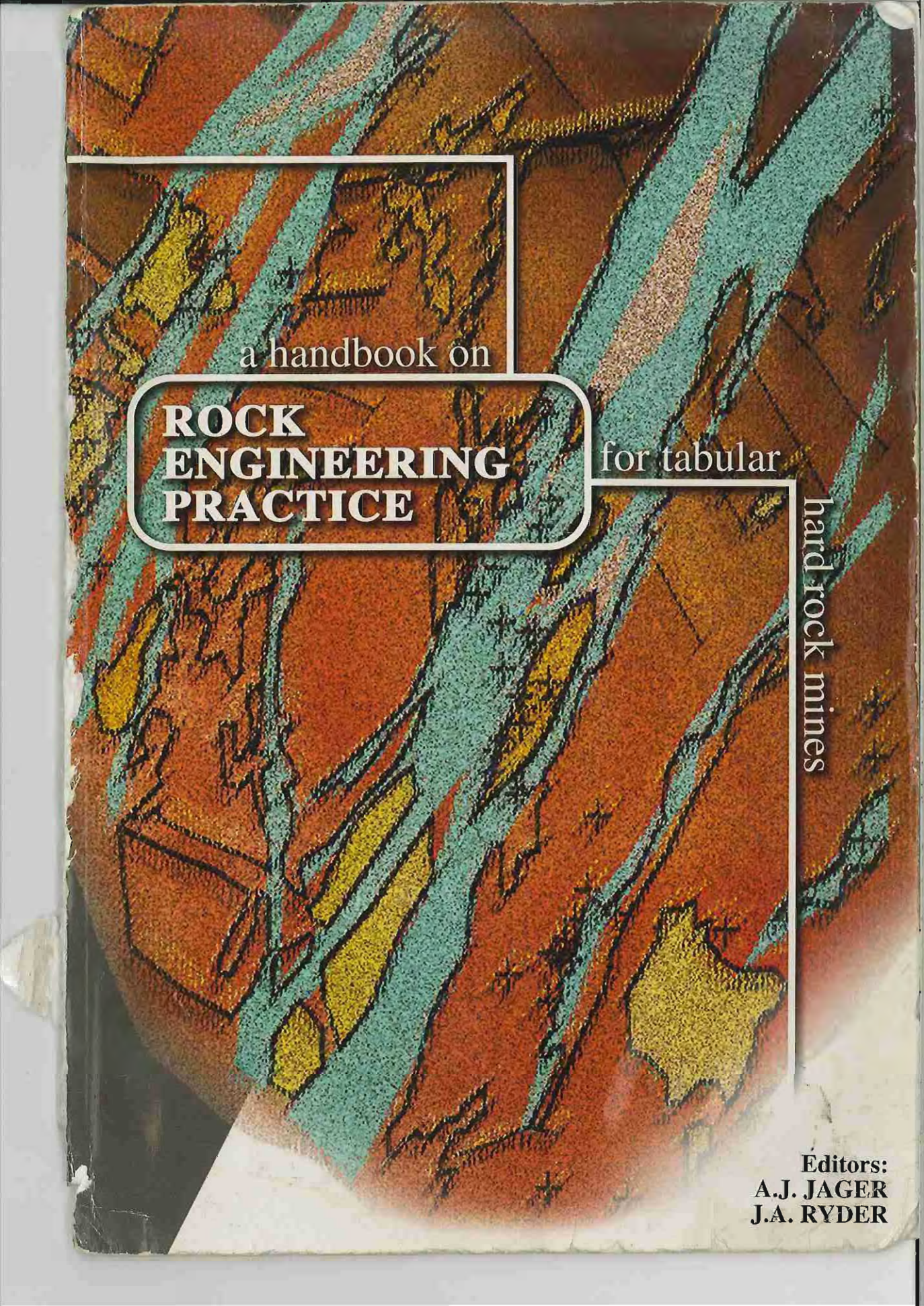


The background of the book cover is a detailed geological map. It features a complex network of lines representing geological structures, with areas shaded in various colors including red, orange, yellow, and teal. The map is oriented diagonally, with the top-left corner showing a more detailed view of a specific geological feature.

a handbook on

**ROCK
ENGINEERING
PRACTICE**

for tabular

hard rock mines

Editors:
A.J. JAGER
J.A. RYDER

A HANDBOOK ON
ROCK ENGINEERING PRACTICE
FOR TABULAR HARD ROCK MINES

Editors A J Jager, J A Ryder

The Safety in Mines Research Advisory Committee (SIMRAC)
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FOREWORD

The most recent South African books dealing with rock mechanics and rock engineering in tabular hard rock mining have been "Rock Mechanics in Mining Practice" (S. Budavari) and "An Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts" (Chamber of Mines Research Organization), published in 1983 and 1988 respectively.

Since then, advances have occurred in rock engineering practice, and significant changes have also arisen in the regulation of the South African mining industry. In 1994, the Leon "Commission of Inquiry into Safety and Health in the Mining Industry" sat to examine evidence and make recommendations as to changes it saw necessary. In 1996, the new Mine Health and Safety Act was promulgated and, as with subsequent statutory regulations, account was taken of the recommendations of this Commission. Among many issues, an important provision was that it is legally incumbent on mine management to take all "reasonably practicable" measures to remove or mitigate the hazards that are present on their mines, fully utilizing the "state of knowledge reasonably available". A further significant new requirement is that mines must compile codes of practice to combat rockfall and rockburst accidents, according to guidelines provided by the Department of Minerals and Energy. In addition, a strong focus throughout South Africa, and no less in the mining industry, is being brought to bear on effective education and training of all employees.

A number of new statutory bodies have also been introduced, including SIMRAC (the Safety in Mines Research Advisory Committee). This tripartite body (government, labour and mineowners) was tasked to advise the minister on mining safety and health research requirements, and to administer and control the annual research programme. Funding for this research, which in 1998 amounted to some R 38 million, is provided by a levy collected from all mines. Taking into account the new statutory requirements, as well as the numerous advances that SIMRAC and other research and development endeavours have introduced to the mining industry in recent years, the GAPREAG (Gold and Platinum Rock Engineering Advisory Group of SIMRAC), considered it necessary that up-to-date reference books on rock mechanics and rock engineering be written. In particular the existence of the latter text, covering sound rock engineering practice on both gold and platinum mines, would assist mine management in meeting their new obligations. The CSIR: Division of Mining Technology (MiningTek) was commissioned to carry out this task.

Insofar as rock engineering is a combination of science, art and engineering, and deals with rock masses that can vary significantly from place to place and reef to reef, it is not possible for recommendations and descriptions of best practice in this text to be fully prescriptive. Moreover, in dealing effectively with the vagaries of underground conditions, there is always considerable room and need for innovation and controlled experimentation. However, where mines choose to employ practices or design methods which are considered appropriate for particular situations but which differ significantly from what is presented in this text, these should best be *motivated* and recorded in either the mine's code of practice or in specific planning documentation.

K R Noble
Chairman, GAPREAG

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- To the many authors for their contribution of source material. The editors of this book were the principal authors of **Chapters 1 to 4**. Further original text was received from the following contributors (their affiliation, unless otherwise indicated, being CSIR: MiningTek):

Chapter 3 Dr T O Hagan, Prof M U Ozbay (University of the Witwatersrand), G York, F Vieira, Dr R G Gurtunca, R More O'Ferrall (Consultant), D A Arnold (Avmin), R A Johnson, D J Adams, Dr S M Spottiswoode, Prof E Esterhuizen (University of Pretoria).

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Chapter 10 Dr A P Squelch.

Chapter 11 N Lightfoot.

Chapter 12 R A Johnson, D J Adams.

- Where little change has occurred to existing practice, substantial input was drawn from the 'Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts' (COMRO, 1988).
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CONTENTS

Foreword and Acknowledgements	iii
Chapter 1 OVERVIEW OF THE ROCK ENGINEERING CHALLENGE	1
1.1 INTRODUCTION	1
1.2 STATISTICS OF THE ROCKFALL/ROCKBURST PROBLEM	2
1.2.1 Casualty Trends as a Function of Time	3
1.2.2 Casualty Trends with Depth	6
1.2.3 Location of Accidents	7
1.3 FACTORS GOVERNING ROCK BEHAVIOUR	10
1.3.1 Rock Stresses	10
1.3.2 Rock Strength and Friction Properties	13
1.3.3 Rock Types	14
1.3.4 Geological Weaknesses	16
1.3.5 Stope and Tunnel Fracturing and Deformation Patterns	16
1.3.6 Time-Dependent Fracturing and Deformation	20
1.4 SEISMICITY AND ROCKBURSTING	21
1.4.1 Quantitative Seismology and Rockbursts	21
1.4.2 Seismicity Patterns	22
1.4.3 Seismic Event Mechanisms	24
1.4.4 Strong Ground Motion and Support Design	25
1.5 CONCLUSIONS	26
Chapter 2 OVERVIEW OF ROCK ENGINEERING STRATEGIES	29
2.1 INTRODUCTION	29
2.2 SHALLOW MINING (Typically <1000 m depth)	31
2.2.1 Shallow Mining Environment	31
2.2.2 Shallow Mining Strategies	32
2.3 MEDIUM DEPTH (SCATTERED) MINING	33
2.3.1 Medium Depth Mining Environment	33
2.3.2 Medium Depth (Scattered) Mining Strategies	34
2.4 DEEP & ULTRA-DEEP MINING	35
2.4.1 Deep and Ultra Deep Mining Environment	35
2.4.2 Deep Mining Strategies	36
2.4.3 Ultra-Deep Mining Strategies	37
2.5 GENERAL ROCK ENGINEERING STRATEGIES	38
2.5.1 Seismic Monitoring	38
2.5.2 Advance Exploration of Geological Structures	39
2.5.3 Ground Control Districts	40
2.5.4 Risk Assessments	40
2.5.5 Auditing	40
2.5.6 Support Quality Control	41
2.5.7 Training	41
2.6 CODES OF PRACTICE	41
Chapter 3 STOPING LAYOUTS	45
3.1 INTRODUCTION	45
3.2 MINE LAYOUT DESIGN CRITERIA	46
3.2.1 Energy Release Rate (ERR)	46
3.2.2 ERR: A Measure of Rockburst Hazard in Longwall Mining	50
3.2.3 Excess Shear Stress (ESS)	51
3.2.4 Volume Excess Shear Stress (VESS)	53
3.2.5 Rockburst Hazard Index (RHI)	53
3.2.6 Face Shape Index (FSI)	55
3.2.7 Average Pillar Stress (APS)	55

3.2.8	Pillar Strength Criteria	56
3.2.9	The RCF Criterion for Tunnel Layout and Support Design	58
3.3	REGIONAL SUPPORT	60
3.3.1	Water-Barrier and Boundary Pillars	60
3.3.2	Stabilizing Pillars	61
3.3.3	Bracket Pillars	65
3.3.4	Concrete Pillars	67
3.3.5	Backfill as a Support Medium	69
3.4	STANDARD LAYOUTS	76
3.4.1	Shallow/ Overstoped Areas	76
3.4.2	Medium Depth (Scattered) Mining	82
3.4.3	Deep Longwall Mining	86
3.4.4	Deep 'Sequential Grid' Mining	90
3.4.5	Deep (and Ultra Deep) 'Room and Pillar' Mining	92
3.4.6	Ledging and Raising	93
3.4.7	Gully Layouts	94
3.5	LAYOUTS IN SPECIAL MINING AREAS	101
3.5.1	Remnant Extraction	101
3.5.2	Mining Adjacent to and Negotiation of Geological Structures	102
3.5.3	Re-Establishing Panels	104
3.5.4	Mining in Anomalous Stress Conditions	105
3.5.5	Caving	105
3.6	LAYOUTS FOR SPECIAL OREBODY GEOMETRIES	107
3.6.1	Wide Reef Mining Layouts	107
3.6.2	Multi-Reef Mining Layouts	109
3.6.3	Steep Mining Layouts	111
Chapter 4 STOPE SUPPORT		113
4.1	INTRODUCTION	113
4.2	FACTORS GOVERNING THE DESIGN OF STOPE SUPPORT SYSTEMS	114
4.2.1	Mining Depth	115
4.2.2	Minor Geological Features	125
4.2.3	Rock Mass Deformation	130
4.2.4	Mining Environments and Ground Control Districts	133
4.2.5	Characteristics of Support Units	137
4.3	SUPPORT SYSTEMS - GENERAL CHARACTERISTICS	141
4.3.1	General Definitions	141
4.3.2	Area Coverage of Support Systems and Support Spacing	143
4.3.3	Support Resistance and Support Spacing	144
4.3.4	Energy Absorption	147
4.4	DESIGN AND EVALUATION OF SUPPORT SYSTEMS	147
4.4.1	Setting of Design Criteria	147
4.4.2	Establishment of Closure Rates for Geotechnical Areas	151
4.4.3	Support Design Methodology for Rockfall Conditions	151
4.4.4	Support Design Methodology for Rockburst Conditions	156
4.4.5	Cost Analysis, Risk Analysis and Factor of Safety Evaluation	161
4.4.6	Field Trials	162
4.4.7	Implementation	162
4.4.8	Backfill as Local Support	163
4.4.9	Gully Support	167
4.4.10	Application of RYHPs & Prestressable Elongates	169
4.5	EXAMPLES OF PANEL SUPPORT SYSTEMS	172
4.5.1	Reef Pillars with Simple Timber Prop Panel Support	172
4.5.2	Timber-Based Elongate Support Systems	174
4.5.3	Pack Support Systems	176

4.5.4	Backfill Systems	178
4.5.5	Cave Mining Support Systems	180
4.5.6	RYHP System with Diagonal Blast Barricades	181
4.6	STOPE SUPPORT IN SPECIAL MINING SITUATIONS	183
4.6.1	Support for Remnant Conditions	183
4.6.2	Support Near Geological Structures	183
4.6.3	Stopped Panel Support	184
4.6.4	Support While Re-establishing Panels	184
4.7	STOPE SUPPORT FOR SPECIAL OREBODY GEOMETRIES	185
4.7.1	Wide Reef Support	185
4.7.2	Multi-Reef Stope Support	186
4.7.3	Steep Stope Support	190
4.8	STOPE SUPPORT – QUALITY CONTROL	191
Chapter 5	TUNNEL LAYOUTS	193
5.1	INTRODUCTION	193
5.2	ROCK MASS ENVIRONMENT	193
5.2.1	Rock Mass Strength Characteristics	194
5.2.2	Stress Environment	195
5.3	EXCAVATION SIZE AND SHAPE	196
5.3.1	Excavation Shape	197
5.3.2	Excavation Size	198
5.4	EXCAVATION POSITIONING	200
5.4.1	Orientation	200
5.4.2	Excavation Interaction	201
5.4.3	Seismic Hazards	203
5.5	EXCAVATION TECHNIQUE	204
5.5.1	Blasting Practice	204
Chapter 6	TUNNEL SUPPORT	207
6.1	INTRODUCTION	207
6.2	SUPPORT SYSTEM CHARACTERISTICS	208
6.3	DESIGN CONSIDERATIONS	214
6.3.1	Geotechnical Controls	214
6.3.2	Stress Environment	215
6.3.3	Seismicity	215
6.4	DESIGN METHODOLOGIES	216
6.4.1	Structural/Mechanistic Analyses	216
6.4.2	Rock Mass Classification Systems	218
6.4.3	Empirical Design Rules	221
6.4.4	Design Philosophy	221
6.5	SPECIAL CONSIDERATIONS	222
6.5.1	Corrosion	222
6.5.2	Rehabilitation	222
6.5.3	Support Quality Control	222
Chapter 7	LARGE CHAMBERS AND SHAFTS	225
7.1	INTRODUCTION	225
7.2	LARGE UNDERGROUND CHAMBERS	225
7.2.1	Siting of Major Excavations	225
7.2.2	Stress Environment and Effect of Subsequent Mining	226
7.2.3	Shape and Size	227
7.2.4	Excavation Sequence	228
7.2.5	Blasting	232
7.2.6	Support	232

7.3	SHAFTS	235
7.3.1	Vertical Shaft Design	236
7.3.2	Vertical Shaft Sinking and Development	240
7.3.3	Orepasses	241
7.3.4	Incline Shafts	244
7.3.5	Early Removal of the Shaft-Reef (Vertical Shafts)	244
7.3.6	Shaft Pillar Extraction	245
Chapter 8	ROCKBURSTING	249
8.1	INTRODUCTION	249
8.2	ROCKBURSTS – DEFINITIONS	249
8.2.1	Rockbursts Classified in terms of the Source Mechanism	250
8.2.2	Rockbursts Classified in terms of the Damage Mechanism	251
8.3	DISTRIBUTION AND INTENSITY OF ROCKBURST DAMAGE	253
8.3.1	Rockburst Investigation Methodology	253
8.3.2	Case History 1: Pillar Burst	253
8.3.3	Case History 2: Rockburst Damage to Tunnels	258
8.3.4	Case History 3: Shakedown Burst, Slip on Dyke	264
8.3.5	Case History 4: Shakedown Burst, Shaft Pillar Failure	271
8.3.6	Synthesis of Rockburst Investigations	276
8.3.7	Concluding Comments	281
8.4	ROCKBURST CONTROL: Preconditioning	281
8.5	ROCKBURST CONTROL: Prediction of Large Rock Mass Instabilities	285
Chapter 9	A GUIDE TO ROUTINE SEISMIC MONITORING IN MINES	287
9.1	SEISMIC EVENT, SEISMICITY AND STATE OF THE ROCK MASS	287
9.2	OBJECTIVES OF SEISMIC MONITORING IN MINES	289
9.3	LOCATION OF SEISMIC EVENTS	290
9.4	QUANTIFICATION OF SEISMIC SOURCES	291
9.5	SEISMIC HAZARD AND SEISMIC EXPOSURE	293
9.6	GENERAL GUIDELINES FOR INTERPRETATION	299
9.7	CHARACTERISTICS OF SEISMIC MONITORING SYSTEMS	303
9.8	LIMITATIONS OF MONITORING AND FUTURE DEVELOPMENT	308
Chapter 10	MONITORING AND AUDITING	311
10.1	INTRODUCTION	311
10.2	WHY CONDUCT UNDERGROUND MONITORING?	311
10.3	WHAT CAN BE MONITORED?	312
10.4	INSTRUMENTATION ACCURACY AND ERRORS	316
10.5	WHAT TO MONITOR, WHERE AND WHY	317
10.5.1	Stopes	317
10.5.2	Tunnels	320
10.5.3	Shafts	321
10.5.4	Large Service Excavations	322
10.6	INSTRUMENTATION AND ANALYSIS	322
10.7	ROCK ENGINEERING AUDITING	326
10.7.1	Technology Auditing	326
10.7.2	Skills Auditing	327
Chapter 11	NUMERICAL MODELLING	329
11.1	INTRODUCTION	329
11.2	ASPECTS OF NUMERICAL MODELS	331
11.2.1	Eulerian and Lagrangian Analyses	331
11.2.2	Dimensionality	332
11.2.3	Disturbing the Equilibrium – Static & Dynamic Analyses	332

11.2.4 Analytical and Numerical Methods	333
11.3 NUMERICAL MODELLING METHODS	334
11.3.1 The Finite Difference Method	335
11.3.2 The Finite Element Method	336
11.3.3 The Distinct Element Method	336
11.3.4 The Fictitious Stress Method	336
11.3.5 The Displacement Discontinuity Method	337
11.3.6 The Direct Boundary Integral Method	338
11.3.7 Hybrids	338
11.3.8 Keyblock Theory	338
11.4 PRACTICAL ASPECTS	338
11.4.1 Discretisation, Iteration and Convergence	338
11.4.2 Design Criteria	339
11.4.3 Application of Numerical Models	339
11.4.4 Specific Numerical Modelling Packages & Programs	340
11.4.5 Program Overview	344
11.4.6 A Modelling Methodology	344
Chapter 12 ROCK ENGINEERING TRAINING REQUIREMENTS	347
12.1 INTRODUCTION	347
12.2 ALL PERSONS WHO GO UNDERGROUND	348
12.3 STAFF WHOSE PRIMARY FUNCTION IS STOPING OR TUNNELLING	349
12.4 MINERS AND SHIFT BOSSES	351
12.5 SENIOR OFFICIALS	353
12.6 ROCK ENGINEERING PRACTITIONERS	356
12.7 IMPLEMENTATION OF NEW TECHNOLOGIES	356
12.8 RISK ASSESSMENTS	358
Glossary of Terms and Index	361
Glossary 1. Tunnel support terminology	361
Glossary 2. Parameters used in seismic monitoring and hazard assessment	363
Index	367



OVERVIEW OF THE ROCK ENGINEERING CHALLENGE

1.1 INTRODUCTION

It is a truism that underground mining - the art of extracting minerals from deep within the Earth's crust - is among the more intrinsically hazardous, and technically challenging, of the varied activities of mankind. This is largely because (unlike in ship-building, or in bridge design, for example) the basic engineering materials involved - the heterogeneous fabric of the rock mass in which mining has to take place - is pre-ordained by nature and is thus not under the *direct* control of the mining engineer.

South Africa is a country richly endowed with minerals of many kinds, including the world's largest deposits of gold, platinum, chromium and manganese. It ranks as the world's largest producer from **underground tabular hard rock** deposits, and the rock engineering aspects of this particular branch of mining engineering constitute the subject matter of this book.

Significant innovations have emerged from South African mining operations. These have included the development of expertise in deep shaft sinking and equipping, mining refrigeration technology, the hydro-power concept and systems, effective rescue capabilities, amongst many others. In the field of rock engineering, which in the 1950s began evolving from simple empiricism towards becoming a much more scientific and quantitative engineering discipline, significant advances were also made. These included analytical/numerical methods for better understanding and modelling the stress and energy changes associated with mining (and hence providing means for the more rational design of mining layouts), quantifying the strength and stability of coal/reef pillars in shallow mines, exploiting the concept of regional support in the form of stabilizing pillars in deep mines, implementing the placement of backfill over vast areas in tabular stopes, installing district-wide mine seismic networks, developing and implementing innovative forms of yielding and other rockburst-resistant support systems in stopes and tunnels. These developments were aimed not only at improving mining productivity, but primarily towards reducing the hazards of underground tabular mining.

Yet over the years, the South African hard rock mining industry has sustained, and continues to sustain, a high level of accidents, and the resulting rate of human casualties (injuries and deaths) has been, in world-wide terms, unacceptably high. Of these accidents, a high proportion (over 50 %) have been *rock-related*, that is, the result of rockfalls or of rockbursts.

There are underlying reasons for these high rock-related hazards. Many were identified by the 1994 Leon 'Commission of Inquiry into Safety and Health in the Mining Industry':

- The bulk of South African hard rock mining is carried out in *narrow tabular* orebodies with mining heights less than 1.5 m. The mines are large and operations are widespread: each year, some thirty million square metres of ground are extracted, accessed by some 800 km of newly developed tunnels in addition to the 10 000 km currently in use. While high standards of strata control and support are aimed at, the vast areas involved lead to deficiencies in implementation, supervision and quality control which do not always ensure that these standards are met.
- The hard/abrasive rock types involved, together with the restricted working heights, has so far thwarted the development of effective bulk mechanized mining systems and as a result, South African hard rock mining is still *labour-intensive* with some half-million men employed in underground work. This huge labour force is deployed over the vast areas already mentioned, and is therefore continuously exposed to the hazards of tabular mining - hazards which continually change as faces advance and fresh geological structures are encountered.
- The host rocks are generally strong, but *brittle*. In all medium and deep mines, the exposed rockwalls are highly *fractured* and this facilitates hazardous falls of ground.
- *Weak bedding partings or incompetent strata* in the immediate hangingwall, encountered in various localities, also promote falls of ground and these hazards are not always immediately obvious to casual inspection.
- The *heavy faulting* encountered in many of South Africa's mining districts generates seismicity, particularly once large areas of ground have been extracted. Some of the largest mining seismic events recorded ($M > 5$) have occurred in such districts. Moreover, the negotiation of faulting often poses severe rock engineering problems, with potential increases in the hazards of falls of ground or of damaging seismicity.
- All deep and ultra-deep mines are subjected to very high states of rock *stress* with concomitant problems, the most serious being the high incidence of seismicity and severe *rockbursting*.

Most of these problems are, in scale and intensity, **unique**. In fact, the long-term viability of South African mining will revolve around the further development and stricter application of practical and cost-effective procedures aimed at **reducing these rock-related hazards**.

It is appropriate to begin this book with a detailed review of the severity of current South African safety-related problems. Also covered in this chapter are the levels of virgin rock stresses to be contended with; and some basic properties of the rock types and geological weaknesses, typical fracturing and deformation patterns, and seismicity levels commonly encountered. These issues together constitute the **rock engineering challenge** confronting mining in South African hard rock conditions. Practical technical and managerial strategies aimed at addressing this challenge are summarized in Chapter 2 and are dealt with in further detail in the balance of this book.

