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A geomechanics classification system for the rating of rock mass in mine design

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SYNOPSIS

The mining rock-mass rating (MRMR) classification system was introduced in 1974 as a development of the CSIR geomechanics classification system to cater for diverse mining situations. The fundamental difference was the recognition that *in situ* rock-mass ratings (RMR) had to be adjusted according to the mining environment so that the final ratings (MRMR) could be used for mine design. The adjustment parameters are weathering, mining-induced stresses, joint orientation, and blasting effects.

It is also possible to use the ratings (RMR) in the determination of empirical rock-mass strength (RMS) and then in the application of the adjustments to arrive at a design rock-mass strength (DRMS). This classification system is versatile, and the rock-mass rating (RMR), the mining rock-mass rating (MRMR), and the design rock-mass strength (DRMS) provide good guidelines for the purposes of mine design. However, in some cases a more detailed investigation may be required, in which case greater attention is paid to specific parameters of the system.

Narrow and weak geological features that are continuous within and beyond the stope or pillar must be identified and rated separately.

The paper describes the procedure required to arrive at the ratings, and presents practical examples of the application of the system to mine design.

SAMEVATTING

Die mynrotsmassa-aanslag-klassifikasiestelsel (MRMR) is in 1974 ingevoer as 'n ontwikkeling van die WNNR se geomeganikaklassifikasiestelsel om vir uiteenlopende mynboutoestande voorsiening te maak. Die fundamentele verskil was die erkenning van die feit dat *in situ*-rotsmassa-aanslae (RMR) volgens die mynbou-omgewing aangesuiwer moes word om die finale aanslae (MRMR) vir mynontwerp te kan gebruik. Die aansuiweringsparameters is verwering, mynbougeïnduseerde spannings, naatoriëntasie en die gevolge van skietwerk.

Dit is ook moontlik om die aanslae (RMR) by die bepaling van empiriese rotsmassasterkte (RMS) te gebruik, en dan by die toepassing van aansuiwerings om 'n ontwerprotsmassasterkte (DRMS) te kry. Hierdie klassifikasiestelsel is veelsydig en die rotsmassa-aanslag (RMR), die mynrotsmassa-aanslag (MRMR), en die ontwerprotsmassasterkte (DRMS) verskaf goeie riglyne vir die doeleindes van mynontwerp. Daar kan egter in sommige gevalle 'n uitvoeriger ondersoek nodig wees waarin daar meer aandag aan spesifieke parameters van die stelsel geskenk word.

Smal en swak geologiese aspekte wat deurlopend is in en verby die afbouplek of pilaar, moet geïdentifiseer en afsonderlik aangeslaan word.

Die referaat beskryf die prosedure wat nodig is om die aanslae te kry en gee praktiese voorbeelde van die toepassing van die stelsel op mynbou-ontwerp.

INTRODUCTION

The classification system known as the mining rock-mass rating (MRMR) system was introduced in 1974 as a development of the CSIR geomechanics classification system[1,2]. The development is based on the concept of *in situ* and adjusted ratings, the parameters and values being related to complex mining situations. Since that time, there have been modifications and improvements[3-5], and the system has been used successfully in mining projects in Canada, Chile, the Philippines, Sri Lanka, South Africa, the USA, and Zimbabwe.

This paper consolidates the work presented in previous papers and describes the basic principles, data-collection procedure, calculation of ratings (RMR), adjustments (MRMR), design rock-mass strength (DRMS), and practical application of the systems.

An important development of this classification makes it suitable for use in the assessment of rock surfaces, as well as borehole cores.

Taylor[4] reviewed the classification systems developed by Wickham, Barton, Bieniawski, and Laubscher and concluded that

> Thus, the four systems chosen as being the most advanced classifications are based on relevant parameters. Each technique undoubtedly yields meaningful results, but only Laubscher's geomechanics classification and the 'Q' system of Barton offer suitable guidelines for the assessment of the main parameters; namely, the joint attributes. For general mining usage and where the application of a classification varies widely, Laubscher's geomechanics classification has the added advantage of allowing further adjustments to the rating for different situations. This, coupled with the fact that the technique has been in use for six years, gives no reason for changing to another system which offers no substantial improvement.

The figure below shows a 98 per cent correlation between the RMR of the MRMR system and the NGI system based on the classification by Taylor[4] of thirty sites ranging from very poor to very good. Thus, if NGI data are available, this information can be used in the practical applications.

PRINCIPLES

A classification system must be straightforward and have a strong practical bias so that it can form part of the normal geological and rock-mechanics investigations to be used for mine design and communication. Highly sophisticated techniques are time-consuming, and most

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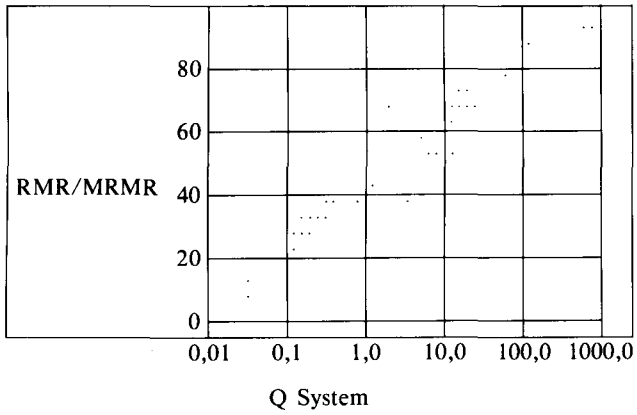

mines cannot afford the large staff required to provide complex data of doubtful benefit to the planning and production departments.

The approach adopted involves the assignment to the rock mass of an *in situ* rating based on measurable geological parameters. Each geological parameter is weighted according to its importance, and is assigned a maximum rating so that the total of all the parameters is 100. This weighting was reviewed at regular intervals in the development of the system and is now accepted as being as accurate as possible. The range of 0 to 100 is used to cover all variations in jointed rock masses from very poor to very good. The classification is divided into five classes with ratings of 20 per class, and with A and B sub-divisions.

A colour scheme is used to denote the classes on plan and section: class 1 blue, class 2 green, class 3 yellow, class 4 brown, and class 5 red. Class designations are for general use, and the ratings should be used for design purposes.

The ratings are, in effect, the relative strengths of the rock masses. The accuracy of the classification depends on the sampling of the area being investigated. The terminology *preliminary, intermediate,* and *final* should be applied to assessments to indicate the state of drilling and development. It is essential that classification data are made available at an early stage so that the correct decisions are made on mining method, layout, and support requirements.

In the assessment of how the rock mass will behave in a mining environment, the rock-mass ratings (RMR) are adjusted for weathering, mining-induced stresses, joint orientation, and blasting effects. The adjusted ratings are called the mining rock-mass ratings or MRMR.

It is also possible to use the ratings to determine an empirical rock-mass strength (RMS) in megapascals (MPa). The *in situ* rock-mass strength (RMS) is adjusted as above to give a design rock-mass strength (DRMS). This figure is extremely useful when related to the stress environment, and has been used for mathematical modelling.

The classification system is versatile, and the rock-mass rating (RMR), the mining rock-mass rating (MRMR), and the design rock-mass strength (DRMS) provide good guidelines for mine design purposes. However, in some cases where a more detailed investigation is required, examples of these situations are described in which specific parameters of the system are used.

Since average values can be misleading and the weakest zones may determine the response of the whole rock mass, these zones must be rated on their own. Narrow and weak geological features that are continuous within and beyond the stope or pillar must be identified and rated separately.

GEOLOGICAL PARAMETERS, SAMPLING, AND RATINGS

The geological parameters that must be assessed include the intact rock strength (IRS), joint/fracture spacing, and joint condition/water. Before the classification is done, the core or rock surface is examined and divided into zones of similar characteristics to which the ratings are then applied. These parameters and their respective ratings are shown in Table I.

Intact Rock Strength (IRS)

The IRS is the unconfined uniaxial compressive strength of the rock between fractures and joints. It is important to note that the cores selected for testwork are invariably the strongest pieces of that rock and do not necessarily reflect the average values; in fact, on a large copper mine, only unblemished core was tested. The IRS of a defined zone can be affected by the presence of weak and strong intact rock, which can occur in bedded deposits and deposits of varying mineralization. An average value is assigned to the zone on the basis that the weaker rock will have a greater influence on the average value. The relationship is non-linear, and the values can be read off an empirical chart (Fig. 1).

