMR system, and how these were applied to the field sites.

As was indicated early in this report, (Section 1.1) the rock mass/support system in tunneling differs widely from that in mining, especially in caving production blocks. In formulating the adjustment structure, the most important factors giving rise to this difference were identified. Preliminary ratings were assigned to these factors, based initially on judgement and theory. The rating structure was critically reviewed to ensure that, to the greatest extent possible, the ratings values would be explicitly defined by readily measurable, objective data. The ratings were then applied to the field data and varied as necessary to fit the observations of support performance. Whenever installed support appeared to be inadequate or overdesigned, the system was adjusted accordingly.

The result is a system that will generate support recommendations that agree with adequate support practices in the data base, and offer improvements on inadequate or overconservative support practices. Such improvements allow for what are the most probable causes of the problem. Since the data base covers a wide range of geological settings and conditions, the developed classification system should be useful to many mine planners.

Section 6 deals with the development of the MBR classification parameters. Sections 6.1 and 6.2 develop important background concepts; Section 6.3 describes the adjustments in more detail. Section 6.4 shows how the studied mines fit into the MBR system. It will be helpful in understanding Section 6.4, if the reader is familiar with the MBR procedure as illustrated in Volume II (Manual) of this report. In Section 6.5, the adjustment process is critically reviewed.

6.1 Support Failure

There are as many views of what constitutes unacceptable support performance as there are philosophies of mine operation. Most would agree that the limit is reached when one is "run out" because the drift presents safety hazards, is impassable, or restricts ventilation. However, the decision to repair, re-mine, or abandon, ordinarily takes place before this limit occurs. The need to repair at all impedes production, an event that is to be avoided whenever possible.

Most block caving supports can tolerate considerable distress; so long as the failure stops without impeding ore extraction, no real problem is experienced. However, once damaged, monolithic concrete as well as steel sets are far less effective in supporting the ground, and the failure is not likely to stop unless the initiating mechanism is removed or repair measures are instituted. Consequently, even moderate cracking of concrete linings is a cause for concern in production drifts, as is visible deformation of steel sets.

Concrete linings in production drifts are designed to resist the very high abutment stresses that result from cave initiation. They are not, however, as effective at resisting the large rock mass strains that occur at other times in the caving process. Tensile strains very readily crack unreinforced, monolithic concrete linings. Compressive strains cause stress concentrations in the comparatively unyielding linings and produce spalling at the ribs or back. Deformations tending to shear or roll the lining can be extremely damaging.

Caving is by nature a mining method that induces large rock mass strains. Large-scale rock mass movements are encouraged in some areas and resisted in others. It is the resistance to strain that sets up high drift support loads. For this reason, the strain behavior of the rock mass, and the response of various support systems to this behavior, constitute the framework of the following discussion.

6.2 Progression of Rock Mass Strains

6.2.1 Generalized Strain-Time/Distance Relation

In general, there are several sequential steps in the drift/rock mass deformation history, each with characteristic magnitudes, durations, and areal extents depending on ore body competence:

1. Prior to initiating caving, but subsequent to drift development, the rock mass and drift are in a state of near-equilibrium. Zones of stress concentration have stabilized, as for a tunnel.

2. For a newly undercut panel, the rock mass and drifts directly beneath the undercut respond by dilating towards the void created, and undergo a release of compressive strain that manifests itself as tensile failures.

3. For adjacent areas, the rock mass is compressed as the load is shed from the undercut level.

4. As undercutting proceeds and the mined area advances, formerly compressed areas are relieved. The extending caving mass causes further dilation into the caving area of the surrounding rock mass and drifts.

5. When the mining area is in full production, high and erratic compressive deformations are once more manifested, owing to the manner of drawing of ore and the consequent dynamic behavior of the caved rock mass.

Panek (1982) has developed a synthesis of caving (undercutting) effects (Figure 5) that is both elegant and simple. From this diagram one can see the various strain zones referred to above.

These strain zones can also be seen in the field. Figure 6 illustrates the location of these zones and their manifestations for one instance studied during the field work, in this case, at Mine D. Figure 6 shows the tendency for thin, hairline cracks, commonly at springline, to form directly beneath the undercut. The location of the abutment zone (Zone B) is marked by widespread, shallow spalling of ribs and springline. Further away (Zone A), near-vertical cracks form diagonally across the slusher drifts, sub-parallel to the cave line trend.

Similar observations at other mines lead to the description of a generalized strain-distance (or strain-time, if the cave front advances uniformly) relationship. This is shown on Figure 7. The steps previously listed as 2 through 5 above are recognizable in Figure 7 as zones B through D, respectively.

The several zones delineated in Figure 7's strain history have all been reported by most, if not all, prior workers. However, some zones are less well developed or apparently absent in some cases. The presence and size of strain zones are related to the properties of the rock mass.

6.2.2 Dependence of Strain Zones on RMR

We have selected the Geomechanics System of Bieniawski as the basic system to be adjusted and modified. The next step is to characterize the rock mass conditions in various caving environments by assigning a representative value of RMR, and then relating peak strain values for each zone to the RMR for the involved rock mass.

Since complete observations of the mining cycle and resultant drift deformations were not possible at Mines A, B, C, and D, such data were sought in the literature for mines of a similar nature. Tables 18 and 19 present published data on the deformation histories of drifts in several mines, using the strain zones of Figure 7 and the basic RMR ratings as keys.

These data in Tables 18 and 19 are approximate and somewhat generalized. In order to reduce all data to a common basis, it was assumed that drift convergence measurements may be converted to

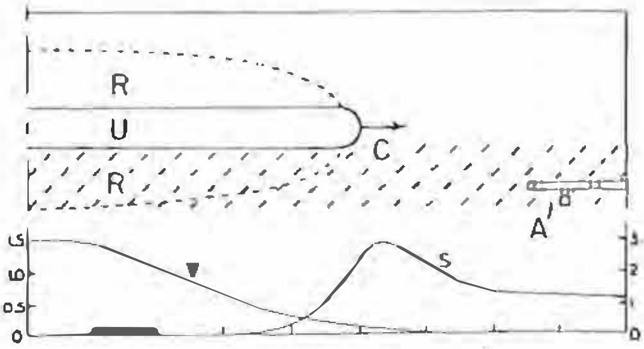


Figure 5 - Main Effects of Undercutting (Vertical Section) (after Panek, 1982, p. 1457). U is undercut slot; R is zone of relieved vertical compressive stress (contains tensile stresses if ratio of horizontal to vertical lithostatic stress is less than one-half); C is zone of increased vertical compressive stress; A is potential access openings coming within the zone of influence of the advancing undercut; s is vertical compressive stress (as multiple of vertical lithostatic stress Sv) at points in hatched zone; v is uplift of points in hatched zone (as multiple of SvL/2G), G is modulus of ridity [based on plane-strain elasticity solution (method of Muskhelishvili) for an ovaloidal slot, width 2L/height = 16]. The graphs are representative of all points in the hatched zone, the height of which is one-eighth the width of the undercut, the zone where access openings are typically located. Used with permission of the Society of Mining Engineers AIME.

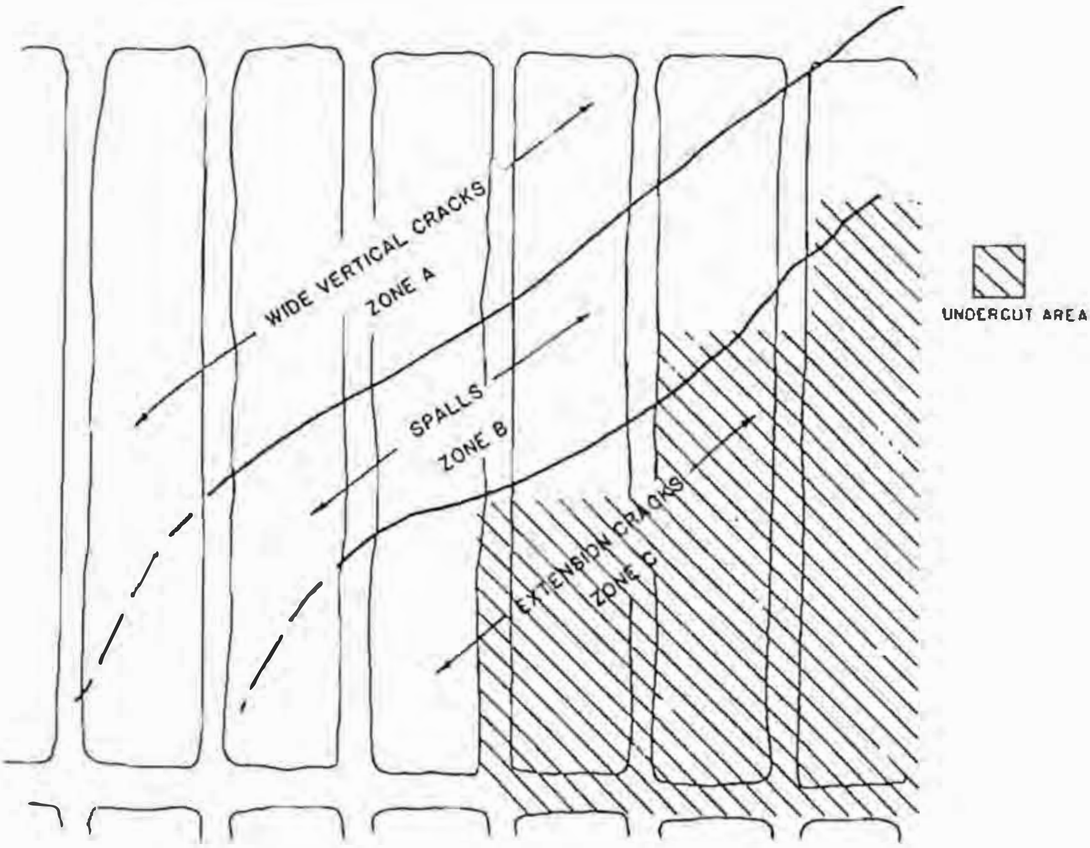


Figure 6 - Field Observations in a Partially Undercut Block. Shading shows extent of undercutting at time of observations. The three zones shown are generalized descriptions of concrete lining behavior as seen in the slusher drifts.

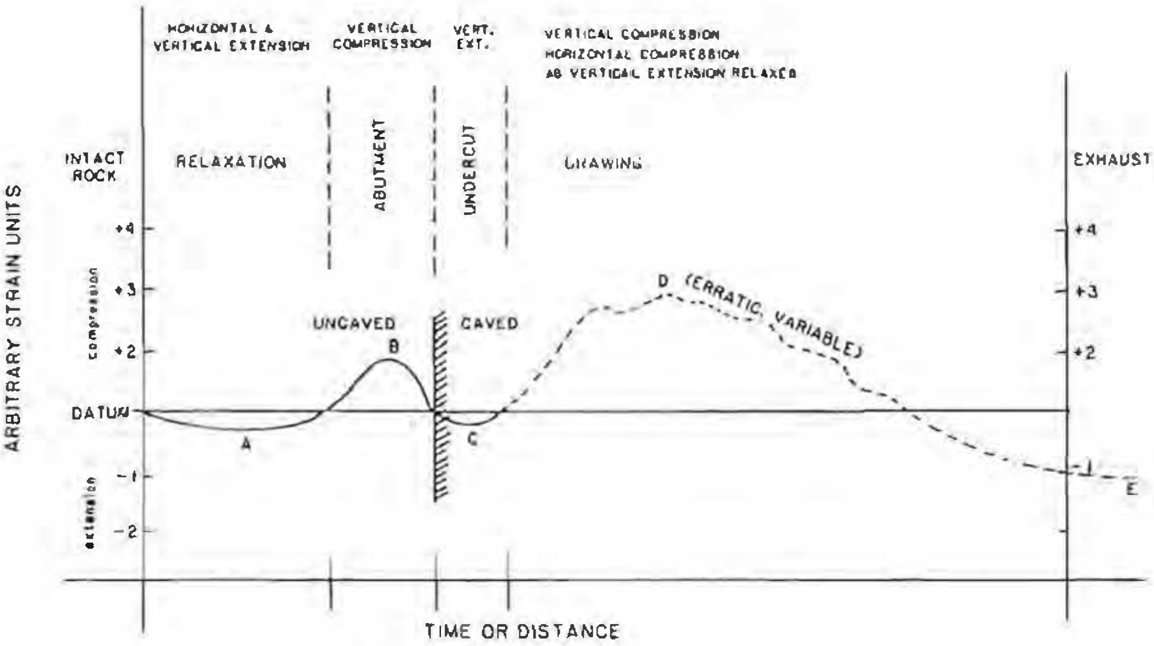


Figure 7 - Generalized Rock Mass Behavior in the Production History of a Caving Area. A, B, C, D, and E are principal strain zones. Senses of strain (+ or -) are <u>generally</u> applicable as shown.

TABLE 18 Strain Data for Caving Mine Drift Strain Zone

Mine	RMR	Strain Zone of Figure 7 in./in.	Reference
		ZONE A	
Henderson	60+	0(?)	Brumleve & Maier (1981)
San Manuel	35	-0.008	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	-0.01	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	-0.012	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	-0.018	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	-0.008	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	-0.008	Panek (1981) and Panek and Tesch (1981)
Climax	55	-0.0001	Bolmer (1965)
Climax	55	-0.0001	Bolmer (1965)
Climax	55	-0.0001	Bolmer (1965)
Climax	55	-0.0001	Bolmer (1965)
Climax	55	-0.0004	Bolmer (1965)

TABLE 18 (continued)

Mine	RMR	Strain Zone of Figure 7 in./in.	Reference
		ZONE B	
Henderson	60+	+0.033	Brumleve & Maier (1981)
Henderson	60+	+0.077	Brumleve & Maier (1981)
Henderson	60+	+0.004	Brumleve & Maier (1981)
San Manuel	35	+0.018	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	+0.004	Panek (1981) and Panek and Tesch (1981)
San Manue	35	+0.006	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	+0.012	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	+0.018	Panek (1981) and Panek and Tesch (1981)
San Manuel	35	+0.01	Panek (1981) and Panek and Tesch (1981)
Climax	55	+0.0028	Julin (1964)
Climax	55	+0.0019	Julin (1964)
Climax	55	+0.0002	Bolmer (1965)
Climax	55	+0.0005	Bolmer (1965)
Climax	55	+0.001	Bolmer (1965)
Climax	55	+0.0004	Bolmer (1965)

TABLE 18 (continued)

Mine	RMR	Strain Zone of Figure 7 in./in.	Reference
		ZONE C	
Henderson	60+	-0.003	Brumleve & Maier (1981)
Henderson	60+	+0.014	Brumleve & Maier (1981)
Henderson	60+	+0.0058	Brumleve & Maier (1981)
San Manuel	35	-0.008	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	-0.006	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	-0.004	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	-0.02	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	-0.016	Panek (1981) and Panek & Tesch (1981)
Climax	55	-0.0022	Julin (1964)
Climax	55	-0.0022	Julin (1964)
Climax	55	-0.0006	Bolmer (1965)
Climax	55	-0.0006	Bolmer (1965)
Climax	55	0	Bolmer (1965)
Climax	55	-0.0012	Bolmer (1965)
		ZONE D	
San Manuel	35	+0.012	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	+0.008	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	+0.0012	Panek (1981) and Panek & Tesch (1981)

TABLE 19 - Strain Zone Length Data for Caving Mine Drifts

Mine	RMR	Strain Zone Length, ft	Reference
		ZONE A	
Henderson	60+	0	Brumleve & Maier (1981)
San Manuel	35	200	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	150	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	320	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	200	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	300	Panek (1981) and Panek & Tesch (1981)
San Manuel	35	450	Thomas (1971)
Climax	55	75	Bolmer (1965)
Climax	65	60	Bolmer (1965)
		ZONE B	
Henderson	60+	130	Brumleve & Maier
Henderson	60+	225	Brumleve & Maier
San Manuel	35	300	Thomas (1971)
San Manuel	35	300	Thomas (1971)
Climax	55	300	Julin (1964)
Climax	55	100-300	Julin (1964)
Climax	55	245	Julin (1964)

TABLE 19, Continued

Mine	RMR	Strain Zone Length, ft	Reference
	ZONE C		
Henderson	60+	190	Brumleve & Maier
San Manuel	35	0-15(?)	Thomas (1971)
Climax	55	280-400	Julin (1964)
Climax	55	210-300	Julin (1964)

strain by dividing the closure or dilation by the drift diameter. Also, directions are not necessarily normal to the cave line front. The possible errors in such techniques are fully recognized, but nonetheless, the data are the only data available. Also, it is noted that, as reported by Panek (1981), for at least one instance, the drift linings and drifts are moving along with the deforming rock mass, so that drift deformations approximate rock mass deformations, and vice-versa.

The data from Tables 18 and 19 can be combined into strain curves based on the general case (Figure 7), to show the effect of rock mass quality (Figure 8). Also, from these data, the curves in Figure 8 and the tabulated strain data can be plotted for each zone. Figure 9 shows the resulting relationship between RMR and rock mass strain, for each major recognized strain zone. Also shown on Figure 9 are the strain limits that may be used for various types of commonly used drift linings (Neville, 1974; Watstein and Bresler, 1974; Cuevas, Robles, and de Cossio, 1974). Figure 10 was compiled from Figure 8 data and shows how the length of strain zones A, B, and C varies with RMR.

Throughout this discussion, there has been no mention of the role of draw control and production blasting. Considerations have been limited to control of pre-production strains, as has been a basic assumption throughout this report. It should be remembered, however, that draw control and blasting can be overriding factors in determining drift performance.

6.2.3 Effect of Strains on Lining Integrity

Replotting the now-generalized strain curves for measured field geotechnical data and incorporating the strain limits for installed supports, Figure 11 shows the predicted behavior of several field areas. Detailed observations at these various mines agree remarkably well with the predicted drift behavior, and also show that some instances exhibit overdesign, while others predict trouble, which did, in fact, occur.

In order to illustrate the effect of panel size, the curves of Figure 10 were re-plotted (Figure 12) at 50% of their magnitudes to allow for the peaks of deformations to be outside the mined zone. The heavy lines in Figure 12 are the resulting smaller strains, that would be experienced within panels of the indicated sizes.

The curves in Figure 13 summarize the effect of panel size on geologic environments of varying RMR. This graph shows that only RMR is important in determining support requirements for panels wider than about 200 ft. However, lighter support is possible for smaller blocks. Another aspect of Figure 13 is that workings nearest the

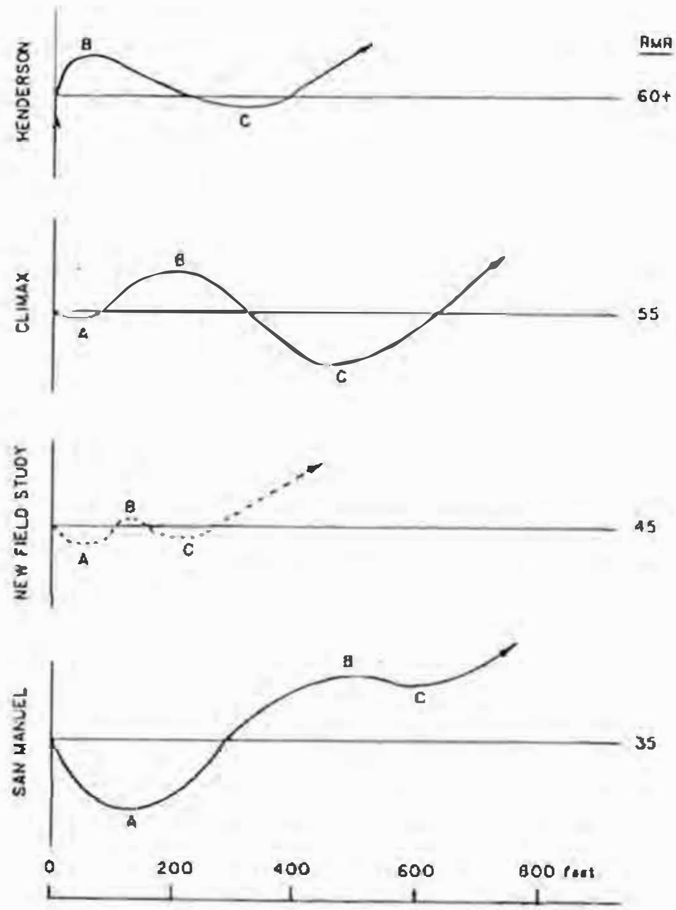


Figure 8 - Generalized Strain - Distance Relationships for Rock Masses of Different Competence. These show the locations of the various strain zones introduced in Figure 7 at an instant when the undercut is well under way. Similar relationships would occur for a given station, if time were plotted rather than distance, on the horizontal axis.