1.2 STATISTICS OF THE ROCKFALL / ROCKBURST PROBLEM

In this section, most attention will be focused on the South African gold mining industry, which not only has higher total casualties than the other hard rock mines,

but also experiences more severe hazards, notably a high incidence of rockbursting in the deeper gold mines. Some figures relating to rockfall hazards in the platinum mines are nevertheless included for purposes of comparison.

Figures relating to rock-related casualties are available from the turn of the 20th century, but these early statistics have little relevance to modern conditions and will not be discussed here. Over the past eight years (1990-1997) on the gold mines, **rockfall and rockburst fatalities** have accounted for about one-half of the total that occurred for any reasons (that is, some 200 rock-related deaths *each year*, out of a total of about 400). Of these, 57 % were attributed to rockfalls and 43 % to rockbursts. During this period the total workforce declined from 456 000 to 317 000, and the proportion of underground labour increased slightly from 76 % to 79 %.

1.2.1 Casualty Trends As A Function Of Time

The **fatality rates** (conventionally expressed as the number of fatalities per 1000 people in service per annum) are depicted in Figure 1.2.1a for the combined rockfall and rockburst components in the South African **goldfields**. Between 1970 and 1997 this rate has fluctuated around a value of about 0.7 per thousand in service.

The rate expressed per 1000 employees working underground more realistically represents the *overall risk* from rockfalls and rockbursts. The combined (rockfall + rockburst) rate is plotted as the upper curve in Figure 1.2.1a; values fluctuating around 0.85 per thousand employed underground are indicated. The subdivision into rockfall and rockburst (underground) rates is shown in the lower two curves. The rockfall rate shows no declining trend and averages about 0.5. This statistic is disappointing because, with improved technologies and methodologies available since about 1993, it could have been expected that the relatively easier rockfall problem would have shown an improvement. The rockburst rate likewise shows no significant trends, and has fluctuated around the value of 0.35 per thousand employed underground.

Figure 1.2.1b shows the total (*rockfall*) fatality rates experienced in the South African **platinum** mines, and exhibits no clear trends [serious *rockburst* incidents in the platinum and chrome mines of the Bushveld mining district are currently extremely rare, though this could well change in the long term if mining depths increase significantly]. The average rockfall fatality rate for underground personnel is about 0.35, somewhat lower than the average for the gold mining industry as a whole, but no better than for some of the individual gold mines or goldfields.

Figure 1.2.2a is the same type of graph as Figure 1.2.1a but depicts **reportable injury rates** (i.e. injuries resulting in off-work periods of greater than 13 days) in the **goldfields**. Unlike the situation for fatalities, the graph shows a commendable 60 % reduction in the rock-related injury rate between 1974 and 1986. Since then the rate

has been roughly steady at about 6.0 per 1000 in service, i.e. about 8.0 per 1000 employed in underground work. This means that in the early 1970s the rock-related injury rate was 20 times that of the fatality rate, while currently that figure has reduced to approximately 10. The ratio of rockburst injuries to that of rockfalls in the goldfields is 0.28 : 1 compared to the ratio of fatalities of 0.76 : 1. These figures indicate that the severity of the consequences of rockbursts in the goldfields is currently about three times that of rockfalls.

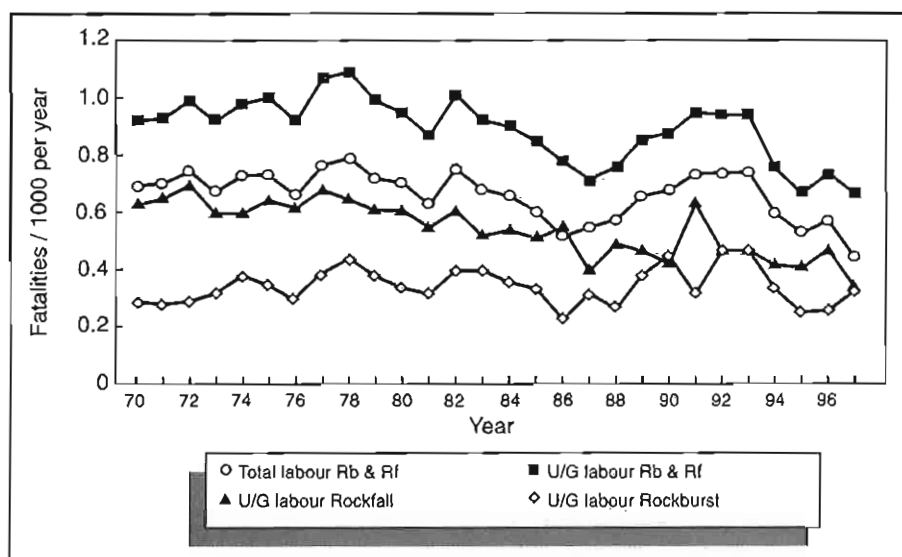


Figure 1.2.1a Fatality rate trends with time in the South African gold mines

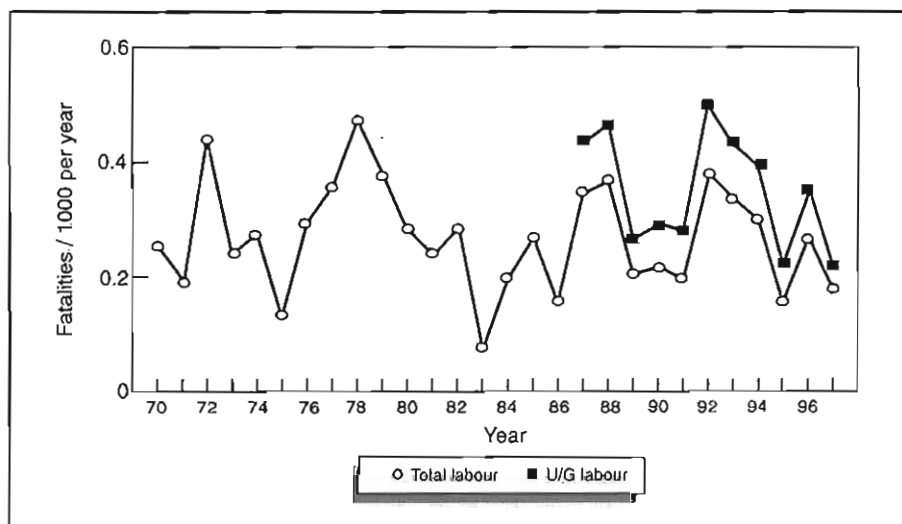


Figure 1.2.1b Fatality rate trends with time in the South African platinum mines.
(Underground labour complements available only subsequent to 1987).

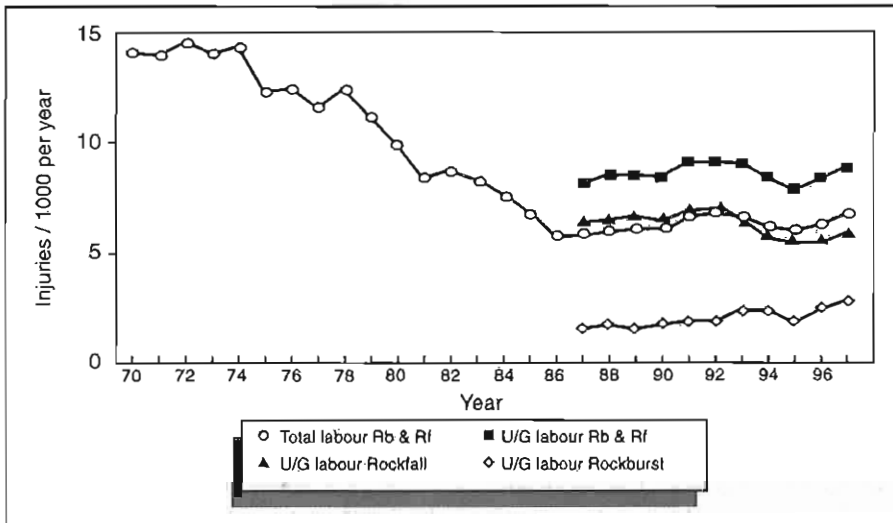


Figure 1.2.2a Injury rate trends with time in the South African **gold** mines

The injury rates in the **platinum** mines, Figure 1.2.2b, show essentially similar trends. The current (rockfall) injury rate for underground personnel is about 3.0, which is significantly lower than the comparable figure of about 6.0 for the goldfields. This can be partly attributed to the more intense fracturing to be found in the rockwalls of gold mine excavations, and the greater presence of weak bedding partings in the immediate hangingwall.

The figures discussed so far illustrate only the *overall* severity of rock-related hazards. The following sub-sections attempt more detailed analyses so as to focus attention on those situations which require particularly concentrated safety-related action. These include the evident strong increase in rockburst hazards with mining depth, and the clear concentration of severe accidents close to the working stope faces.



Figure 1.2.2b Injury rate trends with time in the South African **platinum** mines

1.2.2 Casualty Trends With Depth

Data pertaining to the proportion of production coming from various **depth** ranges are not readily available, nor are figures routinely collated for the number of people at risk at various mining depths. The lack of these and similar data was identified in a recent industry-wide high-level risk assessment to be an area requiring attention. Their availability would allow the root cause of accidents to be better assessed, the effectiveness of implementation of new strategies to be better tracked, and areas for research and development to be better identified. Currently, the influence of depth on the accident rate has to be evaluated indirectly.

An analysis of data in the CSIR-Miningtek/SIMRAC accident data base, populated with information gleaned from all reports made at fatal accident inquiries since 1990, is plotted in Figure 1.2.3. The rockfall and rockburst fatality rates normalized with respect to production are shown for each goldfield, plotted in sequence of increasing average mining depth. This clearly shows a relative independence of *rockfall* fatalities with depth, and contradicts the common (but incorrect) view that poor ground conditions correlate directly with depth and Energy Release Rate (ERR) - c.f. Chapter 3.2.1. However, there is a strong increase in *rockburst* fatalities with depth. Confirming data are shown in Figure 1.2.4 in which the proportion of rockburst to rockfall fatalities (together with absolute numbers) is plotted as a function of depth. The clear increase in this proportion confirms the significantly *increasing rockburst hazard with depth*. Further confirmation can be found from the Chamber of Mines safety competition statistics which indicate that their 'ultra-deep' mines (those principally mining at depths exceeding 2100 m) suffer rockburst fatality rates *in excess of 0.8 per thousand at work*, whereas their 'shallow' and 'deep' mines experience rates of 0.25 or less.

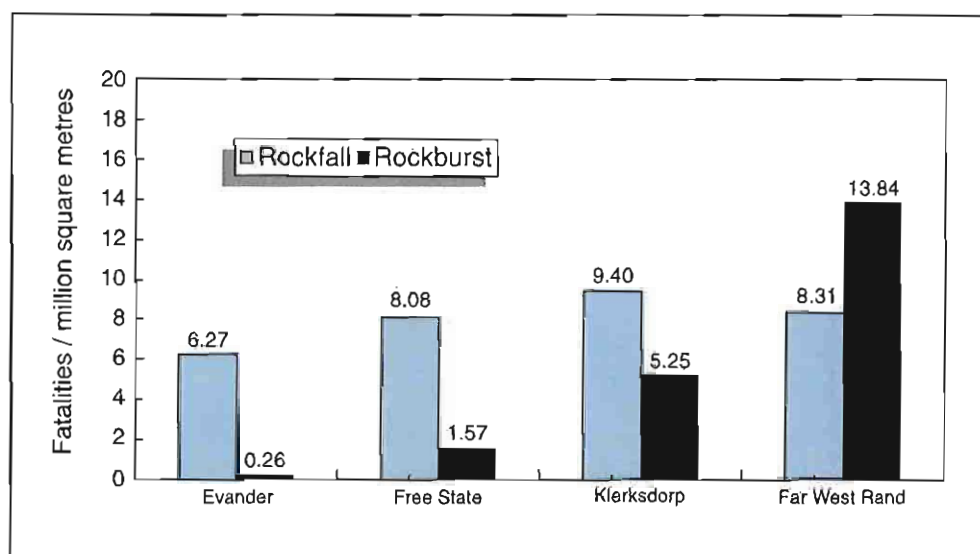


Figure 1.2.3 Fatality rates in South African gold mining districts, plotted in order of increasing mining depth (from 'intermediate' to mainly 'deep'). Period 1990-97.

Since gold mining is continuing to proceed to ever-increasing depths, these statistics clearly pose a formidable and immediate challenge to the industry: to devise and implement significantly improved layout, support and mining methodologies aimed at reducing and containing these deep-level rockburst hazards.

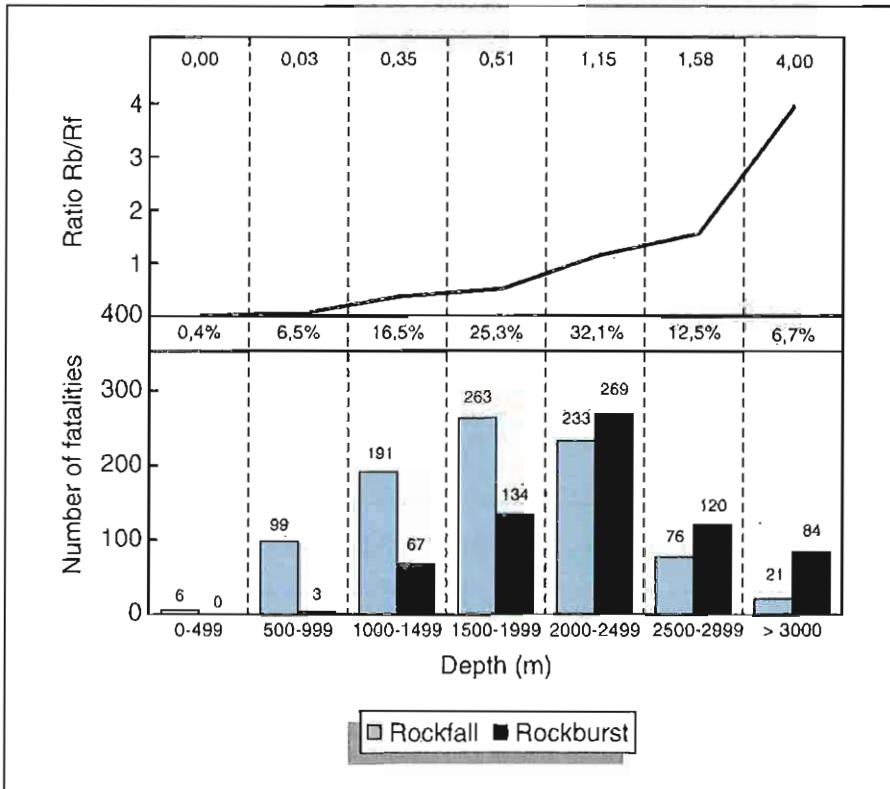


Figure 1.2.4 Rockburst and rockfall fatalities, and their ratio, gold mining 1990-97.

1.2.3 Location Of Accidents

The **locations** where rockfall and rockburst casualties occurred (in the goldfields) are analyzed in Figure 1.2.5a. Quite clearly, the dominant risk area for both rockfall and rockburst accidents was the **stope face area** (within 4 m of the face, where the majority of work is carried out) - Figure 1.2.5b. This highlights the overwhelming importance of maintaining sound *face area ground control procedures* and *support* at all times.

Figure 1.2.6a shows the difference in rock-related fatality rates for different **reefs**. These differences reflect important variations in geotechnical conditions, in addition to depth differences and other factors. They clearly motivate the use of specific layout and support standards (for different '*mining environments*' and '*ground control districts*' in a mine, see chapter 4.2.4) which are tailored to individual requirements and conditions. Figure 1.2.6b is a similar plot for two neighbouring mines mining in essentially the same geotechnical conditions. The differences here reflect, from a

rock engineering point of view, mainly differences in support and mine layout strategies and demonstrate that appropriate design and strict implementation of sound strategies can make a significant impact on safety.

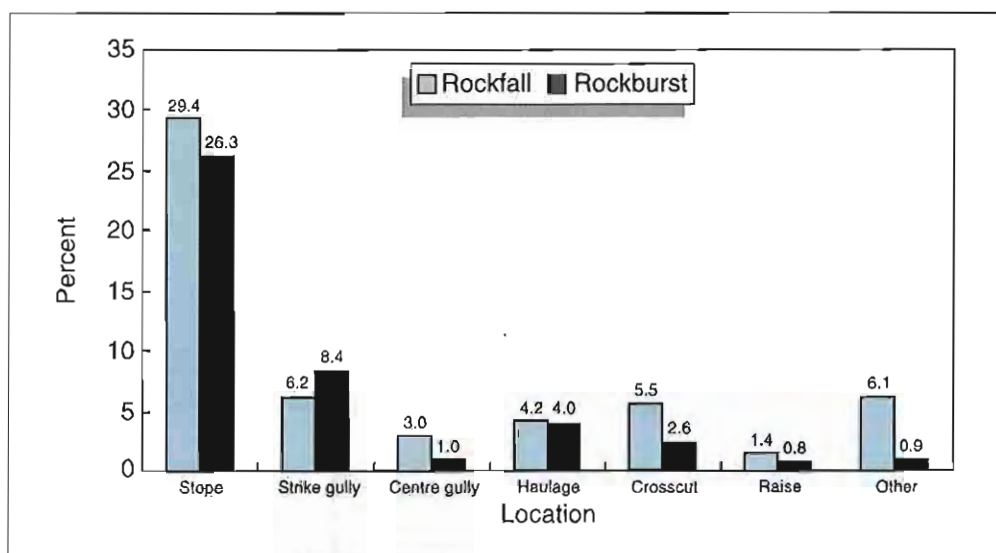


Figure 1.2.5a Location of fatal accidents, gold mining 1990-97

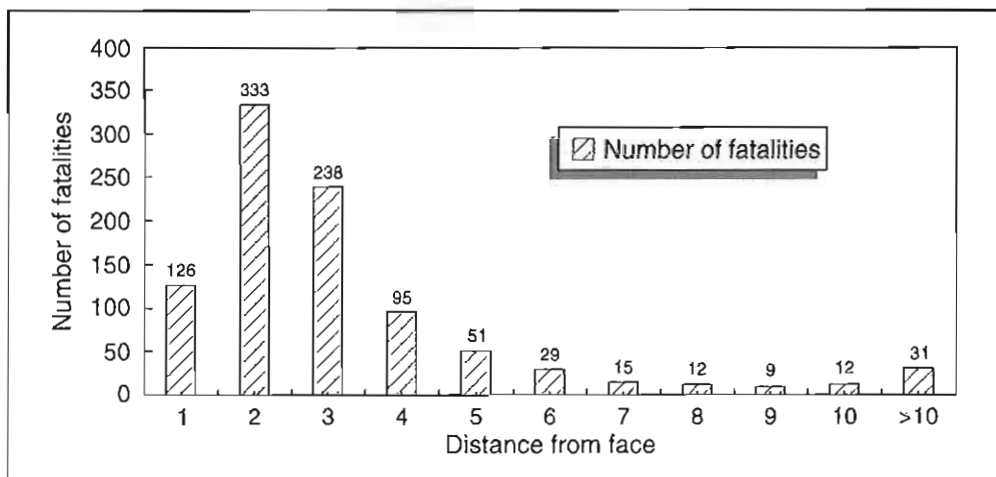


Figure 1.2.5b Analysis of distance from face where stoping accidents took place, gold mining 1990-97

As with stopes, many accidents in **tunnels** occur close to the face of developing ends as shown in Figure 1.2.7. The high incidence of accidents further than 20 m behind the face indicates some deficiency in the secondary support in use. A possible contributing factor to this is deterioration of the support with time as a result of corrosion, continued stress-induced deformation or the accumulation of damage from many seismic loadings. The high relative rate in the area close to the face indicates that more attention also needs to be given to the design of the primary support and implementation of temporary support.

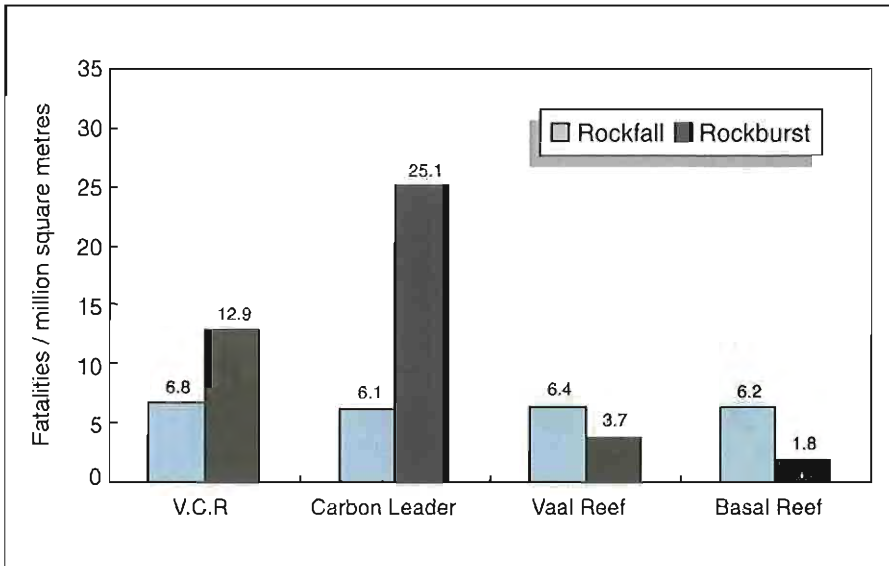


Figure 1.2.6a Fatality rates for major gold mining reefs, 1990-97

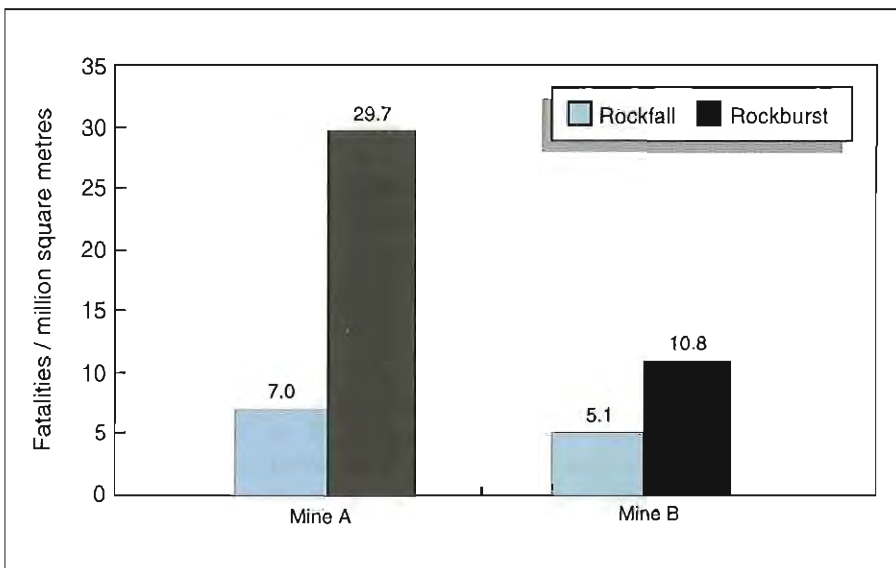


Figure 1.2.6b Fatality rates on two neighbouring gold mines, 1990-97

(Data used in the compilation of these analyses of casualties was drawn from the Department of Minerals and Energy's SAMRASS database, from the Chamber of Mines of South Africa's annual Statistical Tables, and from CSIR-MININGTEK's database of mining accidents.)

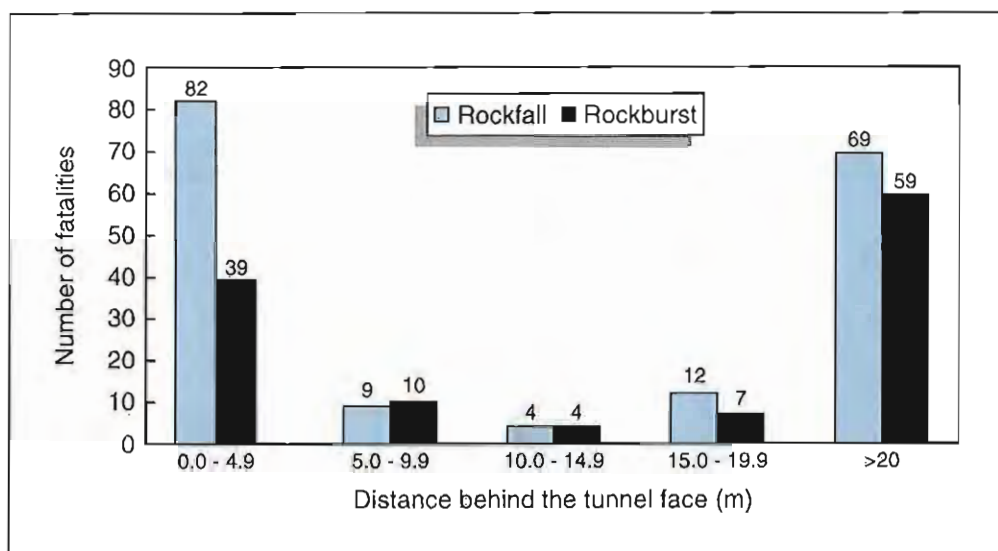


Figure 1.2.7 Analysis of distance from face where tunnel accidents took place, gold mining, 1990-97. Distances >20 m relate mainly to fatalities in established tunnels, other data relate to accidents incurred during development.