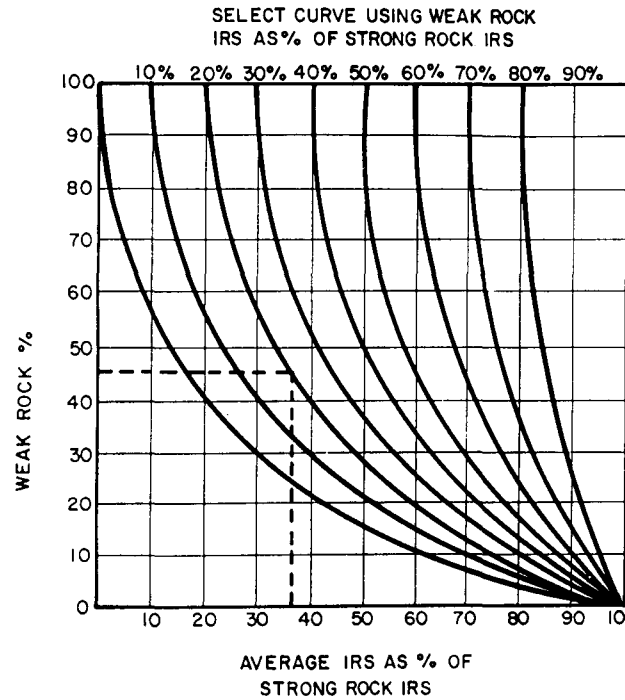

Fig. 1—Determination of average IRS where the rock mass contains weak and strong zones

Example:
Strong rock IRS = 100 MPa
Weak rock IRS = 20 MPa

$$\frac{\text{Weak rock IRS}}{\text{Strong rock IRS}} \times 100 = 20\%$$

Weak rock IRS = 45%
Average IRS = 37% of 100 MPa = 37 MPa

The rating range is from 0 to 20 to cater for specimen strengths of 0 to greater than 185 MPa. The upper limit of 185 MPa has been selected because IRS values greater than this have little bearing on the strength of jointed rock masses.

Spacing of Fractures and Joints (RQD + JS or FF)

Spacing is the measurement of all the discontinuities and partings, and does not include cemented features. Cemented features affect the IRS and as such must be included in that determination. A joint is an obvious feature that is continuous if its length is greater than the width of the excavation or if it abuts against another joint, i.e. joints define blocks of rock. Fractures and partings do not necessarily have continuity. A maximum of three joint sets is used on the basis that three joint sets will define a rock block; any other joints will merely modify the shape of the block.

Two techniques have been developed for the assessment of this parameter:

- the more detailed technique is to measure the rock quality designation (RQD) and joint spacing (JS) separately, the maximum ratings being 15 and 25 respectively;
- the other technique is to measure all the discontinuities and to record these as the fracture frequency per metre (FF/m) with a maximum rating of 40, i.e. the 15 and 25 from above are added.

Designation of Rock Quality (RQD)

The RQD determination is a core-recovery technique in which only cores with a length of more than 100 mm are recorded:

$$\text{RQD, \%} = \frac{\text{Total lengths of core} > 100\text{ mm}}{\text{Length of run}} \times 100.$$

Only cores of at least BXM size (42 mm) should be used. It is also essential that the drilling is of a high standard.

The orientation of the fractures with respect to the core is important for, if a BXM borehole is drilled perpendicular to fractures spaced at 90 mm, the RQD is 0 per cent. If the borehole is drilled at an inclination of 40 degrees, the spacing between the same fractures is 137 mm; on this basis, the RQD is 100 per cent. As this is obviously incorrect, it is essential that the cylinder of the cores (sound cores) should exceed 100 mm in length. At the quoted 40 degree intersection, the core cylinder would be only 91 mm and the RQD 0 per cent. The length of core used for the calculation is measured from fracture to fracture along the axis of the core.

In the determination of the RQD of rock surfaces, the sampling line must be likened to a borehole core and the following points observed:

- experience in the determination of the RQD of core is necessary;
- do not be misled by blasting fractures;
- weaker bedding planes do not necessarily break when cored,
- assess the opposite wall where a joint forms the sidewall,
- shear zones greater than 1 m must be classified separately.

Joint Spacing (JS)

A maximum of a three-joint set is assumed, i.e. the number required to define a rock block. Where there are four or more joint sets, the three closest-spaced joints are used. The original chart for the determination of the JS rating has been replaced by that proposed by Taylor[4]. From the chart in Fig. 2 it is possible to read off the rating for one-, two-, and three-joint sets.

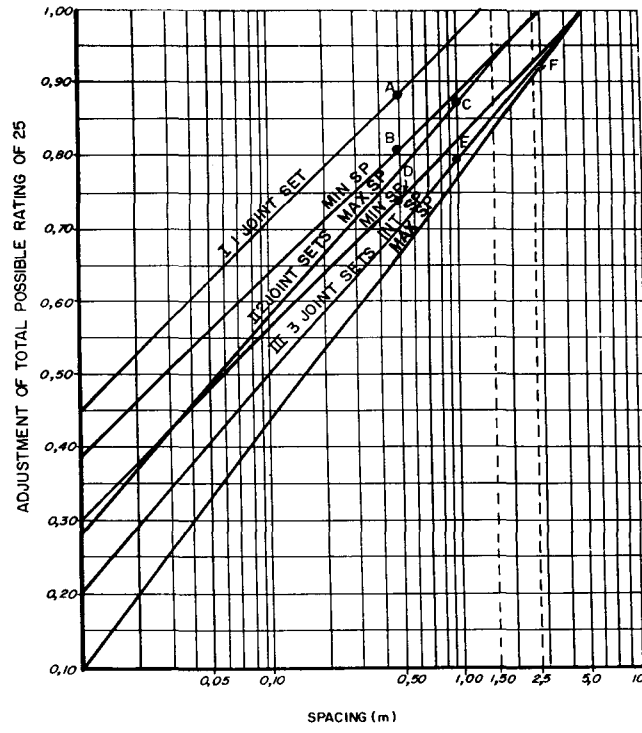

Fig. 2—Assessment of joint-space rating (after Taylor[4])

NB x = spacing × 100

A 1-Joint set R = 25 × ((26,4 × $\log_{10}$ x) + 45)/100

B 2-Joint set R = 25 × ((25,9 × $\log_{10}$ x_{min}) + 38)/100 × ((30,0 × $\log_{10}$ x_{max}) + 28)/100

C 3-Joint set R = 25 × ((25,9 × $\log_{10}$ x_{min}) + 30)/100 × ((29,6 × $\log_{10}$ x_{int}) + 20)/100 × ((33,3 × $\log_{10}$ x_{max}) + 10)/100

Example:

One set, spacing at 0,5 m = A, rating = 0,88 × 25 = 22

Two sets, spacing at 0,5 m and 1,0 m = B + C, rating = 0,81 × 0,86 × 25 = 17

Three sets, spacing at 0,5 m, 1,0 m, and 3,0 m = D, E + F, rating = 0,74 × 0,80 × 0,93 × 25 = 14

Fracture Frequency per Metre (FF/m)

This apparently simplified system requires the measurement of all the discontinuities that are intersected by the sampling line. It is important to determine whether a one-, two-, or three-joint system is being sampled. For the same FF/m, a rock mass with a one-joint set is stronger than one with a two-joint set, which is again stronger than one with a three-joint set. The rating allocation in Table I makes provision for the different joint sets.

In the case of core, it is also necessary to know whether only one or two joints of a three-joint system are intersected.

Underground measurement of fracture frequency is done on the sidewalls and hanging of drifts, tunnels, or stopes, depending on the orientation of the features. The

TABLE I
GEOLOGICAL PARAMETERS AND RATINGS

1. Meaning of the ratings

	1	2	3	4	5
Class	A B	A B	A B	A B	A B
Rating	100 – 81	80 – 61	60 – 41	40 – 21	20 – 0
Description	Very Good	Good	Fair	Poor	Very Poor
Colour	Blue	Green	Yellow	Brown	Red

Distinguish between the A and B sub-classes by colouring the A sub-class full and cross-hatch the B.

2. Parameters and ratings

IRS–MPa rating %		RQD rating %		Joint spacing m	Fracture frequency, FF/m: Average per metre	Rating: 1 set	2 set	3 set
>185	20	97–100	15	0<—>25	0,1	40	40	40
165–185	18	84–96	14	See Fig. 2	0,15	40	40	40
145–164	16	71–83	12		0,20	40	40	38
125–144	14	56–70	10		0,25	40	38	36
105–124	12	44–55	8		0,30	38	36	34
85–104	10	31–43	6		0,50	36	34	31
65–84	8	17–30	4		0,80	34	31	28
45–64	6	4–16	2		1,00	31	28	26
35–44	5	0–3	0		1,50	29	26	24
25–34	4				2,00	26	24	21
12–24	3				3,00	24	21	18
5–11	2				5,00	21	18	15
1–4	1				7,00	18	15	12
					10,00	15	12	10
					15,00	12	10	7
					20,00	10	7	5
					30,00	7	5	2
					40,00	5	2	0
					ALLOW FOR CORE RECOVERY			

TABLE I (continued opposite)

TABLE II
FACTORS TO GIVE AVERAGE FRACTURE FREQUENCY

Sampling procedure	Factor
a. One set of three sets on a line, or one set only	1,0
b. Two sets of three sets on a line or two sets only	1,5
c. All of the sets on a line or borehole core	2,0
d. Two sets on one line and one on another	2,4
e. Three sets on three lines at right-angles	3,0

TABLE III
BOREHOLE LOG SHEET

Borehole No: **Date:**

Zoning of borehole
Interval length (A)
Total sound core (B)

RQD, % $\frac{B}{A} \times 100$

Joint spacing	Low angle 0–29	Number Mean spacing (A) True distance = A × sin (0,26)
	Moderate angle 30–59	Number Mean spacing (B) True distance = B × sin (0,71)
	High angle 60–90	Number Mean spacing (C) True distance = C × sin (0,97)

Average frequency = $\frac{\text{Sum of individual FF/m (inverse of spacing)}}{2}$

IRS **Final Rating**
RQD
Joint spacing
Joint condition
Total
Remarks
Sin values 0–29 = 0,26 30–59 = 0,71 60–90 = 0,97