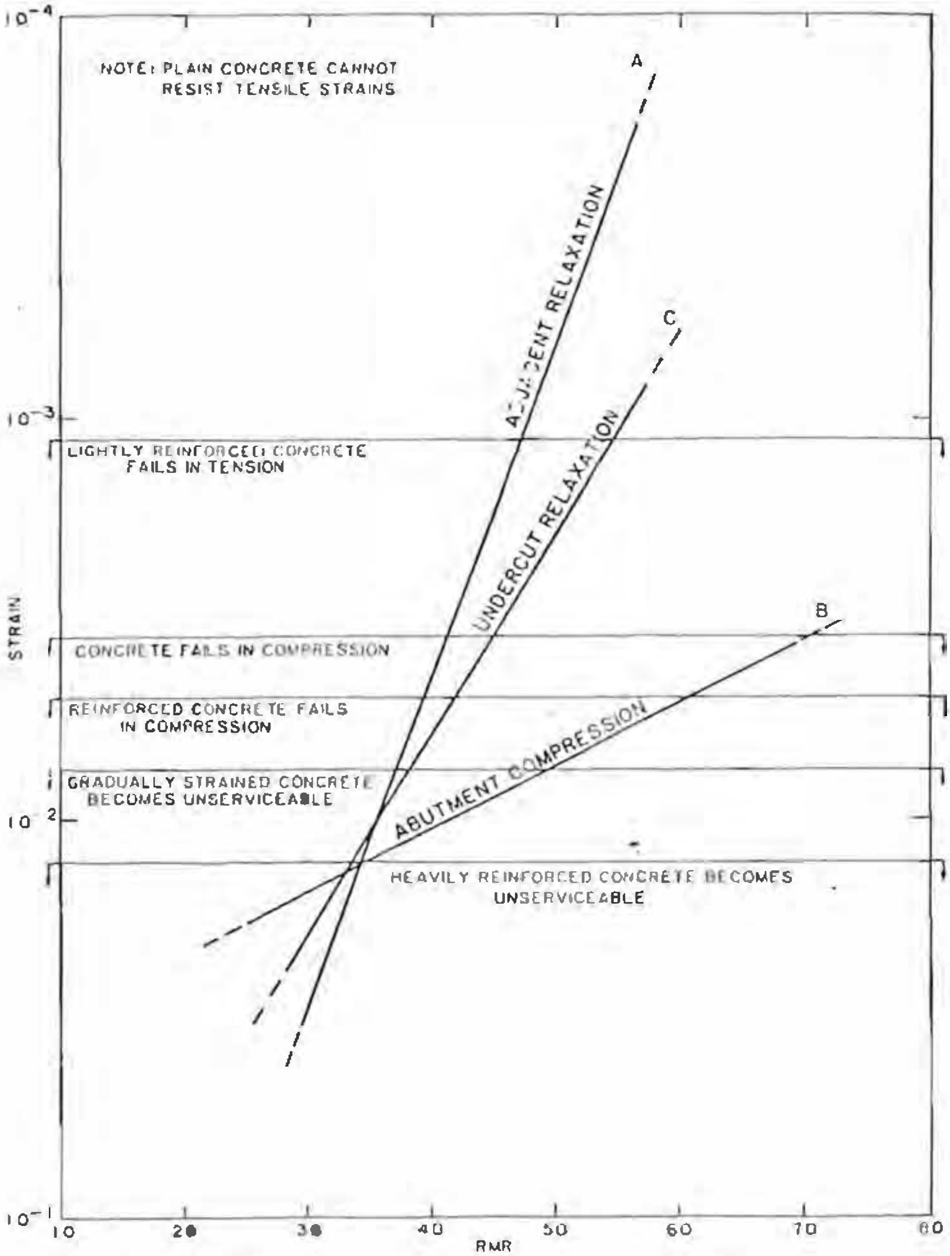


Figure 9 -RMR Values Leading to Concrete Lining Distress for Various Strain Zones. This figure is based on the data in Tables 18 and 19.

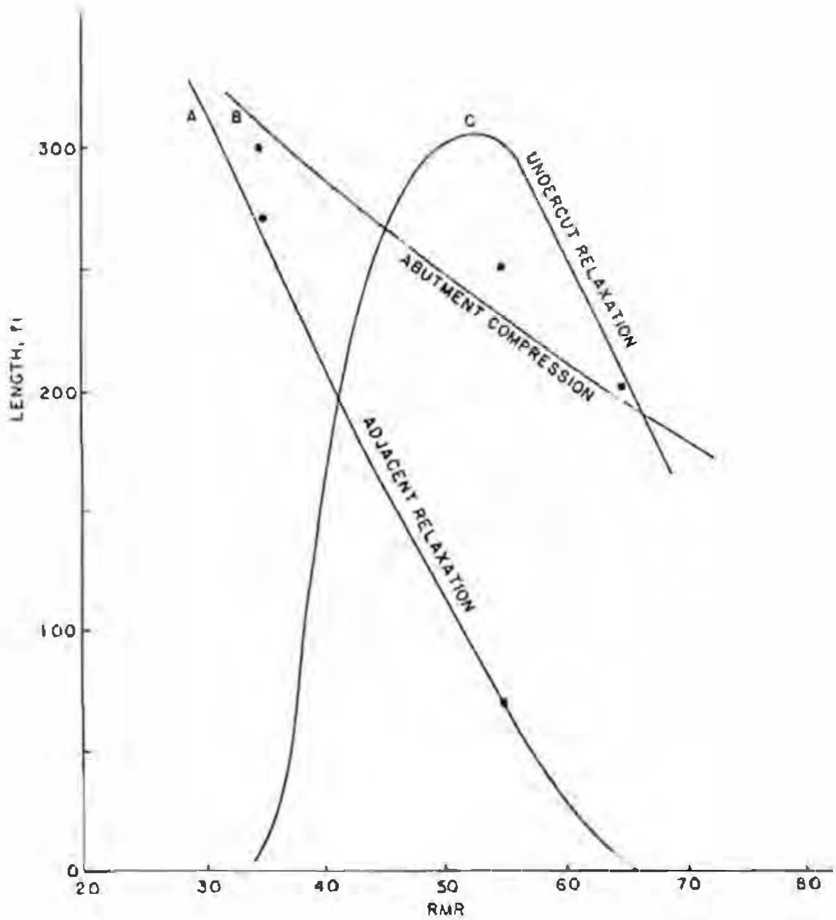
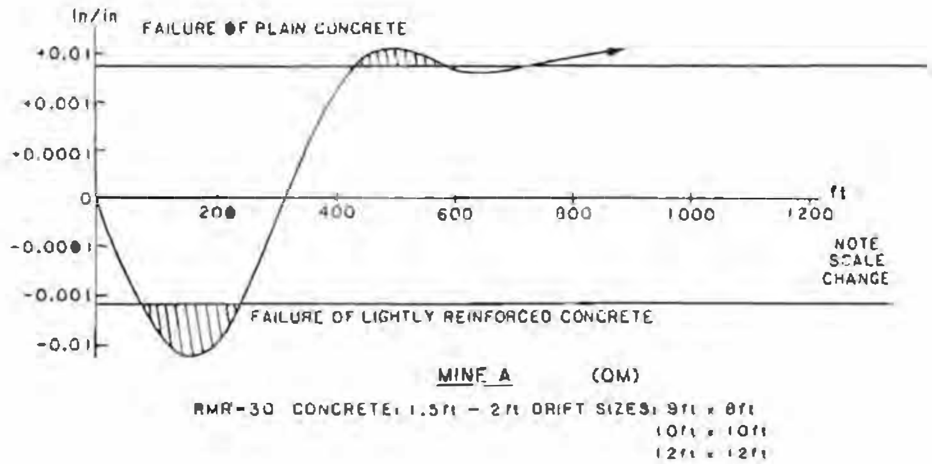


Figure 10 - Dependence of Strain Zone Size on Rock Mass Competence. Length is taken perpendicular to cave line. With respect to Zone C, weak rock masses are capable of releasing only very little strain, whereas very strong rock masses strain less under abutment loading.



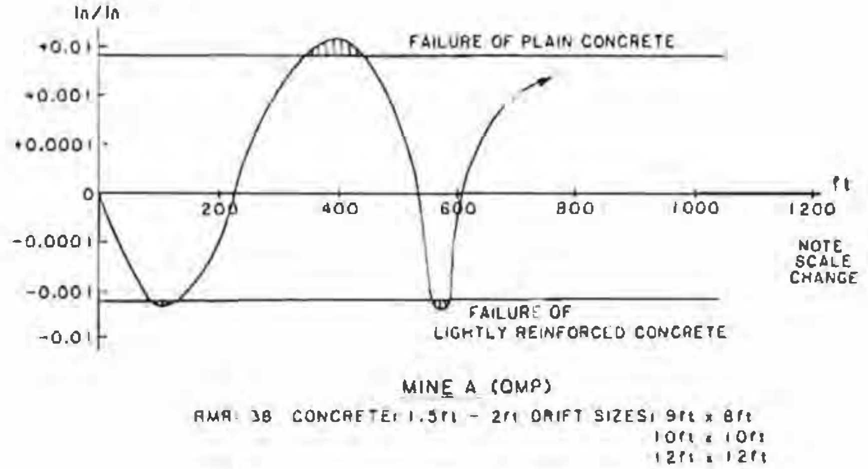


Figure 11 - Predicted Lining and Rock Mass Behavior for Mines in Rock of Varying Competence. From these curves, one can estimate the type, location, magnitude, and approximate time of lining distress.

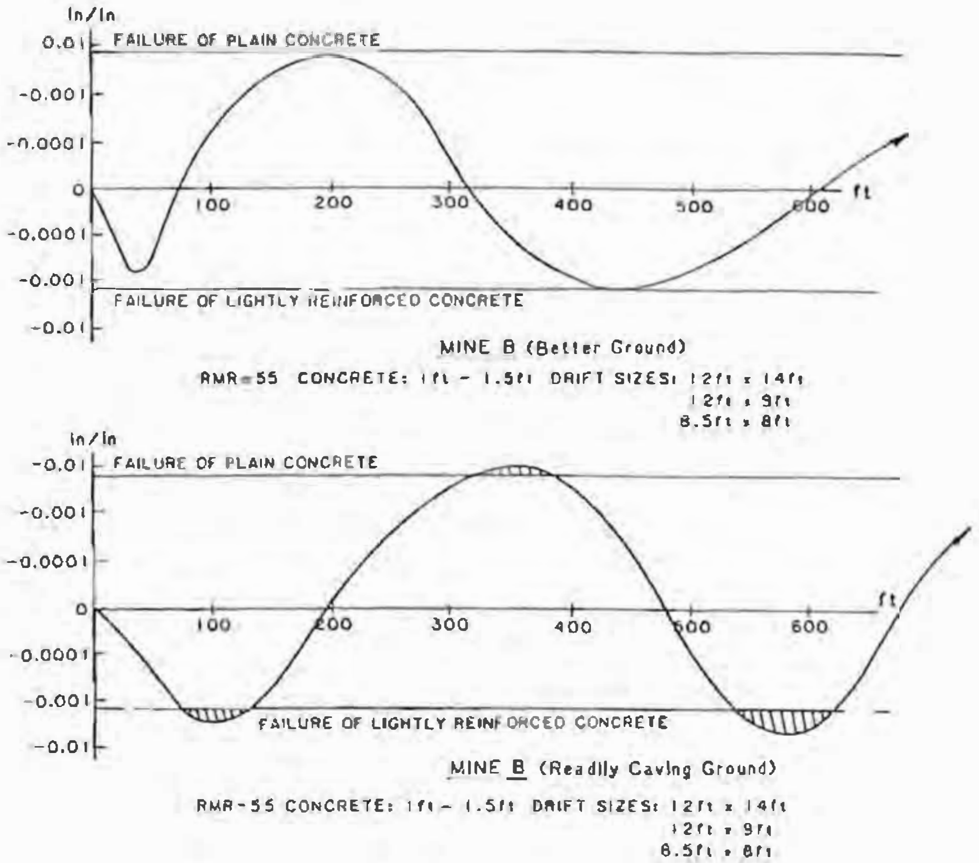
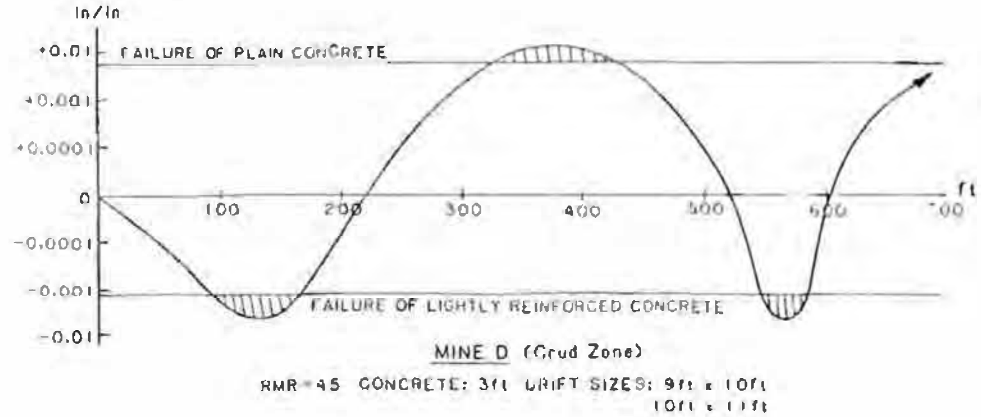


Figure 11 - Continued.



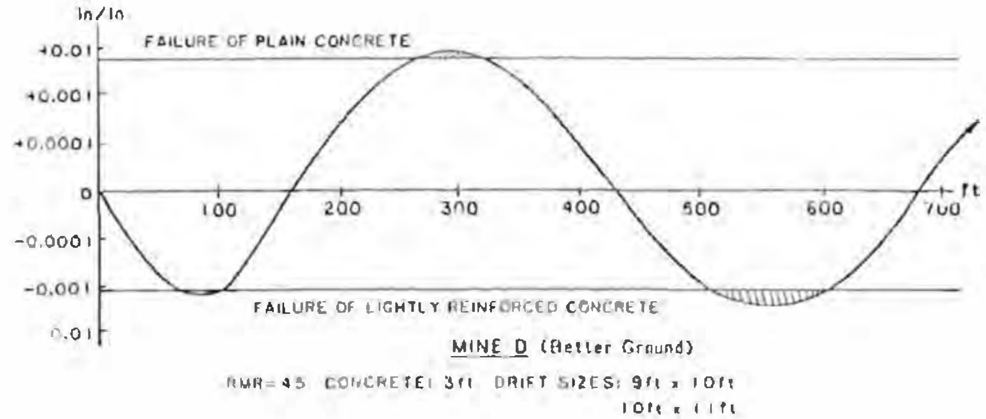


Figure 11 - Continued.

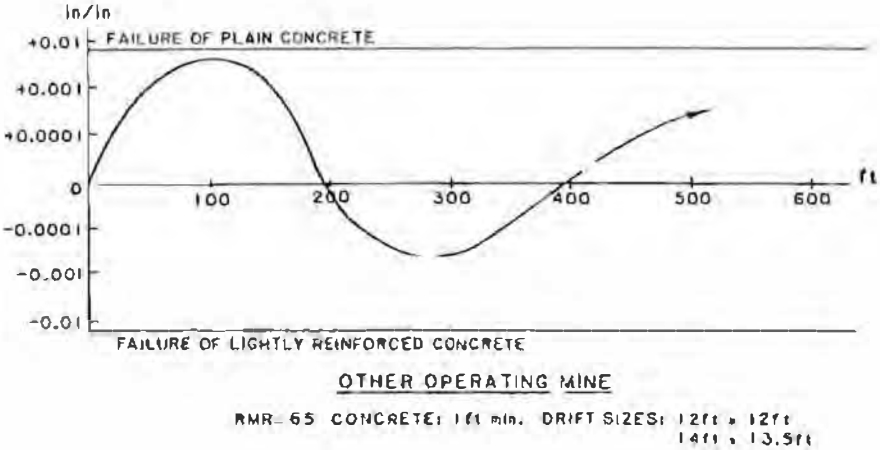


Figure 11 - Continued.

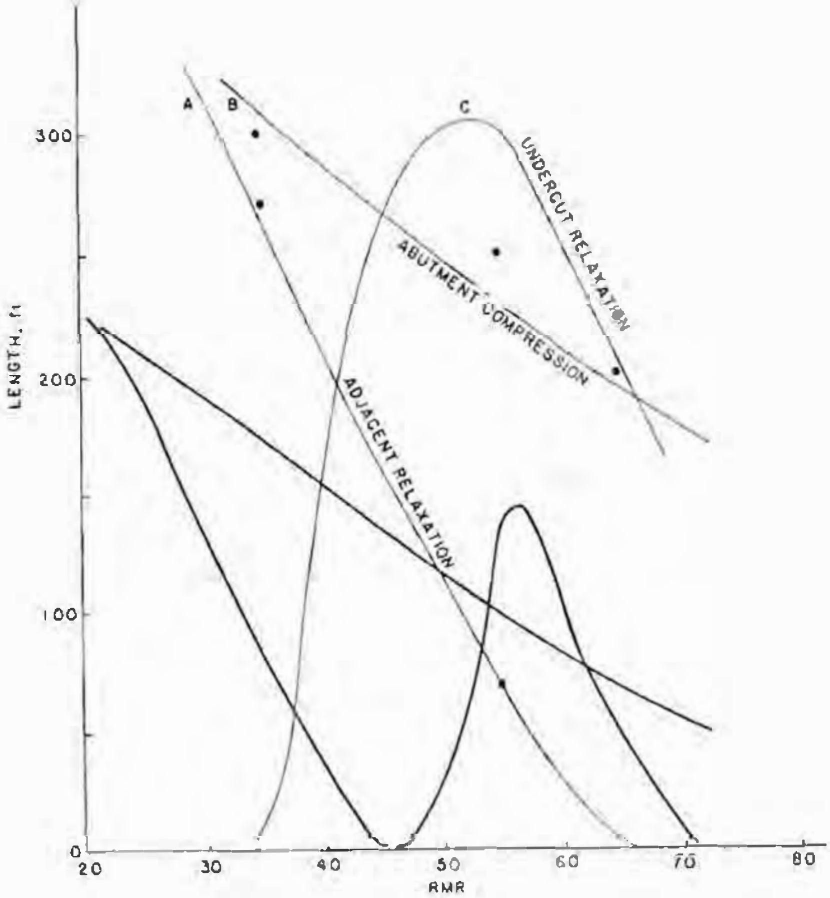


Figure 12 - Panel Strains versus RMR. The heavy lines show a reduced length of strain zone, to allow for peak rock mass deformations to be outside the active panel area. Redrawn from Figure 10 with the assumption that if peak strains occur outside the panel, total deformations are less.

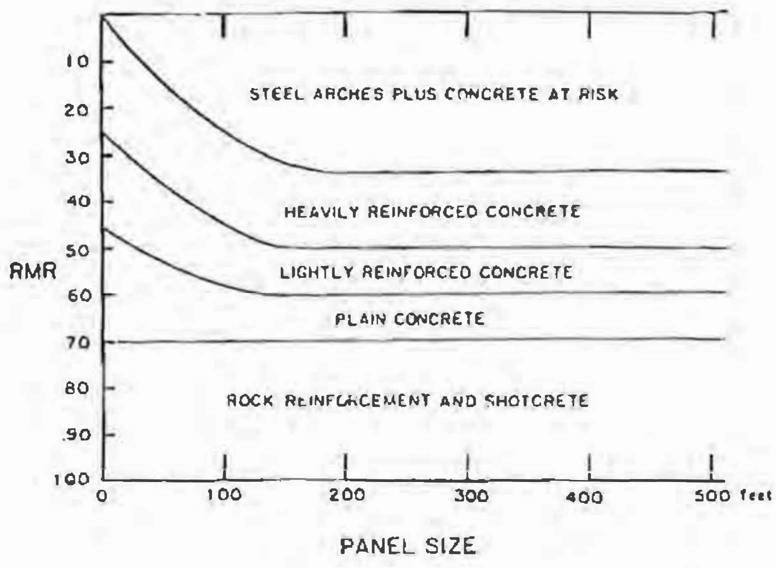


Figure 13 - Support Prediction for Panels of Various Sizes. These recommendations combine the possible concrete failure modes distinguished earlier.

point of cave initiation in a block may experience less abutment stress, simply because less ground has been opened up, than will workings at greater distances of cave advance.

The derivation of Figure 13 provides the basis for one of the key geometric factors. The implications of Figure 13 are drawn upon in the following section.

6.3 Ratings and Adjustment Factors

This section discusses the adjustment terms in detail, as well as gives the rationale for the overall approach to the MBR System. This section is a complement to, and not a replacement for, Volume II (Manual) of this Report. Information on the use and application of the ratings and adjustments is fully presented in the Manual and is not repeated here. Although figures from the Manual are reproduced in this section for completeness, the reader will find it helpful to become familiar with the Manual after completing this section and before beginning Section 6.4.

It was recognized that caving mine drift development is a sequential process that can take considerable time. Thus the production openings serve different functions during development, until caving begins, and consequently, different support requirements exist.