1.3 FACTORS GOVERNING ROCK BEHAVIOUR

1.3.1 Rock Stresses

South African hard rock mining is often carried out at great depths below the Earth's surface. Those mines which in this book are classified as being of 'medium' depth (1000 - 2250 m, c.f. Chapter 2.1), would most probably be deemed 'very deep' elsewhere in the world. As depths increase, so do stress levels, with concomitant increases in stress fracturing and mining-induced seismicity. It is therefore appropriate to review what is known about the nature of the stresses which are encountered in South African mining.

The **virgin** ('primitive') stresses which exist in the rock mass prior to mining can be measured in the field, though the techniques involved are onerous and prone to measurement error (Chapter 10.3.2). Results from South African underground measurements (mainly in gold mines) are illustrated in Figure 1.3.1a.

The **vertical** component q_v of the virgin stress tends to increase according to the deadweight of the overburden, that is $q_v = \rho gh$ where ρ is the average rock density (typically 2750 kg/m³), g is gravitational acceleration (9.8 m/s²), and h is the depth below surface in metres. In MPa units, $q_v \approx 0.027 h$. Thus, at a depth of 2000 m the vertical virgin stress is likely to be close to 54 MPa, at 4000 m to 108 MPa, and so on. Hard rock mines operating in this range of depths will suffer not only intense face and tunnel fracturing, but will be subjected to increasingly severe incidences of seismic events and damaging rockbursting.

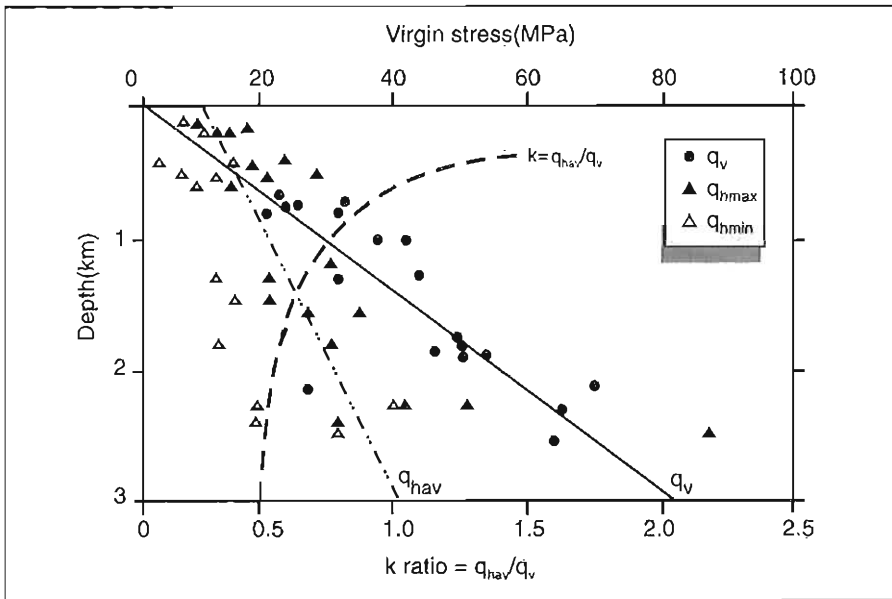


Figure 1.3.1a Virgin stress measurements (mainly South African gold mines)

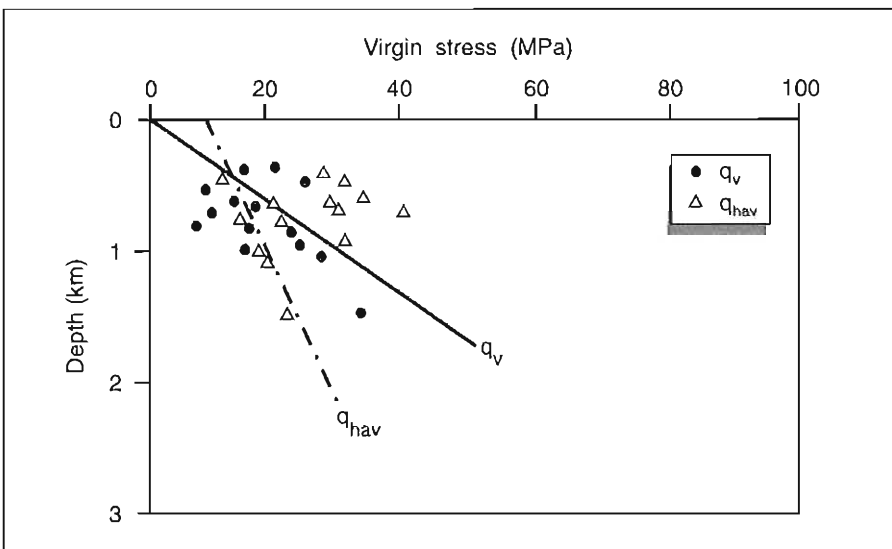


Figure 1.3.1b Virgin stress measurements (W. Bushveld platinum mines)

The measured **horizontal** virgin stresses q_h are subject to considerable scatter, but seem to indicate an average constant (tectonic) stress of about 10 MPa, acting together with a small linear increase with depth. Very approximately, $q_{hav} \approx 10 + 0.01 h$. The virgin stress **k-ratio** ($k = q_{hav} / q_v$) accordingly tends to decrease with depth, from quite high values ($\gg 1$) in shallow situations, to relatively low values (< 0.5) in the deepest mines. [Horizontal stresses in South African mines are actually relatively low. Moderate to deep mines in North America or Australia, for example, often experience k-ratios well in excess of 1.]

Figure 1.3.1b shows virgin stress measurements taken in platinum mines in the Western part of the Bushveld Complex. Average horizontal stresses q_{hav} are illustrated. The data are actually too scattered to permit realistic regression lines to be calculated, but the same trend lines as in Figure 1.3.1a are illustrated for comparison purposes. The vertical virgin stresses do not seem to deviate significantly from the expected cover load values, but the horizontal stresses seem to be noticeably higher than in the goldfields. Indeed, at depths < 1000 m, some platinum mines experience 'Gothic-arch' fracturing in their footwall tunnels, indicative of exceptionally high horizontal stresses.

In specific situations, other abnormal virgin stress situations can pertain. Due to tectonic forces in the Earth's crust, the major principal virgin stress can sometimes be oriented sub-normal to the stratification, so that in deep steeply-dipping reefs the k-ratio may exceed 1 (Figure 1.3.2). In geologically-disturbed reefs (in which irregular or partially locked-up slippages may have occurred in the geological past), the stress tensor may also be inclined and peculiar stress levels and k-ratios can pertain in places (Figure 1.3.3).

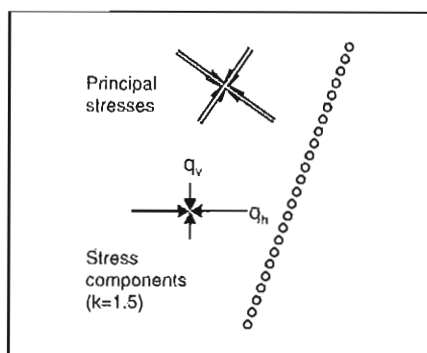


Figure 1.3.2 Virgin stresses, steeply dipping reef

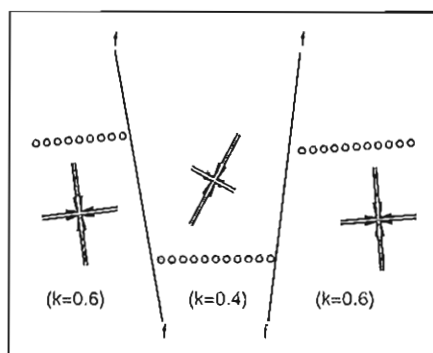


Figure 1.3.3 Virgin stresses, faulted ground

Knowledge of the prevailing virgin stress tensor (or, at the very least, a reasonable estimate of the prevailing k-ratios) is essential input in most numerical modelling exercises (Chapter 11). Physically, abnormal local values of the k-ratio (some of the data points in Figure 1.3.1a, for example, indicate $k < 0.3$) are likely to affect rock pillar strength and foundation stability to some extent, and the incidence and severity of seismicity in deep mines to a considerable extent. A prudent and recommended course of action therefore is that virgin stress levels be established at least in particularly critical situations, such as in orepass layouts or shafts in the vicinity of major faults. Careful observation of dogearing in boreholes, or unusual spalling in tunnels, can indicate where abnormal stresses occur and thus motivate the carrying out of more precise virgin stress measurements.

The act of mining causes **induced** stresses to be generated which, when added to the **virgin** stresses, result in given states of **absolute stress**. These absolute stress levels can be extremely high, and result in the fracturing of the near rock mass described in sections 1.3.5/6 and Chapter 4, and in the seismicity/rockbursting activity

described in section 1.4 and Chapters 8 and 9. The term **field stresses** refers to the state of absolute stress in the rock mass through which tunnels or other service excavations are to be driven.

1.3.2 Rock Strength and Friction Properties

Knowledge of the engineering properties of the rock types on a mine provides a basis for sound design of the excavations involved. As with any geotechnical project, the more data available, the better. In the past, mining companies have tended to test rock specimens on an ad hoc basis, but only rarely have the data been compiled to form a long-term reference for the mine or mining industry as a whole. Such a compilation is currently being done for Southern African hard-rock mines, and will in due course be made available to the mining industry as a source data base of geotechnical information.

The stress-strain behaviour and 'strength' of rock are usually determined by means of laboratory uniaxial and triaxial tests carried out on small cylindrical samples. Typical results for quartzite are illustrated in Figure 1.3.4.

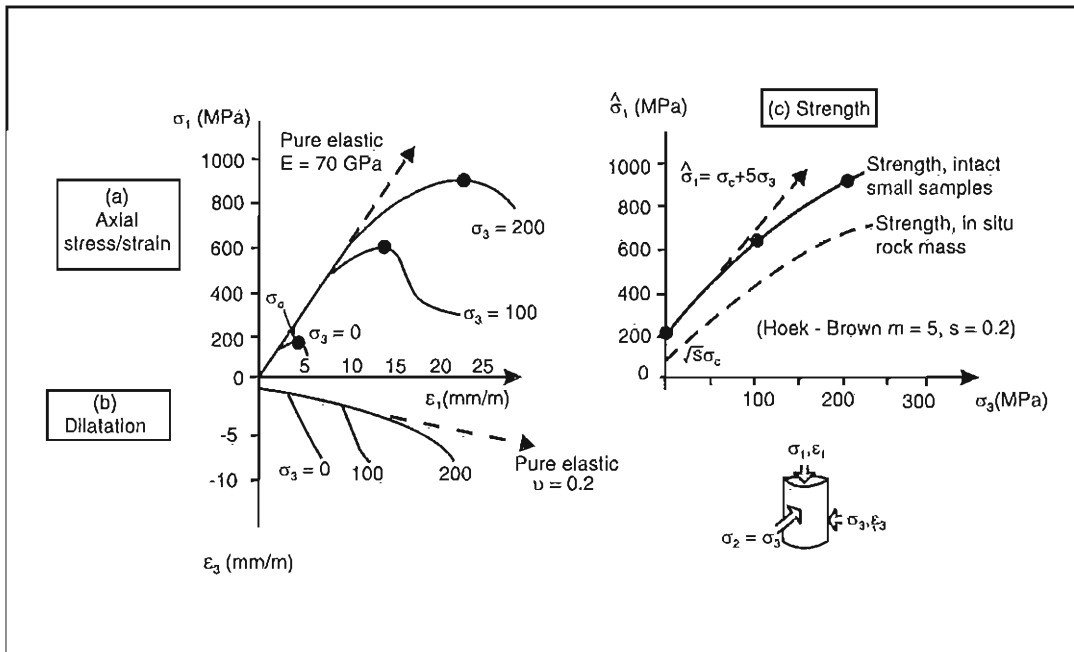


Figure 1.3.4 Quartzite stress-strain, dilatation and strength behaviour

Figure 1.3.4a shows that under uniaxial loading conditions (confinement $\sigma_3 = 0$), the rock axial **stress-strain** behaviour is initially linearly elastic (governed by elastic constants **Young's modulus E** and **Poisson's ratio ν**), but at the **uniaxial compressive strength σ_c** the rock fails and its ability to carry load drops rapidly to zero. At higher confinements ($\sigma_3 > 0$), the strength increases rapidly, but the post-failure behaviour becomes less 'brittle' and increasingly 'ductile' in nature. Figure 1.3.4b illustrates the marked lateral strain (**dilatation**) of rock after failure.

Fracturing-generated dilation leads to beneficial clamping and stabilization of the hangingwall in deep stopes. Also, the high strength of squat support pillars is due to the confining stress provided by, friction-resisted, dilation of the failing edges of the pillar.

Figure 1.3.4c summarises the **strength** behaviour of the rock as a function of confining stress σ_3 . The rapid increase in strength with confining stress is of considerable practical importance: (i) in limiting the depth of fracturing around stressed mining excavations, and in enabling pillars to sustain high average pillar stress levels; (ii) in enhancing the strengthening effect provided by conventional support systems. The strength behaviour depicted in Figure 1.3.4c is often described by a linear ('Mohr-Coulomb') failure criterion: $\sigma_1 = \sigma_c + \beta \sigma_3$, where σ_c is the *uniaxial compressive strength (UCS)*, and the '*strengthening parameter*' β takes on values ranging from 3 up to 10 for the strongest rock types. In practice, the true strengthening behaviour may be slightly curved, and the in situ strength of representative masses of rock transected by multiple weaknesses can be considerably less than that of small intact laboratory specimens (the so-called 'scale effect'). These characteristics are catered for in the Hoek-Brown failure criterion, which has been applied in the design of critical excavations: $\sigma_1 = \sigma_3 + \sqrt{(s \sigma_c^2 + m \sigma_c \sigma_3)}$, where m governs curvature of the failure characteristic (typically $m = 0.05$ to 5); and s governs 'rock mass condition' (typically $s = 0.01$ to 0.6). [Note that the uniaxial strength of the rock mass is now given by the value $\sqrt{s \sigma_c}$]. Appropriate values of m and s can be estimated by carrying out a rock mass classification (survey of joint spacings and other geotechnical parameters) pertaining at a particular site - Chapter 10.3.5. The increased realism thereby obtained is suggested by the dashed line in Figure 1.3.4c.

Rock **friction** characteristics are important in controlling sliding movements which influence the load-bearing capability and fracturing of rockwalls and pillars, and which govern the stability and rupturing potential of planes of weakness in the rock mass. A representative dynamic friction coefficient for sliding quartzitic surfaces is $\mu = 0.6$ (friction angle $\phi = \tan^{-1} \mu = 30^\circ$), though values as low as $\mu = 0.1$ ($\phi = 6^\circ$) can exist for micaceous or serpentinised interfaces which are sometimes encountered in geological discontinuities. The **static** shear strength τ_s along a plane is given by $\tau_s = S_0 + \mu_s \sigma_n$, where S_0 is the 'cohesive strength' (in the order of 40 MPa for intact quartzite, but only 0 to 5 MPa for geological discontinuities), σ_n is the stress normal to the plane, and μ_s is the 'static' coefficient of friction which is generally slightly higher than or equal to the 'dynamic' μ value.

In 'excess shear stress' (ESS) calculations aimed at estimating seismic potentials, $ESS = \tau - \mu \sigma_n$, where τ is the acting absolute shear stress along a potential rupture plane, and μ is the estimated dynamic (sliding) friction coefficient on this plane.

1.3.3 Rock Types

The host rocks in which South African mining is carried out are generally competent, but are certainly not entirely homogeneous. In the gold mines, for example, layers of quartzites of varying compositions are interbedded with weak shale bands, and are

transected by occasional strong dyke and sill intrusives - Figure 1.3.5a. In the platinum and chrome mines, weakish tabular orebodies are embedded in layers of anorthosites/pyroxenites/norites of greatly varying composition and strengths - Figure 1.3.5b.

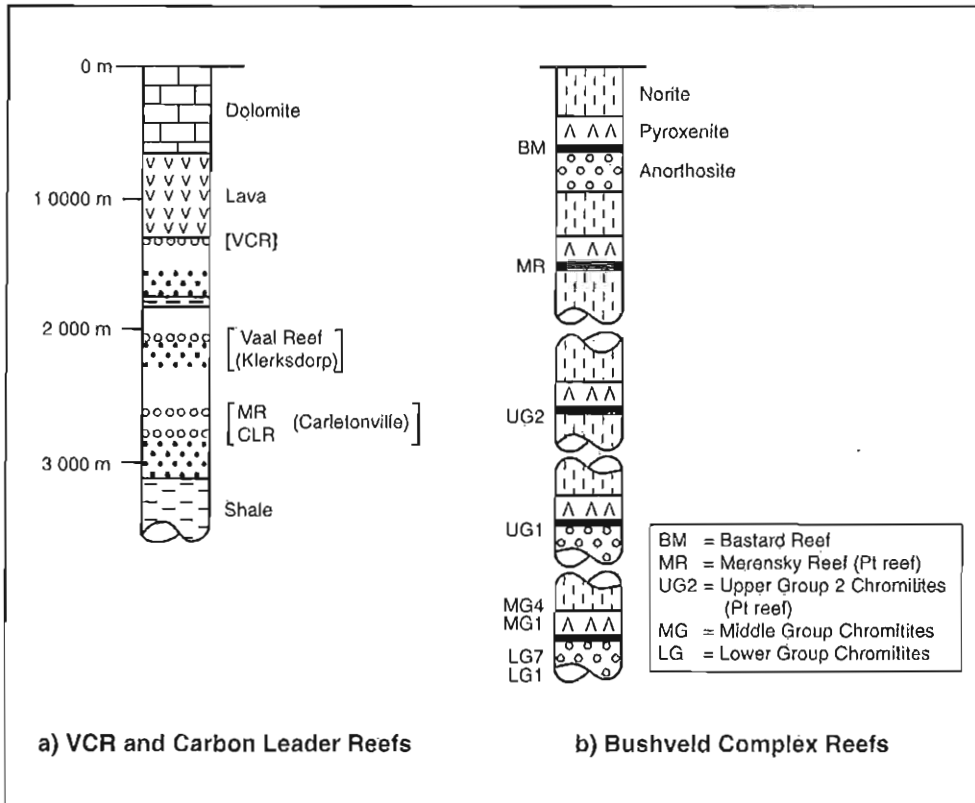


Figure 1.3.5 Simplified stratigraphic columns - Witwatersrand gold mines, and Bushveld mines

There are several practical effects of the presence of such **inhomogeneities**.

- Fracturing in service excavations, pillars and abutments is affected, due to the induction of tensile stresses and resultant fracturing in the stiffer rock layers adjacent to softer ones.
- The presence of shale layers, tuffaceous lava, or other weak materials in the immediate hangingwall or footwall can cause special mining and strata control difficulties; particularly in the neighbourhood of abutments or pillars.
- Dykes, many of which are strong and brittle and which seem to be associated with abnormal virgin stress fields, can fail violently and thus are often associated with enhanced levels of seismicity and rockbursting in the deeper mines.
- Footwall haulages need to be carefully sited to avoid traversing incompetent rock strata. Crosscuts, which necessarily traverse different strata horizons, can require tailored intensities of support along their lengths.

Table 1.3.1 summarises representative properties of a number of common rock types.

Table 1.3.1 Typical rock elastic and strength properties

	E(GPa)	ν	σ_c (MPa)	β
Quartzite (90 %qtz)	80	0.12	200-300	10
Quartzite (60 %qtz)	70	0.20	150-200	5
Shale	40-80	0.25	80-200	2-5
Lava	80	0.21	200-500	5
Soft Lava	65	0.22	80-160	4
Dolomite	75	0.27	300	5
Dyke/Sill	80	0.25	90-300	5
Pyroxenite	60-120	0.15-0.25	100-180	4-10
Anorthosite	90	0.22	230	8
Norite	60-100	0.18-0.25	100-250	6-10
Chromitite	80-110	0.22	50-150	6-9

1.3.4 Geological Weaknesses

Apart from inhomogeneities, other types of geological weakness transect the rock mass; these also play important roles in controlling mining conditions.

(i) **Parting planes**. The presence of parting planes (weak bedding planes in the gold-fields, or weak pseudo-bedding or 'cooling dome' structures in the Bushveld) strongly affect hangingwall control and FOG problems in general. Also, by fostering lateral strata sliding movements, they permit deeper fracturing and weakening to take place in abutments and pillars.

(ii) **Joint sets, minor faults, dyke contacts**. These, if present, often promote falls of ground; they also play a distinct role in generating seismicity.

(iii) **Major faults**. Apart from their obvious control over mining layouts, many of these are known to be the focus of large seismic events.

Geological structures generally play an important role in determining how the rock mass will respond to the imposition of mining-induced stress changes. Unless a representative seismic or other history of a particular structure is available, it is wise to treat all geological features with equal caution whenever they are exposed in mining excavations.

1.3.5 Stope and Tunnel Fracturing and Deformation Patterns

In South African hard rock mining, the great bulk of the rock mass seems to behave elastically (Figure 1.3.6), but stress concentrations around the deeper excavations cause stable, as well as sometimes unstable, fracturing to take place. Stable rock behaviour is reviewed in this section and in Chapters 4-7, while unstable behaviour (seismicity and rockbursting) is discussed in section 1.4 and in Chapters 8 and 9.

The stress concentrations induced by mining are largest immediately in front of **stope faces**, and are particularly severe in abutments, remnants, and pillars. In all but the shallowest stopes, these stresses result in intense and characteristic patterns of fracturing - Figure 1.3.7a and b.

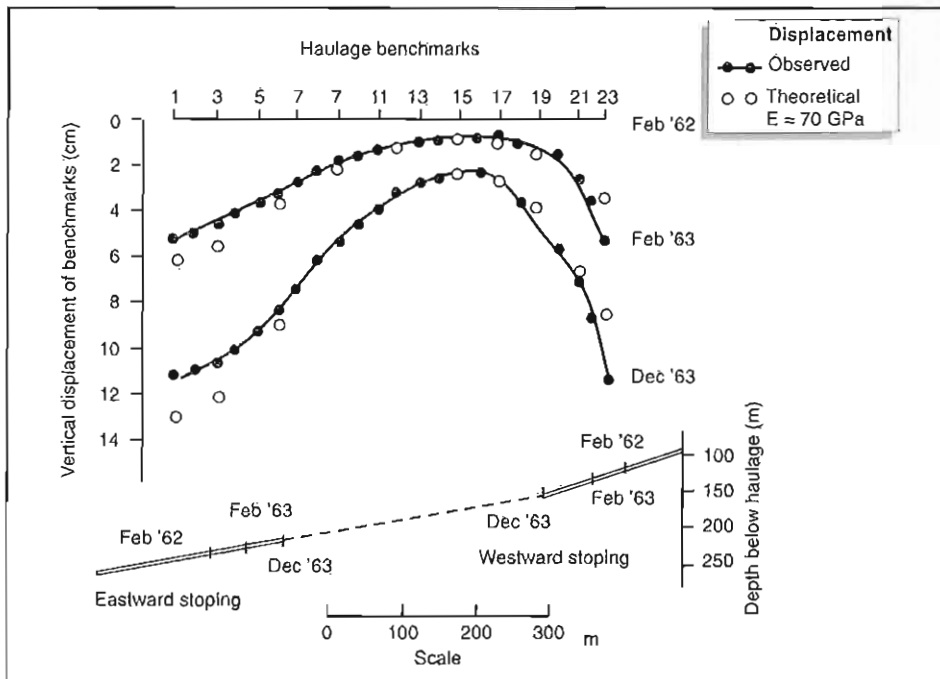


Figure 1.3.6 Elastic rock mass behaviour

Pre-existing **parting-planes P** and **joints J** are generally present. Mining-induced rides (slip displacements) of several centimetres occur on the weaker parting planes. These slip movements destroy the cohesion of the plane and promote dilatatory bed separation movements and enhanced stope closures (heavy lines in Figure 1.3.7b).

Mining-induced **shear fractures, S**, form well in front of the face and are generally 'longwall-parallel' in plan, that is they tend to conform to the broad orientations of mining and do not follow minor irregularities of the face or of small lags. In high-stress situations such as in deep longwalls, they can form as much as 10 m in front of the face and some extend as much as 10-30 m above or below the stope horizon.