Signature:

TABLE I (continued from opposite page) **3. Assessment of joint condition**

Parameter	Description		Accumulative % adjustment of possible rating of 40			
				Adjustment, %		
			Dry	Moist	Mod. pressure 25–125 1/m	High pressure >125 1/m
A	Multi wavy directional		100	100	95	90
	Uni		95	90	85	80
Large-scale joint expression	Curved		85	80	75	70
	Slight undulation		80	75	70	65
	Straight		75	70	65	60
B	Rough stepped/irregular		95	90	85	80
	Smooth stepped		90	85	80	75
Small-scale joint expression	Slickensided stepped		85	80	75	70
	Rough undulating		80	75	70	65
200 mm × 200 mm	Smooth undulating		75	70	65	60
	Slickensided undulating		70	65	60	55
	Rough planar		65	60	55	50
	Smooth planar		60	55	50	45
	Polished		55	50	45	40
C Joint wall alteration weaker than wall rock and only if it is weaker than the filling			75	70	65	60
D	Non-softening and sheared material	Coarse	90	85	80	75
		Medium	85	80	75	70
		Fine	80	75	70	65
Joint filling	Soft sheared material, e.g. talc	Coarse	70	65	60	55
		Medium	60	55	50	45
		Fine	50	45	40	35
	Gouge thickness < amplitude of irregularities		45	40	35	30
	Gouge thickness > amplitude of irregularities		30	20	15	10

Example: A straight joint with a smooth surface and medium sheared talc under dry conditions gives A = 70%, B = 65%, D = 60%; total adjustment = 70 × 65 × 60 = 27%, and the rating is 40 × 27% = 11.

The rock mass rating (RMR) is the sum of the individual ratings.

following situations apply:

- if all the features are present in the sidewalls, establish whether they intersect a horizontal line;
- if they all do not intersect the horizontal line, measure on a vertical line as well;
- if a set is parallel to the sidewall, measure these on a line in the hanging at right-angles to the sidewall.

This conflicting situation of different sampling procedures can be resolved if the sum of the measurements is divided by a factor to arrive at the average frequency. These factors are shown in Table II, which can be appreciated if compared with the sampling of the sides of a cube on different lines on intersection.

The need for accurate sampling cannot be too highly stressed. Often detailed scan-line surveys are done on sidewalls that do not intersect all the features, and then this biased information is analysed in detail.

Where boreholes do not intersect all the features at 45 degrees, a sampling bias will occur unless provision is made for the angle of intersection, as in the log sheet of Table III.

The average fracture frequency per metre (FF/m) is used in Table I to determine the rating. The inverse of this number gives the average fracture spacing. The data from A and B can be used only if the joint spacing for all the sets is approximately the same.

Fig. 3 shows the relationship between FF/m and ratings after the different sampling techniques for core and underground exposures have been adjusted to an average spacing.

Because the FF/m includes both continuous(joints) and discontinuous(fractures) features, the continuity must be estimated to give the joint spacing and rock block size (Fig. 4). Thus, the FF/m will give the rock-mass rating, but this has to be adjusted by the factors given in Table IV.

Core Recovery

As the FF/m does not recognize core recovery, the FF/m must be increased if there is a core loss, which will occur in the weaker sections of the core. The adjustment is done by dividing the FF/m by the core recovery and multiplying the quotient by 100.

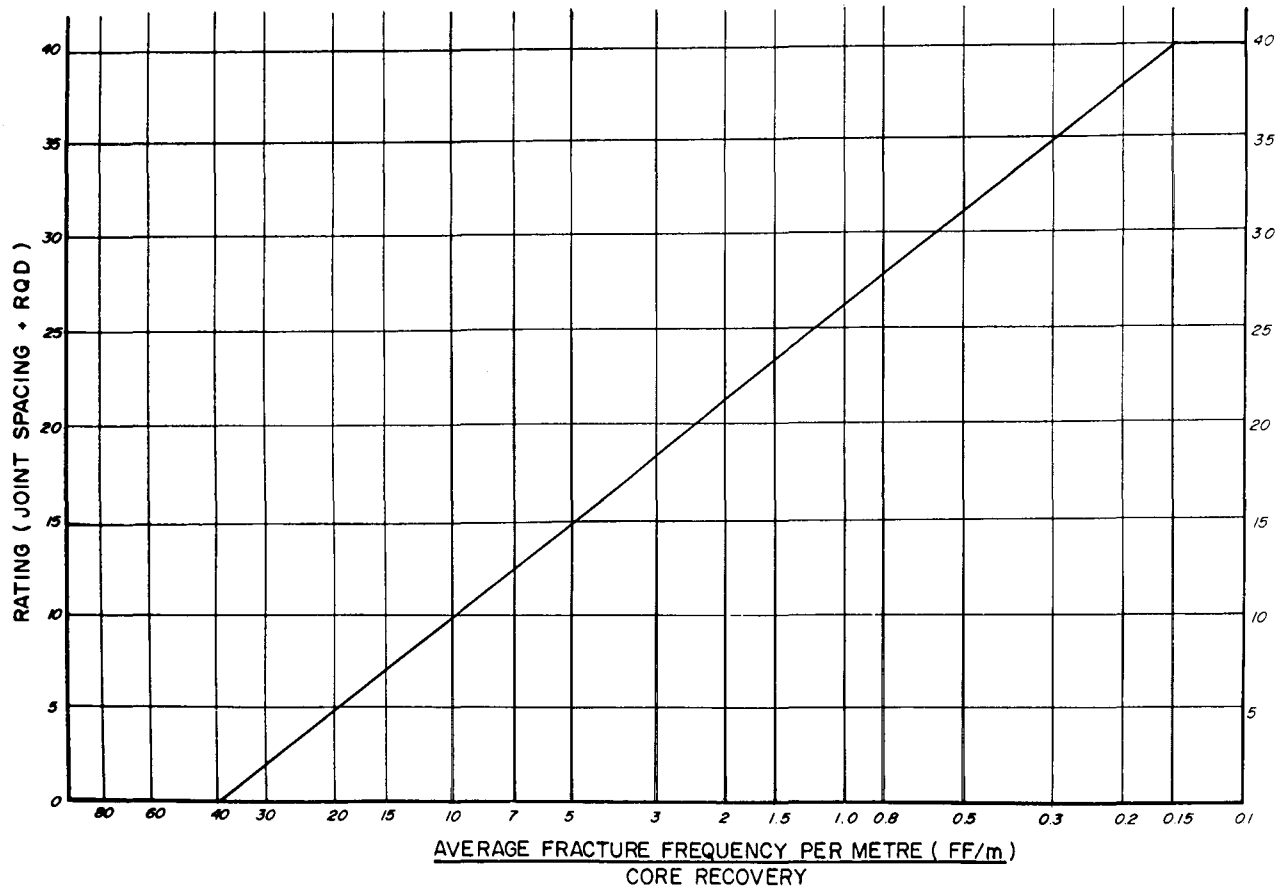

Fig. 3—Ratings for fracture frequency per metre

TABLE IV
FACTORS BY WHICH JOINT FREQUENCIES ARE MULTIPLIED

Continuous features %	Factor
100	1,0
90	0,9
80	0,8
70	0,7
60	0,6
50	0,5

TABLE V
COMPARISON OF TECHNIQUES

Joint spacing m	Rating			Rating FF/m
	RQD	JS	Combined	
0,025	0	1	1	1
0,05	0	1,5	1,5	5
0,10	8	3	11	10
0,20	12	5	17	15
0,50	14	10	24	20
1,00	15	13	28	26
2,00	15	19	34	31
3,00	15	21	36	33
4,00	15	23	38	36
5,00	15	25	40	38

Comparison of the Two Techniques

The advantage of the FF/m technique is that it is more sensitive than the RQD for a wide range of joint spacings, because the latter measures only core less than 100 mm and rapidly changes to 100 per cent. Examples of this are shown in Table V, which assumes that there is a percentage of core greater than 100 mm at joint intersections.

The fracture-frequency technique was first used in Chile in 1985 and then in Canada in 1986. In Zimbabwe, the FF/m technique was used in conjunction with the RQD and JS technique and was found to be just as accurate.

Joint Condition and Water

Joint condition is an assessment of the frictional properties of the joints (not fractures) and is based on expression, surface properties, alteration zones, filling, and water. Originally the effect of water was catered for in a separate section: however, it was decided that the assessment of joint condition allowing for water inflow would have greater sensitivity[3]. A total rating of 40 is now assigned to this section. The procedure for the determination of joint condition is shown in Table I, which divides the joint-assessment section into sub-sections A, B, C, D.

Sub-section A caters for the large-scale expression of the feature, such as across a drift or in a pit face. B assesses the small-scale expression and is based on the profiles shown in Fig. 5. Section C is applied only when there is a distinct difference between the hardness of the host rock and that of the joint wall. Section D covers the variations in joint filling.