Production panel development involves creation of openings that must stand for long periods, in some cases even years, before caving begins. If there is doubt about the stability of the drift over this period, it is common practice to install the final concrete lining at an early stage, especially in cases where clear evidence exists that the drift is moving. Unfortunately, this has the effect of pre-loading the concrete, probably before it has reached its full strength, and leaves less strength to buttress against the caving loads when production is begun. Lining failures are the all-too-common result, and effective repair is less likely since the reduced size of the drift limits the size of steel support that can be practically installed.

To prevent this type of problem, it is preferable to stabilize the drift prior to lining, so that the final support strength increment is fully available to resist caving and production loads.

Accordingly, the adjustments are applied in two stages after the initial, purely geologic, rating is determined, to keep development support criteria distinct from final support criteria.

The initial rating is very similar to the more generally-used RMR, and is likewise applicable to support recommendations for workings analogous to single tunnels--fringe drifts away from the production area, for example. When multiple openings are created, as for the active production area, adjustments pertinent to this step are applied. Since support requirements are greater, the adjustments tend to decrease the rating. Next, the second adjustment is applied, to yield final support recommendations. This accounts for loads originating from production initiation. Production caving loads and other effects of routine production such as blasting in the draw raises, are assumed, with good mining practice, to be less than the abutment loads.

This process is followed in Figure 14, which is the flow chart for the MBR System. The basic input parameters are along the left and top of the chart, and include:

1. MBR inputs

   - intact rock strength
   - RQD
   - discontinuity spacing
   - discontinuity condition
   - groundwater condition

2. Development Adjustment Inputs

   - blasting damage
   - depth
   - stress field
   - extraction ratio
   - fracture orientation

3. Production Adjustments

   - width, location, and attitude of major structural discontinuities
   - distance to cave line
   - block or panel size
   - acceptable level of repair

6.3.1 First Step--MBR Rating

The initial rock mass rating (MBR) is a modification of the Geomechanics System in that it adjusts the values of the ratings slightly through combination of terms, and restructures the rating procedure to enable it to be applied on the basis of drill core data alone. Also, changes were made to make the geological ratings more responsive to characteristics typical of caving orebodies; intense and variable alteration, dense fracturing, and weak rock masses. In order to obtain a purely geological parameter, the geometric relationship of fracturing pattern to drift orientation is left to the

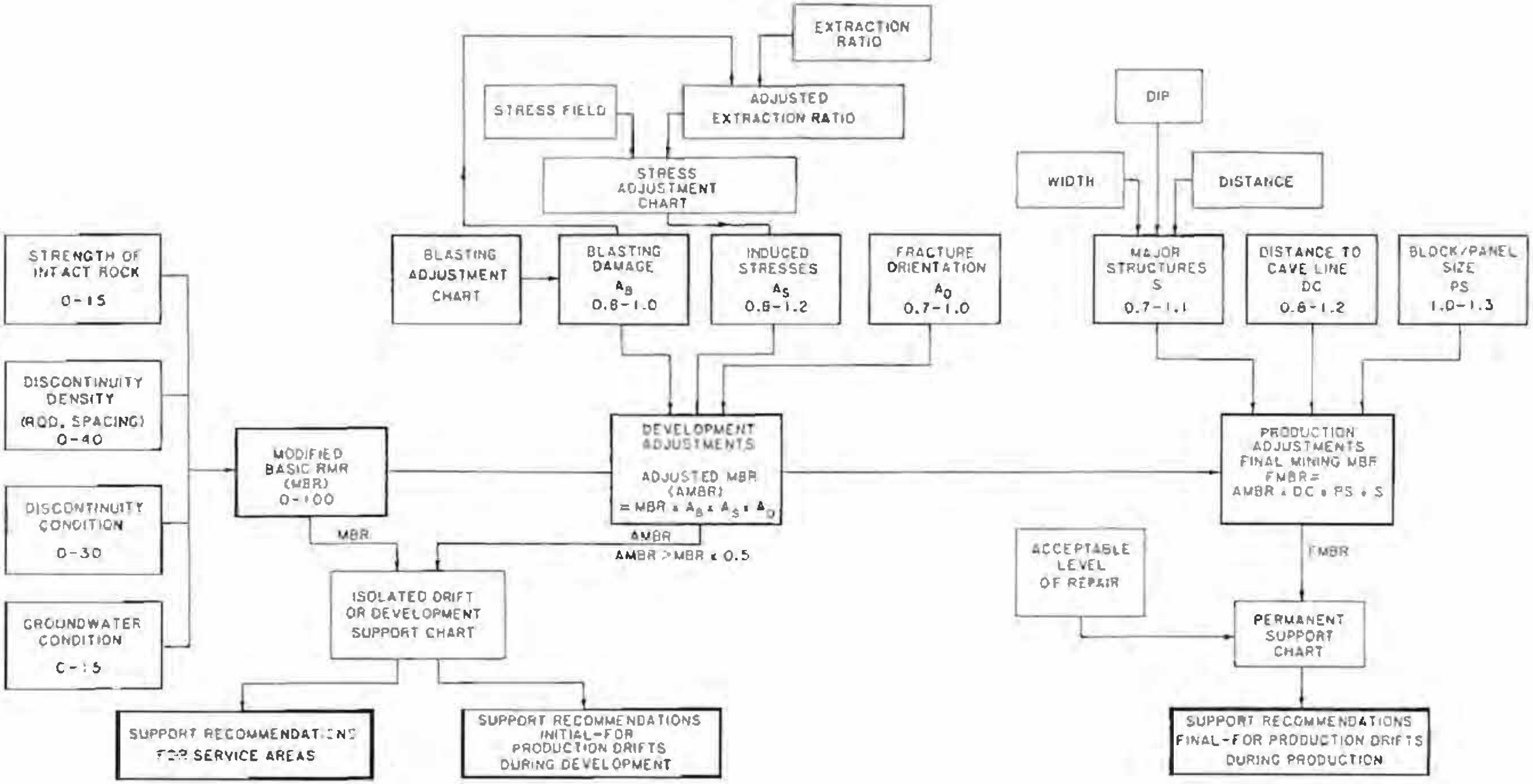


Figure 14 - Organization of the MBR System. The general flow begins at left. Intermediate data inputs originate at the top, and outputs are at the bottom. The multi-stage approach is evident in the groupings of the adjustments.

first adjustment step. Thus the initial rating is called the Modified Basic RMR, or MBR, and is an expression of the geological condition, independent of the type of opening excavated in it.

The Intact Rock Strength rating (Figure 15) is reorganized to allow for different sources of compressive strength data. Many mines routinely collect compressive strength data on new drill core by point load tests, whereas some other mines may prefer the use of laboratory compressive test data. Since these two methods are subject to different sources of bias, and to allow the geologist to select a rating that is defensible and reasonable even with widely variable test data (typical for caving orebodies), the rating is formulated so that the geologist has clear guidelines within a range of possible ratings.

The Geomechanics System calls for both RQD and fracture spacings to be known. However, in many instances at operating caving mines, the user will know one or the other, but not both. Also, RQD is ordinarily measured during exploration, but absolute fracture spacings are not known unless orebody exposures are available. Thus, a relationship from Bieniawski (1979(b)) (Figure 16) allows the user to estimate either RQD or spacing from a knowledge of the other. Then, each can be rated (Figure 17) and the ratings summed to yield the Discontinuity Density. Such a unified term is intended to be a better concept if single-source data are used.

During the field work it became evident that prospective users of the Geomechanics System in the mining community would have the greatest difficulty in assessing the alteration and weathering conditions of the fractures. Typically, joint conditions in caving orebodies are variable in response to contrasting zones of alteration, weathering, and mineralization. Since ratings of such conditions are always at least somewhat subjective and inevitably generalized, the MBR System attempts to develop concepts of joint weakness associated with weathering and alteration through the assembly of general descriptive terms. It was thought that this would be preferable to a more rigorous approach in which a correlation with joint mechanical properties is attempted through numerous refinements of joint descriptions, which may be useful for tunnels in better, less variable ground. The descriptive terms are associated into five highly simplified categories (Table 20) that will permit a rating suitable for mining situations.

The Geomechanics System's rating systems for groundwater conditions is straightforward and appropriate for mining situations (Table 21).

These four geological parameters are summed to obtain the MBR. As with the RMR, a geological rating is obtained that varies from 0 (worst) to 100 (best). A fundamental conviction of the RMR, that

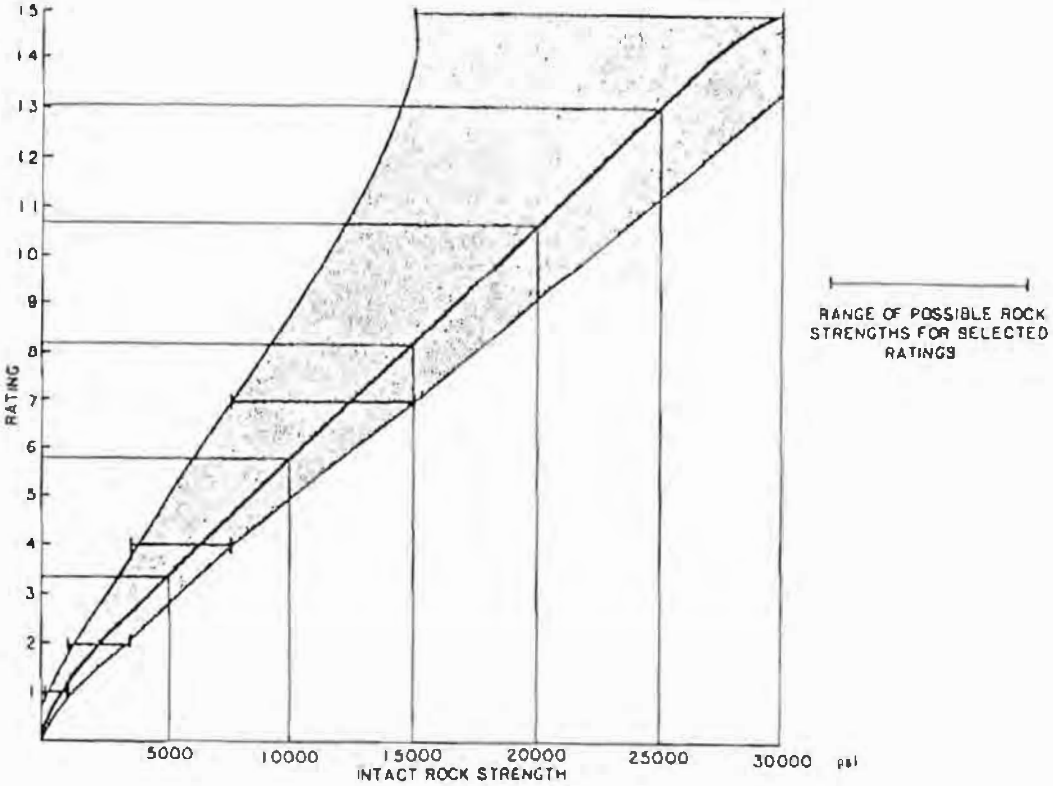


Figure 15 - Rating for Intact Rock Strength. Intended source of data is point load testing. The stippled area allows latitude in assigning ratings where biased test results are suspected. (Derived from relationship of observed rock strengths to ratings in the Geomechanics System. Ranges defined by stippled area are suggested from field and laboratory testing experience.)

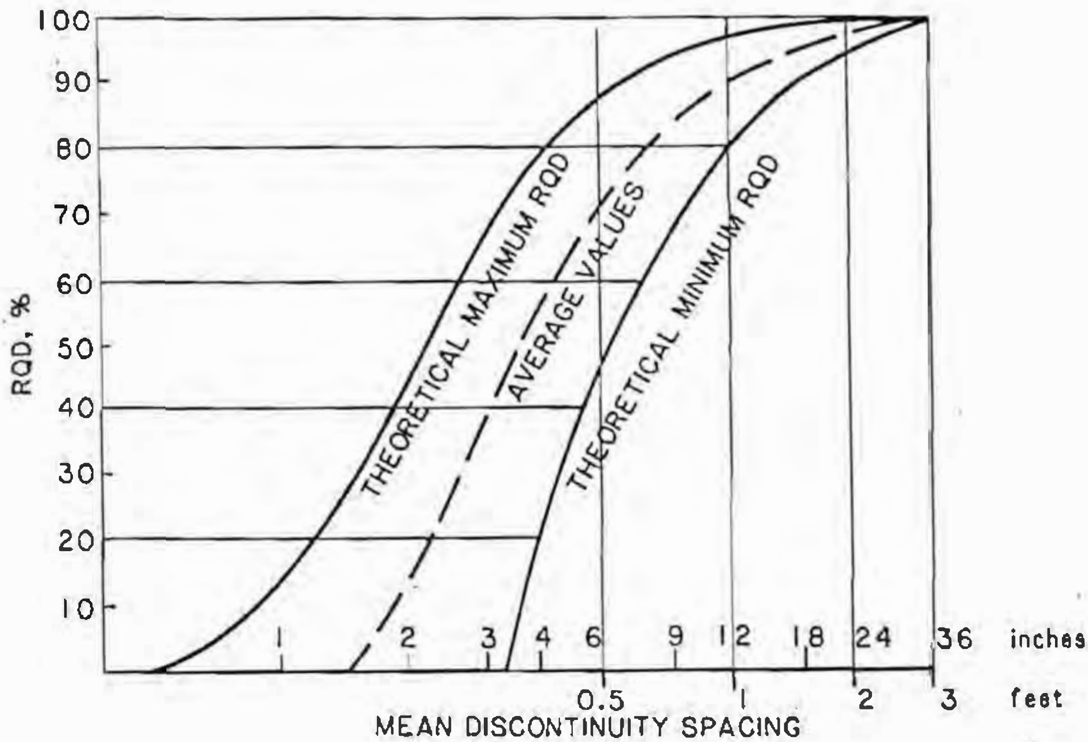
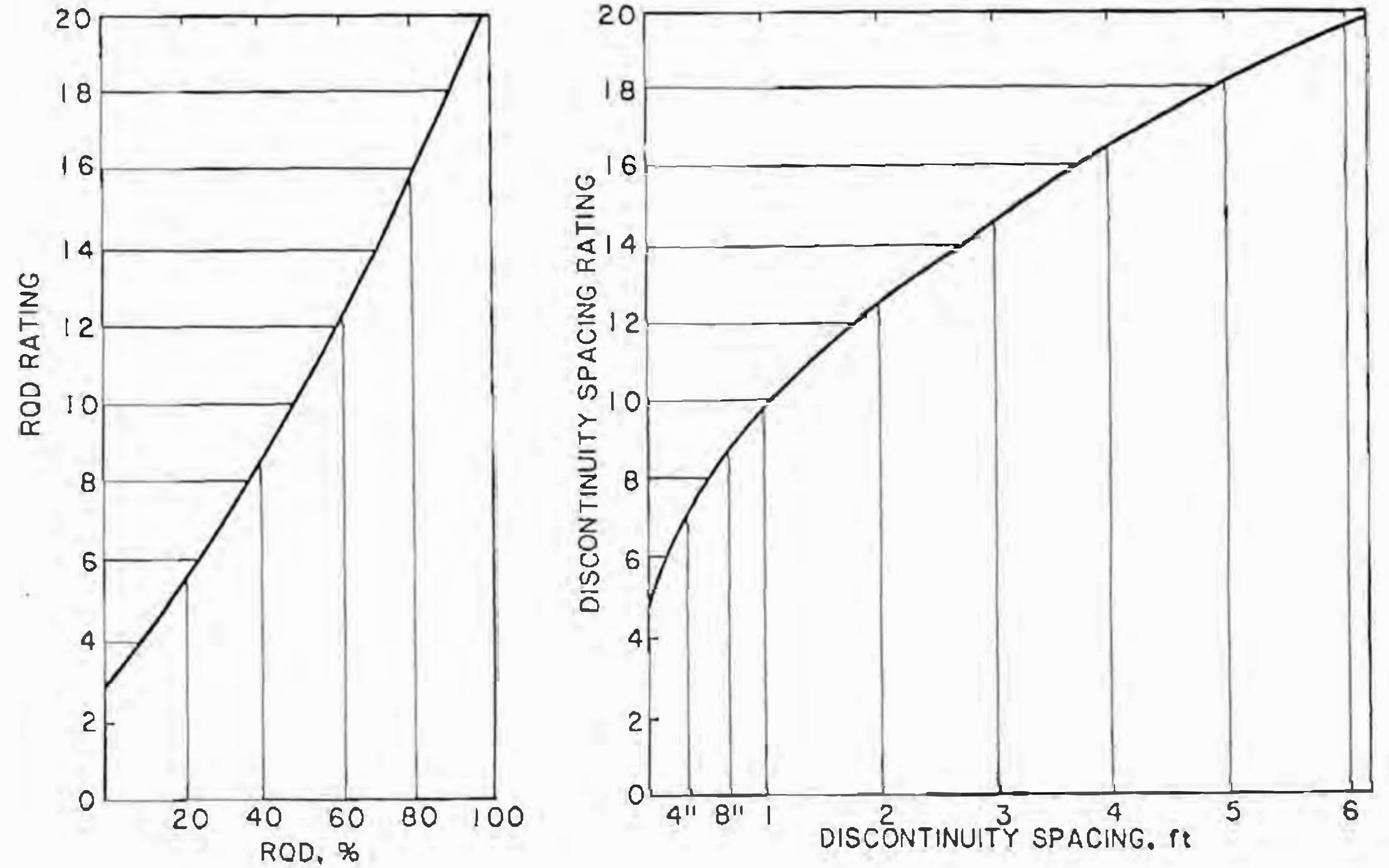


Figure 16 - Theoretical Relationship between RQD and Discontinuity Spacing (after Bieniawski, 1979(2)).



A.

B.

Figure 17 - Ratings for Discontinuity Density.
A - RQD ratings
B - Discontinuity spacing ratings.

TABLE 20 - Discontinuity Condition Ratings

	Description of Discontinuity				
Wall Roughness	VR	R-SR	SR	SM-SK	SK
Wall Separation	None	Hairline	Hairline	<½"	>½"
Joint Filling	None	None	Minor Clay	Stiff Clay, Gouge	Soft Clay, Gouge
Wall Weathering	F	SL	SO	SO	VS
Rating	30	25	20	10	0

VR = Very rough (coarse sandpaper)
R = Rough (medium or fine sandpaper)
SR = Smooth to slightly rough
SM = Smooth but not polished
SK = Slickensided, shiny

F = Hard, unweathered, fresh
SL = Hard, slightly weathered
SO = Softened, strongly weathered
VS = Very soft or decomposed

Table 21 - Groundwater Condition Rating

Water Condition	Completely Dry	Damp	Wet	Dripping	Flowing
Rating	15	10	7	4	0

fracturing is the dominant factor governing rock mass behavior applies especially to caving orebodies. In the MBR, this feature is retained.

The MBR value is found on a chart (Figure 18), to obtain support recommendations for single tunnels and service areas. Support recommendations for very large openings, such as underground crusher halls or hoist houses, may depend on many factors and separate design methods should be used for these.

6.3.2 Second Step--Development Adjustments

The development adjustments recognize that multiple workings in complex arrangements, such as are found in caving mines, interact so that the surrounding near-field stress state is changed. The closer the openings, the greater the degree of stress overlap. This can be related to extraction ratio (the fraction of rock removed from a given area, usually taken as a horizontal plane at springline).

The ability of the rock mass to resist stress interaction effects is influenced by the damage to the intervening pillar due to blasting. Since blasting damage reduces the area of intact rock between openings that is capable of bearing load, the influence of the damage also depends on the pillar width. In addition, the location and magnitude of stress concentrations around multiple openings depends on the orientations of the principal stresses. In high horizontal stress fields, the interior openings are shielded from stress by those at the margins. Finally, the application of the resultant stress field can affect the openings in different ways depending on how free rock blocks are to move. Thus, fracture orientations are important.

It is therefore seen that this step in the rating process incorporates the basic geometric characteristics of the production area--size, orientation, spacing, and location of drifts.