Mining-induced **extension fractures, E**, form at or very near the stope face and are 'face-parallel' in plan, that is they conform to small irregularities in the face shape. In section, they curve around the stope face, and account for the majority of visible fracturing in the hangingwall. Local strata control difficulties arise when these fractures form at low angles or parallel to the bedding - Chapter 3.4.7 and Chapter 4.

Falls of ground are commonly bounded by combinations of face-parallel shear/extension fractures and parting planes which break the hangingwall into distinct blocks; they occur most readily if face-normal weaknesses, such as joints or adversely orientated mining-induced or **blast fractures**, exist.

The movements associated with the formation of shear and extension fractures result in enhanced closure rates in the back areas of the stope; this strongly influences the behaviour of standard support elements - Chapter 4.2.3. Measured 'skin' closure

rates, typically 20 to 50 mm per m of face advance, are many times larger than 'elastic' convergences (those expected if the entire rock mass were competent and elastic); they are also strongly time-dependent (section 1.3.6 below). Inelastic closures are little affected by the presence of conventional back area support, but massive resistances of 5 MPa or more generated by backfill in the back areas can reduce their magnitude by about 50 % - Figure 1.3.7c.

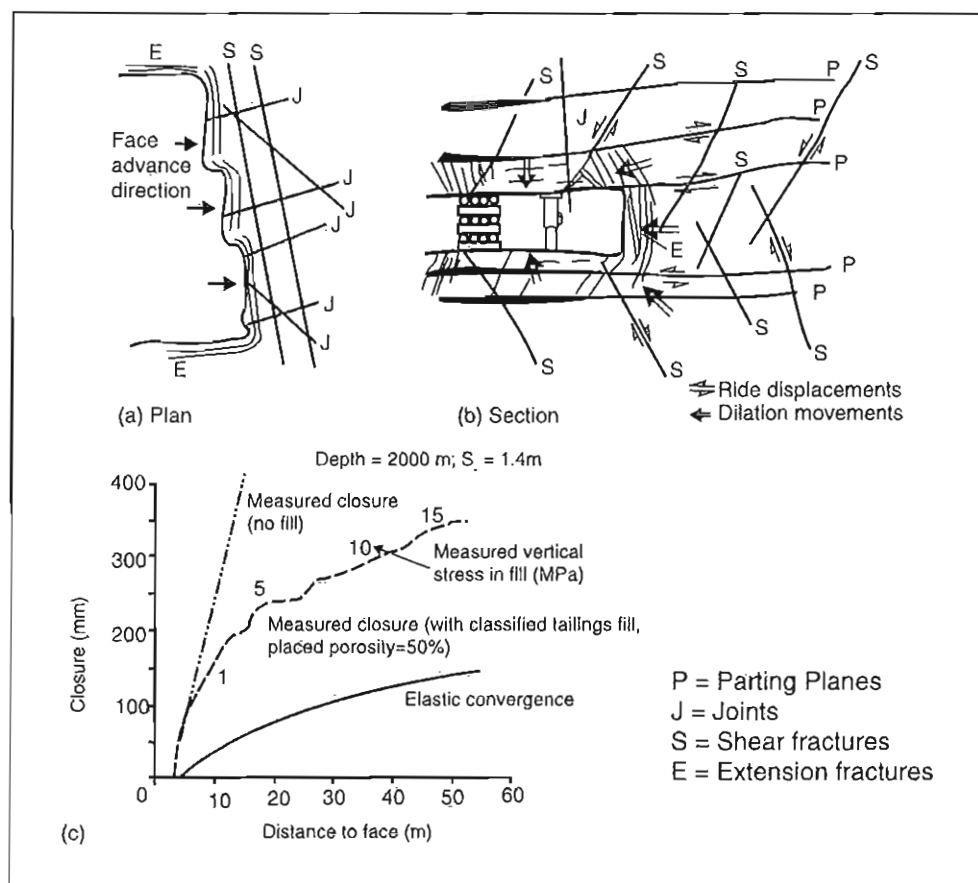


Figure 1.3.7 Fracture and deformation patterns around a deep stope

Dilatationary movements are also important in generating horizontal stresses, thus helping to clamp and strengthen the fractured hangingwall beam above the face area in deep stopes. These effects are absent in shallow stopes and this accounts for some of the special difficulties encountered in such environments - Chapter 3.4.1.

Tunnel Fracturing: Boreholes drilled in high-stress intact rock exhibit immediate spalling or 'dog-earing' - Figure 1.3.8a. The points of onset of spalling are easily explained in terms of elastic theory, Figure 1.3.8b. **Tunnels** driven in exceptional stress conditions experience similar immediate damage, Figure 1.3.9a. To avoid this heavy fracturing and associated dilation, such tunnels are often rather developed in overstoped stress-relieved ground. These 'follow-behind' haulages and crosscuts

are an integral part of conventional deep longwall mining layouts. More commonly, tunnels and other service excavations are developed in lower intrinsic stress conditions but may be subjected to later mining-induced increases (or even major relaxations) in field stresses which cause fracturing and deformation to take place - Figure 1.3.9b. Several factors are involved including: (i) **rock mass condition**, i.e. the effects of bedding and joint properties; (ii) initial **field stress levels** in terms of magnitude and direction, and subsequent changes in these values; and (iii) quantity and quality of **installed support**.

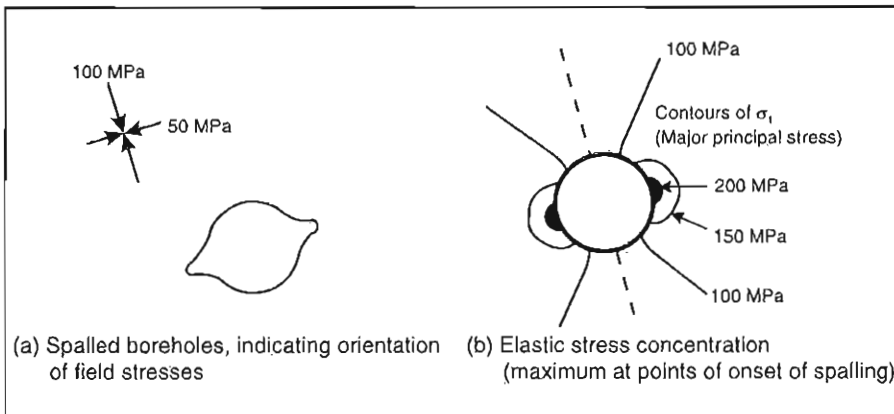


Figure 1.3.8 Borehole spalling in terms of elastic stress concentrations

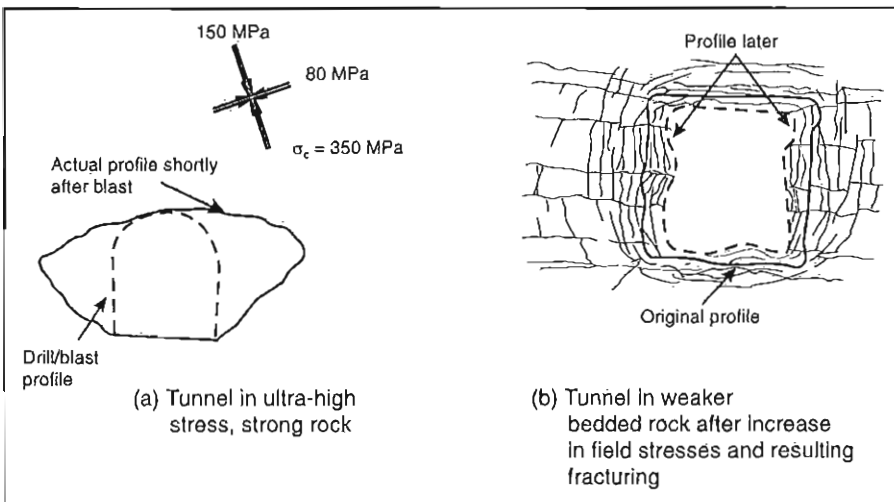


Figure 1.3.9 Tunnel spalling, and fracturing/deformation patterns

Rock mass condition, which is a central factor in governing rockwall integrity in the presence of given field stress levels and installed support, tends to be 'good', or even 'extremely good', in South African hard rock mining situations. There are nevertheless notable exceptions - for example, where heavy jointing or weak hangingwall shale bands or so-called 'running dykes' are encountered, and in quite extensive areas in the Bushveld where serpentinized and blocky rockwalls can be associated

with severe ground control problems. Rock mass condition can be quantified in terms of various indices (developed in the first instance to guide the design of shallow Civil Engineering excavations) such as the 'Q' or 'RMR' systems - Chapter 10.3.5 - but these still require modifications to make them more applicable for deeper tabular mining conditions.

1.3.6 Time-Dependent Fracturing and Deformation

As illustrated in Figure 1.3.6, significant movements take place in the rock mass around hard rock mining excavations, but these generally seem to conform to the expectations of elastic behaviour due to the patterns of mining, and do not change if mining is static. The situation is very different when viewed from within the 'skin' of fractured ground which envelops the excavations themselves. Here, **time-dependent** deformation processes can be of overriding importance - Figures 1.3.10.

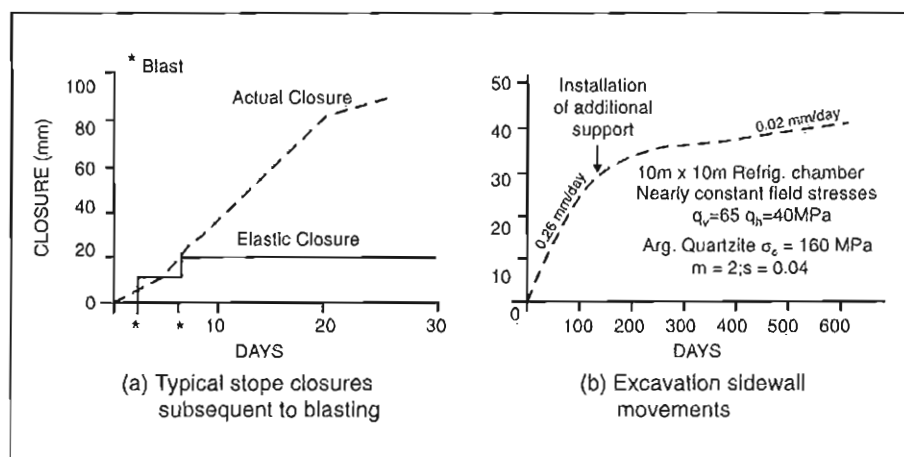


Figure 1.3.10 Time-dependent deformations, stope and service excavation

A straightforward explanation of these phenomena is that fracturing processes in overstressed zones in the immediate vicinity of mining excavations are subject to '**primary creep**', i.e. a finite time in the order of days or weeks is required for fracturing/sliding processes and stress readjustments to reach completion. In the case of long-life service excavations, Figure 1.3.10b, so-called '**secondary creep**' in which extremely slow non-terminating movements take place, can also assume importance. These effects are particularly noticeable in less competent rocks, (e.g. highly argillaceous and bedded quartzites) and at elevated stress levels. In terms of strata control there are at least two important consequences:

- (i) in deep stopes, slow or intermittent rates of face advance lead to a deterioration in hangingwall conditions and enhanced amounts of back-area closure (hence concomitant support problems);
- (ii) in tunnels and other service excavations, especially those subjected to high field stresses or in poor ground, and designed for a working life of 10 or more years, adequate clearance must be provided and support of appropriate quality and yieldability must be installed, preferably from the outset.

1.4 SEISMICITY AND ROCKBURSTING

Seismic networks, in conjunction with application of seismological theory, have given valuable objective information on the occurrence and nature of seismic events and of rockbursting in South African mines; there is in fact a high degree of similarity between earthquakes and mining seismicity. The present discussion is brief, but more comprehensive information can be found in Chapters 8 and 9 of this book.

1.4.1 Quantitative Seismology and Rockbursts

Seismic networks have provided *quantitative* data on seismicity levels and have thereby furnished information relevant to the control of the rockburst hazard. Figure 1.4.1a shows a typical plot of the cumulative frequency of events arranged in order of magnitude; this could pertain to an entire district, a mine (as here), or even a region within a mine. Over a wide range, the log-cumulative frequencies fall off linearly: $\log N = a - bM$. This formula is the Gutenberg-Richter relationship, applied in the first instance to naturally occurring earthquakes. The 'a' parameter describes the overall level of prevailing seismicity, while the slope 'b' describes the relative likelihood of large events taking place. In practice a definite geometry-controlled upper limit exists to the magnitude of the expected largest event in any given mining situation (broken portion of curve, Figure 1.4.1a; in certain situations the excess shear stress (ESS) criterion of Chapter 3.2.3 can be used to estimate this important cutoff magnitude. Figure 1.9.1a shows, for example, that a typical deep gold mine experiences some 300 events of magnitude $M > 2$ every year.

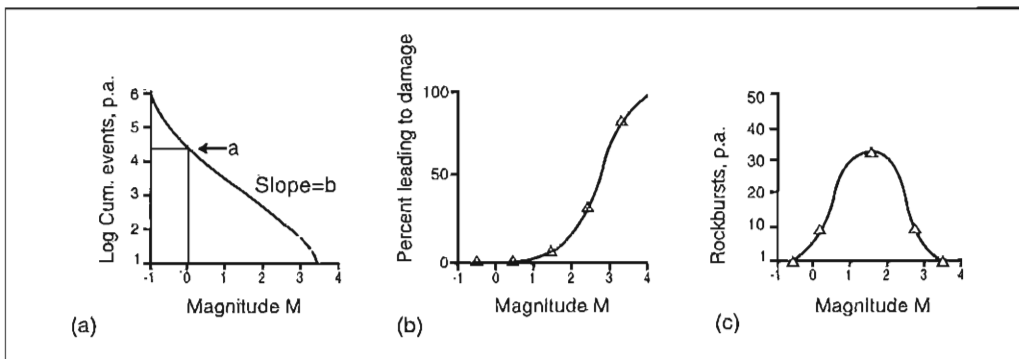


Figure 1.4.1 Seismic event and rockburst frequencies vs event magnitude, for a deep gold mine

A formal definition of rockbursting is given in Chapter 8.1, but in simple terms a 'rockburst' is the occurrence of damage to one or more mining excavations associated with a seismic event. Not all seismic events cause damage, but Figure 1.4.1b illustrates the well-known fact that, as the magnitude of a seismic event increases, so does the probability of its manifesting as a reportable rockburst. For example, about 20 % of events of magnitude 2 are accompanied by actual damage or casualties, while for magnitude 4 events this figure rises to nearly 100 %. This is largely due to the rapid increase with event magnitude of event source dimension, exposing an increasing volume of mining excavations to high ground velocities and probability of damage. Figure 1.4.7 shows, for example, that an $M = 0$ event typically involves a

rupture surface spanning only about 10 m, whereas an $M = 2$ event spans about 100 m and an $M = 4$ event as much as 1000 m. Thus, very few $M < 1$ events manifest as rockbursts, whereas the reverse is true for $M > 3$ events. The combined effect of Figures 1.4.1a and b leads to Figure 1.4.1c. This shows that there exists a range of event magnitudes, here $0 < M < 3$, which expresses the *pragmatic* hazard of rockbursting and which, in principle, should be used to govern the selection of appropriate support systems and mining layouts in any given rockburst-prone situation. Figure 1.4.1c also shows that, for the deep gold mine illustrated, some 60 damaging rockbursts occurred *every year*.

Diurnal analyses show a pronounced peak in seismicity typically 0-2 hours after the blast, motivating on safety grounds the use of *centralized blasting* systems.

A final well-substantiated use of quantitative seismology lies in assessing the benefits of the introduction of regional support systems such as stabilizing pillars - Chapter 3.3.

1.4.2 Seismicity Patterns

Useful results can be obtained by plotting patterns of seismicity, preferably grouped in meaningful magnitude ranges. Figures 1.4.2 illustrate the well-known clustering of events near active mining faces. Figure 1.4.3 shows the contrasting seismicity patterns found when mining obliquely (with diffuse low-intensity events) versus breast-on (with intense activity close to the face) through a pervasive set of joints; these data assisted in motivating a major change in longwall face orientation on the mine concerned. Figure 1.4.4 illustrates an inferred set of foundation failure events on a highly-stressed stabilizing pillar. Figure 1.4.5 shows numerous fault-slip events taking place in heavily-mined geologically-disturbed ground.

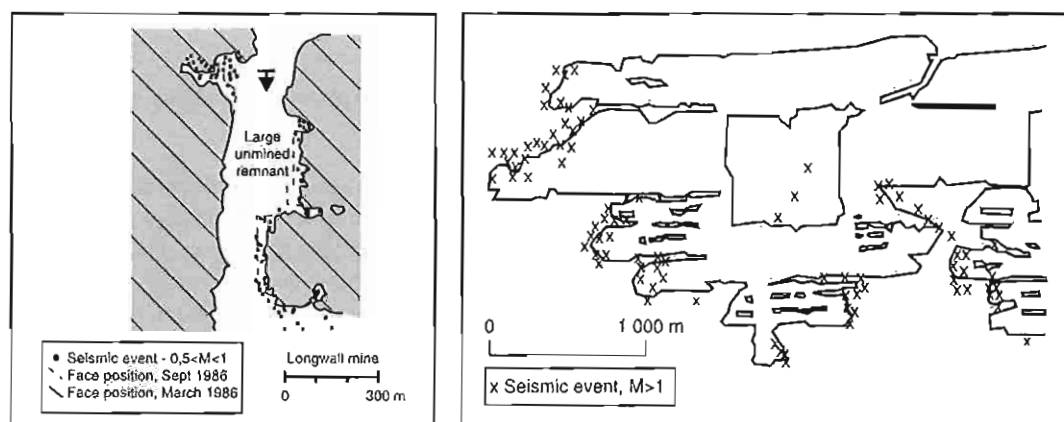


Figure 1.4.2 Seismicity patterns, showing clustering around active mining faces

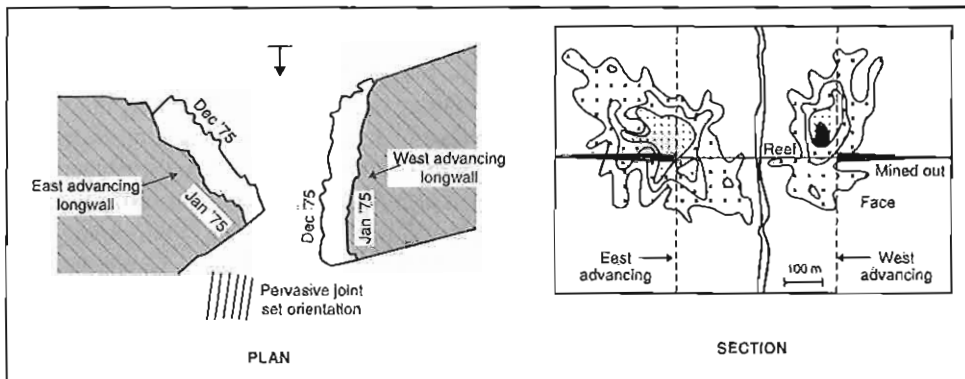


Figure 1.4.3 Seismicity patterns associated with oblique/breast mining through a joint set

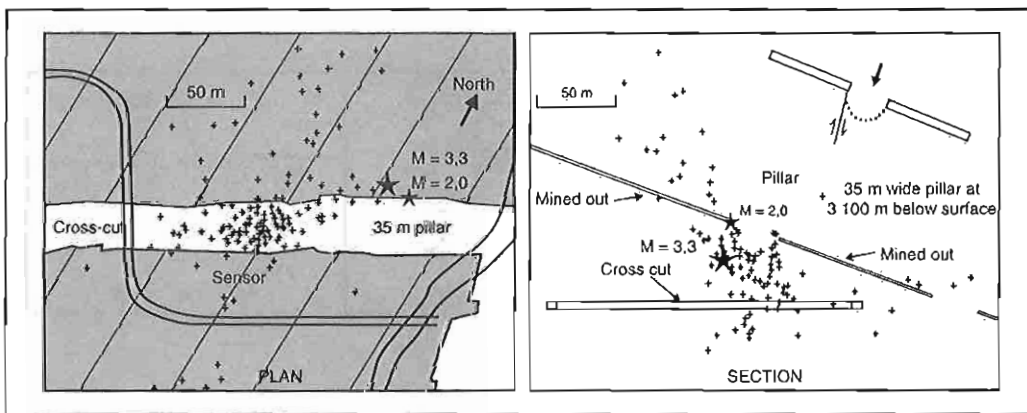


Figure 1.4.4 Inferred foundation failure on a stabilizing pillar, followed by microcrush events

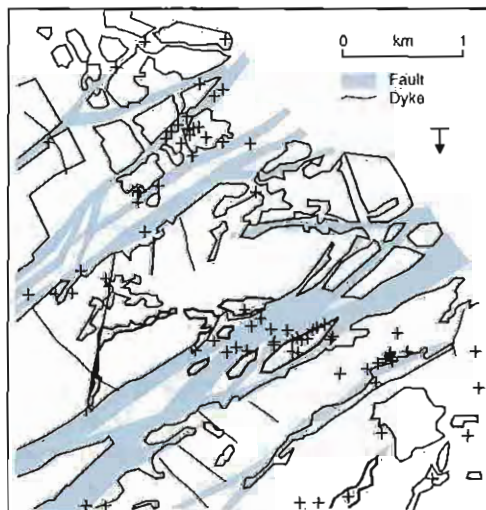


Figure 1.4.5 Seismicity associated with fault planes and dykes, highly geologically-disturbed ground

1.4.3 Seismic Event Mechanisms

Analyses of seismic data are beginning to provide information on the *mechanisms* underlying seismic events and rockbursting. For an event of appreciable magnitude to occur, firstly, a substantial zone of overstressed rock must exist in a state of unstable equilibrium; secondly, a sufficient change in stress must take place to 'trigger' the event, and finally the failing structure must undergo a significant and sufficiently rapid 'stress drop' - Figure 1.4.6a. Careful analysis of geophone 'first-motion' and waveform signatures, of 'aftershock' distributions, and of more recent full 'moment tensor inversions' has suggested that at least two qualitatively different mechanisms pertain in practice - illustrated in highly simplified form in Figure 1.4.6b.

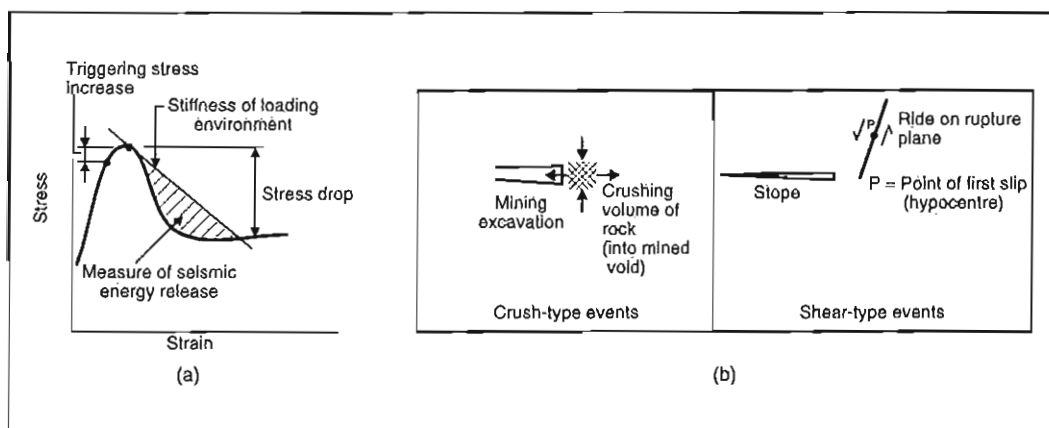


Figure 1.4.6 Seismic event mechanisms (simplified)

(i) So-called 'crush - type' events, which locate near the reef and close to the immediate face, and in which damage is usually limited in extent across a mining panel. These possibly account for in-panel pillar bursts and for many smaller magnitude ($M < 2$) events, in particular most *strain bursts* (c.f. Chapter 8.2.1 i) and many *dyke* - related events. These events exhibit a high degree of volumetric closure relative to a much lower degree of pure shear activity at the source.

(ii) So-called 'shear - type' events, which can locate 50 m or more above or below the reef horizon and which account for most of the large ($M > 2$) widespread-damage events. These take place preferentially on planes of weakness such as major faults, or minor faults/joints/dyke contacts. It has been observed that large events occur preferentially on steeply dipping ($> 60^\circ$) faults of large throw (> 40 m) and especially where the fault is planar, clean and tightly closed. In longwall mining, shear ruptures also occur in intact rock, manifesting as *burst ruptures* above or below faces, or as *foundation failures* in pillars. In recent years, moment tensor inversions have shown that a majority of large events are of shear type, though many also include a large component of coseismic volumetric closure. This suggests that many large events in the deep gold mines are accompanied by significant stope closure, as has in fact been borne out by direct continuous monitoring of closures in deep stopes.