As the conditions of the different joint sets are not

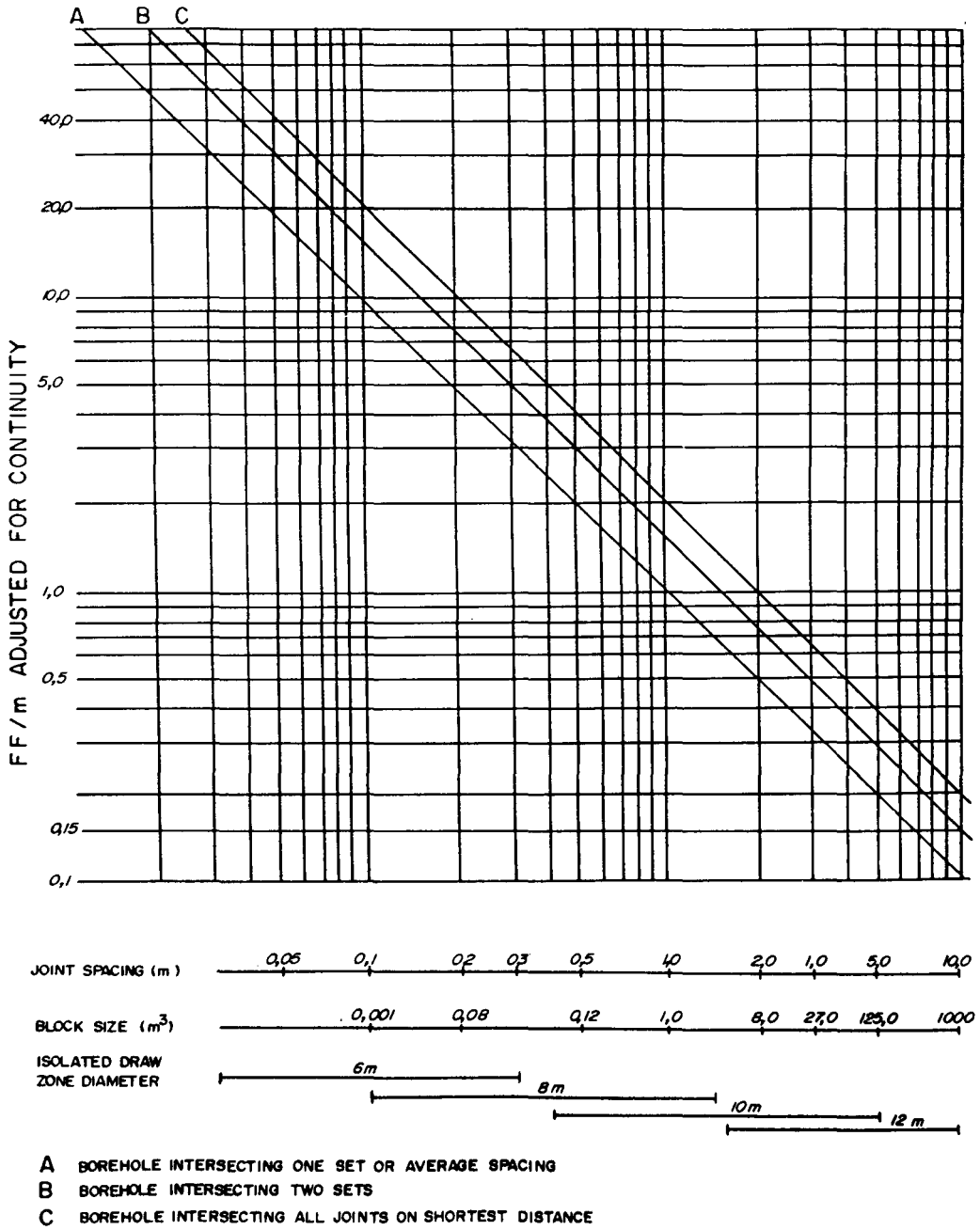

Fig. 4—Joint-spacing diagram

necessarily the same, a weighted average has to be calculated. However, if there is a significant difference in the condition ratings, this should be highlighted in the text or on the plans. A low rating for one joint set could influence the orientation of tunnels and/or the mining sequence.

When there is a preponderance of crosscuts over drives or *vice versa*, a sampling bias can occur, resulting in preference being given to those features that intersect the dominant drifts at a large angle.

ADJUSTMENTS

The RMR is multiplied by an adjustment percentage to give the MRMR. The adjustment percentages are empirical, having been based on numerous observations in the field. The adjustment procedure requires that the engineer assess the proposed mining activity in terms of its effect on the rock mass. For example, poor blasting influences the stability of a drift or pit slope but has no influence on the cavability of the rock mass.

It has been found that there is a better appreciation of the operation when planning personnel have to think in terms of adjustments. The adjustment concepts developed for the MRMR system were used by Engineers International, Inc. to prepare a classification of caving-mine rock mass and a support estimation system[6].

Weathering

Certain types of rock weather readily, and this must be taken into consideration in decisions on the size of opening and the support design. Weathering is time-dependent, and influences the timing of support installa-

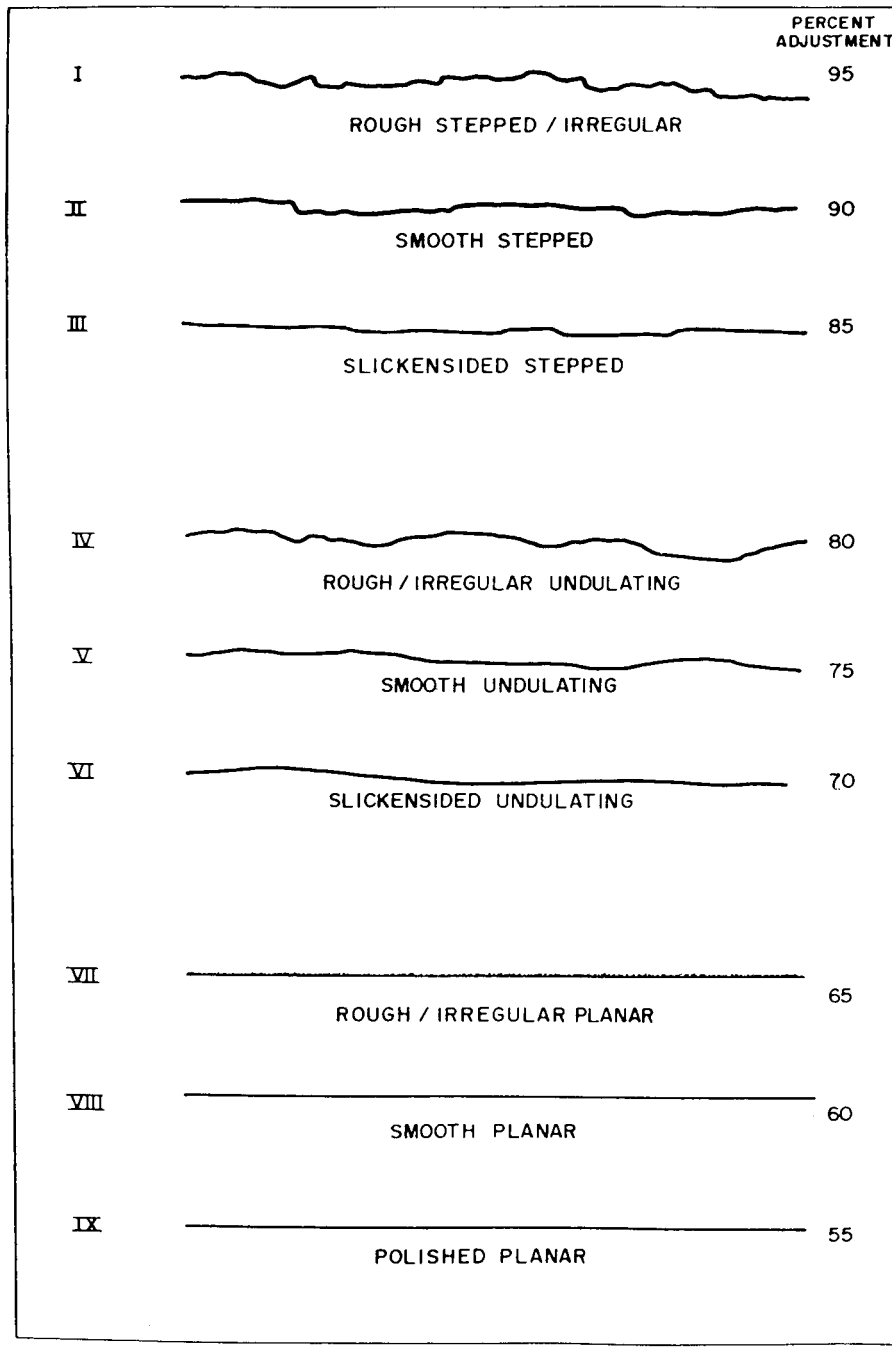

Fig. 5—Joint roughness profiles

tion and the rate of mining.

The three parameters that are affected by weathering are the IRS, RQD or FF/m, and joint condition. The RQD percentage can be decreased by an increase in fractures. The IRS can decrease significantly as chemical changes take place; in fact, there is the situation with kimberlites, where solid hard rock becomes sand in a short time. The joint condition is affected by alteration of the wallrock and the joint filling. Weathering data based on the examination of borehole cores can be conservative owing to the large surface area of core relative to the volume—underground exposures are more reliable.

Table VI shows the adjustment percentages related to degree of weathering after a period of exposure of half, one, two, three, and four-plus years.