In developing criteria for applying extraction ratio concepts, it was necessary to limit the lateral extent of the computations so that fringe drifts and ventilation laterals with intersecting drifts could be regarded as multiple openings, even if they bound the mining area. It is assumed that non-parallel openings occurring further than 1½ diameters into the rib would have little additional effect, and this was chosen as the limit for computing extraction ratios at intersections and turnouts. The extraction ratio for multiple parallel openings can be readily computed without such assumptions. All extraction ratios are computed perpendicular to the anticipated principal compressive strain, that is, a horizontal plane. Some openings, such as ventilation laterals or access drifts, may be viewed as multiple parallel openings with spacings in the hundreds of feet, or as single drifts with multiple intersections. Using the

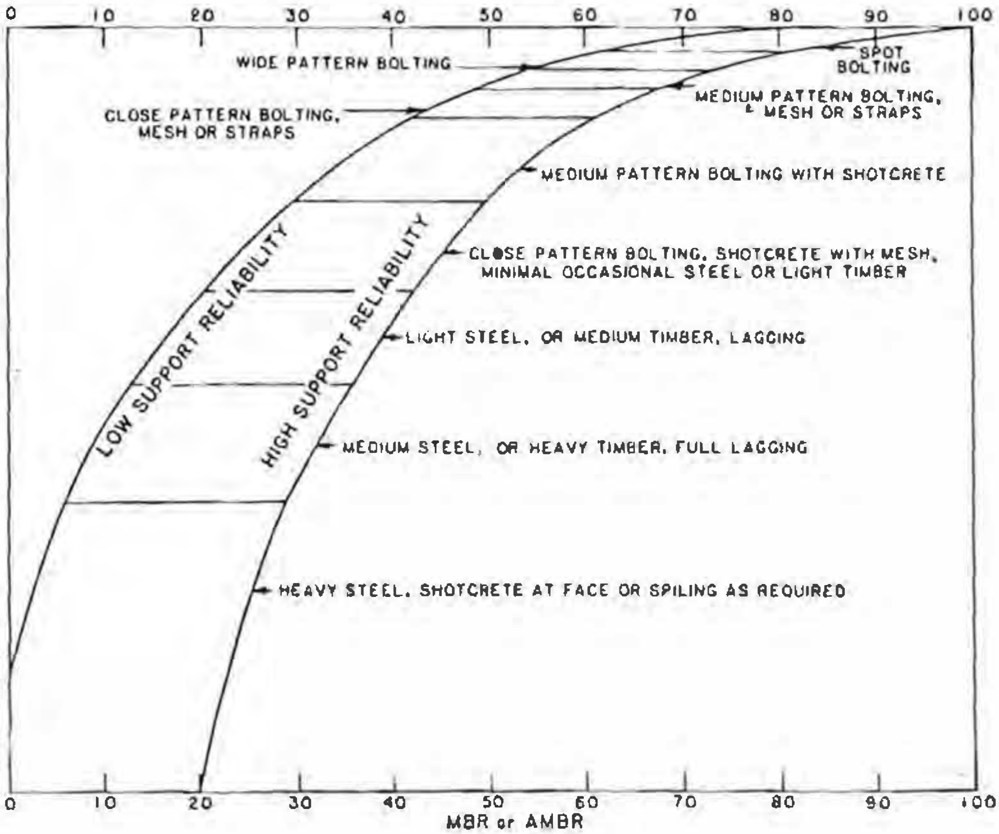


Figure 18 - Isolated Drift or Development Support Chart. Explanations of the support types shown are given in the Manual, Volume II.

extraction ratio concept, it will be readily seen that the best description of these drifts' behavior is as a single opening with multiple intersections, and design can be carried out accordingly.

Being a high-production operation, mine drift development frequently experiences excessive blasting, which damages the rock. There are two principal effects. First, overblasting enhances the development of a loosened zone, which must be taken into account in assessing the effects of applied loads. Second, pillars between openings are effectively reduced in width, fractures are opened, and overbreak occurs. It therefore is recognized that additional loose rock must be supported, and that the effective width of the opening is increased, which changes the extraction ratio.

Thus the MBR provides graphs (Figures 19, 20, and 21) for finding the effective extraction ratio based on blasting practice. In viewing these graphs, it will be seen that one must first have assessed the extent of blast damage.

The appropriate terms are found from Table 22. In computing blast damage adjustments it was assumed that the loosened zone from blasting extends 2 ft into the rock for "light" damage, 5 ft for "moderate" damage, and 8 ft for "severe" damage. It was assumed that this broken rock contributes insignificantly towards resisting stress, so the measured extraction ratio is adjusted (increased) accordingly to yield the effective extraction ratio.

Since the loosened zone extends into the crown as well as rib, development support of this loosened, non-load-bearing rock must also be increased. This results in a direct adjustment A_B for blasting damage (Table 22).

The adjusted extraction ratio, eff e_r, is not used directly as an adjustment to MBR but operates in conjunction with the prevailing state of stress on the level to determine the stress state in the pillars of intact rock between production openings. The result is an adjustment A_S (Figure 22) for Induced Stresses. This approach recognizes the relationship of the applied stresses in the future production area to the resisting capacity of the rock, which is in turn related to the competence of the rock mass (MBR) and the amount of rock of this competence that remains after development (eff e_r).

At this point in the procedure, the geometrical characteristics of the planned level are applied, including azimuth, dimensions, inclination, and location of each key production drift. Heretofore, it has not been possible to address the favorability or unfavorability of the structural fabric since the relationship of structural orientation to drift orientation could not be incorporated into a purely geological rating. However, in the first sequence of adjustments it is possible to recognize that the orientation of fractures

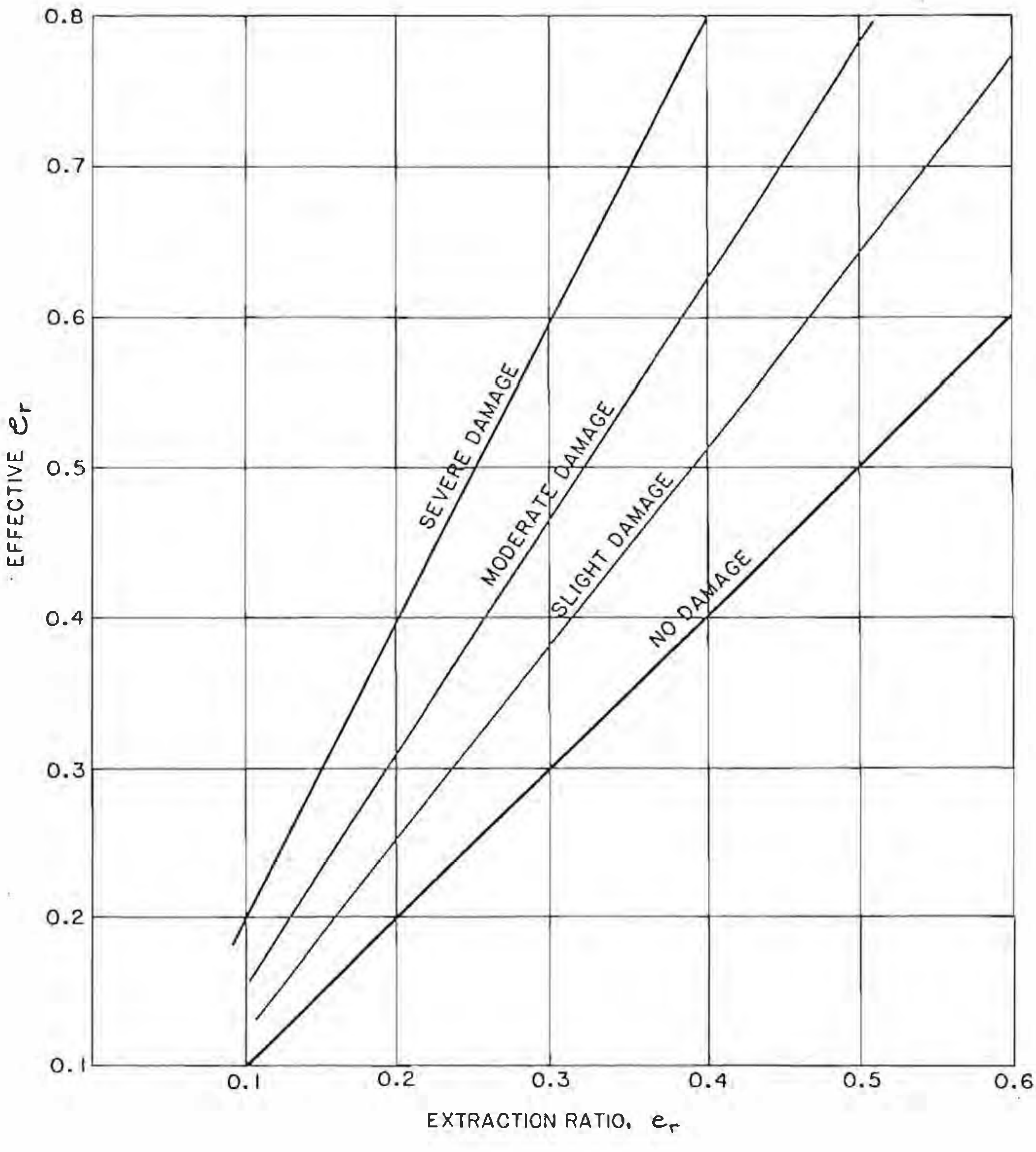


Figure 19 - Effective Extraction Ratios for 7-Ft-Wide Drifts, for Various Degrees of Blasting Damage.

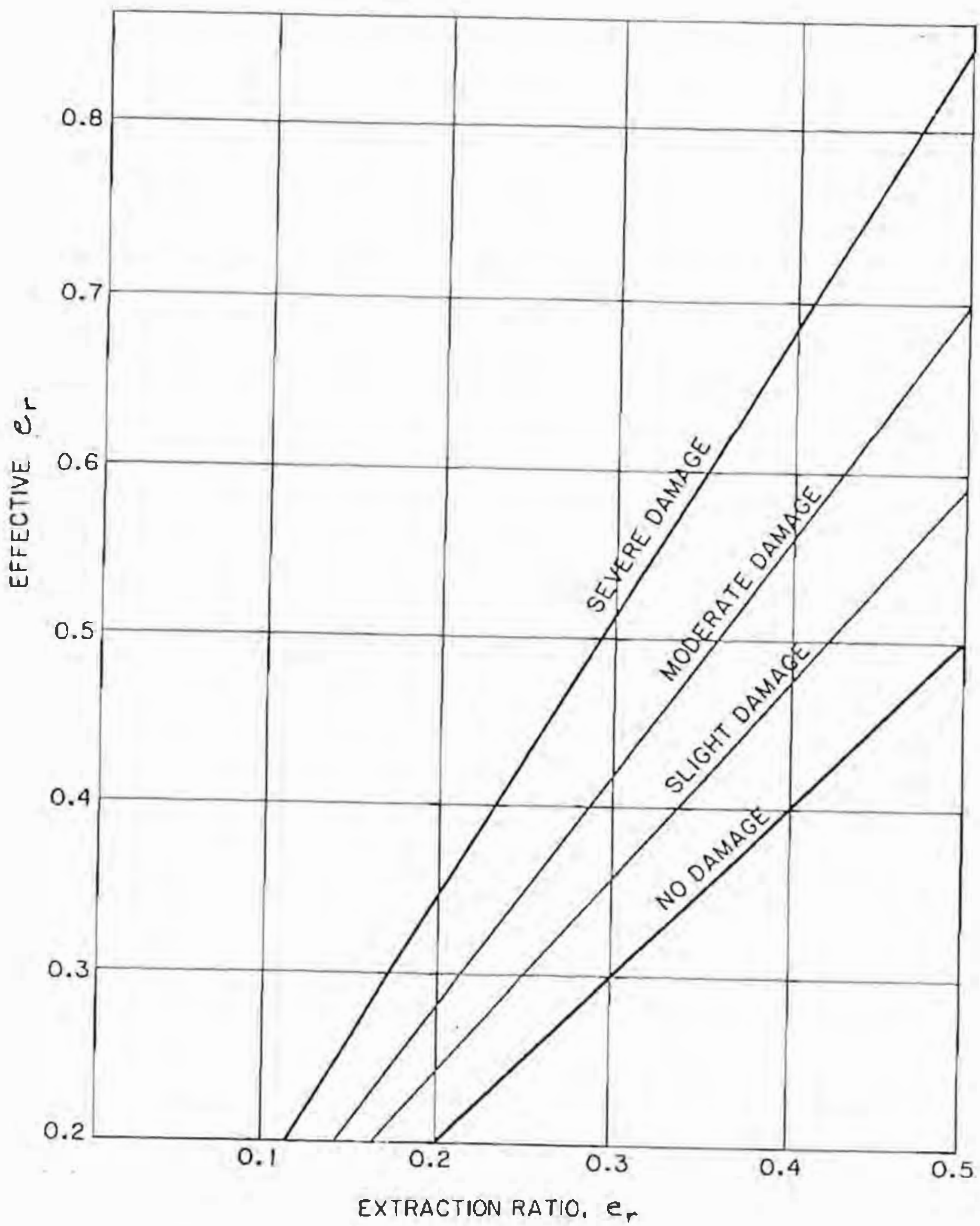


Figure 20 - Effective Extraction Ratios for 10-ft-Wide Drifts for Various Degrees of Blasting Damage.

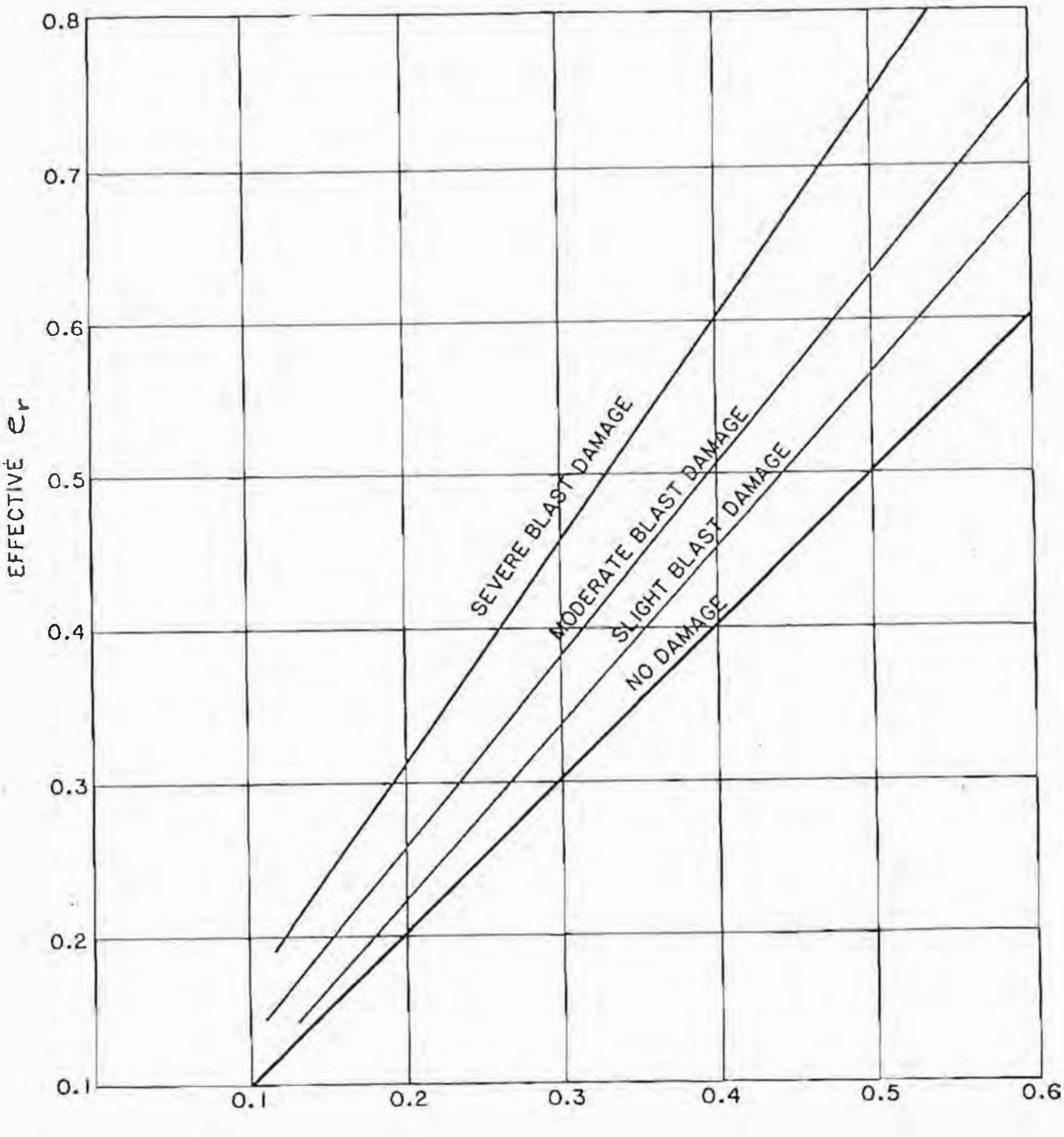


Figure 21 - Effective Extraction Ratios for 15-ft-Wide Drifts for Various Degrees of Blasting Damage.

TABLE 22 - Blasting Damage Adjustment A_B

Conditions/Method	Applicable Term	Adjustment A_B
1. Controlled Blasting a. Practically all hole traces preserved b. No loosened blocks or opened joints c. Overbreak: always less than 1 ft d. Minor to no new inter-joint cracking	Slight Damage	0.94 to 0.97
2. Good conventional blasting a. Some perimeter hole traces preserved. b. Some loosened blocks and slabs, some barring down necessary but not extensive. Some joints opened. c. Overbreak: 1 ft, locally higher. d. Tendency for cracks to develop within rock blocks, between joints, even in harder areas.	Moderate Damage	0.90 to 0.94
3. Poor conventional blasting a. Few to no hole traces preserved. b. Many loosened blocks in crown and ribs. Moderate to extensive barring down required, may impede production. c. Overbreak: almost always greater than 1 ft, locally 3 ft or more.	Severe Damage	0.90 (best) to 0.80 (worst)
4. No experience whatsoever in this rock	Moderate Damage	0.90 (nominal)

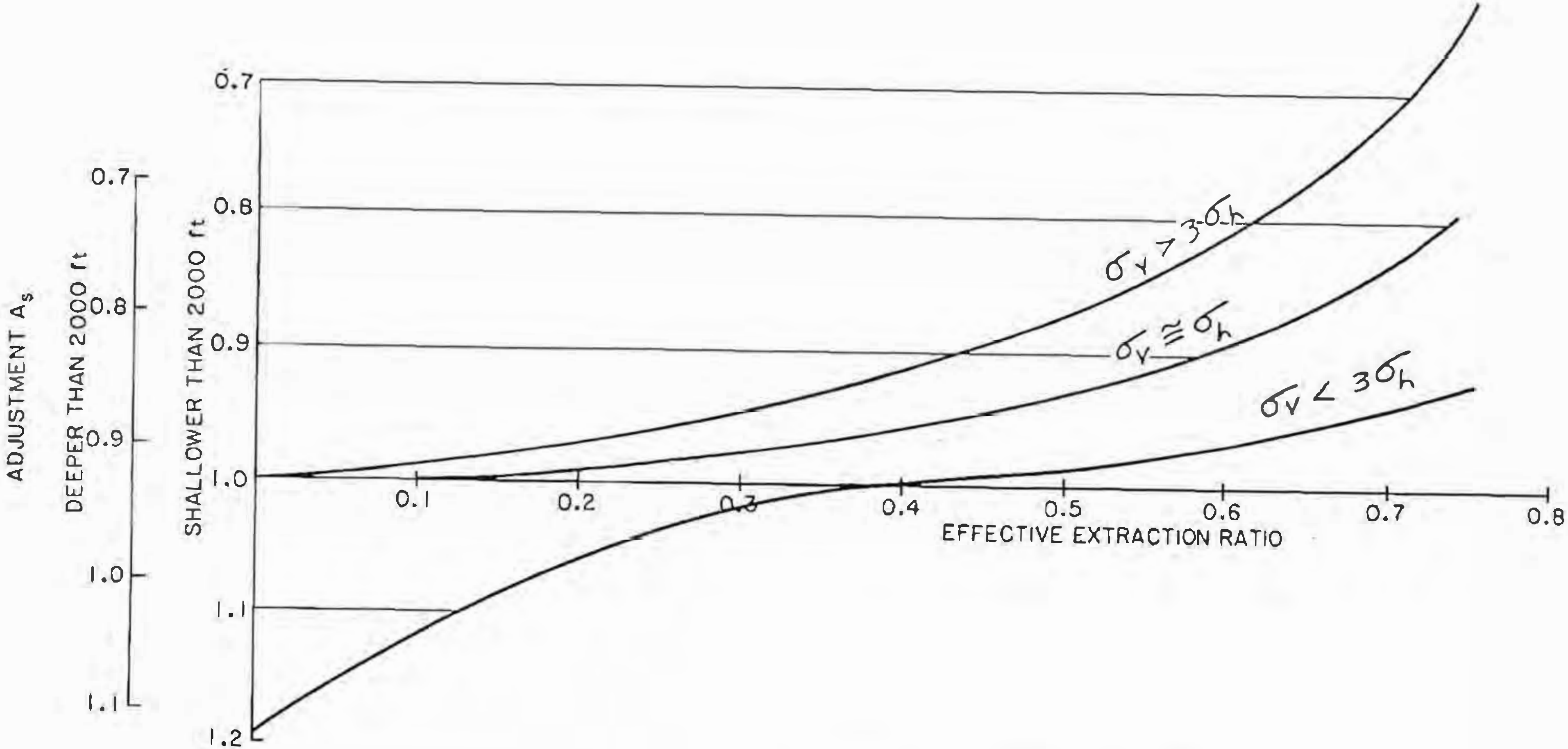


Figure 22 - Adjustment A_S for Induced Stresses Due to Multiple Openings. Curves are based on occurrence of elastic stress concentrations around multiple openings. The use of the effective extraction ratio allows the stress concentration to be regarded as taking place beyond the immediate zone of loosening. The dual scale for depth is projected from field observations.

in the rock mass will determine, for various drift orientations, whether rock blocks will tend to key together, enhancing stability, or separate through shear or tension, creating an incipient failure that must be prevented through increased support. The basic concept was borrowed from the Geomechanics System that fractures parallel to the long axis of an opening are inclined to be pulled apart in the destressed region over the crown and are thus more unfavorably oriented than are perpendicular fractures. (In the MBR System, it is assumed that an exceedingly high horizontal stress field would be required to put such fractures in sufficient compression to override this effect. This is why the fracture orientation adjustment is not somehow tied to the stress field or the Induced Stresses Adjustment A_S). As with the Geomechanics System, it is also assumed that both development and support are facilitated by fractures that dip away from the heading rather than towards it, and that steep dips are preferable to shallow dips.