1.4.4 Strong Ground Motion And Support Design

Seismological theory and direct observations from seismic networks help explain the form of Figure 1.4.1b and therefore have direct relevance to the design of improved support systems. For example, Figure 1.4.7 gives a useful indicative set of relationships (similar to those of the popular simple **Brune model**) between magnitude M , seismic moment M_0 , source dimension L_s , and approximate maximum slip R in a pure shear type event.

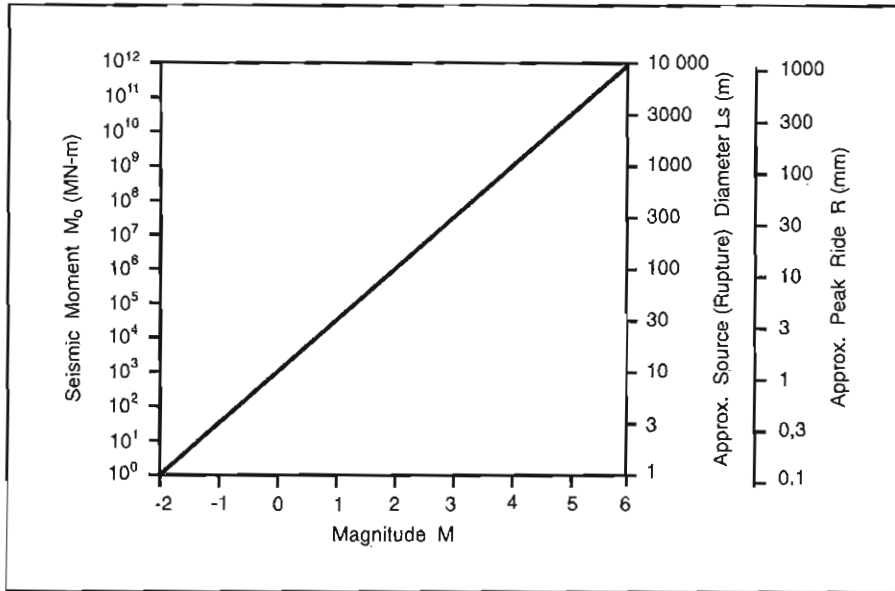


Figure 1.4.7 Approximate relationship between seismic parameters M , M_0 , L_s and R (for shear rupture with a circular ESS profile, peak stress drop = 8.5 MPa, av. stress drop = 5 MPa)

Peak ground velocities, which are thought to control rockburst damage potential, are illustrated in Figure 1.4.8 as measured in the 'far-field' (outside of the source region). Unfortunately, the form of the critical dashed portions, which correspond to the 'near-field' or *strong ground motions* of seismological theory, is not yet well determined, and is probably highly dependent on details of rupturing within the source region. Based on measurements of far-field peak *accelerations*, a limiting strong ground motion of about 4 m/s has been inferred - but for a particular source model, and subject to perhaps a 100 % level of uncertainty. Again, for a particular source model, the peak *near-field* ground velocity is simply given by $v_{\max} \equiv \tau_e / \sqrt{\rho G} \equiv 0.1 \tau_e$, where τ_e is the shear stress-drop across a rupturing plane (in MPa), ρ is the rock density, and G is the shear modulus. Since the confined excess in situ shear strength of even the strongest intact rocks in the South African goldfields probably does not exceed about 40 MPa, this again suggests an upper bound on *near-field* velocities of about 4 m/s; and this, only near to a rupture of intact strong rock or an intact strong asperity on a sliding plane of weakness.

There are also complications when a seismic wave impinges on the rockwall of an excavation: some energy is lost in penetrating the "cushion" of rock making up the fracture zone, but large amplifications of ground velocity can also occur due to surface waves and possible resonance effects. Actual observations, from instruments installed in stopes and tunnel sidewalls, have so far indicated peak ground velocities of no more than 2 m/s associated with nearby seismic events. Current support designs for rockburst conditions assume a maximum block ejection velocity of 3 m/s; and this value, weighing the above observations, may represent a reasonable compromise for support design in being able theoretically to contain the damaging potential of all but the most violent (and rare) rockburst events.

The consequences of strong ground motion include: (i) the potential punching of prop-type supports, or their toppling due to rapid lateral movements (countered by use of appropriate headboards/footboards); (ii) the ejection of loosened blocks of fractured rockwall in stopes or service excavations (countered by providing adequate areal coverage in conjunction with appropriate levels of support resistance and energy absorption capability - Chapter 4).

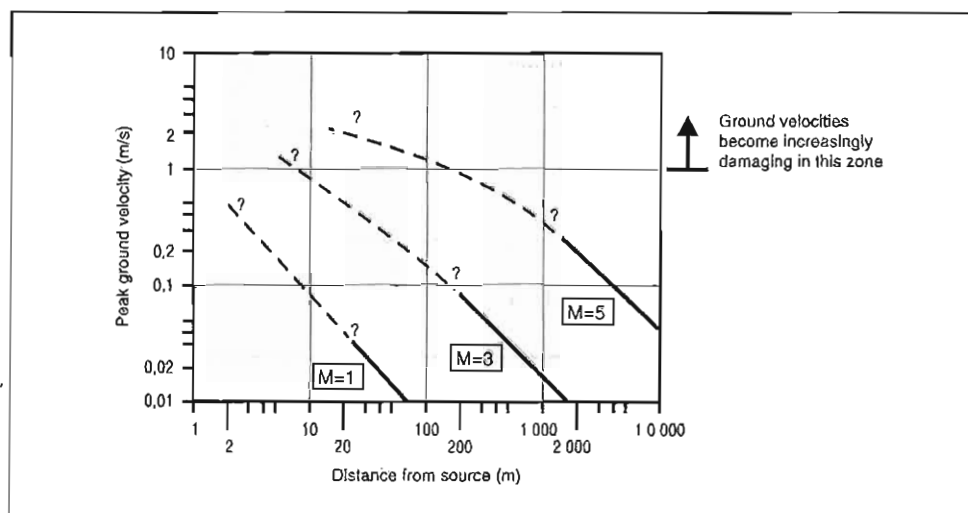


Figure 1.4.8 Decrease in ground velocity with distance from seismic event - far field data

1.5 CONCLUSIONS

In this chapter, elements of the South African rock engineering challenge were identified and reviewed. These include the high rates of rock-related casualties, notably in accidents near the working stope faces and due to rockbursting in the deep mines. A major causative factor is the labour-intensive wide-spread nature of South African mining in which there is continuous exposure to the hazards of high rock stresses, fractured or otherwise unstable hangingwall rock, faulting, and seismicity.

Clearly, the overall challenge is to significantly reduce rock-related casualty rates by reducing the incidence and effects of rockfalls and of rockbursts. Using no more than the guidelines given in this book, together with sound innovations adapted to local

conditions, a significant reduction is believed to be attainable. To achieve this goal, however, mining personnel will need to address the following issues in a concerted and energetic fashion:

- (a) Implement sound layout strategies (Chapters 2, 3, 5 and 7) with the objective of reducing seismicity, ensuring stable mining conditions, and otherwise containing rockburst and rockfall hazards to more tolerable levels.
- (b) Select and implement sound support and strata control systems, with particular attention to providing adequate areal coverage and support resistance close to the working faces and in gullies (Chapters 4, 6 and 7).
- (c) Gain a better understanding of the causation of rockbursting by on-sight investigations (Chapter 8), and through innovative seismic monitoring and analysis (Chapter 9). Consider the use of preconditioning techniques (Chapter 8.4).
- (d) Improve rock engineering designs by the use of appropriate monitoring and auditing processes (Chapter 10) and numerical modelling (Chapter 11). Improve the general level of understanding of rock-related hazards and rock engineering counter-measures through appropriate training programmes (Chapter 12).
- (e) Ensure the implementation of sound practices through appropriate quality-control procedures, risk analyses, auditing, and the preparation and continuous updating of sound mine codes of practice (Chapter 2.5, 2.6).
- (f) Pro-actively detect and deal with incipient hazards not only through the above formal procedures, but by conscious scrutiny at regular planning meetings and informal risk analyses. It is known that many severe accidents could have been prevented by these means.

In addition, further *research and development challenges* will be to address (on a time scale of 5-10 years) inadequacies in knowledge and technology with relevance to both present mining, and the ultra-deep mining of the future where depths of 4-5000 m are envisaged:

- (a) Develop improved means of analysis of seismic data coupled with numerical modelling, for improved planning, and identification and quantification of future seismic hazards.
- (b) Address the rockburst problem through a better understanding of seismic source and rockburst damage mechanisms. Devise practical applications.
- (c) Improve numerical modelling of fracturing, time-dependent and dynamic processes, for better quantitative design of pillars, service excavations, and support system requirements.
- (d) Establish and calibrate improved criteria, and a rock mass characterization scheme tailored for tabular mining, for the better design of layouts, and regional and local support systems.
- (e) Design and implement improved areal support systems for use near the working face.
- (f) Devise viable methods for the development and support of tunnels in very high field-stress regimes.



OVERVIEW OF ROCK ENGINEERING STRATEGIES

2.1 INTRODUCTION

A distinguished mine manager, immersed in the challenges of improving the viability of deep level mining, once said that rock mechanics was the 'soul' of mining!

Rock engineering is the discipline that uses the science of rock mechanics to attempt to harness the natural strengths and weaknesses of the rock mass in which mining takes place so as to counter the stresses and deformations that nature and mining impose; in short, to treat the complex mining environment as an *engineering structure*. Thus, the role of rock engineering within the larger discipline of mining engineering is to establish principles for the design and monitoring of the layout and support of mining excavations, so as to maximize their **safety, stability and cost-effectiveness**.

In tabular mining, the central importance of rock engineering precepts can be seen at a glance. Contrast typical shallow room and pillar or crush pillar layouts (designed to prevent large-scale collapses of the hangingwall), with ultra-deep operations which feature stabilizing pillars or stiff backfill (designed to provide regional control of rockbursting hazards). The presence of these control structures dominates any given mining plan, and close inspection of intermediate-depth mining layouts will also reveal many features which are in place for purely rock engineering reasons (shaft pillars, overstoping, bracket pillars). Many other details of layouts, as well as the support systems in place, will be found to change their fundamental character as a function of mining depth; or, more precisely, the *field stress level relative to the local rock strength*.

Thus the effective **mining depth** is a central factor influencing global mining strategies. The following three sections in this chapter give an overview of the environment and appropriate mining strategies applicable to shallow, medium, deep and ultra-deep mining conditions. These environments are illustrated schematically in Figure 2.1.1. An attempt has been made to quantify them in terms of common rock engineering parameters in Table 2.1.1.

Factors in addition to effective mining depth also play important roles. Local geological structures and prevailing rock mass condition and strength affect mining strategies at any depth; and often assume overriding importance in shallow mining operations. Other natural features, such as steep angles of dip of the ore body and thick or multiple reefs, will also obviously influence mining procedures and standards. Strategies appropriate to these situations are discussed in the later portions of Chapters 3 and 4.

Planning and managerial factors, such as inadequate understanding of, for example, the importance of close face-area support (via analysis of accidents and appropriate risk assessments), and inadequate auditing, training, and support quality assurance procedures, can lead to avoidable human casualties and costly mining disruptions. These issues are overviewed in the final section of this chapter, where the precepts underlying sound mine **codes of practice** are also discussed.

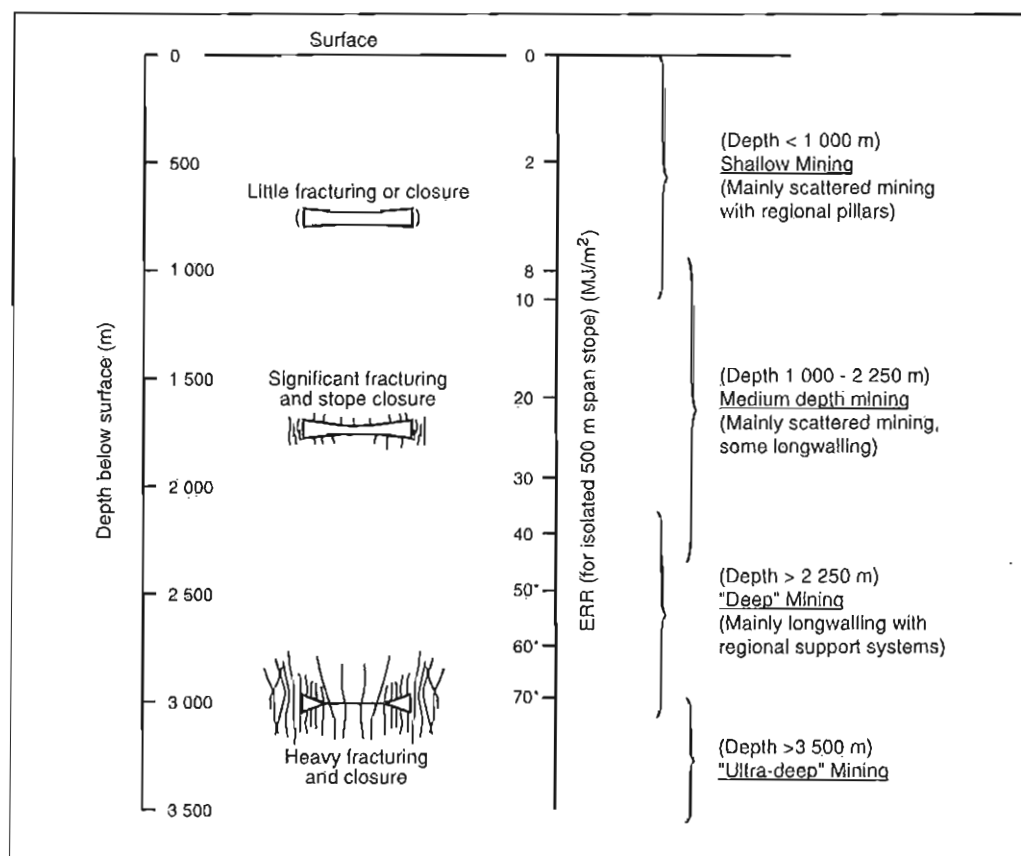


Figure 2.1.1 'Shallow', 'Medium', 'Deep' and 'Ultra-Deep' mining environments.
(Boundaries are approximate, and vary with local conditions)

Table 2.1.1 Values of parameters associated with different mining depths.

Parameter	Shallow	Medium	Deep	Ultra-deep
Typical depth(m)	<1000	1000-2250	2250-3500	>3500
Typical ERR(MJ/m ²)	<8	8-40	40-80*	>80*
Vertical virgin stress(MPa)	<25	25-60	60-95	>95
Stress fracturing	Little/None	Moderate	Deep	V.deep
Stope closure(mm/m advance)	Low(<10)	Mod.(10-30)	High(30-60)*	V.High(>60)*
Infl.of geology on h/w stability	Strong	Moderate	Moderate	Moderate?
Possible extent of FOG's	Can be large	Often small	Usually small	Small?
Rockburst hazard	Minimal	Mod.-Severe*	Severe*	V.Severe*?

* If regional support were not used

2.2 SHALLOW MINING (Typically <1000 m depth)

2.2.1 Shallow Mining Environment

In South Africa, extensive shallow tabular hard-rock mining is carried out in platinum, chrome, manganese and some gold mining operations. Shallow mines experience problems that are peculiarly their own. Some of these problems can also be encountered in multi-reef *stress-relieved* (overstoped or understoped) situations in gold mines at much greater mining depths, for similar underlying reasons.

These problems include: (i) the presence of an extended 'tensile zone' in the hangingwall (largely due to deadweight) - Figure 2.2.1 - which promotes bed-separation and the potential for massive collapses; (ii) the virtual absence of face fracturing, which in deeper mines generates horizontal dilational stresses that act to knit the hangingwall into a coherent self-supporting beam; and (iii) relatively high and sometimes highly variable virgin horizontal stresses.

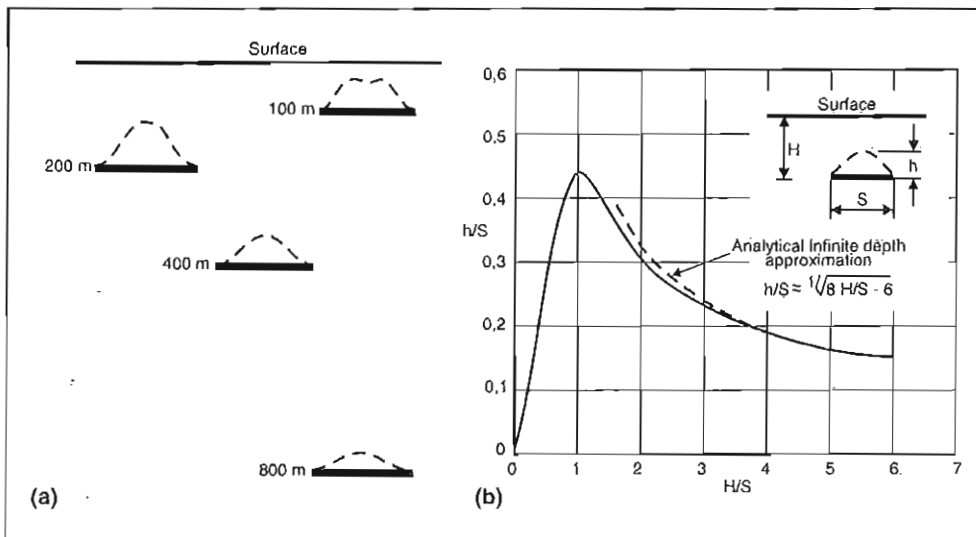


Figure 2.2.1 Tensile zone (areas where the vertical stress is tensile, promoting bed-separation and h/v failure) above shallow stopes having little or no in-panel pillar protection. In (a), 200 m span stopes are illustrated at the depths indicated. In (b), tensile zone heights h are plotted normalized with respect to the mining span S .

As a result, potential failures are defined by relatively widely-spaced tensile fractures or opened-up geological discontinuities, and can take the form of hangingwall beam failure, keyblock dropout followed by large scale hangingwall unraveling, or even more massive stope **backbreaks**. The latter can occur either rapidly or slowly and can involve great thicknesses of hangingwall rock. These factors account for one of the more striking paradoxes of mining: that shallow stopes generally require support of a much *more* robust nature than do their deeper counterparts.

Smaller-scale but potentially hazardous **rockfalls** are at least as much a problem in shallow mining as in deeper operations - Figures 1.2.1 and 1.2.3 - particularly in incompetent or geologically disturbed ground.

Thus shallow mining strategies are aimed at controlling the **tensile zone** on a regional basis to prevent large-scale rock failure, and at stabilizing the immediate **stope hangingwall** to prevent local falls of ground in the working area. The presence of highly competent or highly incompetent beds, well-defined bedding planes or intensive jointing can be critical in the detailed design of appropriate shallow mining (low-stress) pillar layouts and support systems. Likewise knowledge of prevailing **k-ratios**, particularly the presence of abnormally *high* virgin horizontal stresses, can assist in the appropriate design of stoping and tunnel layouts and support standards.

2.2.2 Shallow Mining Strategies

Shallow mining layouts are '**scattered**' in nature, but are subject to major provisos.

(i) Regional mining spans should be kept to no more than about one-half of the mining depth to reduce the possibility of beam failure, stope collapse or backbreak; and (through 'compartmentalization') to eliminate the possibility of disastrous mine-wide collapses occurring. This can be achieved by using a system of appropriately-spaced regional **barrier pillars**, which can comprise or incorporate *major geological structures and/or unpay or other unmined ground*. Barrier pillars need to be substantial (>10 m wide, width:height ratio at least 10:1) since they must be able to carry the appropriate weight of overburden strata for indefinite periods of time.

(ii) In order to stabilize the massive stope hangingwall between these regional pillars, support systems with very high support resistance are required. This is best achieved with the use of small '**crush pillars**'; or, in very shallow mines, larger '**non-yield pillars**' which are loaded within their elastic limits. The dimensions and spacing of these pillars need to be chosen with care, since instances of 'pillar bursting' and even more catastrophic 'pillar runs' have been well documented in the past in inappropriately designed shallow mining layouts. The presence of any pervasive weak joint sets is a vital factor in deciding mining directions and safe pillar dimensions and orientations. Notably in the platinum mines, the presence of weak layers and pervasive partings close to the reef horizon can lead to foundation instability and pillar punching, with consequent severe footwall heave and stope closure which must be accommodated with the use of stiff but yielding in-panel support. By and large, however, well-designed pillar layouts counter the problems of shallow mining extremely effectively.

(iii) Since closure rates in shallow environments are low and horizontal clamping stresses are absent, **stiff local support** is required to control the immediate hangingwall. Appropriate units include concrete-based packs, stiff elongates/props, reinforcing tendons, or cemented backfill. The design of specific face and back area support systems should reflect the local geological conditions and other factors identified for each of the significant *ground control districts* present on a given mine.

Shallow (low-stress) mining strategies are illustrated in Figure 2.2.2.

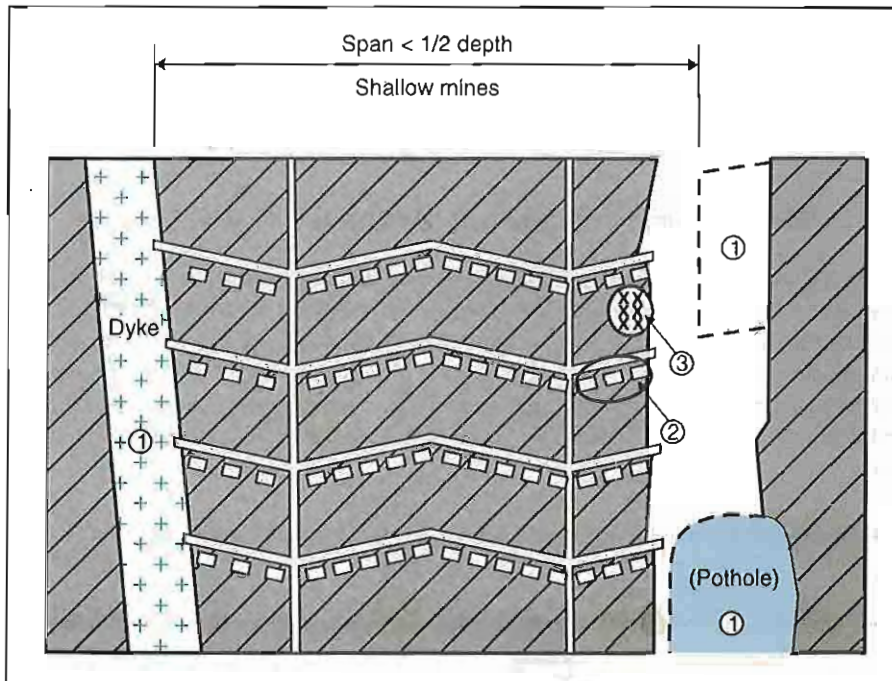


Figure 2.2.2 Illustrative SHALLOW (LOW-STRESS) mining strategies

- 1: Barrier pillars (often incorporating blocks of unpay ground or geological structures).
- 2: Non-yield or crush in-panel pillars, typically along gullies; face orientation or mining direction (e.g. up-dip mining) often changed to negotiate joint sets, etc.
- 3: Stiff face and back-area support (sandwich packs, elongates).

2.3 MEDIUM DEPTH (SCATTERED) MINING (Typically 1000-2250 m depth)

2.3.1 Medium Depth Mining Environment

Medium depth mining conditions occur in the majority of South African gold mines, as well as in some of the deeper platinum mines. The environment is 'relatively' congenial, and scattered mining methods (which are both flexible and operationally convenient) are generally practiced.

- (i) Stress conditions range from low to moderately high, with k -ratios generally < 1 ; and, in contrast to shallow mining, the excavations are normally surrounded by an envelope of *fractured* rock.
- (ii) As a result, compressive *horizontal* stresses are induced in the immediate stope hangingwall and footwall (as a result of dilation on stress-induced fracturing ahead of the face) which help to stabilize these strata. Only in localized situations (e.g. where flat-dipping mining-induced fracturing has occurred, or when negotiating geologically-disturbed or otherwise incompetent ground) does hangingwall instability pose real problems.

(iii) Stope *closure rates* vary widely, but are generally large enough to ensure adequate build-up of support resistance in conventional soft support elements such as mat packs.

(iv) In extensively mined-out ground, the build-up of ESS (excess shear stress) levels on fault planes or remnants can lead to damaging *seismicity*.

2.3.2 Medium Depth (Scattered) Mining Strategies

Scattered mining layouts in shallow or medium depth environments give the mining engineer a great deal of latitude in carrying out ad hoc advance exploration (determining the disposition of geological structures and payable reserves), and in opening up new blocks of ground for selective mining. The routine use of good standards for layout and support, tailored if necessary to local 'geotechnical-area' requirements, leads to few problems. Nevertheless, special circumstances require individual attention.