TABLE VI
ADJUSTMENTS FOR WEATHERING

Degree of weathering	Potential weathering and adjustments, %				
	½ y	1 y	2 y	3 y	4 + y
Fresh	100	100	100	100	100
Slight	88	90	92	94	96
Moderate	82	84	86	88	90
High	70	72	74	76	78
Complete	54	56	58	60	62
Residual soil	30	32	34	36	38

Joint Orientation

The size, shape, and orientation of an excavation af-

fects the behaviour of the rock mass. The attitude of the joints, and whether or not the bases of blocks are exposed, have a significant bearing on the stability of the excavation, and the ratings must be adjusted accordingly. The magnitude of the adjustment depends on the attitude of the joints with respect to the vertical axis of the block. As gravity is the most significant force to be considered, the instability of the block depends on the number of joints that dip away from the vertical axis. The required adjustments are shown in Table VII.

TABLE VII
PERCENTAGE ADJUSTMENTS FOR JOINT ORIENTATION

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

The orientation of joints has a bearing on the stability of open stopes and the cavability of undercut rock masses.

The adjustments for the orientation of shear zones with respect to development are as follows: 0–15° = 76%, 15–45° = 84%, 45–75° = 92%.

Advance of the ends in the direction of dip of structural features is preferable to development against the dip. An adjustment of 90 per cent should be made to previous adjustments when the advance is against the dip of a set of closely spaced joints. This is because it is easier to support rock blocks that have the prominent joints dipping with the advance.

The adjustment for shear-zone orientation does not apply to 'jointed rock'. The maximum rating is therefore joint orientation multiplied by direction of advance, which is 70% × 90% = 63%.

The effect of joint orientation and condition on stability is clearly displayed in bridge arches made from high-friction rock blocks.

Joint-orientation Adjustment for Pillars and Sidewalls

A modified orientation adjustment applies to the design of pillars or stope sidewalls. Adjustments are made where joints define an unstable wedge with its base on the sidewall. The instability is determined by the plunge of the intersection of the lower joints, as well as by the condition of the joints that define the sides of the wedge (Table VIII).

TABLE VIII
PERCENTAGE ADJUSTMENTS FOR THE PLUNGE OF THE INTERSECTION OF JOINTS ON THE BASE OF BLOCKS

Average rating	Plunge degree	Adjustment %	Plunge degree	Adjustment %	Plunge degree	Adjustment %
0–5	10–30	85	30–40	75	>40	70
5–10	10–20	90	20–40	80	>40	70
10–15	20–30	90	30–50	80	>50	75
15–20	30–40	90	40–60	85	>60	80
20–30	30–50	90	>50	85		
30–40	40–60	90	>50	90		

Mining-induced Stresses

Mining-induced stresses result from the redistribution of field(regional) stresses that is caused by the geometry and orientation of the excavations. The magnitude and ratio of the field stresses should be known. The redistribution of the stresses can be obtained from modelling or from published stress-redistribution diagrams[7,8]. The redistributed stresses that are of interest are maximum, minimum, and differences.

Maximum Stress

The maximum principal stress can cause spalling of the wall parallel to its orientation, the crushing of pillars, and the deformation and plastic flow of soft zones. The deformation of soft intercalates leads to the failure of hard zones at relatively low stress levels. A compressive stress at a large angle to joints increases the stability of the rock mass and inhibits caving. In this case, the adjustment can be up to 120 per cent, i.e. improving the strength of the rock mass.

Minimum Stress

The minimum principal stress plays a significant role in the stabilities of the sides and back of large excavations, the sides of stopes, and the major and minor apexes that protect extraction horizons. The removal of a high horizontal stress on a large stope sidewall will result in relaxation of the ground towards the opening.

Stress Differences

A large difference between maximum and minimum stresses has a significant effect on jointed rock masses, resulting in shearing along the joints. The effect increases as the joint density increases (since more joints will be unfavourably orientated) and also as the joint-condition ratings decrease. The adjustment can be as low as 60 per cent.

Factors in the Assessment of Mining-induced Stress

The following factors should be considered in the assessment of mining-induced stresses:

- drift-induced stresses;
- interaction of closely spaced drifts;
- location of drifts or tunnels close to large stopes;
- abutment stresses, particularly with respect to the direction of advance and orientation of the field stresses (an undercut advancing towards maximum stress ensures good caving but creates high abutment stresses, and *vice versa*);
- uplift;
- point loads from caved ground caused by poor fragmentation;
- removal of restraint to sidewalls and apexes;
- increases in size of mining area causing changes in the geometry;
- massive wedge failures;
- influence of major structures not exposed in the excavation but creating the probability of high toe stresses or failures in the back of the stope;

- presence of intrusives that may retain high stress or shed stress into surrounding, more competent rock.

The total adjustment is from 60 to 120 per cent. To arrive at the adjustment percentage, one must assess the effect of the stresses on the basic parameters and use the total.

Blasting Effects

Blasting creates new fractures and loosens the rock mass, causing movement on joints, so that the following adjustments should be applied:

Technique	*Adjustment, %*
Boring	100
Smooth-wall blasting	97
Good conventional blasting	94
Poor blasting	80.

The 100 per cent adjustment for boring is based on no damage to the walls; however, recent experience with roadheader tunnelling shows that stress deterioration occurs a short distance from the face. This phenomenon is being investigated since good blasting may create a better wall condition.

It should be noted that poor blasting has its greatest effect on narrow pillars and closely spaced drifts owing to the limited amount of unaffected rock.

Summary of Adjustments

Adjustments must recognize the life of the excavation and the time-dependent behaviour of the rock mass:

Parameter	*Possible adjustment, %*
Weathering	30–100
Orientation	63–100
Induced stresses	60–120
Blasting	80–100.

Although the percentages are empirical, the adjustment principle has proved sound and, as such, it forces the designer to allow for these important factors.

STRENGTH OF THE ROCK MASS

The rock-mass strength (RMS) is derived from the IRS and the RMR[5]. The strength of the rock mass cannot be higher than the corrected average IRS of that zone. The IRS has been obtained from the testing of small specimens, but testwork done on large specimens shows that their strengths are 80 per cent of those of small specimens[4]. As the rock mass is a 'large' specimen, the IRS must be reduced to 80 per cent of its value. Thus, the strength of the rock mass would be IRS × 80% if it had no joints! The effect of the joints and its frictional properties is to reduce the strength of the rock mass.

The following procedure is adopted in the calculation of RMS:

- the IRS rating(B) is subtracted from the total rating(A) and, therefore, the balance, i.e. RQD, joint spacing, and condition are a function of the remaining possible rating of 80;
- the IRS(C) is reduced to 80 per cent of its value,

$$\text{RMS} = \frac{(A-B)}{80} \times C \times \frac{80}{100},$$

e.g. if the total rating was 60 with an IRS of 100 MPa and a rating of 10, then

$$\text{RMS} = 100\,\text{MPa} \times \frac{(60-10)}{80} \times \frac{80}{100} = 50\,\text{MPa}.$$

DESIGN STRENGTH OF THE ROCK MASS

The design rock-mass strength (DRMS) is the strength of the unconfined rock mass in a specific mining environment. A mining operation exposes the rock surface, and the concern is with the stability of the zone that surrounds the excavation. The extent of this zone depends on the size of the excavation and, except with mass failure, instability propagates from the rock surface. The size of the rock block will generally define the first zone of instability. Adjustments, which relate to that mining environment, are applied to the RMS to give the DRMS. As the DRMS is in megapascals, it can be related to the mining-induced stresses. Therefore, the adjustments used are those for weathering, orientation, and blasting. For example, if

weathering = 85%, orientation = 75%, blasting = 90%, total = 57%, and RMS = 50, the adjustment = 57% and the DRMS = 50 × 57% = 29 MPa.

Therefore, the rock mass has an unconfined compressive strength of 29 MPa, which can be related to the total stresses.

PRESENTATION OF DATA

The rating data for the rock mass should be plotted on plans and sections as class or sub-class zones. If the A and B sub-divisions are used, the A zones can be coloured full and the B cross-hatched. These plans and sections now provide the basic data for mine design. The layouts are plotted with the adjusted ratings (MRMR), which will highlight potential problem areas or, if the layout has been agreed, the support requirements will be based on the MRMR or DRMS. In the case of the DRMS, the values can be contoured.

Practical Applications

The rock mass can now be described in ratings or in megapascals; in other words, these numbers define the strength of the material in which the mining operation is going to take place. Excavation stability or instability has been related to these numbers. On the mines in which the system has been in operation, its introduction was welcomed by all departments from those dealing with geology to those involved in production.

Within the scope of this paper, the practical applications are described in broad terms to indicate the benefits achieved from the use of this system.

Communication

Communication between various departments has improved since the introduction of the classification system because numbers are used instead of vague descriptive terms. It is well known that the terminology used to describe a particular rock mass by personnel experienced in the mining of good ground is not the same as that used by personnel experienced in the mining of poor ground.

Support Principles

The RMR is taken into consideration in designing support even though the adjusted ratings (MRMR) are used. The reason is that a class 3A adjusted to 5A has reinforcement potential, whereas an *in situ* class 5A has no reinforcing potential.

Support is required to maintain the integrity of the rock mass and to increase the DRMS so that the rock mass can support itself in the given stress environment. The installation must be timed so that the rock mass is not allowed to fail and should therefore be early rather than late. A support system should be designed and agreed before the development stage so that there is interaction between the components of the initial and the final stages. To control deformation and to preserve the integrity of the rock mass, the initial support should be installed concurrently with the advance. The final support caters for the mining-induced stresses.