To put these concepts into practice, it was recognized that there are several types of data that the user may have available. Accordingly, three possible procedures are outlined in the MBR System for arriving at the Fracture Orientation adjustment A_O.

For situations where drift exposures are available, but no detailed, statistically valid determination of fracture tendency has been made, the user can make the ratings adjustment by observing the degree of freedom of rock blocks to move at several localities. Table 23 has been provided for this, based on the one proposed by Laubscher (1975, 1976).

If statistical data on joint orientations are available, such as computer-generated analyses of detailed fracture survey data, then the sets can be ranked and treated individually, considering the orientation of each. A weighted average adjustment, based on the relative strengths (number of points measured) of the joint sets is computed based on guidelines (Table 24) related to joint orientation favorability.

In most instances, the detailed fracture statistics are found from an underground survey. However, at preliminary development stages, there may be no underground exposures in the production area, and drill core must be relied upon. In mining exploration, this core is almost never oriented. In order to arrive at a rating, intact intervals of core are assembled and the number of distinct strikes and dips are noted. The procedure suggested in the Manual recognizes the relatively lower likelihood of intercepting steeper fractures in vertical core. Using judgement, a geologist should be able to compare the orientations of fractures he sees in the core and estimate the tendency of the rock mass to develop fracturing sets with various relative (to each other) orientations. Since the absolute orientations are not known, a steep set is arbitrarily designated as

TABLE 23 – Fracture Orientation Rating A_0 Based on Direct Observation in Drift

Number of Fractures Defining Block	Number of Non-Vertical Faces 1	2	3	4	5	6
3	–	0.95	0.80	–	–	–
4	–	0.95	0.85	0.80	–	–
5	1.0	0.95	0.90	0.85	0.80	–
6	1.0	1.0	0.95	0.90	0.85	0.80

TABLE 24 – Fracture Orientation Rating A_0 Based on Indirect Observation of Fracture Statistics

Strike	Perpendicular				Parallel		Flat dip
Heading Direction	With dip		Against		–		–
Dip Amount, degrees	45–90	20–45	45–90	20–45	45–90	20–45	0–20
Fracture Orientation Rating, A_o, Adjustment	1.0	0.95	0.90	0.85	0.80	0.90	0.85

unfavorable (parallel to, say, the haulage drift axes) which defines the orientations of the other hypothetical sets. Again, judgement is called upon to rank the sets according to intensity, and the rating process proceeds as if with measured fracture statistics.

Having concluded this process, the user has applied adjustments for Blasting Damage (A_B), Induced Stresses (A_S), and Fracture Orientation (A_O). The MBR rating is first multiplied by A_B, then this result is multiplied by A_S; this result is then multiplied by A_O, to yield the Adjusted MBR (AMBR). The user is then ready to again consult the support chart (Figure 18) for support recommendations to support the drifts for the entire pre-production period. Because this is a separate support determination, the temporary, preproduction support selected will be sufficient for stability.

6.3.3 Third Step--Permanent Support

The third step is to apply adjustments that account for those factors that most influence drift behavior during caving. During the field visits, it became clear that the greatest drift damage was associated with the front abutment zone. Major discontinuities in the rock mass can act as bounding influences, concentrating the abutment loads in some places while preventing its application in others. These effects are pronounced at Mines B and D. The total load to be transferred is dependent on the amount of ground opened up, so the undercut area of the block or panel is influential. Also, drifts are at varying distances from the production area and, depending on location, will "see" a varying percentage of the abutment stress. The shortest distance to the cave line is a measure of this effect.

It was assumed that the additional stress increment due to abutment pressure is so much more dependent on undercut area than depth that depth need not be considered separately, for purposes of classification. The role of depth has already been considered (Adjustment A_S). This assumption appeared to be valid at the range of depths in the data base. Further experience at greater depths may require this assumption to be modified.

In developing the adjustment for major discontinuities (S), a variety of scenarios was formulated, using different strikes, dip magnitudes, dip directions, and thicknesses of postulated weakness zones. Weakness zones were regarded as faults filled with gouge that are only sparingly capable or incapable of transferring shear stresses. A reduction of at least one MBR class is required to qualify a weakness zone. A uniform vertical stress was then applied, corresponding to an abutment stress that is approaching the weakness zone. Using well-known principles of load distribution around faults, a likely stress distribution was arrived at along the involved drifts in each scenario. It was assumed that:

1. Faults dipping towards the observer will cause a concentration of compressive strain at the toe of the fault;

2. Faults dipping away from the observer will tend to disperse compressive strains beneath;

3. Moderate dips affect the greatest area and produce the greatest effect;

4. Shallow dips permit the transmittal of a significant fraction of the applied stress, normal to the dip;

5. Near-vertical to steep dips have the least effect;

6. Zones striking parallel to the drift in question produce a greater effect than zones striking perpendicular to it, especially when the attitude is such that the drift tends to be rolled over;

7. The wider the fault, the greater the effect;

8. As the distance to the disturbed zone increases, the effect diminishes rapidly for faults dipping towards the observer and somewhat less rapidly for faults dipping away.

The adjustment framework (Table 25) was based on these assumptions. Other guidelines were formulated to enhance the practicality and simplicity of the adjustment. To recognize that a weakness zone need not be a fault to require classification, but should be a significant feature, the adjustment is termed Major Structures (S).

The adjustment for distance to cave line DC assumes that abutment loadings dissipate more rapidly below the production area than laterally away from it. Also stresses will be felt further away in weaker rock than in stronger rock; thus, MBR was incorporated into the rating structure (Figure 23). The adjustment framework was developed on the basis of prevailing mining practices, field observations, and published data on abutment loads and stress distributions around large openings.

The adjustment for the panel size permits the user to gain a more favorable rating by limiting the amount of ground opened up in the caving process. In smaller blocks, the peak abutment zone may actually occur outside the block boundary, which greatly benefits the drift support system. The rating structure was arrived at after considering the rock mass strain history, as discussed in Section 6.2.3. The Panel Size adjustment PS (Figure 24) used for computing FMBR is a rearrangement of Figure 13 of that Section.

TABLE 25 - Major Structures Adjustment S

Adjustments are >1 if abutment stresses tend to be carried away from the excavation; <1 if stresses are concentrated.

Fault strike vs. heading direction	Most nearly perpendicular						Most nearly parallel*					
Fault dip direction with respect to works	Towards			Away			Towards			Away		
Amount of Dip	Sh	M	St	Sh	M	St	Sh	M	St	Sh	M	St
Distance to nearest fault zone, ft (W = Zone Width)												
<0.1 W	0.8	0.75	0.75	-	0.9	0.95	0.75	0.7	0.7	-	0.9	0.95
0.1 W to 1.0W	0.85	0.8	0.8	0.90	0.9	1.0	0.8	0.75	0.9	0.85	1.0	0.95
1.0 W to 10.0W	0.95	0.85	0.9	0.95	1.05	1.05	0.9	0.9	0.95	0.90	1.05	1.10
10.0W to 50.0W	1.0	0.95	1.0	1.05	1.05	1.0	1.0	1.0	1.0	1.0	1.10	1.0
>50 W	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.05	1.0	1.0
Within Zone	0.85	0.80	0.90	0.85	0.80	0.90	0.80	0.75	0.85	0.80	0.75	0.85

Sh = Shallow (<30°)
M = Moderate (30° to 60°)
St = Steep (>60°)

<0.1 W factor not to be applied for W<10 ft
>50W factor not to be applied for W>10 ft

*For those workings not "screened" from fault effect by caved volume. If "screening" exists, use 1.0.

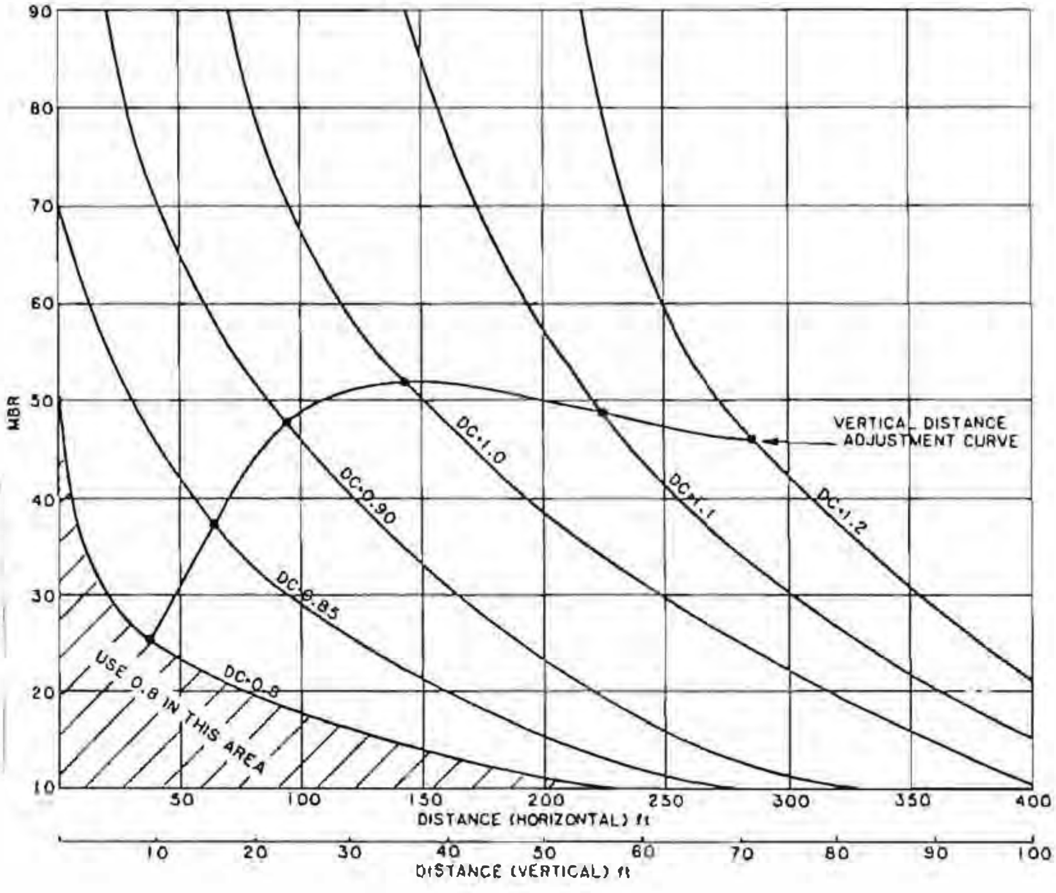


Figure 23 - Adjustment, DC, for Distance to Cave Line. For drifts beneath the caving area, the vertical distance is projected up to the single Vertical Distance Adjustment Curve; the rating is read by interpolating between the multiple curves. For workings horizontally removed from the caving area, the horizontal distance is projected up to the MBR value and the rating is interpolated at that point from the multiple curves. For workings both beneath and to the side, ratings are computed both ways, and the lowest value is taken.

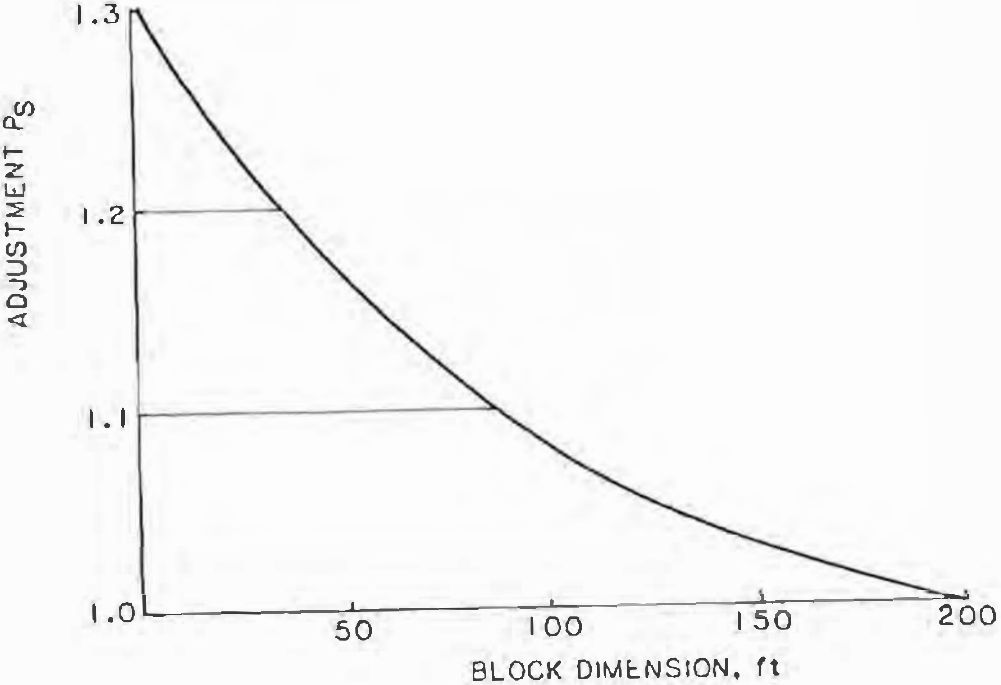


Figure 24 - Block/Panel Size Adjustment P_S.

These production adjustments, Major Structures (S), Distance to Cave Line (DC), and Panel Size (PS), are applied as sequential multipliers to the AMBR. The result is the Final MBR (FMBR), which is related to support recommendations by means of a support chart (Figure 25). The chart is different from that for the MBR and AMBR, to allow a more rigorous support to be specified. The Permanent Support Chart in Figure 25 is set up so that a general increase in support pressure is followed from top towards bottom. The support recommendations are slightly conservative, to allow for natural variation within rated drift intervals.

6.4 Support Recommendations at Studied Field Sites

In this section, the relationship of the data base to the MBR System is illustrated by applying the ratings and adjustments to data from Mines A, B, and C. Calculations for Mine D are carried out in detail in the Manual, Volume II of this Report, as an example of the use of the MBR System, and are not repeated here.

6.4.1 Mine A

At Mine A, we consider four types of drifts: panel, grizzly, fringe, and haulage, and their behavior in three geologic environments: faulted, intrusive porphyry, and host granite.

Essential geological data for the faulted area are given in Figure 26. Data sheets for the Host Granite and Intrusive Porphyry are given as Figures 27 and 28 respectively. In collecting these data, some parameters had to be assessed using judgement. In practice, statistical summaries would be used as data became available.

From Figure 15, the rock strengths rate as follows:

	Faulted	Host Granite	Intrusive Porphyry
Value	8,000 psi	13,500 psi	19,300 psi
Rating	5	8	8

The ratings for Granite and Porphyry are adjusted within the shaded area, to account for suspected bias in sampling.

The Discontinuity Density ratings are derived from Figure 17 as follows:

	Faulted	Host Granite	Intrusive Porphyry
RQD	15%	40%	30%
Rating	5	8	7
Spacing	0.2 ft	0.45 ft	0.4 ft
Rating	5	7	7
Total Rating	10	15	14

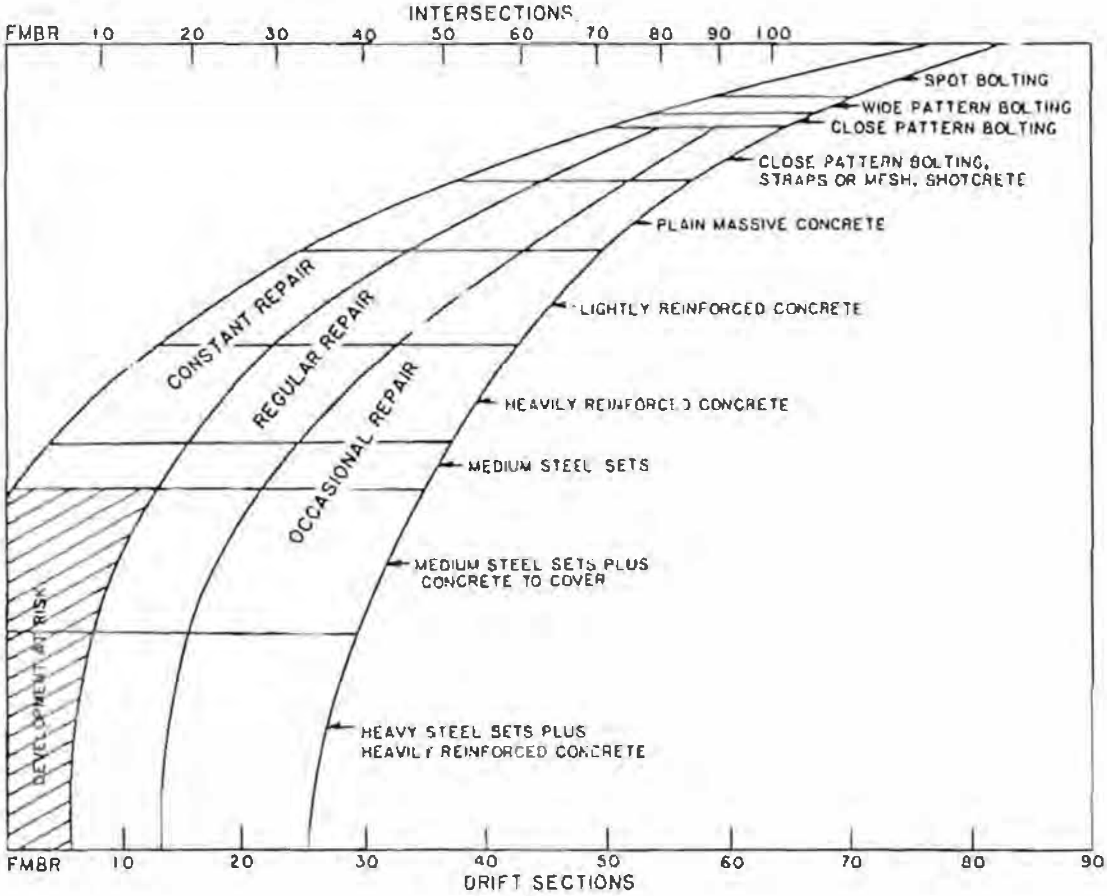


Figure 25 - Permenent Support Chart. Gives recommendations for permanent support of production area drifts. Further explanations of the support types are given in the Manual, Volume II.

Project Name RQI Site of Survey Mine A By RAC Date 2/82

1. Geologic Region: Faulted Area, P.26 Rock Type mixed Location Panel 26

2. Compressive Strength: Average 8,000psi Range 6,000psi Method UCS Comment Fracture influence

3. Core Recovery: Interval typical Average 60% Range 30%-80%

4. RQD: Interval typical Average 15% Range 0%-40%

5. Discontinuity Spacing: Average 0.2 ft Range <0.1ft-1.0ft Comment generalized

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common	S/x	<1/4"	Clay	SO
Intermediate	SR	>1/4"	Crushed rx	SL
Least Common	R	H	py	VS
Consensus	S/x	<1/4"	clay, gouge	SO

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N30W	N30W	E-W	N/S	—
Dip/Dir	80/NE	80/SW	Vert.	15/E	—
Rank	1	2	5	6	—

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: Panel 26 Zone	N30W	30-50	SW	75ft	75ft southwest, parallel to 25/26 Pan. D.
Name: E-W Trend	E-W	Vert	-	30ft	Multiple parallel zones @ 30'-50' spcg.
Name:					

10. Stress Field σ_1: Direction Vert. Magnitude 2,000 psi Measured? yes
σ_3: Direction Horiz., NE/SW Magnitude 1,600psi Measured? yes

11. Source of Geological Data Mine geological maps, fracturing trends from published data, other parameters estimated on basis of conversations with mine personnel and other experience.

Figure 26 - Geological Data, Mine A, Faulted Area.

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Project Name RQI Site of Survey Mine A By R.K. Date 2/82

1. Geologic Region: Host Granite Rock Type Host Granite Location Panels 18-23

2. Compressive Strength: Average 13,500psi Range 13,000psi Method Comp test Comment Orientation-dependent

3. Core Recovery: Interval Typical Average 90% Range 60%-100%

4. RQD: Interval Typical Average 40% Range 10% -50%

5. Discontinuity Spacing: Average 0.45 ft Range <0.2 ft-2.0 ft Comment Average of 3 sets

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common	R	0.03"	Clays	SC
Intermediate	Slr	0.05"	Chlorite	VS
Least Common	—	0.10"	Py	Disaggregated
Consensus	Slr-SR	H-<1/4"	Clay	SO

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N75E	N40W	N40W	N20E	—
Dip/Dir	80/NW	80/NE	70/SW	15/SE	—
Rank	1	2	3	4	—

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: Panel 26 Zone	N30W	30-50	SW	~75'	Up to 700 ft away
Name: —					
Name: —					

10. Stress Field σ_1: Direction Vert. Magnitude 2,000psi Measured? Yes

σ_3: Direction Horizontal/NE-SW Magnitude 1,600psi Measured? Yes

11. Source of Geological Data Published reports, mine geological maps, mine files. Drilling data estimated after underground reconnaissance.