Medium-depth (scattered) mining strategies are illustrated in Figure 2.3.1.

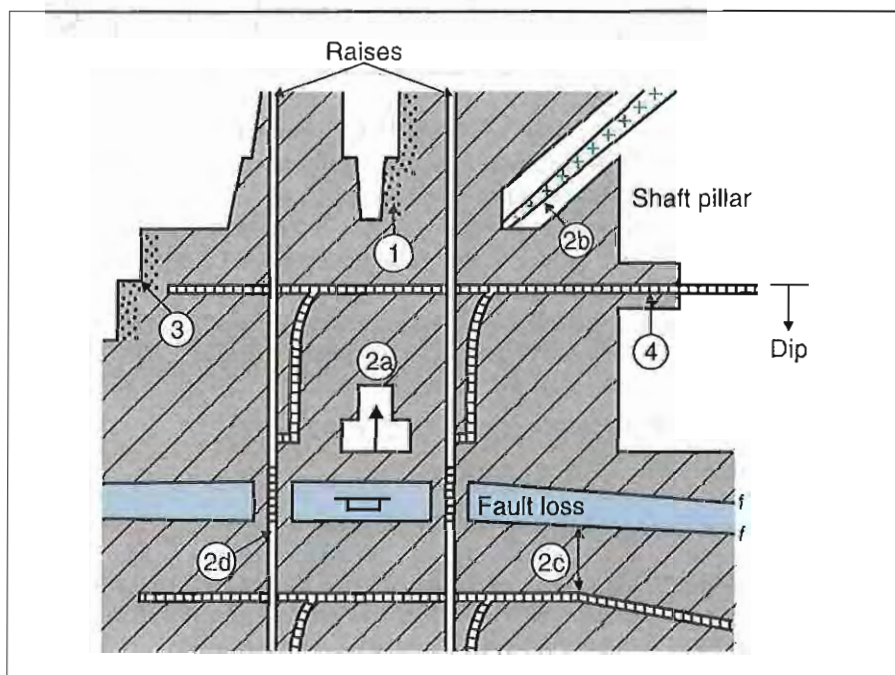


Figure 2.3.1 Illustrative MEDIUM DEPTH (SCATTERED) mining strategies

- 1: Remnant support and layout precautions in force.
- 2a: Remnant mined away from seismically active fault.
- 2b: Bracket pillars left against seismically active dyke.
- 2c: Haulage avoids proximity to seismically active fault.
- 2d: Off-reef raise ledged and protected through fault loss.
- 3: Standard support used.
- 4: Tunnel overstoped in some cases to protect from excessive stresses.

(i) **Remnants** arise frequently and can involve significant mining hazards in scattered mining, if surrounded by expansive (say >200 m span) areas of fully mined-out ground. Tailored extraction sequences and other *remnant precautions* need to be adopted.

(ii) Seismic events and resulting **rockburst** damage due to build-up of stress on remnants or geological weaknesses can arise from time to time, and seismic monitoring is essential in these conditions. Appropriate regional design criteria (such as excess shear stress ESS) can be used to assist in evaluating and modifying mine layouts to minimize the risks of seismic activity. Standard precautions should be adopted, such as the use of *bracket pillars* around hazardous structures, or the mining-out of such features at an *oblique angle* (> 35°).

(iii) The **support systems** stipulated in a particular mine's *code of practice* need to be tailored to the conditions anticipated in identified *ground control districts*. For example, low-convergence (low ERR) areas will require stiffer systems than those for higher convergence areas.

(iv) **Tunnels** in high-stress or incompetent ground require enhanced support standards, and may occasionally require appropriate overstoping protection.

2.4 DEEP and ULTRA-DEEP MINING

(Typically 2250-3500 m, >3500 m)

2.4.1 Deep and Ultra-Deep Mining Environment

Some of the features described for medium depth mining are also associated with deep level mining, but are more intensely developed and give rise to more severe mining conditions.

(i) *Stress and energy release rate* (ERR) levels are potentially high giving rise to highly fractured rock conditions. Fracture zones surrounding unprotected excavations are normally extensive: only overstoped tunnels and other protected service excavations escape very high levels of field stress. At ultra-depths, even development of tunnels in virgin ground can pose difficulties due to heavy fracturing, dilation and strain bursting.

(ii) *Hangingwall strata* are clamped by high horizontal stresses and are, generally speaking, virtually self-supporting under static conditions. The presence of adverse low-angle fracturing or jointing can nevertheless continue to give rise to problems of hangingwall stability; this is particularly evident in stope gullies.

(iii) *Stope closure rates* are high and, while rapidly generating support loads in passive soft support units, can also rapidly exceed the useful yieldability range of many such units. Total closure of the back areas is common, and stresses here regenerate slowly back to virgin levels - sometimes leading to problems in nearby conventionally-supported service excavations.

(iv) *Seismicity* associated with ruptures on geological structures is common; and bursts caused by large-scale rupture of previously intact rock also tend to increase markedly with depth, in both cases, most notably around *remnants* or other highly-stressed *abutments*. Most deep level mines therefore experience a severe *rockburst* problem.

2.4.2 Deep Mining Strategies

Deep level mining strategies are aimed at minimizing stress-related problems in stopes and in service excavations, and at reducing the incidence and effects of rockbursts.

Longwall mining layouts - introduced to minimize the formation of remnants - are often employed. The use of substantial *regional support* (stabilizing pillars, stiff backfill, and bracket pillars on geological weaknesses) is generally imperative to reduce the risks of seismic activity and resulting damaging rockbursting. *Stabilizing pillars* at about 85 % extraction are commonly laid out on strike.

So-called '*sequential grid*' mining is currently gaining favour: this involves a grid of pre-development similar to scattered layouts, with breast mining up to dip pillars left permanently unmined (these '*planned remnants*' serving as stabilizing pillars). A variant of this method features a pair of relatively long oblique-faced panels mining down-dip (or up-dip) feeding into a central raise, and flanked by dip stabilizing pillars. These '*scattered mining with dip pillars*' (SMDP) methods give considerably greater flexibility in pre-exploration and negotiation of geological structures than can conventional strike stabilizing pillar layouts.

The following general considerations apply in deep level mining (Figure 2.4.1).

- (i) Appropriately-designed **regional support systems** (stabilizing pillars and/or use of stiff backfill or concrete pillars) need to play a central role in the overall mining layout. Their function is to reduce levels of regional closure and ERR, thereby reducing the magnitude of the disturbances and stresses induced by mining. In some (though not all) senses, the use of an efficient regional support system leads to a reduction in the *effective* mining depth, in terms of the severity of the mining conditions likely to be encountered.
- (ii) Detailed mining **layouts and sequences** need to be designed to keep stress and average energy release rate levels within tolerable limits for the duration of mining activities. The formation and mining of exposed solid peninsulas and remnants need to be minimized.
- (iii) Procedures for the negotiation and stabilization of seismically-active **geological structures** need to be implemented. These could include mining obliquely through and destressing these features, or leaving specially designed *bracket pillars* to reduce their potential for causing seismicity and rockbursting.
- (iv) **Seismic monitoring**, with appropriate interpretation of results, provides a basic tool for ongoing planning and review of mining layouts and standards.
- (v) The use of methods of **rockburst control** (such as *preconditioning* blasting of highly-stressed areas) needs to be considered.
- (vi) **Local support** must cater for high intensities of fracturing, large deformations, and potential seismic loading. Rapid-yield hydraulic props and backfill are examples of suitable working area support, though properly designed pre-stressed elongates may also fulfill this role adequately.
- (vii) **Follow-behind haulages** are commonly used to avoid the very high field stresses found under and ahead of the mining faces. Overstopping protection of other service excavations is often required. For example, early overtopping of '*sequential grid*' development tunnels may be necessary in deep layouts.
- (viii) **Shaft pillars** have either to be made excessively large, or generally avoided altogether by early shaft-reef extraction (Chapter 7.3).

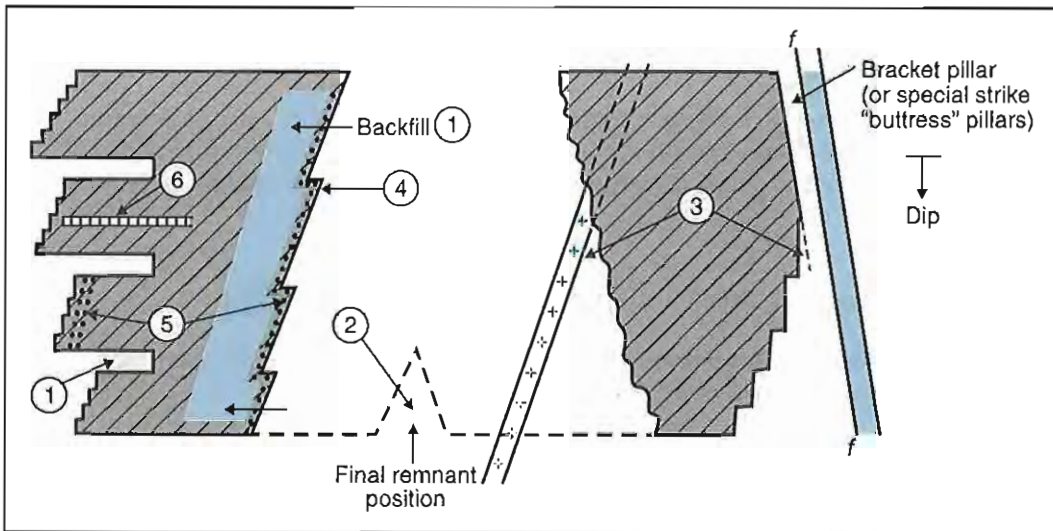


Figure 2.4.1 Illustrative DEEP mining strategies

- 1: Use of regional support (stabilizing pillars, stiff backfill kept close to face).
- 2: ERR control: number of remnants minimized (ideally against unmined abutment)
- 3: Geological structures approached obliquely, and mined through or given substantial bracket pillar protection.
- 4: Longwall faces with controlled leads/lags.
- 5: Local support tailored to high closure rates and potential seismicity (rapid-yield hydraulic props, prestressed elongates, moderate yield-force packs on gullies).
- 6: Follow-behind haulages avoiding high face stresses.

2.4.3 Ultra-Deep Mining Strategies

No mines currently operate deeper than 3600 m, yet very large potentially payable reserves are known to exist at depths down to 5000 m. The high rock temperatures and stresses, coupled with potential severe rockburst hazards, obviously require significantly improved mining methods for viability at these depths. A number of issues are currently under investigation.

- (i) Single-lift shafts operating to depths in excess of 4000 m. These are necessary for economical early exploitation of the orebodies, and will require (probably attainable) improvements in winding ropes and other winding technology.
- (ii) Drastically increased *face advance rates* through the use of innovative forms of stope mechanization. The resulting more concentrated forms of mining would facilitate cooling and ventilation, and make the necessarily more expensive forms of regional and local support more economically viable.
- (iii) Reduced *extraction ratios* and smaller *open spans*. These could reduce even ultra-deep mining energy release rate (ERR) levels to as little as 10-30 MJ/m², maintain a largely elastic regional environment, and contain maximum stope closures to 200 mm or less.
- (iv) The use of *stiff backfill* and *concrete fill* materials as regional support to supplement the use of stabilizing pillars for improved overall extraction ratios, as well as to assist in cooling and ventilation control.

(v) Improved *stope support* systems to cater for the novel mining methods envisaged, and to counter the probably increased levels of seismicity.

(vi) Drastically improved *tunnel development and support* methods to counter the large field stresses to be expected. Such improvements will be essential in order to mine deep orebodies, especially those located in less competent rock, and to exploit the considerable potential advantages of 'sequential-grid'-type mining at ultra-depths.

2.5 GENERAL ROCK ENGINEERING STRATEGIES

A number of important overall rock engineering strategies are available to expedite the mining considerations outlined so far.

2.5.1 Seismic Monitoring

The installation of one or more seismic monitoring networks is an essential strategy on any mine faced with a significant incidence of seismicity and rockbursting. The complex and evolving issues involved are discussed in depth in Chapter 9, but a simplified introduction is presented here.

A standard **seismic network** comprises a minimum of four, but in practice at least six to 30 operational geophones, arrayed around an area of interest. For acceptable vertical location accuracies, a minimum number of these must be situated well below and/or above the reef horizon. Geophones detect *ground velocities* incident from a seismic event. Signals are transmitted over telephone wires or radio links and storage and processing of data are carried out on central mini- or micro-computer systems.

A typical *regional* network spans an entire mining district, and depending on the number of geophones installed, the achievable location accuracies range from 50 to 200 m. Typical close-in or 'mini' networks may cover an area of around 1 km² and can achieve location accuracies of 10 m. They are being increasingly used for monitoring seismically active structures. Specialized *microseismic* networks have been used for research purposes only. They can achieve location accuracies of the order of a few metres, and have been used to delineate fracture processes, to sense precursory activity prior to major events, and to plot aftershock areas.

In order of increasing complexity, seismic networks can:

- (i) detect arrival times of seismic signals and 'locate' events, that is, compute the co-ordinates of source hypocentres, and detect energy or amplitude characteristics and assign *magnitudes* (*M*) to the events. These are minimum requirements for building up patterns of seismicity for mine planning purposes, and for co-ordinating rescue operations in the case of individual large events or rockbursts;
- (ii) detect, store and process full signal *waveforms*. This is a topic of on-going research, but important practical applications are beginning to emerge. For example, waveform processing permits estimation of event *seismic moment* (proportional to area of rupture times mean slip) which allows more reliable assignments of magnitude *M*. Waveform analysis can also enable full moment tensor inversion for identifying the orientation and nature of rupture planes and thus the structures associated

with particular significant events. Other potentially important parameters, including stress drop and source dimension, can likewise be quantified - Chapter 9.

Prior to the occurrence of a major seismic event, there is sometimes precursory activity in the form of clustering of smaller events in both time and space, as well as shifts in other parameters of quantitative seismology - Chapter 9. Attempts to use in-depth seismic monitoring as a fully practical tool for prediction of rockbursts and avoidance of risk exposure have so far proved elusive, but research in this area is continuing.

Another area of research is in rockburst control, addressing the two forms of seismic activity illustrated in Figure 1.4.6b. The first concept here is of *preconditioning* (destressing): the use of controlled blasting to remove or withdraw the burst potential of remnants, dykes or other zones of hazardous ground at or near the working face. This technique is beginning to show promise in the extraction of remnants - Chapter 8.4 - and is doubtless applicable in other situations, for example the pre-fracturing of crush pillars which could otherwise pose problems in certain shallow mining layouts. The second, so far unsuccessful, concept is of *controlled fault slip* ('triggering'): the injection of pressurized fluid or of blasting to induce over-stressed fault planes to slip at a controlled time.

Apart from the obvious importance of seismology in rockburst research, seismic networks are already bringing clearcut operational benefits to mines where they are installed:

- (1) Source locations can be supplied to management within minutes of the occurrence of large events, and this information can greatly assist in the rapid planning of appropriate rescue operations.
- (2) Patterns of seismicity can be built up which are more reliable than statistically-sparse rockburst damage reports, and which can assist significantly in medium-term mine planning. Examples include the detection of critical remnant dimensions, of unfavourable face orientations, and of seismically-active structures such as faults or dykes which traverse as yet unmined sections of a property. Remedial layout and support strategies can then be applied in good time.
- (3) The overall efficacy of rockburst control strategies adopted by a mine can be assessed by comparing with seismicity levels pertaining in the past, or between neighbouring areas using differing strategies. Examples include the efficacy of stabilizing pillar, bracket pillar or backfill control strategies (Chapter 3.3), or of preconditioning or triggering experiments.

2.5.2 Advance Exploration Of Geological Structures

Advance exploration techniques (long surface or underground boreholes, surface vibro-seismic mapping) are expensive, yet the returns often justify the expense ten-fold or more. The costs of inappropriate 'blind' development or stoping into faulted-out or otherwise non-viable areas is immense and, in deep-level stoping, there are major hazards in stoping breast-on up to unexpected geological planes of weakness.

The appropriate use of advance development (where possible), together with forward drilling and other available geophysical mapping technologies, is therefore strongly advocated.

2.5.3 Ground Control Districts

The concept of '*ground control districts*' (sometimes denoted '*geotechnical areas*') is a pragmatic one: the identification of distinct areas on a mine where different support (or strata control) standards need to be applied. The criteria for distinguishing separate ground control districts vary, but include obvious characteristics such as major changes in dip or stope width, as well as less obvious parameters such as field stress level or hangingwall rock quality (c.f. Chapter 4.2).

2.5.4 Risk Assessments

Formal *risk assessments* (Chapter 12.8) were introduced in the mines a few years ago, and have been very successful in improving safety standards, hazard identification, job training, and communication between senior staff and operators. Rock-related risk assessments can be greatly enhanced by analyses of accidents and other major incidents - Chapter 1.2 gave industry examples in which the stope face and face/gully areas were identified as especially hazardous localities. Similar studies on a particular mine can highlight local '*hot spots*'. Examples include critical dimensions of small blocks of ground for which remnant precautions need to be adopted; hazardous geological structures; and abnormally dangerous localities. Remedial measures, including appropriate changes in mine standards and codes of practice, can then be more rationally motivated.

As part of this process, detailed records of rockfall and rockburst injury-causing incidents need to be kept and analyzed to determine *root causes* of the accidents. These causes can be much deeper than a mere identification of how a particular incident occurred. The adequacies of training, supervision, control systems, managerial structures and in fact the entire risk management system may need to be probed.

The number of rock-related casualty-causing incidents on a mine and in individual ground control districts is limited. There is nevertheless a great deal of potentially useful data on a mine involving ground collapses which is not usually recorded and analyzed. For example, it is not generally recorded how many rockbursts occur per annum, nor how the rockburst rate varies with mining depth or with energy release rate. There is strong motivation therefore that each mine should establish a *data base* including pertinent details of all significant rockfalls and rockbursts, both those that involve casualties and those that do not. The type of data that is useful to collect is indicated in the new fatality accident data capture form 13, designed for the Department of Minerals and Energy. A system, to ensure that data from all significant incidents are captured, should be described in the code of practice and the minimum essential data that needs to be recorded for all incidents should be prescribed, e.g. depth, location, dimensions, rockfall or rockburst. These data and other production data (including annual m^2 for each reef mined and underground staff complements in appropriate depth ranges), necessary to normalize the figures, will then be available to be collated on a mine-wide and ultimately industry-wide basis.

2.5.5 Auditing

In view of the shortage of rock engineering expertise on the mines, it may often be prudent to undertake *auditing* exercises (Chapter 10.7). Here, consultants (experts

from sister mines, head offices, universities or consulting organizations) can be asked to *audit* proposed or existing layouts, mine standards and codes of practice. If such an exercise were to lead to just *one or two* significant improvements, the resulting impact on improved safety and reduction in costs could be very considerable.

2.5.6 Support Quality Control

The implementation of a sound *support quality control programme* (Chapters 4.8, 6.5.3), covering standards of installation and integrity of support elements used, is necessary to ensure the ongoing safety and cost-effectiveness of operations on a mine.

The integrity of support elements may be checked by carrying out random *sample testing* (in the case of timber units which are susceptible to drying-out, both on receipt and immediately prior to being sent underground). Particular attention needs to be paid to *hydraulic prop control*: special procedures need to be set up for the efficient testing, maintenance and deployment of these units. Similar requirements apply to *backfill* quality control.

The importance of standards covering *making-safe* procedures as well as *quality of installation* of temporary and permanent support units cannot be over-emphasized. These standards will vary considerably according to local conditions, but appropriate ongoing management attention to aspects of standard-setting, training, discipline, and supervision, will pay significant dividends in terms of enhanced safety and productivity on the mine.

The unpopular but necessary aspect of *support monitoring* cannot be overlooked: the checking on a random basis that standards are being adhered to, that installed support units meet required pull-test standards, that inappropriate short-cuts are not being followed, and that contingency support systems are available and are being used wherever necessary.

2.5.7 Training

A better understanding of rock engineering principles by all underground workers will contribute directly to improvements in safety and mining productivity. A syllabus of *training topics*, tailored for individual levels of mine staff, is suggested in Chapter 12.

2.6 CODES OF PRACTICE

A mine's *code of practice*, whether compiled on a mandatory or voluntary basis, is an excellent means of developing and documenting comprehensive and efficient strategies to combat rock-related hazards.

Guidelines for the preparation of codes of practice (prepared by the tripartite MRAC Task Group on Rock-Related Safety) are issued by the Chief Inspector of Mines. These guidelines, though not prescriptive in detail, thus allowing knowledge and experience of local conditions to be applied, do identify all issues which need to be covered in the document. Mine standards and other managerial instructions, which need not form part of the actual code of practice, must nevertheless comply implicitly with the strategies outlined in the code.

While compiling the code of practice, the drafting committee must bear in mind that both management and workers should contribute to the document and have ownership thereof. To facilitate this, the document should contain those day-to-day aspects for which strategies or standards are already in place (such as early and mid-shift examinations, which have proved highly successful in containing falls of ground in the area adjacent to working faces).

The development of the various strategies should follow an ordered procedure. Initially, the *overall mining environments* need to be described. These will be governed by major factors including depth and dip, reef width, areas of multiple reefs, degree of faulting, potential for seismicity, influence of previous mining both in the lease area and adjacent properties, surface structures and environmental features which may affect mining.

Within these environments, specific *ground control districts* may need to be defined and delineated. The discrimination of these areas should be based on known hazards associated with various geological and mining conditions, and must take into consideration how these conditions will affect local and regional support design, stoping layouts, mining methods and mining direction and any other rock engineering strategies pertinent to the mine. It should be noted that ground control districts for off-reef and on-reef excavations may be different, and that the method of defining such areas could also be significantly different. The factors used in delineating ground control districts should be discussed and described in the code of practice.

To assist in identifying the hazards associated with the various geotechnical conditions, formal *risk assessments* should be carried out in accordance with the Mine Health and Safety Act - see section 2.5.4 for a discussion of these important issues.

Having defined the mining environment, established the potential hazards and evaluated current risk control measures, the next aspect in the code of practice is to establish the **overall rock-engineering strategies**. These comprise three main issues.

- (i) The overall stability of the mine and shaft systems. This will usually involve *regional support* systems, as well as *overall mining layouts and sequences*. In this regard *numerical modelling* requirements and methodologies need to be outlined.
- (ii) *Local support* requirements for stopes, tunnels and large excavations, together with the specification of layout features which influence strata control such as gullies, lead/lags, and excavation sequences for large chambers. Other rock-related risk control measures such as early examination/making-safe procedures and material quality control should be described.
- (iii) *Rockburst control*. The issues that need to be considered are firstly seismic monitoring requirements, data analysis and how the results will be incorporated into mine planning procedures; secondly how mining in the vicinity of seismically active structures will be carried out; and thirdly if control measures such as preconditioning are to be implemented, under what circumstances they must be used and how the systems should be designed.

Both *long and short term* planning and monitoring procedures need to be elucidated. This involves describing the chronological sequence of events, who must be involved, who is responsible for what and what sections need to be countersigned by

whom, record keeping and follow up procedures and the distribution of the final plan or decisions made at a particular meeting.

Design methodologies used in the above strategies need to be described in some detail and, where *criteria* are applied, the values used need to be motivated. For example, with the ERR criterion on the Carbon Leader Reef, a figure of 20 - 30 MJ/m² is often accepted as a reasonable value for design purposes. However, there are situations where a lower value is necessary or a higher value may be acceptable. These values should be established by back-analysis for a particular mine, mining environment or reef, and discussed in the code of practice. Where strategies or design methods differ for different geotechnical conditions, these need to be motivated and described.

Remnant conditions and 'special areas' could be considered a subset of ground control districts (for example, wherever the ERR exceeds say 50 MJ/m²) with the appropriate stratagem described. However, particular issues such as the procedure for the declaration of a special area and the constitution of a Special Areas Committee as described in the Chief Inspector of Mines guidelines, need to be addressed.

Monitoring is an important aspect of a rock engineering strategy, and the required procedures must be described. The monitoring could vary from simple daily reporting of visual assessments of rock conditions, through routine closure or backfill porosity measurements, to sophisticated instrumentation programmes. Regular review of mine plans by suitably qualified rock engineering personnel to determine adherence to mine layout criteria is also considered a form of monitoring. Assessing the quality of implementation of rock engineering design and support installation is another important issue to be described.

The final technical issue requiring elaboration in the code of practice is **blast design**. This may vary between ground control districts, and the blasting procedures for special situations such as ledging and undercutting require description.