An integrated support system consists of components that are interactive, and the success of the system depends on the correct installation and the use of the right material. Experience has shown that simple systems correctly installed are more satisfactory than complicated techniques in which the chances of error are higher. The supervisory staff must understand and contribute to the design, and the design staff must recognize the capabilities of the construction crews and any logistical problems. The construction crews should have an understanding of the support principles and the consequences of poor installation.

Layout of Support Guide for Tunnels Using MRMR

Table IX shows how the support techniques, in alphabetical symbols, increase in support pressure as the MRMR decreases. Both the RMR and the MRMR are shown as sub-classes.

TABLE IX
SUPPORT* PRESSURE FOR DECREASING MRMR

	RMR									
MRMR	1A	1B	2A	2B	3A	3B	4A	4B	5A	5B
	← Rock reinforcement—plastic deformation →									
1A										
1B										
2A										
2B	a	a								
3A	b	b	a	a						
3B	b	b	b	b	b	c				
4A	r	r	c	c	c	d	d			
4B				d	e	f	f	c+1		
5A						f/p	h+f/p	h+f/1	h+f/1	
5B							h+f/p	f/p	t	t

* The codes for the various support techniques are given in Table X.

Adjusted ratings must be used in the determination of support requirements. In specialized cases, such as drawpoint tunnels, the attrition effects of the drawn caved rock and secondary blasting must be recognized, in which case the tunnel support shown in Table IX would be supplemented by a massive lining.

The support techniques shown in Table X are examples of a progressive increase in support pressures and are not a complete spectrum of techniques. Where weathering is likely to be a problem, the rock should be sealed on exposure.

TABLE X
SUPPORT TECHNIQUES

Rock reinforcement
- a Local bolting at joint intersections
- b Bolts at 1 m spacing
- c b and straps and mesh if rock is finely jointed
- d b and mesh/steel-fibre reinforced shotcrete bolts as lateral restraint
- e d and straps in contact with or shotcreted in
- f e and cable bolts as reinforcing and lateral restraint
- g f and pinning
- h Spilling
- i Grouting

Rigid lining
- j Timber
- k Rigid steel sets
- l Massive concrete
- m k and concrete

} Low deformation

- n Structurally reinforced concrete

Yielding lining, repair technique, high deformation
- o Yielding steel arches
- p Yielding steel arches set in concrete or shotcrete

Fill
- q Fill

Spalling control
- r Bolts and rope-laced mesh

Rock replacement
- s Rock replaced by stronger material
- t Development avoided if possible

Layout of Support Guide Using the DRMS

The support guide for tunnels using the DRMS and the support techniques of Table IX are shown in Figs. 6 and 7.

Stability and Cavability

The relationship between the ratings adjusted for stability or instability (MRMR) and the size of excavation is shown in Fig. 8. The examples of different situations were taken from operations at the following mines:

- Freda, Gaths, King, Renco, and Shabanie Mines in Zimbabwe
- Andina, Mantos Blancos, and Salvador Mines in Chile
- Bell and Fox Mines in Canada
- Henderson Mine in the USA.

The diagram refers to the stability of the rock arch, which is depicted in three empirical zones:

- a stable zone requiring support only for key blocks or brows, i.e. skin effects
- a transition zone requiring substantial penetrative support and/or pillars, or provision to be made for dilution owing to failure of the intradosal zone,

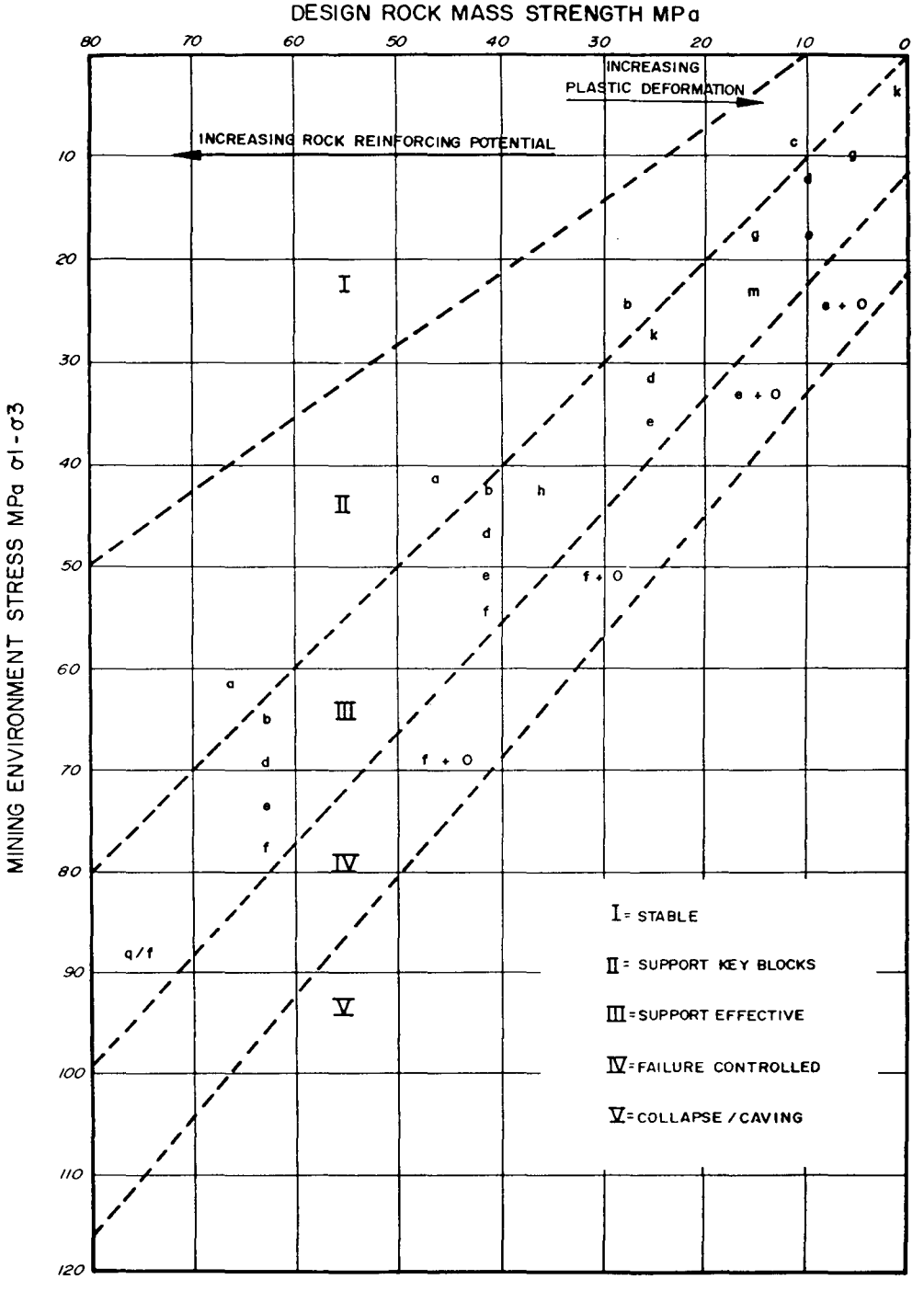

Fig. 6—Support requirements for maximum stress

- a caving or subsidence zone in which caving is propagated provided space is available or subsidence occurs.

The size of the excavation is defined by the 'hydraulic radius' or stability index, which is the plan area divided by the perimeter. Only the plan area is used for excavations where the dip of the stope or cave back is less than 45 degrees. Where the dip is greater than 45 degrees, the area and orientation of the back with respect to the major stress direction must be assessed.

For the same area, the stability index (SI) will vary depending on the relationship between the maximum and the minimum spans. For example, 50 m × 50 m has the same area as 500 m × 5 m, but the SI of the first is 12,5 whereas the SI of the second is only 2,5. The large 50 m × 50 m stope is less stable than a 500 m × 5 m tunnel, and this is well illustrated by the difference in the SI.

Indestructible pillars(regional) reduce the spans so that the SI is applied to individual stopes. Small pillars, as in a post-pillar operation, apply a restraint to the hangingwall, which results in a positive adjustment and, as such, a higher rating, so that the overall stope dimensions can be increased within the dictates of regional stability.

In a room-and-pillar mine, the pillars are designed to ensure regional stability.

The stability or cavability of a rock mass is determined by the extent and orientation of the weaker zones.