Figure 27 - Geological Data, Mine A, Host Granite.

Project Name RQI Site of Survey Mine A By RAC Date 2/82

1. Geologic Region: Intrusive Porphyry Rock Type Intrusive Porphyry Location Panels 18-23

2. Compressive Strength: Average 19,300psi Range 4,000psi Method Comp. test Comment Specimen selection reportedly difficult

3. Core Recovery: Interval Typical Average "low" 60%? Range –

4. RQD: Interval Typical Average 30% Range 10%-50%

5. Discontinuity Spacing: Average 0.4 ft Range <0.2ft-2.0ft Comment similar for all 3 sets

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	R	H	Clay	50

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N60E	N30W	N10E	–	–
Dip/Dir	80NW	75 SW	15 SE	–	–
Rank	1	3	4	–	–

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: Panel 26 Zone	N30W	30-50	SW	75 ft	Up to 700 ft away
Name: –					
Name: –					

10. Stress Field σ_1: Direction Vert. Magnitude 2,000psi Measured? Yes
σ_3: Direction Horiz, NE/SW Magnitude 1,600 psi Measured? Yes

11. Source of Geological Data Published reports, mine geological maps, mine files. Drilling data and fracturing condition estimated on the basis of underground reconnaissance.

Figure 28 – Geological Data, Mine A, Intrusive Porphyry.

The Discontinuity Conditions are rated as follows, from Table 20:

	Faulted	Host Granite	Intrusive Porphyry
Condition	SK, <½", gouge, SO	SK-SR, H-<½", clay, SO	R, H, clay, SO
Rating	10	15	17

The Groundwater Condition rating is straightforward, from Table 21:

	Faulted	Host Granite	Intrusive Porphyry
Condition	Wet	Dry/Damp	Dry/Damp
Rating	7	13	13

These ratings are summed to yield the MBR, which is compared with RMR in Table 26:

TABLE 26. MBR and RMR Ratings for Mine A

	Faulted	Host Granite	Intrusive Prophyry
MBR	32	51	52
RMR	20	51 & 52	53 & 44

Engineering data sheets for the four types of drifts considered at Mine A are given in Figures 29, 30, 31, and 32, for Grizzly Panel, Grizzly, Grizzly Fringe, and Haulage drifts, respectively.

The adjustments for these data were derived using the procedures outlined in the Manual. Using the figures and tables in the previous section, the reader can verify the adjustments and resulting AMBR and FMBR ratings, given in Table 27.

Consulting Figure 18, we find that steel sets or timber would be required for all workings in the fault zone area, whereas close bolting and shotcrete or medium bolting and shotcrete (depending on reliability desired) would be required to temporarily support the multiple drifts. The single drifts would require only bolts and mesh. This, in general agrees well with mine practice, although it is slightly more conservative since the mine does not have a stated objective of stabilizing its drifts during development.

The Permanent Support Chart (Figure 25) at Mine A shows that, in the faulted area, constant repair would be required for grizzly and panel drifts supported with heavily-reinforced concrete. Recall that

Project Name RQI Site of Survey Mine A By RAC Date 2/1982

1. Type of Drift(s) Panel 2. Orientation(s) NW/SE Ps. 18-23 NNW/SSE P. 26 3. Design Life 3-4 yr

4. Design Dimensions Width 10' Width variation None
Height 10' Height variation None

5. Drift Spacing (Horizontal) 150' C-C

Other Openings Type Grizzly Size 8' w. x 9' h. Spacing 35' C-C

6. Extraction Ratio
Multiple Openings: Excavated Area — Unexcavated — e_r —

Single Opening: 1.5 (width) 15' Excavated 590 ft^2 Unexcavated 810 ft^2 e_r 0.42

7. Distance below undercut - drift floor to undercut floor 15'
drift crown to undercut floor 5'

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces Few are visible. Most gone.
Rib or Crown Looseness Generally tight after scaling. Mesh used routinely.
New or Existing Cracks Some
Overbreak & Barring-Down O.B. 1 ft. Barring-down moderate
Other Criteria Blasting practice generally pretty good.

10. Intersections, turnouts: Type Grizzly & fringe Location throughout ~~100'-300' outside~~ Max. Span 15', 25'

11. Block Dimensions: Side 140' Orientation NE/SW End 100' Orientation NW/SE

12. Cave Line Direction ~N/S Direction of Progress SE → NW

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave)
Between blocks. 100'-300' of length outside caving area. P. 25/26 grizzly panel drift is parallel to fault zone, 70' away, dips 30°-50° towards.

Figure 29 - Engineering Data, Mine A, Panel Drifts.

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Project Name R[redacted]I Site of Survey Mine A By RAC Date 2/'82

1. Type of Drift(s) Grizzly 2. Orientation(s) NE/SW Ps.18-23 ENE/WSW P.26 3. Design Life 1-2 yr.

4. Design Dimensions Width 8' Width variation 9.5' at drawpoints
Height 9' Height variation none

5. Drift Spacing (Horizontal) 35 ft c-c

Other Openings Type Panel/Vent Size 10 ft by 10 ft Spacing 150 ft c-c
Fingers 5 ft dia. 17.5 ft c-c

6. Extraction Ratio
Multiple Openings: Excavated Area 220 ft² Unexcavated 391 ft² e_r 0.36 +

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r

7. Distance below undercut - drift floor to undercut floor 15'
drift crown to undercut floor 6'

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces Few visible. Most gone.
Rib or Crown Looseness Generally tight after barring-down. Mesh used.
New or Existing Cracks Some
Overbreak & Barring-Down O.B. 1-1½ ft. Moderate barring-down
Other Criteria Joints apparently remain tight

10. Intersections, turnouts: Type perpendicular to panel drifts Location at ends Max. Span 15'

11. Block Dimensions: Side 140' Orientation NE/SW End 100' Orientation NW/SE

12. Cave Line Direction ~N/S Direction of Progress SE→NW

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave)
No major structures except in Panel 26 where faulting is perpendicular.

Figure 30 - Engineering Data, Mine A, Grizzly Drifts.

Project Name RQI Site of Survey Mine A By RAC Date 2/82

1. Type of Drift(s) Grizzly Fringe (GFD) 2. Orientation(s) NE/SW Ps. 18-23 ENE/WSW P.26 3. Design Life >15 yr.

4. Design Dimensions Width 12' Width variation None
Height 12' Height variation None

5. Drift Spacing (Horizontal) Single Drift

Other Openings Type Panel Drift Size 10'x10' Spacing 300'

6. Extraction Ratio
Multiple Openings: Excavated Area — Unexcavated — e_r —

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r 0.25 (Turnouts)

7. Distance below undercut - drift floor to undercut floor —
drift crown to undercut floor —

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces None Seen
Rib or Crown Looseness ~~Nott~~ Not significant
New or Existing Cracks Some Joints appear tight.
Overbreak & Barring-Down 1 to 1.5 ft on ribs & crown. Moderate barring-down.
Other Criteria Generally good blasting practice

10. Intersections, turnouts: Type Turnouts @ Panel Drifts Location Outside block Max. Span 25 ft

11. Block Dimensions: Side 140' Orientation NE/SW End 100' Orientation NW/SE

12. Cave Line Direction ~N/S Direction of Progress SE → NW

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Outside panel area. Proximity to cave area @ side of block varies 100 ft to 300 ft.

Figure 31 - Engineering Data, Mine A, Grizzly Level Fringe Drifts.

Project Name RQI Site of Survey Mine A By RBC Date 2/82

1. Type of Drift(s) Haulage 2. Orientation(s) NW/SE 3. Design Life 3-5 yr

4. Design Dimensions Width 11 ft. Width variation None
Height 11 ft. Height variation None

5. Drift Spacing (Horizontal) 70 ft c-c

Other Openings Type None Size ______ Spacing ______

6. Extraction Ratio
Multiple Openings: Excavated Area 11' Unexcavated 70' e_r 0.16

Single Opening: 1.5 (width) ______ Excavated ______ Unexcavated ______ e_r 0.25 (Turnouts)

7. Distance below undercut - drift floor to undercut floor 75'
drift crown to undercut floor 64'

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces None Seen
Rib or Crown Looseness Occasionally found
New or Existing Cracks New joints - none. Existing joints - some are opened.
Overbreak & Barring-Down 1 ft - 2 ft. Worse in poorer rock
Other Criteria Generally good blasting practice

10. Intersections, turnouts: Type Turnouts @ Fringe Location Outside cave Max. Span 35 ft

11. Block Dimensions: Side 140' Orientation NE/SW End 100' Orientation NW/SE

12. Cave Line Direction ~N/S Direction of Progress SE → NW

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Beneath active cave area, extending outwards towards haulage fringe drifts. Parallel to vent and panel drifts overhead.

Figure 32 - Engineering Data, Mine A, Haulage Drifts.

TABLE 27 - Ratings and Adjustments at Mine A

	Panel 26 Faulted						Panel 20											
							QM						MP					
	P	G	F_1	F_2	H_1	H_2	P	G	F_1	F_2	H_1	H_2	P	G	F_1	F_2	H_1	H_2
Intact Rock Strength	8000 psi					5	13,500 psi					8	19,300 psi					8
RQD/Spcg.	15%/0.2'						40%/0.45'						30%/0.4'					
Discontinuity Density	5 + 5					10	8 + 7					15	7 + 7					14
Discontinuity Condition	SK>½",gouge,SO					10	SK-SR,H-<½",clay,SO					15	R,H,clay,SO					17
Groundwater	Wet					7	Dry/Damp					13	Dry/Damp					13
MBR						32						51						52
A_B	.92	.88	.90	.90	.90	.90	.92	.88	.90	.90	.90	.90	.92	.88	.90	.90	.90	.90
eff e_r/A_S	.56/ .91	.6/ .9	1.0	.34/ .97	.2/ 1.0	.35/ .97	.56/ .91	.6/ .9	1.0	.34/ .97	.2/ 1.0	.35/ .97	.56/ .91	.6/ .9	1.0	.34/ .97	.2/ 1.0	.35/ .97
A_O	.82	.92	.92	.92	.82	.82	.86	.88	.88	.88	.86	.86	.86	.88	.88	.83	.86	.86
AMBR	22	23	26	26	24	23	37	36	40	39	39	38	37	36	41	40	40	39
S	.75	.8	.9	.9	.75	.75	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
DC	.8	.8	.90	.90	1.15	1.15	.80	.80	1.15	1.15	1.15	1.15	.80	.80	1.15	1.15	1.15	1.15
PS	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03
FMBR	13	15	22	22	21	20	30	30	47	46	46	45	30	30	49	47	47	46

P=Grizzly Panel Drift/F_1=Grizzly Fringe Drift Section/H_1=Haulage Drift
G=Grizzly Drift /F_2=Grizzly Fringe Intersection /H_2=Haulage Intersection

the plain concrete support used, failed. Fringe and haulage drifts are to be supported with steel sets and concrete, or reinforced concrete if some repair can be tolerated. Presently, the mine uses the more conservative support. In the other rock types, reinforced concrete is sufficient for permanent support of grizzly and panel drifts, with plain concrete sufficient for fringe and haulage drifts. This compares well with mine practice (Sections 5.1.3, 5.1.6, and 5.1.7) while reflecting a somewhat more conservative approach.

6.4.2 Mine B

At Mine B, geological conditions were grouped into "very weak," "competent," and "very competent" categories (see Table 8). Data and ratings for these are shown in Figures 33, 34, and 35, respectively.

The resulting MBR ratings are given in Table 28:

Table 28. MBR and RMR Ratings for Mine B

	Very Weak	Competent	Very Competent
MBR	24	52	81
RMR	21	49	70

Engineering data for Slusher, Fringe, and Haulage drifts are given in Figures 36, 37, and 38, respectively. Slusher lanes were considered for drift sections and the sections over the haulage. Haulage intersections were also considered.

The adjustments and resulting AMBR values and FMBR values are shown on Table 29. In this general approach at Mine B, the classification numbers are generalized somewhat. The adjustment S is not as specific as it might be if actual locations were to be considered, and PS assumes a uniformly advancing cave front. Nonetheless, Table 29 shows the sensitivity of the MBR System to workings having different purposes. In particular, the benefit of having the fringe drifts 300 ft from the mining area, is readily seen. It is apparent that the slusher lanes and fringe drifts comprise the end points of a spectrum of ratings.

In the weak rock, the recommended temporary support (from Figure 18) for production openings is light to medium steel sets or heavy timber, with the heavier support in the slusher lanes over the haulage. Temporary support in the intermediate ("competent") rock varies from medium pattern bolting and shotcrete in the slusher lanes to close pattern bolting with shotcrete and mesh in fringe drifts. The more conservative support is preferred in the fringe drifts. In the competent rock, slusher lanes over the haulage receive only medium pattern bolting with mesh, and fringe drifts receive wide pattern bolting or spot bolts only, as temporary support.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Geologic Region: Throughout level Rock Type Very weak rock Location Typical

2. Compressive Strength: Average 1,000psi Range — Method Compr. test Comment Estimated

3. Core Recovery: Interval Typical Average 50% Range always < 80%

4. RQD: Interval Typical Average 10% Range 0-20%

5. Discontinuity Spacing: Average < 2 in. Range < 0.05 ft - 3' Comment Estimated/General

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	SM-SK	1/4"	clay	SO

7. Water Condition Dry **Damp** Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N45W	N30E	N/S	N25E	–
Dip/Dir	65/NE	70/SE	60/E	75/NW	–
Rank	1	2	3	4	–

9. Major Structures

Name	Strike	Dip	Dip Dir.	Width	Location/Comment
South fault	NW	Mod.	NE	10'-40'	{General assumption, 2 ft fault @40',
NE faults	NE	Mod.-Steep	SE	10'	{steeply-dipping, unfavorably oriented.
Weak zones	Variable distribution				

10. Stress Field σ_1: Direction Vertical Magnitude 900psi Measured? No
σ_3: Direction Horizontal Magnitude 300psi Measured? No

11. Source of Geological Data Published and unpublished reports. Several parameters estimated on the basis of visual observations underground and published narrative descriptions

Figure 33 - Geological Data, Mine B, Very Weak Rock.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Geologic Region: Throughout level Rock Type Competent Rock Location Typical
2. Compressive Strength: Average 16,000psi Range – Method Comp. test Comment Estimated
3. Core Recovery: Interval Typical Average ~100% Range 80%-100%
4. RQD: Interval Typical Average 60% Range 40%-75%
5. Discontinuity Spacing: Average 5 m. Range 3m. to 8m. Comment Generalized
6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	SM/SK	< 1/4"	minor clay	SL-SO

7. Water Condition Dry Damp Wet Dripping Flowing
8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N45W	N30E	N/S	N25E	–
Dip/Dir	65/NE	70/SE	60/E	75/NW	–
Rank	1	2	3	4	–

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: South fault	NW	Mod.	NE	10'-40'	General assumption, 2 ft fault
Name: NE faults	NE	Mod.-Steep	SE	10'	@ 40 ft, steeply-dipping, unfavor-
Name: Weak zones	Variable distribution				ably oriented

10. Stress Field σ_1: Direction Vertical Magnitude 700psi Measured? No
σ_3: Direction Horizontal Magnitude 300psi Measured? No

11. Source of Geological Data Published and unpublished reports. Several parameters estimated on the basis of visual observations underground and published narrative descriptions.

Figure 34 - Geological Data, Mine B, Competent Rock.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Geologic Region: Throughout level Rock Type Very Competent Location Typical

2. Compressive Strength: Average 27,000psi Range 25-30,000 Method compr. test Comment Estimated

3. Core Recovery: Interval Typical Average 100% Range 100%

4. RQD: Interval Typical Average 80% Range 75%-100%

5. Discontinuity Spacing: Average 1ft Range 0.5ft-2ft Comment Generalized

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	SR	H	none	SL

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	N45W	N30E	N/S	N25E	-
Dip/Dir	65/NE	70/SE	60/E	75/NW	-
Rank	1	2	3	4	-

9. Major Structures

Name	Strike	Dip	Dip Dir.	Width	Location/Comment
South fault	NW	Mod.	NE	10'-40'	General assumption, 2 ft fault @40'
NE faults	NE	Mod.-steep	SE	10'	steeply dipping, unfavorably
Weak zones	Variable Distribution				oriented.

10. Stress Field σ_1: Direction Vertical Magnitude 700psi Measured? No
σ_3: Direction Horizontal Magnitude 300psi Measured? No

11. Source of Geological Data Published and unpublished reports. Several parameters estimated on the basis of visual observations underground, and published narrative reports.

Figure 35 - Geological Data, Mine B, Very Competent Rock.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Type of Drift(s) Slusher 2. Orientation(s) E/W 3. Design Life 6 mo. to 1½ yr.

4. Design Dimensions Width 9' Width variation Varies at LCO and other locations.
Height 12.5' Height variation Varies at places

5. Drift Spacing (Horizontal) 34 ft C-C along haulage, 66 ft C-C between haulage and vent drifts

Other Openings Type Finger raises Size 9'x11' @45° Spacing ~30 ft C-C average

6. Extraction Ratio
Multiple Openings: Excavated Area 1304 ft² @ Haul. 870 ft² @ drift sect. Unexcavated 1,960 ft² 3,210 ft² e_r 0.40 over haulage 0.21 along drift

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r —

7. Distance below undercut - drift floor to undercut floor 32.5'
drift crown to undercut floor 20'

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces
Rib or Crown Looseness
New or Existing Cracks
Overbreak & Barring-Down
Other Criteria Generally good to fair. Poor in poorer rock.

10. Intersections, turnouts: Type LCO Location end drift Max. Span 10 ft

11. Block Dimensions: Side 100 ft Orientation N/S End 97 ft Orientation E/W

12. Cave Line Direction N/S Direction of Progress Irregular, across level, mass caving

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) General location in level, major structures evenly distributed.

Figure 36 - Engineering Data, Mine B, Slusher Drifts.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Type of Drift(s) Fringe 2. Orientation(s) Looping 3. Design Life 5-8 yr

4. Design Dimensions Width 14' Width variation None
Height 12' Height variation None

5. Drift Spacing (Horizontal) Single drift

Other Openings Type Haulage Size 14'x12' Spacing 200 ft C-C

6. Extraction Ratio
Multiple Openings: Excavated Area — Unexcavated — e_r —

Single Opening: 1.5 (width) 21 ft Excavated 555 ft² Unexcavated 2,090 ft² e_r 0.21 (haulage turnouts)

7. Distance below undercut - drift floor to undercut floor —
drift crown to undercut floor —

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces
Rib or Crown Looseness
New or Existing Cracks
Overbreak & Barring-Down
Other Criteria Generally good. Worse in poorer rock.

10. Intersections, turnouts: Type Location Max. Span

11. Block Dimensions: Side Orientation End Orientation

12. Cave Line Direction Direction of Progress

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Peripheral to production area, 300 ft away, laterally

Figure 37 - Engineering Data, Mine B, Fringe Drifts.

Project Name RQI Site of Survey Mine B By RAC Date 2/82

1. Type of Drift(s) Haulage 2. Orientation(s) N/S 3. Design Life ~5 yr.

4. Design Dimensions Width 14 ft Width variation none
Height 12 ft. Height variation None

5. Drift Spacing (Horizontal) 200 ft C-C

Other Openings Type Vent Size 9 ft by 7 ft Spacing between and parallel to haulages @ 200 ft C-C

6. Extraction Ratio
Multiple Openings: Excavated Area — Unexcavated — e_r c.15

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r —

7. Distance below undercut - drift floor to undercut floor 44 ft
drift crown to undercut floor 32 ft

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces
Rib or Crown Looseness
New or Existing Cracks
Overbreak & Barring-Down
Other Criteria Generally Good. Damage may be severe in poor rock.

10. Intersections, turnouts: Type Double curves Location Haulage Intersects. Max. Span 24 ft

11. Block Dimensions: Side 160 ft Orientation N/S End 97 ft Orientation E/W

12. Cave Line Direction N/S Direction of Progress Irregular, across level, mass caving.

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Perpendicular to slusher drifts and below them. Extend across level.

Figure 38 - Engineering Data, Mine B, Haulage Drifts.