Apart from the technical issues, three other important matters need to be addressed in the code of practice. These are:

- To define who is *responsible* for controlling certain hazards or procedures and the managerial procedures or standards which must be applied.
- The adequate and effective *training* in strata control of all personnel up to a standard such that they can perform their duties safely and effectively; as a result of an understanding of the nature of the hazards, the reason for the implementation of particular procedures and the implications of not carrying out the procedures properly.
- The function of the *rock engineering service*. It is important at the outset to identify the rock engineering requirements of a particular mine. This will depend to a large extent on the complexity of the rock-related problems and hazards expected and the size of the mine. The requirements could vary from periodic visits from an outside consultant, to a fully staffed rock engineering department integrated with a seismic section and with a formally coordinated and functional relationship with the geological department on the mine.

3.1 INTRODUCTION

Tabular hard-rock mining commenced on a large scale in South Africa with the discovery of the extensive Witwatersrand goldfields. In the early decades of this century, incline shafts had opened up huge extents of Main Reef, and other closely associated reef horizons. Scattered mining on mine pole or mat pack supports, often supplemented with pillars and sandfill or waste packing, was used with considerable success. However, remnant-generated rockbursting as well as strata control in reef drives and elsewhere, became serious problems by the 1950s and led to the general adoption of longwall layouts in the deeper mines in the 1960s. At this time, major advances in rock mechanics theory and rock engineering practice began to emerge. Stabilizing pillars made significant contributions to regional support in the deep mines along with the introduction of effective rock-burst-resistant support systems (rapid-yield hydraulic props, tailored elongates, yielding tendons, backfill). Numerical modelling programs began to reveal their power in explaining and quantifying the stresses and deformations resulting from mining operations.

Room and pillars remained the layout of choice in shallow tabular base-metal (chrome and manganese) operations. Crush pillars, an innovation from moderate-depth gold mining practice, proved highly successful in the platinum mines of the Bushveld Complex, as these mines too started mining down to greater depths.

Despite the advances that have been made, rockfalls and rockbursts continue to pose major problems for the Industry (Chapter 1.2). Well-designed **stopping layouts** provide both strategic and tactical means for alleviating these problems.

As discussed in Chapter 2, the importance of stopping layouts with respect to the control of rockfalls and rockbursts increases with increasing mining depth. At shallow depth, provided appropriate pillar configurations are adhered to and adverse geological structures are not encountered, stope layout details tend to be less relevant to the control of rockfalls than the use of proper local support standards. As depth increases, this latitude diminishes until, at great depth, stope layouts are of vital importance in controlling the hazards of rockfalls and rockbursts.

In deep mines, optimum stope layouts attempt to maintain face stresses as low and as uniform as possible during the extraction of an entire area. The energy release rate (ERR) criterion has been used as an aid in the assessment of average stress levels, as well as an indicator of possible seismic incidence in deep geologically-undisturbed mining situations. Similarly, the excess shear stress (ESS) criterion may be used to give a semi-quantitative assessment of the effect of stope layouts on the seismic potential and stability of geological

structures. These and other commonly-used design criteria are reviewed in the following sections. In practice, no 'perfect' layout exists, and a process of comparing various layouts needs to be undertaken in order to reconcile and optimise often conflicting requirements.

3.2 MINE LAYOUT DESIGN CRITERIA

Rock engineering is not an exact science, and many of its precepts are qualitative in nature. Nevertheless, certain *quantitative* criteria have been proposed over the years, and a number have been shown to have application in mine design, or in mine layout monitoring. The more popular and relevant of these *layout design criteria* (which generally require numerical modelling for their estimation – Chapter 11) are outlined in the following sections. In addition, simple *rock strength criteria* are reviewed in Chapter 1.3.2, specialised *seismological criteria* in Chapter 9, *tunnel support design criteria* in Chapter 6, and *stope support design criteria* in Chapter 4.

3.2.1 Energy Release Rate (ERR)

In deep mines the virgin rock stresses are high, increasing by about 27 MPa per 1000 m of depth. Removal of this rock through mining results in energy changes due to sag of the massive overlying strata, and in redistributions of stress from the mined to the unmined areas of the reef. Stress concentrations build up to high levels, particularly in front of the mining faces and most noticeably in lagging corners of the face, in peninsulas and in remnants. The ERR concept, introduced in the 1960s, is a convenient and easily calculated measure of these energy changes and stress concentrations, and of some of their effects on the tabular mining environment. The average level of ERR (expressed in MJ/m²) pertaining at a particular set of mining faces takes into account the effects of *depth* and *geometry of neighbouring mining*. It is related to the extent of **volumetric convergence** taking place, or being permitted to take place, in the back areas of the stope.

If a horizontal stope is enlarged by mining an area ΔA , and the resulting change in volumetric convergence in the stope is ΔV , then the change in potential energy of the overlying strata is $q_v \Delta V$, where q_v is the virgin vertical stress. If no significant support (such as backfill, or total closure on the reef horizon) is present, then one-half of this energy change is stored as extra strain energy in the rock mass. The remaining one-half is immediately *released* (largely in the form of shearing/crushing/heating of the fracture zone in front of the face). This released energy, in MJ per unit face advance area ΔA , is called the (spatial) **Energy Release Rate, ERR**:

$$ERR = \frac{1}{2} q_v \Delta V / \Delta A \quad (\text{fully open stope})$$

If significant support is present, then the proportion of stored strain energy decreases and the balance making up the ERR increases, until in a limiting case, such as a very large-span heavily-closed stope, the ERR accounts for nearly 100 % of the potential energy change. There is no further increase in stored strain energy, though a zone of high stresses continues to migrate through the rock mass ahead of the stope as mining progresses. If the effective stope width is S_w , the formula now reads

$$ERR = q_v \Delta V / \Delta A = q_v S_w \quad (\text{heavy total closure present})$$

If the stope is not horizontal, then the value of q to be used is the component normal to the reef horizon. In a dipping stope, there are in fact further energy changes associated with *ride* in the stope, but these are an order of magnitude smaller than the normal ERRs and are usually ignored.

A second, and more useful, method for computing ERR can be explained in terms of a simple 'thought experiment'. Imagine that an area ΔA of the rock in front of a 'stiff' (virtually incompressible) mining face is removed and instantly replaced with notional 'support units' which carry the existing stress σ_b . No change in convergence, stress or energy has yet taken place. Now, the loads on the 'supports' are reduced to zero (simulating mining), and the surrounding rock mass converges and does work on the 'supports' to the extent of $\frac{1}{2}(\sigma_b \Delta A) S_a$, where S_a is the convergence on the 'supports' behind the new face *after* completion of the mining step. Invoking a well-known theorem from engineering statics, an equal quantity of energy will be released in the rock mass during this process of doing work on the 'supports', and therefore $ERR = \frac{1}{2} \sigma_b S_a$; that is, the ERR at a specified point at a (stiff-reef) mining face is 'one-half the product of face convergence and face stress'. Many popular numerical modelling programs publish estimates of ERR based on this simple formula ($ERR = \frac{1}{2} \sigma_b S_a$), and a meaningful ERR level for a *set* of points on a panel face or entire longwall can then be established by manual averaging.

What is the 'Energy Release Rate'? It certainly is *not* a direct expression of seismic energy releases due to mining. In fact, most of the ERR energy (> 97 %) is released aseismically in the form of fracturing, crushing and sliding of blocks in the fracture zone in front of the face (ending up as heat, adding a few percent to a deep mine's refrigeration requirements). Seismic events, on the other hand, are unstable time-dependent failures which release strain energy already stored in an over-stressed rock mass; however, the probability and magnitude of such failures tends to increase with *stress level* and hence, as will be seen, with average ERR.

ERR is actually a concept borrowed from engineering fracture mechanics. In this discipline, the intensity of stress at a point located a (small) distance r from the tip of a crack is described by a 'stress-concentration factor' K_I , such that $stress \propto K_I / \sqrt{(2\pi r)}$. The ERR associated with enlarging the crack is *directly related* to K_I , via the elastic constants E and ν , according to $ERR = (1-\nu^2) K_I^2 / E$. These entities retain their relevance even when a small non-linear 'process zone' of damaged material (similar to the non-linear fracture zone in mining stopes) is acknowledged to exist at the crack tips. Figure 3.2.1 illustrates the situation of a 100 m span stope at 3000 m depth, in which the 1 m reef horizon is modelled not only with realistic compressibility, but also with finite triaxial strength ($UCS = 200$ MPa). In this example, the 'process zone' (fracture zone of fully or partially crushed rock) that arises in front of the face is about 3 m deep. However, this together with the reef compressibility, merely has the effect of enlarging the effective span and associated ERR by about 3 %; and the K_I factor continues to describe the stress-concentration profile *in front of the immediate fracture zone*.

It follows that, contrary to an opinion sometimes expressed, the relevance of ERR is *not* limited to *perfectly elastic* situations. Provided the rock mass as a whole remains

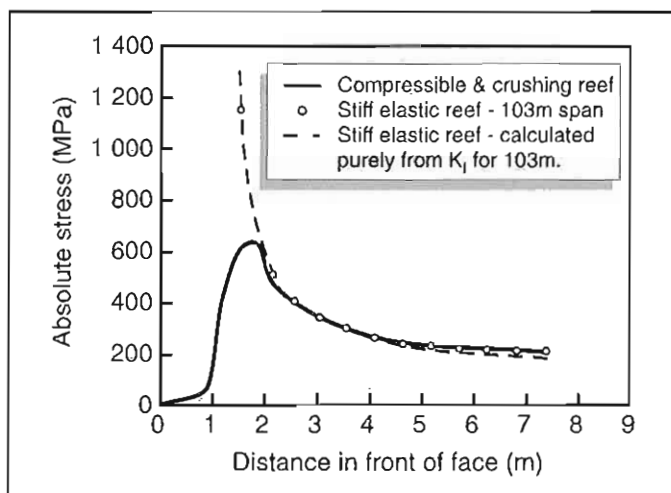


Figure 3.2.1 Modelled stresses in front of face of 100 m span stope at 3000 m depth. Solid curve – reef with finite triaxial strength and realistic compressibility. Points – perfectly elastic and stiff reef, but span enlarged to 103 m. Dashed curve – corresponding stresses evaluated purely from the K_1 stress concentration factor for the 103 m span stiff elastic reef.

largely elastic and the fracture zones represent less than say 5-10 % of the mined spans (both reasonable assumptions in most South African hard rock environments), ERR values calculated by traditional methods are acceptably accurate and relevant. This is not to say there are *no* difficulties in correctly evaluating ERR; but these problems are mainly of a numerical nature or are concerned with selection of appropriate modelling parameters for deep mining environments, such as the correct behaviour of pillar foundations or best choice of equivalent elastic moduli or stope widths (c.f. section 3.3.2).

Thus K_1 (measured in units of $\text{MPa}\cdot\text{m}^{1/2}$), or equivalently ERR (expressed in the more familiar units of MJ/m^2), give a direct measure of the *stress concentration* in front of a given mining face, and other significant aspects of the expected mining environment. Figure 3.2.2 summarises these expected effects of ERR, and schematically contrasts them for low and high stress mining environments.

Average ERR can therefore serve as a *descriptive measure of the mining environment*, and can aid in the selection of layout or support systems appropriate for different areas of a mine.

Nevertheless, ERR does *not* appear to correlate well with *rockfall* hazards - Figure 3.2.3, and also Chapter 1.2.2. This is partly due to the effectiveness of support systems currently in use (which tend to be tailored to prevailing ERR and related seismicity levels), and partly to the fact that high ERRs do not necessarily imply poor hangingwall conditions. The greater depth of fracturing and fracture movement in high ERR environments tends to be stabilised by correspondingly higher degrees of dilatatory clamping forces and generated support resistances (entries 4 and 5 in Figure 3.2.2).

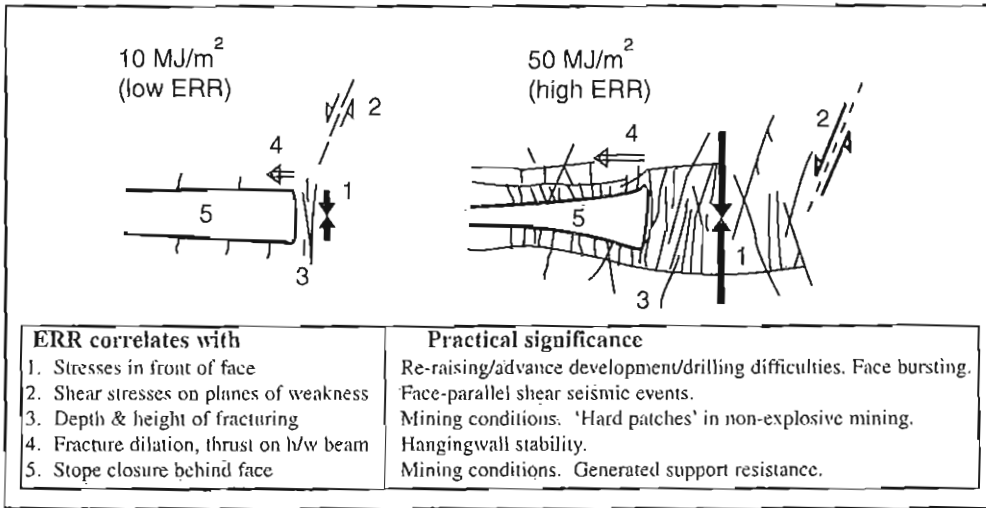


Figure 3.2.2 Expected effect of ERR on mining conditions

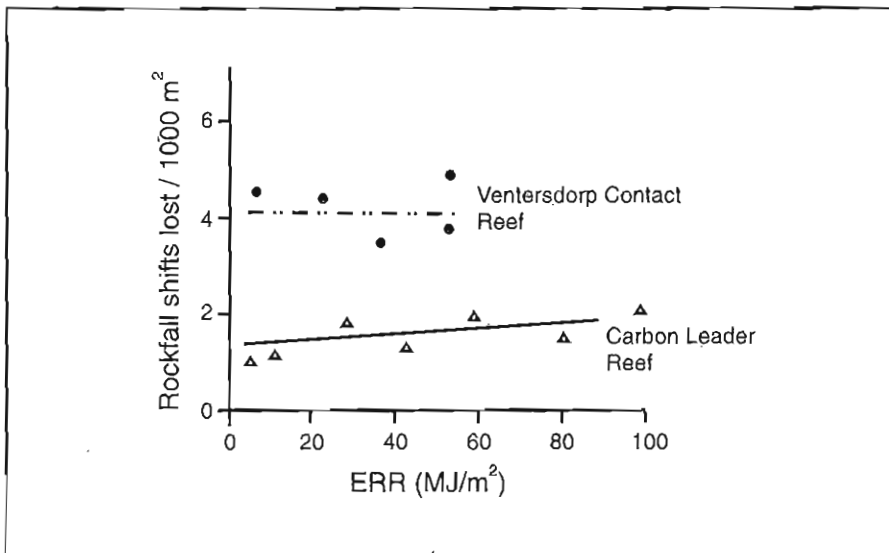


Figure 3.2.3 Weak correlation of FOG incidents with ERR level for a longwall mine

Rather, the *rockfall* hazard can be described by three *strata control* factors which in general are not related to ERR, but which may be exacerbated in situations where ERR is also high: (i) the presence of adverse geological factors such as transverse joints, weak partings, or shale in the hangingwall; (ii) the presence of adverse low-angle fractures such as are encountered in ledging stopes, in stope leads and face irregularities, in high-stress advance gullies, and where jointing is sub-parallel to stope faces; and (iii) the quantity and quality of installed support.

3.2.2 ERR: A Measure of Rockburst Hazard in Longwall Mining

Energy release rate may be equivocal in predicting *rockfall* hazards, but the situation is quite different when considering *rockbursts*. Studies carried out in deep longwall-type mines, have shown a convincing correlation between average ERR level and seismicity or incidence of damaging rockbursting - Figure 3.2.4. Analyses of casualty data (Chapter 1.2.2) have moreover suggested that, for the mining industry as a whole, the risk of fatal accidents due to rockbursting increases with depth at much the same rate as does the general level of ERR, quite unlike the situation with respect to rockfalls. This behaviour is expected from entries 1 and 2 in Figure 3.2.2.

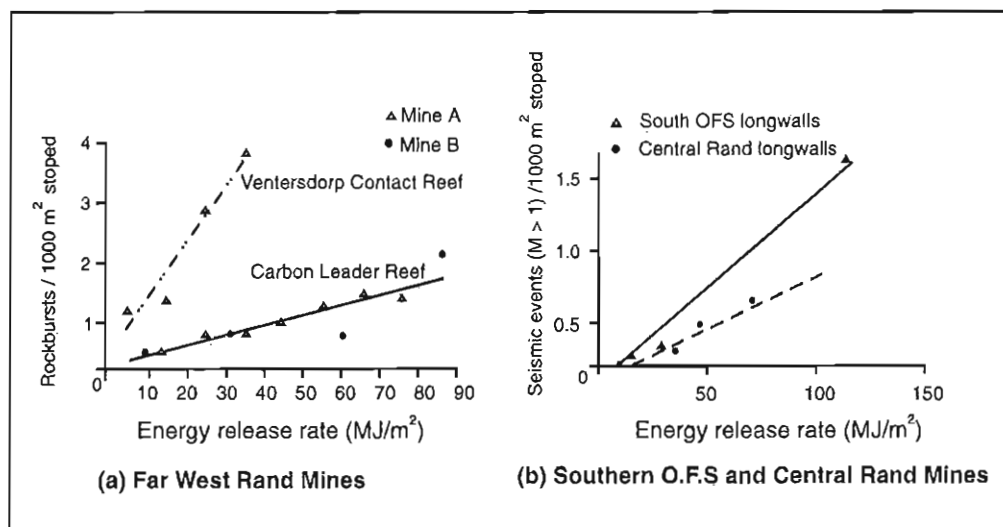


Figure 3.2.4 Rockbursts and seismicity versus ERR. Data from four longwall mines prior to the introduction of stabilizing pillars.

Thus, the design of deep geologically-undisturbed mine layouts can realistically be guided in terms of the ERR criterion. **Regional support** systems - stabilizing pillars or extensively placed stiff backfill, section 3.3 - can be expected to have a containing influence on the rockburst hazard at source, since they directly act to reduce stope closures and hence ERR levels and stresses.

The critical levels of average ERR at which particular design standards should be exercised is however strongly dependent on *local geotechnical conditions*. For example, for the mines in the environment and at the time illustrated in Figure 3.2.4a, a much lower ERR target would have been appropriate on the VCR than on the Carbon Leader mining horizons, or else other drastic control measures introduced in the mining standards for the former reef.

It is necessary therefore that mines analyse the incidence of seismic events and rockbursts for different mining environments, in order to set appropriate values of their ERR criteria for evaluating mine layouts and mining sequencing.

3.2.3 Excess Shear Stress (ESS)

Numerous statistical and seismological studies have confirmed that mining in the vicinity of geological structures - faults, dykes, joint sets - greatly increases the incidence of seismic events and accompanying rockbursts. In geologically-disturbed mining districts (for example, in the Klerksdorp and Northern OFS goldfields), virtually all large seismic events seem to take place *on geological structures* rather than on the mining faces themselves - Figure 3.2.5. Some events take place (presumably in abnormal tectonic stress situations) even when the nearest mining may be many hundreds of metres away. Moreover, in the relatively undisturbed Far West and Central Rand districts, the presence of occasional faults or dykes very significantly increases the frequency of rockbursts - Figure 3.2.6.

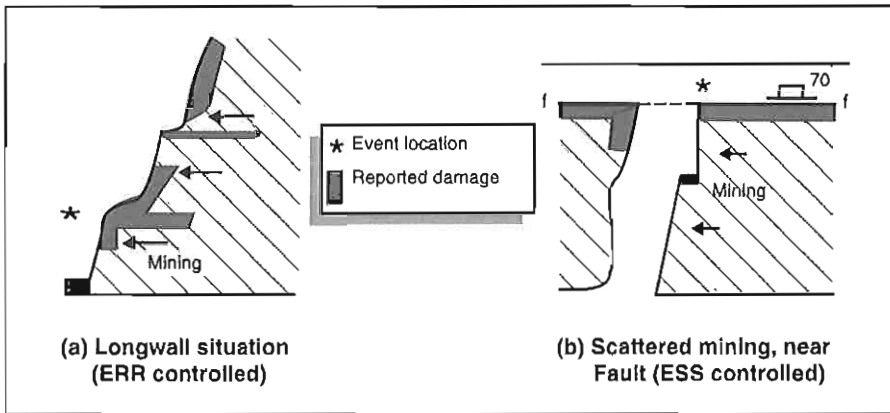


Figure 3.2.5 Location of typical seismic events in geologically-undisturbed and in highly-faulted mining areas.

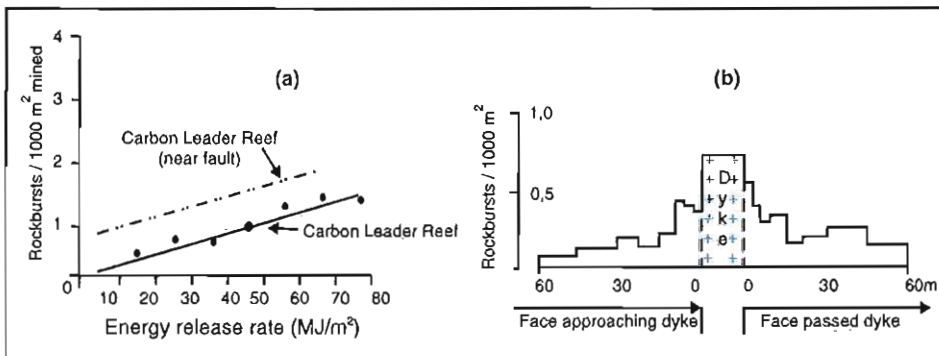


Figure 3.2.6 Effect of proximity to faults/dykes on rockburst incidence.

The shear-type seismic event mechanism sketched in Figure 3.2.7 provides a simple explanation for many of these effects. A plane of weakness in an otherwise strong rock mass can be subjected to shear stresses induced by mining, and, once the cohesive strength of such a plane is exceeded, overstressed asperities will shear and a slip or rupture event can take place. **Excess shear stress** ($ESS = \tau - \mu \sigma_n$) measures the level of effective driving shear stress on a plane of weakness or potential rupture

plane. Areas where the ESS is negative signify probable stability, whereas lobes of **positive ESS** in the rock mass signify *potential instability* together with a quantifiable estimate of the *maximum likely event magnitude*. Calculations have shown that this mechanism can account for seismic events covering a wide range of magnitudes, from less than 1 up to 4 or more - Figure 3.2.8.

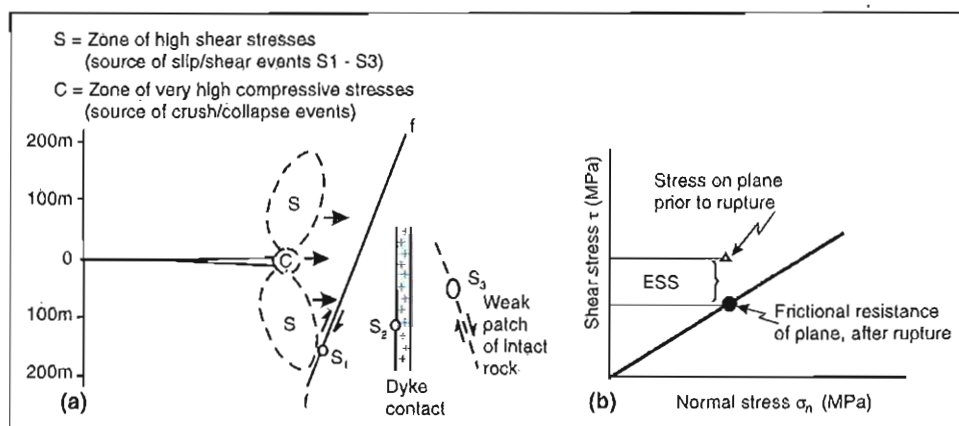


Figure 3.2.7 Mining-induced shear stresses (ESS) triggering seismic events

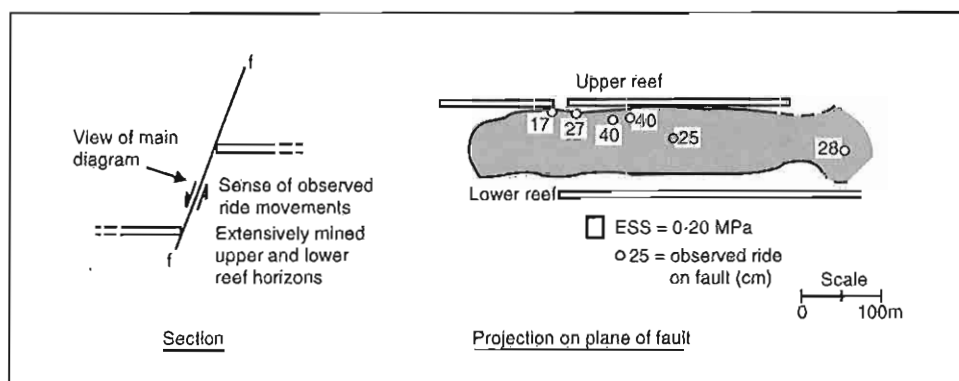


Figure 3.2.8 Back-analysis of a large ($M = 4.5$) seismic event

While ERR may be useful in estimating *overall* levels of seismicity, ESS is capable of explaining a number of more specific rockburst situations. Thus, ESS levels immediately account for the hazard-amplifying effect of faults and joint sets, or of abnormal tectonic stress situations. The desirability of mining obliquely up to fault planes, or 'north-siding' of remnants, is explained in terms of the reduction in size of the ESS lobes acting on the structures concerned, whereas ERR is insensitive to these aspects of the mining geometry.