There is a distinction between massive and bedded

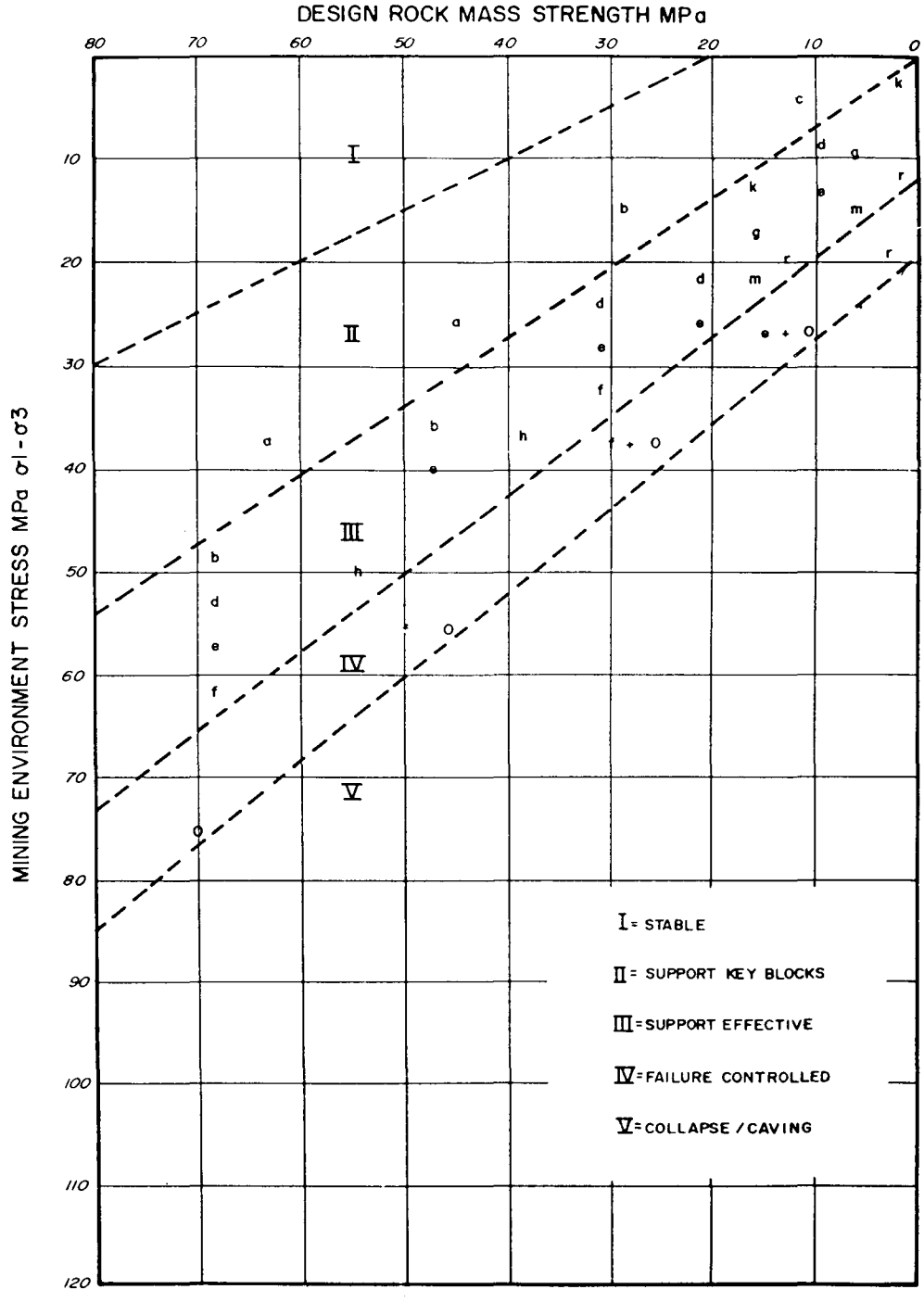

Fig. 7—Support requirements for various stress differences

deposits in that the bedding could be a dominant feature.

Stability of Open Stopes

Large, open stopes are generally mined in competent ground, the size of the stope being related to the criteria for regional stability (Fig. 8). The stability of the stope hangingwall has to be assessed in terms of whether the personnel are to work in the stope or not.

If personnel are to work in the stope, the back must be stable immediately after mining. In order to achieve this stability, potential rock falls need to be identified and dealt with. If the environment and mining rate permit it, or if skilled personnel are available, the support can be designed for the local situation. However, if the mining rate is high, or the identification of potential falls is difficult, a blanket-support design is required.

In the case of open stopes where the activity is from sub-levels outside the stope, the local instability affects the amount of dilution before a stable arch has formed. In the worst situation, the intradosal zone can have a height that is 25 per cent of the span.

By use of a combination of joint-condition ratings and joint-orientation data, a condition/orientation percentage can be derived. These percentages are shown in Table XI.

These percentages can be used to define areas requiring support as follows:

60% – 70% Highly unstable, collapse with blast, requires presupport

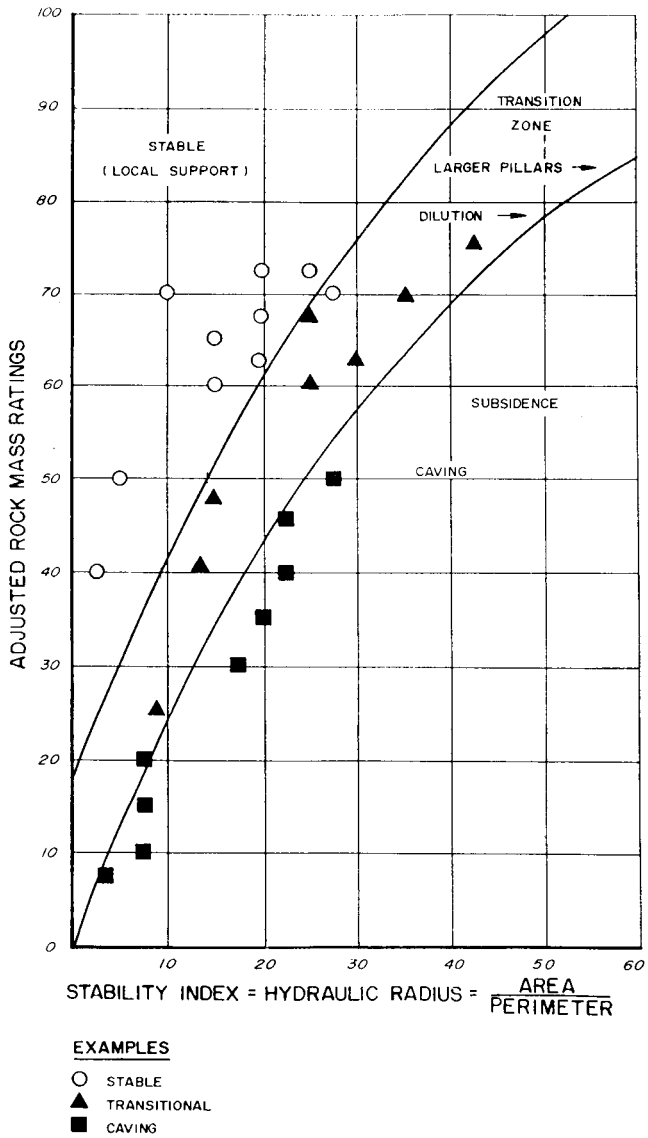

Fig. 8—Stability/instability diagram

70% – 80%	Unstable, time-dependent falls, may require presupport
80% – 90%	Relatively stable, requires support or scaling, even light blasting
90% – 100%	Stable.

In the case of bedded deposits, thinly bedded and massively bedded zones must be rated as distinct units. Bed separation occurs in the thinly bedded zones, while the massive zones contribute to the stability.

Cavability

The joint patterns bear directly on the cavability and fragmentation of the rock mass, and can be used in assessments of whether a cave-mining method can be employed. It is imperative that the hangingwall zone for at least the height of the orebody should be classified. Diagrams like that shown in Fig. 7 are used to define the undercut area for different rock masses.

Fragmentation

Caving results in primary fragmentation, which is the particle size developed in the failure zone of an advancing cave. Primary fragmentation is determined by the stresses in the cave back and by the strength(condition) and orientation of the joints with respect to those stresses. The size of the potential rock blocks is based on the adjusted FF/m in Fig. 4.

Secondary fragmentation is the breaking up of the primary rock block in the draw column. For comminution to occur, the stresses generated must exceed the strength of the rock block, which is unlikely if the block is moving and cushioned by finer material or softer rocks. This is evident in heterogeneous orebodies that contain classes 3, 4, and 5. In these cases, class 4 and 5 zones fragment readily, but class 3 zones arrive at the drawpoint as large blocks even though they contain joints with high joint-condition ratings.

Extent of Cave and Failure Zones

The result of a block cave is the formation of a zone of caved material that has differential rates of movement within boundaries defined by the cave angle. Beyond the cave boundary, a failure zone is developed with fractures(cracks) and limited movement. As shown in Table XII, the strength of the rock mass, the amount of drawdown, and the major structures dictate the angle of the cave and the extent of the failure zone.

Mining Method as Related to MRMR

Table XIII shows how the MRMR varies with mining method.

Pillar Design

Pillars are designed to ensure regional stability or local support in stopes and along drifts, or to yield under a measure of control. In all cases, the strength of the material and the variations in strength must be known both for the pillar and for the roof and floor. The shape of the pillar with respect to structure, blasting, and stresses is significant, and is catered for by the adjustment procedure. For example, for a width-to-height ratio of less than 4,5:1, the following formula uses SI and DRMS[8]:

$$\text{Pillar strength } Ps = k\,\frac{W^{0,5}}{H^{0,7}},$$

where

$$k = \text{DRMS in MPa},\ W = 4 \times \frac{\text{Pillar area}}{\text{Pillar perimeter}}\ (\text{SI}),$$

$$H = \text{height}.$$

Initial Design of Pit Slopes

Table XIV can be used in the design of the initial pit slopes. If the rock mass is homogeneous, the angles shown are comparatively accurate. However, in a heterogeneous rock mass, the classification data of the significant feature must be used. For example, a shear zone dipping into the pit with a rating of 15 would dominate even if the rest of the rock mass had a rating of 50.