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TABLE 29 - Ratings and Adjustments at Mine B

	Very Weak Rock					Competent Rock					Very Competent Rock				
	HD	HI	SH	SD	F	HD	HI	SH	SD	F	HD	HI	SH	SD	F
Intact Rock Strength	1,000 psi				2	16,000 psi				10	27,000 psi				14
RQD/Spcg.	10%/<2 in.					60%/5in.					80%/1ft				
Discontinuity Density	3 + 3				6	12 + 7				19	16 + 11				27
Discontinuity Condition	SM-SK/½in./clay/SO				7	SM-SK/<½in./m.clay/SL				12	SR/H/None/SL				25
Groundwater	Damp				9	Damp				11	Dry				15
MBR					24					52					81
A_B	0.88	0.88	0.88	0.88	0.88	0.92	0.92	0.92	0.92	0.92	0.90	0.90	0.90	0.90	0.90
eff σ_r/A_S	0.19/ .97	0.48/ .88	0.68/ .75	0.37/ .92	0.33/ .94	0.14/ .98	0.39/ .92	0.57/ .83	0.30/ .95	0.27/ .96	0.16/ .98	0.41/ .91	0.60/ .82	0.32 .94	0.29 .95
A_O	0.84	0.84	0.91	0.91	0.85	.84	0.84	.91	.91	.90	.84	0.84	.91	.91	.90
AMBR	17	16	14	18	17	39	37	36	41	41	60	56	54	62	64
S	.98	0.98	.95	.95	.98	.98	0.98	.95	.95	.98	.98	0.98	.95	.95	.98
DC	.98	0.98	.88	.88	1.2	.98	0.98	.89	.88	1.2	.98	0.98	.88	.88	1.2
PS	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
FMBR	16	15	12	15	20	37	36	30	34	48	58	54	45	52	75

HD=Haulage, Drift Section
HI=Haulage, Intersection (turnout)
F=Fringe Drift

SH= Slusher lane, over haulage
SD=Slusher lane, drift section

Final, production support recommendations (Figure 25) show an even wider range of support requirements. Slusher lanes in poor rock would receive medium steel sets and concrete over the haulage, and steel sets or heavily reinforced concrete along the drift sections. (The areas of the slusher lanes over the haualage are common problem areas at Mine B.) Haulage intersections in very weak rock would also receive steel sets and concrete, possibly reinforced, primarily because of the use of the opening and the span. Haulage drift sections could be supported with medium steel sets (possibly concrete near drop-points), and similar support would be used in fringe drifts.

Permanent support in intermediate ("competent") rock would most likely be plain massive concrete in slusher lanes over the haulage with occasional light reinforcement, and plain concrete along the slusher drift sections. Reinforced concrete support is recommended at haulage turnouts. Haulage drift sections would receive plain concrete support. Fringe drifts would be bolted on a close pattern, with shotcrete and mesh.

In "very competent" rock, slusher lanes over the haulage would be supported with close pattern bolts, mesh, and shotcrete, and drift sections similarly with somewhat better performance. For high reliability, the drifts could be concreted. Haulage turnouts would still be concreted, but pattern bolting with mesh would suffice in drift sections. Fringe drifts would be spot bolted only.

These recommendations do reflect mining practice (compared with Table 15) but are slightly conservative for some applications. Again, this is to avoid problems at localized zones of lower competence within the rated region. The support for "very weak" rock is quite heavy, but would not be overly conservative if the region receiving these low ratings were very extensive. It should be pointed out that it is unlikely that fringe drifts, at 300 ft from the production area, would occupy the same region of sub-20 (MBR) ground that also was extensive in the production area.

6.4.3 Mine C

Although there was never any production at Mine C, the MBR ratings are carried out for completeness.

MBR ratings are carried out for three geologic categories--AQM, HW, and QSF (see Section 5.3.5) and the basic input data are given in Figures 39, 40, and 41 for AQM, HW, and QSF areas, respectively.

The adjustments and support recommendations are carried out for each slusher drift, the grizzly drifts and the haulage drift. The trackless ramp to the grizzly level and the slusher level access/vent drift are similar and not rated separately.

Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Geologic Region: AQM Rock Type AQM Location Block 1.

2. Compressive Strength: Average 1,400psi Range 1200-1600 Method PLT Comment small prisms

3. Core Recovery: Interval Typical Average 80% Range 25%-100%

4. RQD: Interval Typical Average 59% Range 7%-96%

5. Discontinuity Spacing: Average 0.6 ft Range 0.15ft-1.5ft Comment general assessment

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	R	0.1in.	minor clay	SL

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	ENE	NNW	NE	WNW	–
Dip/Dir	steep/SE	steep/SW	steep/NW	steep SW	–
Rank	1	2	3	5	–

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: Middle Fault	~E/W	steep	S	20 ft	300 ft to NE of blocks
Name: Backfill	~E/W	~vert.	–	10-30ft	within block, Block 2
Name:					

10. Stress Field σ_1: Direction vertical Magnitude 2,200psi Measured? No
σ_3: Direction horizontal Magnitude 700psi Measured? No.

11. Source of Geological Data Fracture orientations from mine maps and spot checks. Nearest drillhole data 800ft away. Fracture conditions by direct reconnaissance.

Figure 39 - Geological Data, Mine C, AQM.

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Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Geologic Region: HW Rock Type strongly argillized porphyry Location Block2

2. Compressive Strength: Average 700psi Range — Method — Comment Estimated

3. Core Recovery: Interval Typical Average 50% Range —

4. RQD: Interval Typical Average 15% Range —

5. Discontinuity Spacing: Average 0.2 ft Range <0.2ft-0.4ft Comment

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	SM	0.1 in.	clay	VS

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	ENE	NNW	NE	WNW	—
Dip/Dir	steep/SE	steep/SW	steep/NW	steep/SW	—
Rank	1	2	3	5	—

9. Major Structures

Name	Strike	Dip	Dip Dir.	Width	Location/Comment
Middle Fault	~E/W	steep	S	20ft	300 ft NE of blocks
Backfill	~E/W	~vert	—	10-30ft	center, block2

10. Stress Field σ_1: Direction vertical Magnitude 2,200psi Measured? No
σ_3: Direction horizontal Magnitude 700psi Measured? No

11. Source of Geological Data Fracture orientations from mine maps and spot checks. Nearest drillhole data 800ft away. Fracture conditions by direct reconnaissance.

Figure 40 - Geological Data, Mine C, HW.

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Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Geologic Region: QSF Rock Type silicified, sericitized, ppy. Location Block 2

2. Compressive Strength: Average 16,500 psi Range 22,000-11,000 Method PLT Comment small prisms

3. Core Recovery: Interval Typical Average 75% Range 60% - [illegible]0%

4. RQD: Interval Typical Average 61% Range 40% - 82%

5. Discontinuity Spacing: Average 0.4 ft Range 0.2 ft - 1.3 ft Comment general assessment

6. Discontinuity Condition

	Wall Roughness	Wall Separation	Joint Filling	Wall Weathering
Most Common				
Intermediate				
Least Common				
Consensus	VR	0.1-0.2 in	qtz, ser	F-SL

7. Water Condition Dry Damp Wet Dripping Flowing

8. Fracture Orientations

	Set 1	Set 2	Set 3	Set 4	Set 5
Strike	ENE	NNW	NE	WNW	–
Dip/Dir	steep/SE	steep/SW	steep/NW	steep/SW	–
Rank	1	2	3	5	–

9. Major Structures

	Strike	Dip	Dip Dir.	Width	Location/Comment
Name: Middle Fault	~E/W	steep	S	20 ft	300 ft to NE of blocks
Name: Backfill	~EW	~vert.	–	10-30 ft	Center, block 2
Name:					

10. Stress Field σ_1: Direction vertical Magnitude 2,200 psi Measured? No
σ_3: Direction horizontal Magnitude 700 psi Measured? No

11. Source of Geological Data Fracture orientations from mine maps and spot checks. Nearest drillhole data 800 ft away in similar rock. Fracture conditions by direct reconnaissance.

Figure 41 - Geological Data, Mine C, QSF.

The slusher drift (S4) serving the northwest block is wholly within AQM, and the slusher drift (S5) to the southeast is mostly within backfill, with regions of other weak rock types at places. Thus, hypothetical S5s are rated as if they were wholly within QSF or HW domains, although neither of these is actually extensive enough to contain a whole slusher drift. Engineering data for the actual and hypothetical slusher drifts are given in Figure 42.

The grizzly drifts were never completed and their specific geology is therefore not known, so these are classified together. Had more specific geologic data been known for each grizzly drift, individual support recommendations could likewise be computed. Engineering data are given in Figure 43.

The haulage drift crosses all domains and is rated for each. Figure 44 gives the engineering data. Since there are no intersecting or multiple openings, e_r does not apply. This is not an isolated drift, however, so the AMBR and IMBR are computed, using an A_S of 1.0.

The nearest true major structure is a fault, crossed some 300 ft to the northwest of the block area in the haulage drift. This fault has little effect on the production area. However, the backfilled zone is expected to have an effect. Thus it is arbitrarily designated a major structure with a 20-ft width, E-W strike, and near-vertical dip to the south. For the grizzly drifts, S is assigned a nominal value of 0.90. The ratings for S in the haulage depend on the location of the domain under which the classification is being carried out.

Mine C illustrates the use of an upwards adjustment for the panel size. Since the cave direction is not known, the overall production area dimension of 150 ft is used for the rating.

Table 30 compares the MBR and RMR ratings for Mine C. As with the other mines, the MBR and RMR values agree quite well.

Table 30. MBR and RMR Ratings for Mine C

	AQM	HW	QSF
MBR	60	28	71
RMR (typ.)	71	29	71

Table 31 gives the ratings and adjustments generated by the MBR system for the workings at Mine C. The advantage of having the slusher lanes separated from the cave by an additional level, (due to the use of transfer raises), can be clearly seen. Production adjustments for slusher lanes are quite small. There is an added benefit since the slusher lanes can be spaced further apart, lowering the

Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Type of Drift(s) Slushers 2. Orientation(s) N56E 3. Design Life 1 yr

4. Design Dimensions Width 11 ft Width variation 12 ft @ transfer raises
Height 13 ft Height variation none

5. Drift Spacing (Horizontal) 70 ft c-c

Other Openings Type Transfer Size 5ft×5ft Spacing 17.5ft c-c

6. Extraction Ratio
Multiple Openings: Excavated Area 2575 ft^2 Unexcavated 9675 ft^2 e_r 0.21

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r —

7. Distance below undercut - drift floor to undercut floor 48.5 ft
drift crown to undercut floor 35.5 ft

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces none seen
Rib or Crown Looseness few loose slabs in crown, ribs tight.
New or Existing Cracks some
Overbreak & Barring-Down 1 ft on ribs, up to 3-4 ft, especially in AQM, due to shallow fx.
Other Criteria Blocky in AQM domain, better blasting in HW, v. bad O.B. in backfill

10. Intersections, turnouts: Type perp. intersects w/ ven Location NE limit Max. Span 17 ft.

11. Block Dimensions: Side 150 ft Orientation N56E End 70 ft Orientation N34W

12. Cave Line Direction — Direction of Progress —

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Down center of block (165 ft long). Slusher in AQM is 70 ft NE of backfill zone and ~300 ft SW of Middle Fault zone. Slusher in backfill assumed for classifications as in HW

Figure 42 - Engineering Data, Mine C, Slusher Drifts.

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Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Type of Drift(s) Grizzly 2. Orientation(s) N56E 3. Design Life 1 yr.

4. Design Dimensions Width 9 ft Width variation 10 ft at fingers
Height 10 ft 9 in. Height variation None

5. Drift Spacing (Horizontal) 35 ft C-C

Other Openings Type Finger raises Size 5 ft by 5 ft Spacing 17.5 ft C-C

6. Extraction Ratio
Multiple Openings: Excavated Area 212.5 ft^2 Unexcavated 400 ft^2 e_r 0.35

Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r —

7. Distance below undercut - drift floor to undercut floor 19.5 ft
drift crown to undercut floor ~8.5 ft

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces
Rib or Crown Looseness
New or Existing Cracks
Overbreak & Barring-Down
Other Criteria Never driven. Assume fair to poor practice as for other drifts

10. Intersections, turnouts: Type perp. @ grizzly access Location SW end Max. Span 17 ft

11. Block Dimensions: Side 150 ft Orientation N56E End 70 ft Orientation N34W

12. Cave Line Direction — Direction of Progress —

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Two grizzly drifts per block, emptying into transfer raises to slushers.

Figure 43 - Engineering Data, Mine C, Grizzly Drifts.

Project Name RQI Site of Survey Mine C By RAC Date 2/82

1. Type of Drift(s) Haulage 2. Orientation(s) N34W 3. Design Life 3 yr.

4. Design Dimensions Width 13 ft Width variation None
   Height 13 ft Height variation None

5. Drift Spacing (Horizontal) N/A

   Other Openings Type None Size — Spacing —

6. Extraction Ratio
   Multiple Openings: Excavated Area — Unexcavated — e_r —

   Single Opening: 1.5 (width) — Excavated — Unexcavated — e_r —

7. Distance below undercut - drift floor to undercut floor 61½ ft
   drift crown to undercut floor

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
   Perimeter Hole Traces Very fewseen
   Rib or Crown Looseness Ribs drummy in HW but tight in QSF and HW.
   New or Existing Cracks Existing cracks unaffected. A few new fx.
   Overbreak & Barring-Down Extensive in AQM, moderate elsewhere.
   Other Criteria Fair to poor blasting practice.

10. Intersections, turnouts: Type Turnout Location 300ft to NE Max. Span 26 ft

11. Block Dimensions: Side 150 ft Orientation N56E End 70 ft Orientation N34W

12. Cave Line Direction — Direction of Progress —

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave) Beneath SW margin of block. Passes through HW, AQM, and QSF geological regions. No intersecting openings near block boundaries or within blocks

Figure 44 - Engineering Data, Mine C, Haulage.

TABLE 31. Ratings and Adjustments at Mine C

	AQM			HW			QSP		
	S4	C	H	SS	C	H	SS	C	H
Intact Rock Strength	1,400 psi		2	700 psi		1	16,500 psi		9
RQD/Spcg	59%/0.6ft			15%/0.2ft			61%/0.4ft		
Discontinuity Density	13 + 8		21	5 + 6		11	13 + 8		21
Discontinuity Condition	R/0.1 in./minor clay/SL		22	SM/0.1 in./clay/VS		10	VR 0.2 in./qtz/ser/F-SL		26
Groundwater	Dry		15	Damp & Dripping		6	Dry		15
MRR			60			28			71
A_B	0.82	0.82	0.84	0.90	0.90	0.88	0.92	0.92	0.92
Eff r_r/A_S	0.35/ .85	0.6/ .75	– / 1.0	0.30 .87	0.48 .80	– / 1.0	0.29/ .88	0.45/ .81	– / 1.0
A_O	0.86	0.83	0.92	0.86	0.83	0.92	0.86	0.83	0.92
AMBR	36	31	46	19	17	23	49	44	60
S	0.95	0.90	0.90	0.85	0.90	1.05	0.85	0.90	1.0
DC	1.0	0.8	1.12	1.0	0.8	1.12	1.0	0.8	1.12
PS	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03	1.03
FMBR	35	23	48	17	12	27	43	33	69

S4-Slusher Drift in AQM
S5-Slusher Drift, partly in backfill, classified for QSF, HW

H-Haulage drift
C-Grizzly drifts (all)

extraction ratio and keeping the AMBR value relatively high. Also clear is that the haulage crosscut benefits from the small panel size, the great vertical distance to the cave, and, in places, the shielding effect of the backfill, which minimizes the loads due to caving on those portions of the haulage that are in HW.

Temporary support recommendations are as follows:

Slusher drift 4 in AQM - medium pattern bolts with shotcrete
Slusher drift 5 in HW - light steel or medium timber
Slusher drift 5 in QSF - close pattern bolts with mesh or straps
Grizzly drifts in AQM - close pattern bolting, mesh-reinforced shotcrete
Grizzly drifts in HW - light to medium steel or medium to heavy timber
Grizzly drifts in QSF - medium pattern bolting with shotcrete.
Haulage in AQM - medium pattern bolting with shotcrete
Haulage in HW - light steel or medium timber
Haulage in QSF - medium pattern bolting, straps.

These are about one category more conservative than the temporary support in use at the time of examination. There are three possible explanations. First, as a trial caving project, the natural tendency is to be as frugal as possible. Second, the adjustments for A_S are quite severe, due to the observed adverse effect of blasting and the use of the rating scale for depths in excess of 2,000 ft. Third, it may be that local experience in this ground is that the rock withstands blasting better than most, so that the adjustments A_B are overly severe. It is also likely that the compressive strength for AQM is low, and influenced by the small size of the specimens that had to be used. Size corrections to this extent are not proven for point load tests.

Permanent support in AQM ranges from lightly reinforced concrete in grizzly drifts to close pattern bolting with straps for the haulage. In HW, medium steel sets plus concrete are recommended in the grizzly drifts, heavily reinforced concrete in the haulage. In QSF, the recommended permanent support is plain concrete in grizzly drifts, close pattern bolts and shotcrete in slushers, and wide pattern bolting in the haulage. (Obviously, this last recommendation is a paradox, since the process of caving cannot have a beneficial effect on the haulage in QSF. Clearly, the prescribed temporary support should be adequate.)

This contrasts with the planned permanent support: plain, massive concrete two feet thick in the slushers and grizzly drifts and one foot thick in the haulage. It was also planned to extend the steel sets in the haulage. The most likely explanation for this discrepancy is that the conservatism already reflected in the temporary support recommendations was carried through to the permanent support.

6.5 Evaluation of the Adjustment Process

6.5.1 Scope

The adjustment process addresses the major factors influencing mine drift behavior prior to routine production. Other factors have some bearing as well, but are of secondary significance for caving mines in general. Examples include the position of the drift within the block and interaction of the finger or transfer raise cutouts. Also, the support charts are based on prevailing industry support practices and do not directly address novel support techniques, such as ground consolidation by grouting, use of Split Set rockbolts, Bernold lining, or yieldable arches (although the last are commonly used as a repair technique). Still the scope is broad enough that a competent engineer should find the process useful.

6.5.2 Accuracy

Clearly, some parameters are more easily estimated than others. For example, the intact rock strength can be fairly accurately assessed, but the discontinuity condition remains a somewhat rough estimate of joint shear resistance, and stress field assessments can very widely. Many of the adjustments made in this report are carried to two decimal places, which is sufficient in view of the probable accuracy of the input parameters.

The system yields a range of recommended supports. It is apparent, when considering the examples in this report, that the recommendations tend to be more conservative than those used at operating properties. The user should recognize that miners' support philosophies are affected by numerous factors. The supports in use at these properties have been found through time to be the lightest that will provide support with an acceptably low overall incidence of failures. The MBR, on the other hand, yields support recommendations for planning purposes, which are better if estimated conservatively than if underestimated. Also, operating mines often experience a high incidence of failures at some places, while at other places the supports are overdesigned. Indeed, this report contains some examples where heavier support is in use at operating mines than is recommended by the MBR. This is partly because mine workers are most efficient when using a single support procedure, and partly because engineering staffs don't presently have a readily-applied, reliable means to redesign support as new ground conditions are encountered--a situation the MBR is intended to remedy. Finally, although support failures in mining are an inherent cost of doing business, there is still a reluctance to accept them, because they seriously threaten production and safety.

The MBR allows as specific a support design as the user wishes. It can be very responsive to localized changes in geology, and the response of the support system to these changes is limited primarily by operational factors. Although it is an improvement, the MBR still is not completely objective (the ratings are not defined uniquely by the input information).

The conformance of the ratings to the rock conditions will depend on how representative the data is, and this can be difficult in operating situations. The user should be aware of the following problems and should anticipate their occurrence when applying the MBR:

1. Rock strength data are likely to be widely variable and not well correlated to geologic circumstances, unless special efforts are made.

2. Fracture patterns in caving orebodies tend to be both complex and spatially variable. There is no substitute for detailed fracture surveys from the area of interest, but these are commonly not available.

3. The structural geology and alteration zoning are commonly not well enough understood to permit projection of data from a well-studied area into an unstudied area where ratings are desired.

4. MBR system ratings ordinarily cannot be correlated with mineralization, alteration zoning, or ore grade, and these should not be used as criteria for projecting ratings.

5. Many reconnaissance-level or step-out drilling programs do not collect the necessary geotechnical data, as these programs are directed towards finding ore reserves rapidly and at minimum cost. The core is ordinarily logged in a core shack and not at the drill rig, and soon thereafter is split, with half sent to assay. The remaining core is almost useless as a geotechnical information source. Without compromising production, it should be possible to at least log RQD, photograph the core before it is split, and run a representative number of point load tests, as a part of the normal geologic logging process. Underground exploration drilling should consider acquisition of geotechnical data as an objective as well as the determination of ore values and general mining conditions.

6. When experimenting with the effect of different mining layouts, approximate extraction ratios could be used, but close attention should always be paid to areas of large spans.

7. Other potential problems are related to fracture orientation, fracturing conditions, stress field, and major structures. Instructions for assessing these are given in the Manual, and background may be found in Section 6.3 of this report.

From the points raised in this section thus far, it should be clear that the support recommendations generated by the MBR System will only represent the in-situ rock conditions as well as permitted by the input data. Thus the user is justified in interpreting the ratings based on experience and judgement.