OVERVIEW OF THE SYSTEM

Table XV gives an overview of the MRMR system.

TABLE XI
PERCENTAGE ADJUSTMENTS FOR DEGREE OF DIP

Condition rating	Dip from vertical*				Dip towards vertical*			
	0–40°	40–60°	60–80°	80–90°	90–80°	80–60°	60–40°	40–0°
0–10	60	65	70	75	75	80	90	90
11–15	65	70	75	80	80	85	90	100
16–20	70	75	80	85	90	95	100	
21–25	75	80	85	90	95	100		
26–30	80	85	90	95	100			
31–40	85	90	95	100				

* Angles from horizontal.

TABLE XII
THE ANGLE OF CAVE AND THE FAILURE ZONE

	MRMR 1		MRMR 2		MRMR 3		MRMR 4		MRMR 5	
1. Cave Angle										
Depth, m	Unres	Res	Unres	Res	Unres	Res	Unres	Res	Unres	Res
100	70–90	85–95	60–70	75–85	50–60	65–75	40–50	55–65	30–40	45–55
500	70–80	80–90	60–70	70–80	50–60	60–70	40–50	50–60	30–40	40–50
2. Extent of Failure Zone										
Depth, m	Surf.	U/G	Surf	U/G	Surf	U/G	Surf	U/G	Surf	U/G
100	10 m	10 m	20 m	20 m	30 m	30 m	50 m	50 m	75 m	100 m
500	10 m	20 m	20 m	30 m	30 m	50 m	50 m	100 m	75 m	200 m

Unres = No lateral restraint
Res = Lateral restraint
Surf = At surface
U/G = Underground

CONCLUSIONS

The RMR/MRMR classification system has been in use since 1974, during which period it has been refined and applied as a planning tool to numerous mining operations.

It is a comprehensive and versatile system that has widespread acceptance by mining personnel.

The need for accurate sampling cannot be too highly stressed.

There is room for further improvements by the application of practical experience to the empirical tables and charts.

The DRMS system has not had the same exposure but has proved to be a useful back-up tool in difficult planning situations, and has been used successfully in mathematical modelling.

The adjustment concept is very important in that it forces the engineer to recognize the problems associated with the environment with which he is dealing.

ACKNOWLEDGEMENTS

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TABLE XIII
MINING METHOD RELATED TO MRMR

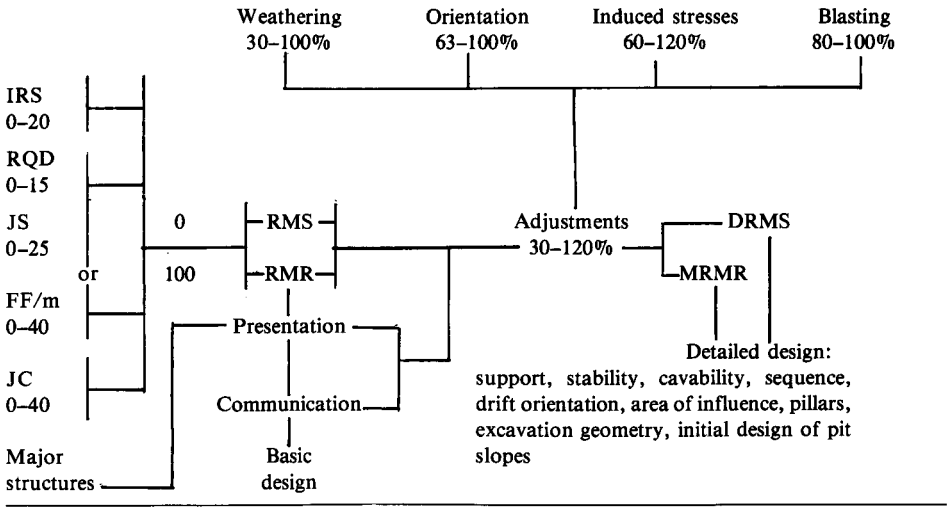
Class rating	5 0–20	4 21–40	3 41–60	2 61–80	1 80–100
Block Caving					
Undercut SI, m	1–8	8–18	18–32	32–50	+50
Cavability	Very good	Good	Fair	Poor	Very poor
Fragmentation, m	0,01–0,3	0,1–2,0	0,4–5	1,5–9	3–20
2nd lay-on blast/drill, g/t	0–50 0–20	50–150 20–60	150–400 60–150	400–700 150–250	+700 +250
Hangups as % of tonnage	0	15	30	45	>60
Dia. of draw zone, m	6–7	8–9	10–11,5	12–13,5	15
Drawpoint span, m					
Grizzly	5–7	7–10	9–12		
Slusher	5–7	7–10	9–12		
LHD, m	9	9–13	11–15	13–18	
Brow support	Steel and concrete Reinf. concrete		Concrete	Blast protection	
Drift support	Lining, rock reinf., repair techniques		Lining, reinf.	Rock reinf.	
Width of point, m	1,5–2,4	2,4–3,5	2,4–4	4	
Direction of advance	Towards low stress		Towards high stress		
Comments	Fine fragmentation, poor ground, heavy support, repairs	Medium fragmentation, good ground, fair support	Medium coarse fragmentation, good drill hangups	Coarse fragmentation, large LHDs, drill hangups	
Sub-level Caving					
Loss of holes	Excessive	Fair	Negligible	Nil	Nil
Brow wear	Excessive	Fair	Low	Nil	Nil
Support	Heavy	Medium	Low	Localized	Nil
Dilution	Very high	High	Medium	Low	Very low
Cave SI, m	1–8	8–18	18–32	32–50	+50
Comments	Not practicable	Applicable	Suitable	Suitable	Suitable, large HW cave area
Sub-level Open Stoping					
Minimum span, m	1–5	5–20	20–30	30–80	100
Stable area, i.e. SI, m	N/A	1–8	8–16	16–35	+35

N/A = Not available

TABLE XIV
APPROXIMATE ANGLES OF PIT SLOPES

Adjusted class	1	2	3	4	5
Slope angle	75	65	55	45	35

TABLE XV
OVERVIEW OF THE MRMR SYSTEM



Technology development——
Innovation in South African equipment*

The synergy of South African research and manufacturing has resulted in several highly innovative metallurgical devices in recent years.

Carbon-concentration Meter

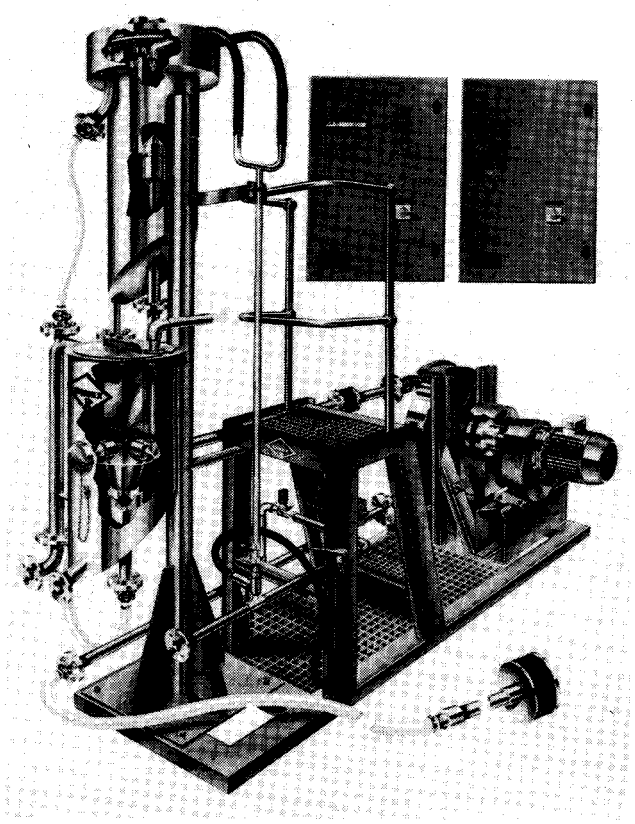
The latest of these, the ultrasound-based carbon-concentration meter, was developed by Mintek with original sponsorship by the Chamber of Mines Research Organization, and is now being manufactured and marketed worldwide by Debex Electronics in Johannesburg.

Designed to achieve precise on-line measurement of critical carbon-concentration levels during the gold-recovery process, the novel instrument makes possible significant improvements and cost savings in metallurgical gold-recovery plants, and has excited international interest as a new and valuable metallurgical tool.

During the carbon-in-pulp gold-recovery process, carbon granules are added to the gold slurry, which follows the initial cyanide-leaching stage. The gold–cyanide complex within the slurry is deposited onto the carbon granules, and the carbon-concentration level is therefore critically important for optimum gold recovery.

Before the development of the new instrument, there was no way of continuously and accurately assessing this level through the six to eight absorption stages involved, since carbon-in-pulp is pumped from one tank to another in counter flow to the flow of pulp or slurry.

Günter Sommer, the Director of the Measurement and Control Division at Mintek, says that the carbon-concentration meter is an international first and was developed



The Debmeter system. On the left, cutaway views of the de-aerator tank and ultrasonic transducer array system. The slurry presentation system is on the right of the drawing.

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