Much attention has been devoted to the importance of elasticity of the ground support system. A stiff support system in yielding ground will develop high stresses, requiring the support in such instances to be disproportionately strong. A design that matches ground deformation characteristics with suitable support elasticity (the convergence-confinement method) still needs considerable refinement for civil tunnels, and the use of the method in mine design is virtually nonexistent. Accordingly, the support systems in the MBR system, which are based on proven mining practice, do not directly incorporate elasticity considerations. In fact, there is doubt as to whether the deformation characteristics of a given rock mass can be predicted accurately enough on the basis of ground classification alone to permit a convergence-confinement approach. Nonetheless, if support systems are to be optimized beyond present practice, this appears to be an avenue of significance.

It is apparent that the adjustment process in the MBR System tends to penalize competent rock masses more than incompetent ones. This is because the application of further, unfavorable factors to an already-weakened rock mass have proportionally less effect. The arrangement of the support charts reflects this, in that the lower the AMBR or FMBR value, the more substantial the increase in support requirement for a unit reduction in rating.

Finally, the support charts assume a continuously increasing support pressure from top (spot bolting) to bottom (steel sets and reinforced concrete). This is a major simplification. Bolts provide active support; steel sets and concrete do not. Steel sets and timber include yielding supports; concrete support does not yield. Steel sets alone may not provide as much support as rock bolts and shotcrete, for some rock masses. With reference to the above comments regarding elasticity, obviously some supports will be more suitable than others for reasons apart from support pressure alone. These effects have been ignored in order to simplify the selection of supports; hopefully, with continuing use, refinements can be instituted.

7.0 ROCK MASS CLASSIFICATION FOR VEIN MINING

Recovery of vein ores is accomplished through a variety of stoping methods that are highly adapted to the orebodies of interest. Veins occur in a wide range of geometries, and the distribution of ore values along vein structures dictates the scheme for extraction. Except for sublevel caving mines operating in unusually wide veins, where multiple extraction headings transverse to strike may be used, the present MBR system probably could not be effectively applied to vein mining methods.

This investigation has focused on block and panel caving methods, which are applied to large, relatively homogenous, low-grade deposits suitable for mass mining. A complete consideration of support prediction in smaller-scale, selective mining methods such as cut-and-fill, open and stull stoping, square-set stoping, and others, has not been undertaken herein. However, it is hoped that the following discussion will aid others who may be interested in developing a system similar to the MBR for use in the design of underground openings for vein mining.

7.1 Nature of U.S. Vein Mining

Until comparatively recently, most of the world's ore production came from veins of relatively high-grade ore in intimate contact with relatively low-grade wall rock. Vein mining is characteristically selective, emphasizing the extraction of regions or shoots of higher grade ore along the vein structure.

Many veins dip steeply, so that the areal extent of a producing vein mine in plan is typically small, compared to caving mines. Provided the vein system is extensive, the mine becomes essentially two-dimensional--along strike and down dip. Vein mines therefore increase in depth more rapidly than they increase in plan, so that depth quickly becomes a significant factor in rock mass behavior.

Compared to block caving, vein mining is a low-tonnage, high grade operation. Major vein mines recover more than one vein--the Star Mine in Idaho, for example, has over 20 veins--but widely separated shoots and veins or clusters of veins are ordinarily stoped individually, depending on the grade, selectivity required, and other factors. Thus, multiple headings may be used, but layouts incorporating a horizontally extensive system ot closely spaced extraction drifts are uncommon.

Level separation in vein mining is most commonly a few hundred feet or less, to keep the need for up-and-down movement of supplies, men, and ore to a minimum. Levels in block caving are normally at greater separations.

Worldwide, vein mining methods incorporate various caving and/or stoping techniques. In the major U. S. districts, present vein recovery is primarily by overhand stoping and backfilling. Mining progresses upward between levels and little ground is opened up at any one time. Although foreign open-stoping mines in dipping veins may experience appreciable abutment pressures, the backfilling methods used in this country tend to minimize these pressures. Large-scale abutment pressures affecting wide areas, which are the major concern in block caving, do not normally develop in domestic vein mines. Abutment zones do develop around stoping areas, however, as a result of elastic stress redistribution. A common problem in cut-and-fill mining is support of the mining area as stoping approaches the sill pillar between levels; the problem is aggravated if high horizontal stresses are present.

Probably the most important vein mining district in the U. S., at present, is the Coeur d' Alene District in the panhandle of Idaho. In this district, a typical mine recovers multiple, near-vertical, narrow, silver-bearing metalliferous veins by cut-and-fill methods. Some mines are quite deep (8,000 ft or more) and rock stress problems are significant. Wallrock is competent quartzite. The horizontal field stress ordinarily exceeds the vertical stress. Thus rib slabbing and rock bursts are problems.

Support in the laterals and level development headings ranges from none to heavy timber, although most workings are bolted. Depending on rock conditions, Split Set bolts, Dywydag bolts, steel mats, chain link, or timber sets may be used. Steel is not favored because of its rigidity, and concrete sees only local use, for the same reason, near the shafts. Dywydag bolts are used in bad rock, so long as alteration and clay content do not preclude good anchorage.

Timber sets are commonly installed with extensive blocking that can be removed for repair in squeezing ground. Raise-up areas at chute lips are routinely timbered.

Split Set bolts have been found effective under the prevailing stress conditions because rock movement may offset the holes during the life of the opening and Split Set anchorage is enhanced when this occurs. Conventional, mechanical-anchor bolts are declining in popularity.

7.2 Vein Mining Classification Requirements

The support methods prescribed by the MBR system include steel sets and concrete which have not been widely used in the Coeur d' Alene district. This and other differences relating to rock stress environment and mine geometry make the MBR system inapplicable to vein mining. Figure 45 shows where these problems occur, using an MBR standard data form. Note that shafts and raises can not be included in the MBR.

MBR INPUT DATA SHEET
Engineering Data

Project Name RQI Site of Survey Typical Mine By RAC Date 1982

1. Type of Drift(s) Various 2. Orientation(s) Northwest 3. Design Life < 5 yr.

4. Design Dimensions Width 10 ft Width variation ± 1 ft
Height 10 ft Height variation ± 1 ft

5. Drift Spacing (Horizontal) Not Applicable

Other Openings Type Crosscuts / Stopes Size 10 x 10 / Variable Spacing Irregular / Variable

6. Extraction Ratio
Multiple Openings: Excavated Area N/A Unexcavated N/A e_r N/A

Single Opening: 1.5 (width) 15 ft Excavated 700 ft² Unexcavated 1600 ft² e_r 0.44

7. Distance below undercut - drift floor to undercut floor N/A
drift crown to undercut floor N/A

8. Method of Excavation: Machine bored Controlled D & B Conventional D & B

9. Excavation conditions:
Perimeter Hole Traces
Rib or Crown Looseness
New or Existing Cracks } Generally average conventional blasting practice
Overbreak & Barring-Down
Other Criteria

10. Intersections, turnouts: Type Intersection Location Various Max. Span 14 ft

11. Block Dimensions: Side N/A Orientation N/A End N/A Orientation N/A

12. Cave Line Direction N/A Direction of Progress N/A

13. Drift Location (in block, with respect to major structures and their dips, with respect to cave)
Haulages are beneath stopes. Crosscuts at right angles. Main development off vein but parallel to it.

Figure 45 - Attempt to Apply the MBR Concept to Vein Mining.

Rock classification in vein mining would emphasize characteristics of vein mining that particularly influence rock mass behavior. Thus, a complete classification scheme should include ratings for the following.

1. Rock mass strength: compressive strength, extent and nature of fracturing for each key geologic horizon. The vein ore itself should be considered separately. Since concepts of fracturing may not apply well to the vein ore itself, the rating may have to be subjective, based on overall apparent competence of the ore.

2. Geologic setting: extent of faulting and its relationship to the workings, groundwater conditions, extent and location of horizons of various ratings.

3. Geometric factors: type of working, location of other workings, proximity to active mining area, orientation with respect to geologic structures, size or maximum span, opening shapes, location and size of stopes.

4. Depth, rock stress conditions (of key significance).

5. Engineering factors: excavation practice, rate of mining progress, specific mining method, extent of excavation.

Output should enable various stoping plans to be compared from the point of view of ground control. Thus, natural support (pillars) should ideally be an output, as well as artificial support (rock reinforcement, timber, and so on). The optimum stope length is largely a matter of economics, but this length is limited by the maximum safe span the ore will hold in the stopes. Stope length will govern dip pillar spacing, if such pillars are used. The need for dip pillars between stopes develops when mining produces high face-end stresses that can result in bursts or severe slabbing. The tendency for this is strongly dependent on stress field (and depth) which for many vein mines cannot be accurately predicted without measurements.

The importance of accounting for the field stresses in vein mining must be appreciated. Stope closure is always a consideration and high stresses normal to the plane of the vein may even dictate that a more supported mining method be used than might otherwise be chosen. Mine economics are affected because the use of strike or dip pillars will influence recovery; if these pillars are highly stressed, it may be impossible to fully recover them later, even with backfilling of adjacent stopes.

In block caving, pillars are normally not left between blocks or panels because they inhibit caving. The size of pillars between extraction drifts depends on drawpoint spacing and distance to undercut. The size of extraction level pillars in block caving is therefore more the result of ensuring efficient and smooth ore fragmentation than of guaranteeing ground support.

Because vein mining methods are in general more flexible than in block caving, pillar size and location in vein mine layouts can be more readily varied.

For vein mining classifications, supports should encompass the spectrum of present practice, emphasizing bolts and timber. Haulageways and other important openings should be considered semi-permanent by approaching the rock reinforcement design so as to develop a restrained arch around the opening. Stopes and more temporary laterals and cross-cuts can be approached from the standpoint of restraining loose slabs for safety and smooth passage. Timber would ordinarily be restricted to moving ground; bolts are preferred where reinforcing potential exists. Steel mats would be fairly routine to prevent the fall of loose slabs between bolts; mesh or chain link would be used in densely fractured areas.

Because vein mines are extensive in an up-and-down orientation, special attention should be given to support of shafts, raises, inclines, and winzes. A separate data base on shafts and the like, would be necessary to develop support criteria for these types of workings.

Foreign experience in the design of underground openings in high and variable stress fields has considered various opening geometries, depending on stress field trajectories. A rock classification system for use in deep vein mines should be sensitive enough to the shape of the opening to allow one to manipulate this parameter, in optimizing support. By orienting, and shaping, the opening favorably with respect to the stress field, it should be possible to reduce or even eliminate artificial support in some workings. Certainly the attractiveness of such an approach depends on ease of extraction and experience and attitude of the work force, but benefits will accrue from the careful design of long-life underground excavations such as crusher stations and underground hoistrooms. Coeur d'Alene experience has shown a marked tendency for square drifts to eventually assume a horseshoe shape. In many cases it may have been better to drive a horseshoe drift at the outset.

7.3 Summary

Tables 32 and 33 show ratings and support recommendations for typical geological conditions in deep vein mines of the Coeur d' Alene mining district, using the Geomechanics System and the Q-System. These geological conditions are highly generalized and are intended only to illustrate the range of values that might be obtained. Neither system has in its data base of support recommendations an

TABLE 32 - Geomechanics Classification in Coeur d'Alene District

	Typical Rock		Fault and Shear Zones	
	Condition	Rating	Condition	Rating
Intact Rock Strength	Revett Quartzite 24,000 psi	14	6,000	5
RQD	80%	16	40%	8
Discontinuity Spacing	0.8 ft	9	0.3 ft	6
Discontinuity Condition	slightly rough, mod. weathered, discontinuous	22	smooth gouge, crushed	9
Groundwater	damp areas	13	wet to dripping	6
Orientations	parallel, steep unfavorable	-9	parallel, steep unfavorable	-8
Rating	66		26	
Stand-up time[a]	years		10 hours	
Support[b]	none or spot bolting		closely spaced bolts, shotcrete or medium steel ribs.	

[a]10 ft by 10 ft drifts, unsupported.
[b]Support is not expressly specified by RMR for small drifts.

TABLE 33 - Q-System Classification in Coeur d'Alene District

	Typical Rock		Fault and Shear Zones	
	Condition	Rating	Condition	Rating
RQD	80%	80	40%	40
Joint Set Number J_n	3 sets plus random joints	10	heavily jointed	15
Joint Roughness Number J_r	rock walls in contact, rough, irregular	2	clay filling, crushed rock filling	1.0
Joint Alteration Number J_a	slightly altered	2.0	crushed rock, clay	6.0
Stress Reduction Factor SRF[a]	$\sigma_c/\sigma_1 = 2$ to 5	8	multiple shear zones, moderate squeezing pressure	7
Joint Water Reduction Factor J_w	minor inflow	1.0	minor to moderate inflow	0.8
$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$	1.0, "poor" rock		0.05, "extremely poor" rock	
Recommended Support[b]	No Support		mesh-reinforced shotcrete, 6 in.	

[a]Depth greater than 3,000 ft.
[b]Excavation Support Ratio (ESR) = 4 for temporary mine openings; span = 3 meters (10 ft).

adequate representation of small, temporary mine openings, so the supports suggested are hypothetical. Neither system considers stope support.

Interestingly, the classifications do compare in general with prevailing District support practice in more competent ground (mats and rockbolts). The Geomechanics System recommends steel sets, or bolts and shotcrete, in fault zones, making no specific allowance for squeezing rock or exceptionally high stress conditions. In practice, steel sets are normally not used, and in ground where bolts may not anchor properly, timber sets are installed. The Q-System, having no identical cases in its data base, nevertheless suggests 6 in. of mesh-reinforced shotcrete, with the observation that space should be left behind the support to allow for deformation. Shotcrete would probably not be preferred in such situations by operators in the district. However, the Q-System does consider various stress fields and, therefore, the role of depth can be allowed for.

Unpublished data from USBM files give values for RSR between 44 and 57. These data were developed during visits to several Coeur d'Alene mines in 1973. These RSR values correspond to a support requirement consisting of rockbolts at 4 ft to 6 ft centers and 2 in. of shotcrete. The shotcrete thicknesses observed during these mine visits were commonly near 2 in. (although often less) and bolts are at 4 ft centers or less, with 2-3 ft spacings in poor rock.

Ground support remains a problem in the district, however, especially at greater (7,000 ft) depths, where severe squeezing of even heavy timber sets occurs, and it appears that improvements in existing classification systems are necessary to adequately allow for effects from stress field, mine layout, and depth.

Certain features of the Q, RMR, and RSR systems appear to have specific value in vein mining. The concept of the Stress Reduction Factor (quotient of stress field and intact compressive strength used in the Q-System), if refined based on case histories, could be used to account for increasing depth and high field stress conditions, proximity of nearby openings, and the effect of mining-induced localized stress concentrations, such as in sill pillars. The simple addition of ratings for RQD, fracture spacing, fracture condition, and water condition has been found useful for characterizing rock mass conditions by users of the RMR, and would also be useful for vein mining.

The geological rating obtained should be modified by a term expressing the degree of faulting. The terminology used in the Q-System for faulting (single or multiple shears; clay or crushed rock content, and so on) but combined with the geological rating by simple subtraction, would seem to be best.

The basic geological ratings could be used directly to determine support requirements for development workings not in ore. For haulageways driven along the vein and for temporary stope support, the vein material and the wall rock should be rated separately, and the relationship of these three ratings used to define the most suitable support. Concepts of relative competence of hangingwall, footwall, and ore already form the basis for selection of method at many mines.

For each support chart, separate curves could be shown for horizontal workings such as drifts or crosscuts, and inclined or vertical workings such as raises, winzes, or shafts. Similarly, different uses or criticality of openings could be allowed for.

The support requirements will depend on mining method, since some methods are more supportive than others. The specific way of handling this might be to use some expression of mined-out volume to total ore volume in the plane of the vein, related to different support requirements. The MBR has done this for the horizontal plane, through the use of the extraction ratio e_r. There are doubtless many other plausible approaches and the selection of the best one should be on the basis of case history studies and mine visits.

As always, development of a rock classification system for vein mining should only be undertaken after an extensive base of case history data has been collected. This data base should encompass the range of mining methods and rock conditions to which the system is to be applied.

8.0 AREAS FOR FURTHER INVESTIGATION

During the course of this investigation, it became clear that there are several areas in which a ground classification approach such as the MBR could enhance production and safety in block caving mines. Also, certain areas require further basic research. These are discussed in the following section.

8.1 Cavability

A key question prior to mining is to determine under what circumstances an orebody can be induced to cave. The classification described by McDonough (1976), and in use at Climax, has been suggested as an index of cavability. McDonough's classes have been related to RMR for Climax and Henderson (Maier, 1980, unpublished data). However, McDonough's classes are subjective and require much experience with the rock in question. During preliminary planning, when cavability estimates are most needed, this experience may not have yet been gained.

McMahon and Kendrick (1969) proposed a relationship of RQD to cavability number, the cavability number being a measure of the caving characteristics of the ore. RQD was selected as a measure of faulting and shearing, thought to be a key factor determining cavability. The data base in support of this relationship is limited, however.

More recently, Nicholas (1981) modified the relation of McMahon and Kendrick, and suggests that fracture spacing data can be used to determine cavability.

The last two approaches simplistically model the caving mechanism, which is complex and not well-understood. Caving is facilitated when the rock mass can no longer support itself by transferring its weight to the sides of the stope. Thus a more general concept of rock mass competence is called for, recalling the influence of fracture condition, stress field, rock strength, and so on, in addition to RQD and fracture spacing.

What is needed is a reliable, predictive relationship based on measurable quantities, that will apply to a representative range of rock types. First, however, a full description of "cavability" is needed, since the acceptability of caving characteristics have been found to depend on the mining method, and because "cavability" is itself almost a purely subjective term.

Still, current beliefs are that caving characteristics are related to measurable geologic parameters. Rock classification approaches are inherently suitable for handling the interaction of these parameters in the assessment of cavability. The established correlation of rock class with span could be used as a basis for predicting required undercut area.

8.2 Secondary Blasting

The need for secondary blasting is germane not only to production efficiency but to safety and drift support as well. Secondary blasting is required when the ore packs or hangs up in the drawpoint. Packing is experienced when very soft, clayey ore consolidates under the local stress conditions prevailing in the drawpoint. Hangups occur when a rock fragment or rock fragments become wedged in place in the drawpoint without breaking up. Packed or hung-up drawpoints impede production, attract stress, subject the production personnel to the hazard of climbing into the drawpoint to set charges, and result in additional blasting shock to the rock mass and support system when the hang-up or pack is shot.

The occurrence of large blocks will be related primarily to the spacing of weak fractures and secondarily to the crushing strength of the rock. Caving stresses seldom approach the compressive strength of most intact rock blocks, so some sort of shear failure along weakness planes must be relied upon to promote the smooth flow of ore.

Rock classification systems, including the MBR, accept fracture spacing, orientation, and condition data and combine it to form a picture of rock mass competence. These same data can be used to predict fragmentation, and, if correlated with production blasting requirements from operating mines, could be used to develop predictive criteria for secondary blasting as a natural outgrowth of other ground classification efforts, such as the use of the MBR.

Similarly, the tendency to pack is related to the occurrence of soft, wet, decomposed rock, which could readily be indicated by combinations of input terms to to RMR or MBR.

8.3 Draw Control

Secondary blasting, cavability, and drift supportability all determine the likelihood for good draw control, which is the essence of drift support during the production period. Assessments of these and other influential parameters could be combined into a factor reflecting the ease of draw control. Mines with a high draw control difficulty factor would anticipate problems with maintaining even production due to drift repair, hangups, stubs, and the like. Parametric studies could then be initiated to vary drawpoint aperture, level separation (ore column height), and so on, to lower this factor.

8.4 Drawpoint Spacing

The selection of the optimum drawpoint spacing is important to reduce the occurrence of hangups and promote smooth production. Current trends recognize that the necessary drawpoint spacing is related to fragmentation; the coarser the fragmentation, the wider the drawpoint spacing (White, 1979). As has been pointed out before in this section, ground classification can be an excellent indicator of fragmentation.

8.5 Basic Research

Throughout this report, and in the Manual, it has been stressed that the MBR System is no substitute for detailed design. Although this is true, the fact is that detailed design of mine drift supports is greatly hampered by lack of knowledge of basic engineering data for the structural elements involved.

Ground classification systems are one means of assessing the material properties of the rock mass. Relationships are needed that will permit specific determination of rock mass strength and, especially, deformation characteristics. This is complicated by the fact that properties in dynamic and tensile modes need to be known as well.

The loads to be borne by the key structural elements, which are comparatively well-known for most engineered structures, also need to be quantifiable and predictable for caving mines, based on measurable characteristics. Thus, the abutment loads, undercut unloadings, and caving stresses must be relatable to depth, stress field, geologic setting, and mining geometry. Although many measurements of abutment stresses have been made to date, there is still no useful relation for predicting these stresses in advance of mining.

In Section 6.2 of this report, a generalized relationship between rock mass condition, as indicated by RMR, and rock mass strain tendency is derived. This relationship assumes a single, representative RMR value for each mine considered--obviously an oversimplification. Reported stress and strain measurements need to be tied to geologic characteristics, (or directly to RMR) so that relationships such as those in Section 6.2 can be refined.

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