Unfortunately ESS analyses require reasonably precise knowledge of the (i) state of virgin stress in the area, including virgin horizontal stress components; (ii) strength properties of the rupture plane - cohesion, dynamic friction coefficient, and planarity; and (iii) history of major recent slip events, which could have re-distributed stresses and drastically modified the cohesion properties of the plane.

In practice, these parameters are known imperfectly at best. Thus, ESS analyses are probably best suited to gaining *qualitative* insights, such as in the following areas: (i) estimating an upper bound of magnitude of potential seismic events near given mining layouts, especially near large known seismically-active faults; (ii) assessing the likely impact of regional support alternatives (pillars or backfill) in containing the magnitude of such events, and the merits (and potential hazards, section 3.3.4) of the common practice of leaving bracket pillars of unmined ground immediately adjacent to active faults or dykes, or of mining on 'retreat' (away from) such hazardous structures; and (iii) designing mining layouts and sequences which break up or reduce the size of ESS lobes, thereby restricting the magnitude of potential seismic events.

Both the ERR and ESS criteria strongly indicate the need for *regional support* in deep mining, that is, systems which provide stiff support in the back areas of large-span stopes so as to reduce volumetric closures and hence stress concentrations in front of the mining faces. Layout design principles for the known methods of implementing regional support - use of stabilizing pillars, stiff backfill, or concrete pillars, as well as the use of bracket pillars to stabilise hazardous geological structures - are discussed in section 3.3.

3.2.4 Volume Excess Shear Stress (VESS)

The use of ESS as a design criterion is based on the identification of geological features, knowledge of their position and properties and the assumption that damaging seismicity concentrates on these features. VESS is complementary to ESS in that it is based on the possibility that rock can fail anywhere that its strength is exceeded. As the rock mass is intersected by numerous faults, joints and dykes, all possible weaknesses are 'seen' as being a possible seismic source if the maximum ESS in *any* direction is positive. Implementation of VESS requires numerical modelling of the mining in small increments and analysing the changes in ESS at numerous points in the rock mass.

Studies at a mine in the Carletonville area suggested that VESS was at least as accurate as, or better than, ERR in modelling the relative hazards of seismicity or rockburst incidence.

If there are dominant joint sets, then a special case of VESS, namely Ubiquitous Joint ESS, UJESS, can be used. UJESS has the special advantage that it can focus the mine design considerations towards mining at oblique angles to dominant joint sets.

3.2.5 Rockburst Hazard Index (RHI)

Both the ERR and ESS concepts suffer from a number of shortcomings when used for practical mine design. For example, ERR ignores the presence of geological structures, while ESS is often difficult to evaluate and interpret when comparing different mining options. A new indicator of the potential for seismicity has been developed called the rockburst hazard index (RHI). This index combines three parameters

that are related to the likelihood of rockbursts occurring: the volume of excess shear stress (VESS) in the rock, the stress at the face according to ERR (section 3.2.1), and the ESS along major geological structures. The RHI is calculated at each reef element mined in a particular mining increment in a MINSIM-type numerical model. It is determined by summing the changes in VESS and ESS and weighting with a factor dependent on the local ERR.

Examples of RHI results are presented in Figures 3.2.9, where the differences between two methods of approaching a major fault are evaluated. The results clearly show a significant increase in the RHI in the last stage of mining breast on to the fault, while the RHI in the oblique mining option remains at lower levels. The RHI has also been shown to correlate well with expert opinion regarding the relative risks of mining in a number of other environments.

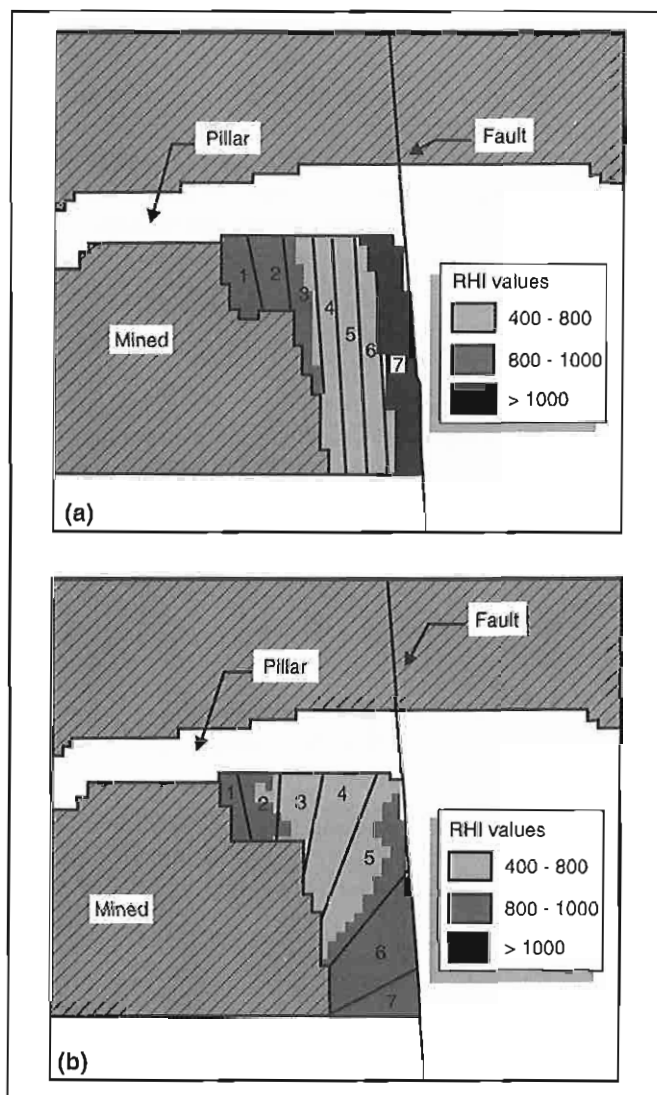


Figure 3.2.9 RHI values a: mining breast-on to a fault; b: mining obliquely

Since the RHI criterion seems to combine the best features of ERR, ESS and VESS in a reasonably appropriate fashion, its further application and refinement is advocated.

3.2.6 Face Shape Index (FSI)

The extent by which the actual and the ideal (planned) face shapes differ, in any given area of a mine, can be simply gauged by use of the Face Shape Index (FSI) criterion. This is not a design criterion as such, but more an instrument of monitoring and control. Figure 3.2.10 illustrates how FSI is calculated. Simply stated, it is the average distance that a set of panels deviate from the ideal (pre-planned) positions. The index can be evaluated at regular intervals (say monthly), and, when plotted as a function of time, gives management a simple tool to assess how successful production staff have been in maintaining ideal face shapes, and whether the trends are improving or otherwise.

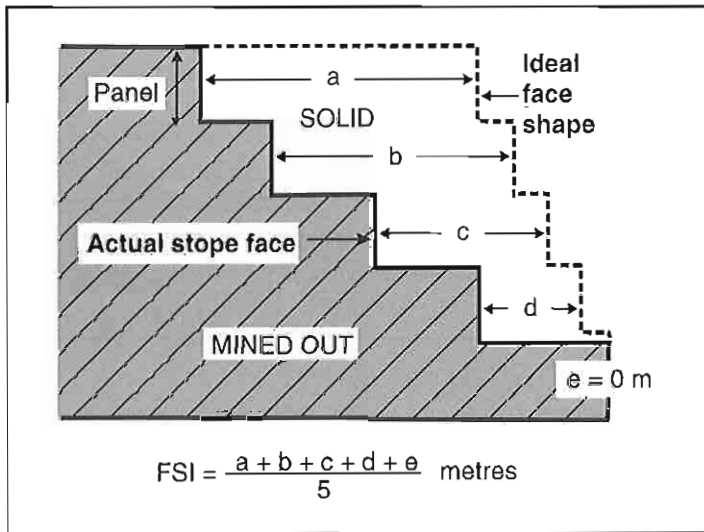


Figure 3.2.10 Plan of a mining area, showing how FSI is calculated.

3.2.7 Average Pillar Stress (APS)

This criterion is basic to designing and assessing the performance of pillars. It is simply the *average* stress carried by a pillar in a given (usually 'worst-case') mining environment. APS values can be estimated quite readily and generally using numerical modelling (Chapter 11).

Alternatively, in situations where the mining is both *regular* and *extensive*, simple 'tributary area' theory can (conservatively) be invoked: $APS \approx q_v / (1 - e)$; where q_v is the virgin vertical stress ($q_v \approx 0.027 h$, where h is depth in m) and e is the extraction ratio. For example, at a depth of 3000 m, $q_v = 81$ MPa; and if the extraction ratio is 85 % ($e = 0.85$), then $APS \approx 81 / 0.15 = 540$ MPa.

The APS criterion is currently used in two ways:

(i) To gauge whether a regional pillar (a stabilizing pillar at great depth, a barrier pillar or even a strong in-stope pillar at shallow depth) is likely to fail by *punching*

('foundation failure'). Regional pillars are generally squat rib pillars (width:height ratio $\gg 10$) and will not fail by crushing in their own right. An approximate and conservative criterion, that $APS < 2.5 \sigma_c$, has been proposed to avoid the occurrence of foundation failure, where σ_c is the UCS (uniaxial compressive strength) of the foundation rock material. In the example given above, the 85 % extraction pillars would (barely) fail this criterion if the UCS of the foundation rocks was 200 MPa. Current ongoing research is aimed at refining this criterion, which probably ought to take into account the friction and dilation properties of the foundation rocks, as well as possibly the prevailing k-ratio (see Chapter 1.2 for definition of these terms). The presence of weak or slippery layers in the immediate foundations (closer than about one-quarter pillar width from the contacts) also appears to induce significant pillar foundation weakening – with concomitant heave in the adjacent panels.

(ii) To gauge whether a pillar is likely to fail in its own right, taking into account the estimated *strength* of the given pillar. Pillar strengths are discussed in the following section.

3.2.8 Pillar Strength Criteria

The stress-strain relationships of hard-rock pillars of various width:height ratios are illustrated schematically in Figure 3.2.11a. It is seen that the behaviour switches from highly *brittle* for *slender* pillars (w:h ratio = 2) to increasingly *ductile* for *squat* pillars (w:h ratio > about 5). There is also a marked increase in (peak) *strength* σ_s with increasing w:h ratio – Figure 3.2.11b – but note that the magnitude of this strengthening seems to be an intrinsic property of the rock type involved.

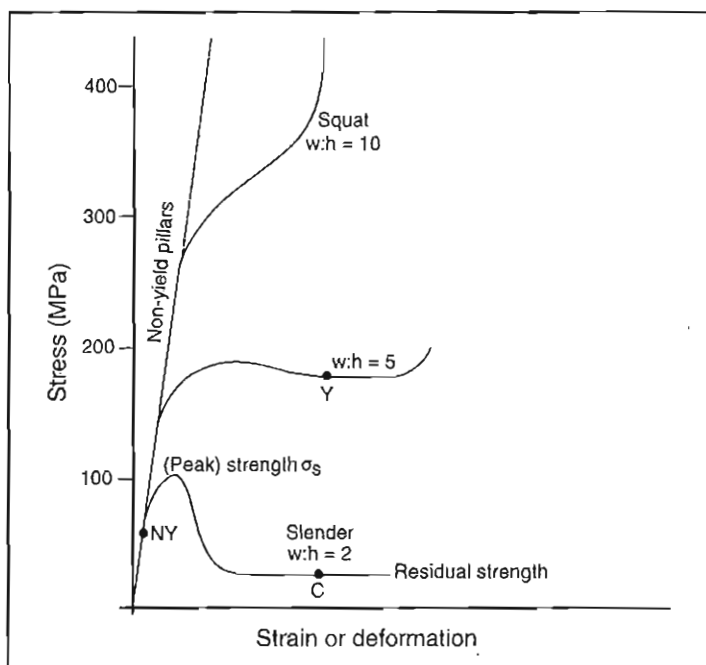


Figure 3.2.11a Stress:strain behaviour of hard-rock pillars of different w:h ratios. Typical operating points are shown for NY (non-yield), Y (yield) and C (crush) pillars.

The strength σ_s of a square coal pillar has historically been described by the Salamon-Munro 'power' equation $\sigma_s \text{ (MPa)} = K w^a / h^b$, where w is the width, h is the height of the pillar, and K , a and b are constants. The values of K , a and b applicable for coal engineering have been well quantified by back-analysis of a large number of coal pillar collapses. A similar formula for hard-rock pillars (based on a very limited sample of collapsed pillars in Canadian mines) has been applied in South African hard-rock situations: $\sigma_s = K w^{0.5} / h^{0.75}$, where the value of K has been variously estimated at between one-third and two-thirds of the UCS of the pillar material.

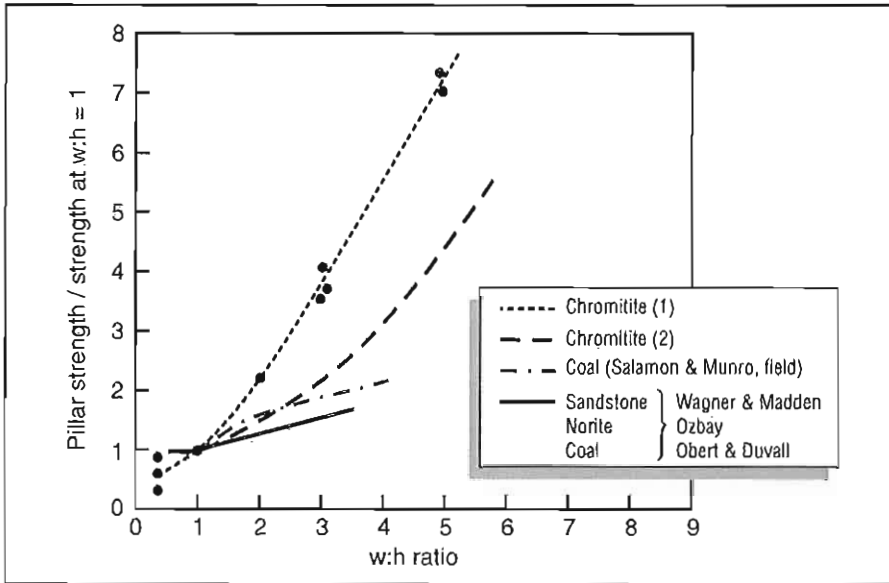


Figure 3.2.11b Strength increase with w:h ratio of laboratory pillars.

A well-substantiated and operationally simpler 'linear' formula, based on the suggestions of Obert & Duvall, for expressing the strengthening effect of width:height ratio on pillar strengths reads as follows: $\sigma_s = K_i [A + (1-A) w/h]$; where K_i is the **in-situ cubic rock mass strength** (similar to Salamon's K , and significantly lower than the rock's laboratory UCS value); and A is a parameter describing the effect of w/h on rock strength. The A parameter can (as far as is presently known) be quantified by means of routine laboratory w/h testing of a given rock type - Figure 3.2.11b. The (downgraded) in-situ strength K_i can in principle be estimated by back-analysis of past *collapses*, by observation of existing pillar *sidewall conditions*, or, in data-impooverished environments, by calculation from drill core strengths and *rock-quality* estimations. For some hard rock types, it may be possible to extrapolate a likely value of K_i directly from a suite of laboratory tests of samples of increasing size up to a diameter of about 200 mm (when the 'scale effect' no longer significantly reduces strength). Further downgrading is necessary if significant jointing or other discontinuities are present in the pillar rock mass. In this regard, current research is focusing on quantifying the effects of spacing and orientation of discontinuities, and of frictional properties of pillar/foundation contacts, on establishing a reasonable estimate of the (fully downgraded) in-situ strength K_i .

Pillar strength formulae refer to *square* pillars (the square outline, from the point of view of maximum support resistance, being in fact the optimum pillar shape for any given extraction ratio). A formula borrowed from coal engineering is sometimes applied to estimate the 'effective width' w_e of hard-rock **rectangular** pillars:

$w_e \approx 4A_p/C_p$, where A_p is the pillar area and C_p is the length of the perimeter. For very long rib pillars, this implies an effective width which is double the actual width, and a strength at least 30 % higher than that of a square pillar. Quite frequently, however, this strengthening effect due to pillar shape is ignored, and the width w is simply (and conservatively) taken to be the *smallest* cross-sectional width of any given pillar structure.

The intrinsic strength of very **squat** pillars ($w:h > 5$) increases much more rapidly than either the Salamon-Munro or Obert-Duvall formulae directly suggest. In fact, for pillars with $w:h > 10$ (the norm for barrier pillars) or $w:h > 40$ (the norm for stabilizing pillars), pillar system strength is governed more by possible foundation failures (section 3.2.7 above) than by failure of the pillar in its own right.

In any given non-yield pillar layout, the ratio of pillar strength to (worst-case) applied pillar stress is called the '**safety factor**': $SF = \sigma_s / APS$. Safety factors of 1.6 are applied in coal engineering, and values at least this high are appropriate for hard-rock non-yield pillar designs.

3.2.9 The RCF Criterion for Tunnel Layout and Support Design

It is pointed out in Chapters 5, 6 and 7 that the stability of tunnels and other service excavations is controlled by rock mass condition, field stress levels, and quantity/quality of installed support. One of the functions of regional design is to control **field stress** levels as far as possible, that is, to aim at keeping the absolute stresses in the rock in which service excavations are located at tolerable levels. This can be accomplished by **siting** off reef excavations sufficiently remote from highly stressed abutments/pillars, by the correct sizing of shaft pillars, and by the design of reef/waste overstepping layouts and sequences.

A recommended design criterion for expressing and controlling tunnel condition is the **rockwall condition factor**: $RCF = (3\sigma_1 - \sigma_3) / F\sigma_c$; where σ_1 and σ_3 are the major and minor field stress components acting normal to the tunnel, σ_c is the uniaxial compressive strength of the host rock, and F is an empirical rock mass condition factor. In normal competent rock, this factor F does not deviate significantly from unity. In highly incompetent rock (in which there are both joints and weak bedding planes spaced less than about 0.1 m apart), an appropriate value of F is approximately 0.5. In large excavations (say 6 x 6 m as opposed to standard 3 x 3 m tunnels), F should be further downgraded by about 20 % to reflect the weakening effect of excavation size on effective rock mass strength. Figure 3.2.12a shows the results of a correlation between RCF in competent quartzites ($F = 1$) and observed tunnel condition; if $RCF < 0.7$ conditions are excellent, but when $RCF > 1$ conditions rapidly deteriorate and increased levels of support resistance and areal coverage are required.

The RCF formula given above assumes a circular tunnel outline, and caters for weak or incompetent host rock conditions. Unlike the more simplistic 'vertical field stress'

criterion sometimes used, it also caters for the inclined stress vectors encountered in the footwall of remnants and pillars - Figure 3.2.12b. The siting and support of tunnels located in such areas has to guard against the high sub-horizontal stress levels which prevail (as well as the possibility of foundation failure events taking place in the neighbouring pillars).

Tunnels are often subjected to substantial changes in field stresses, and hence RCF levels, as a result of neighbouring mining operations - Figure 3.2.12c. Large stress *increases* can cause severe damage unless properly anticipated (Chapter 6); conversely, large stress *relaxations* can cause problems due to shearing and dilation of fractures as well as to lack of resilience in conventional support systems.

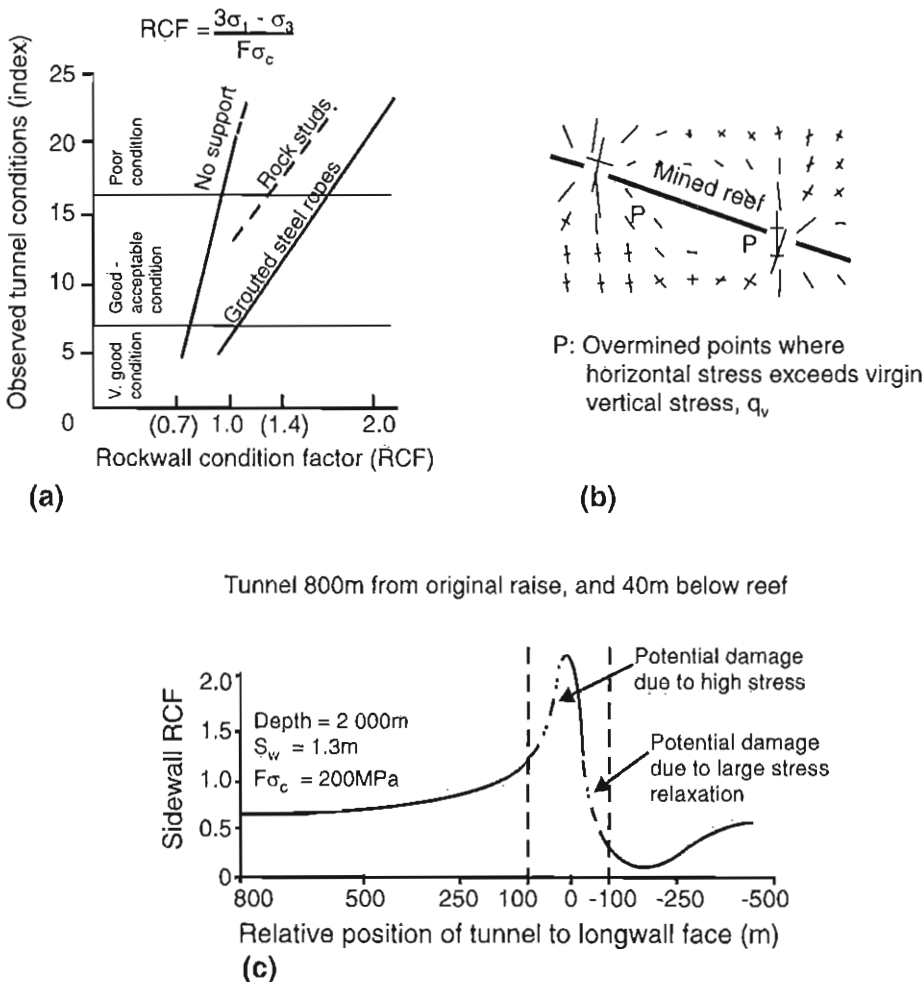


Figure 3.2.12 a Applicability of RCF in tunnels
b High horizontal stresses in overmined ground
c Stress changes (measured by RCF) in a tunnel during overstoping

3.3 REGIONAL SUPPORT

Severe problems of a *regional* nature that can arise in tabular mining (major collapses, water intrushes, mining-induced seismicity) may be countered by the use of appropriate **regional support systems**. These utilize substantial and carefully laid-out *pillars of unmined ground*, or alternatively re-introduced *backfill materials*, of sufficient stiffness to control regional deformations adequately.

Regional support systems play important strategic roles in all mining layouts, but most especially in deep mining where their contribution to controlling the hazards of *rockbursting* is of paramount importance.

Barrier pillars, which are used in shallow mines as insurance against the possibility of extensive collapses and surface subsidence, are a special case of regional support and are discussed in section 3.4.1.

3.3.1 Water-Barrier and Boundary Pillars

All mines make water to some extent, and disused workings in which pumping has ceased, eventually become completely flooded. Massive intrushes, often associated with water-bearing dolomitic strata, can also occur – for example the inundations that afflicted Merriespruit and West Driefontein in the early years of those gold mines.

Water-barrier pillars are the primary means used to isolate portions of a mine or entire mines from intrushes from neighbouring flooded (or potentially flooded) areas of mined ground. **Boundary pillars**, of a total width not less than 18 m, are required by law to be established between adjacent mines, and may not be extracted without due permission.

Certain design criteria have been proposed in the past. It has been suggested that a water-barrier pillar needs to be made sufficiently wide to ensure:

- (i) that it is 'indestructible' in terms of pure crushing (nominal width:height ratio at least 15:1);
- (ii) that along the *vertical centreline* of the pillar, the minor stress σ_3 does not fall below the water pressures involved (to prevent water from opening up sub-vertical fissures); and
- (iii) that the rock here does not fail in triaxial compression according to standard rock failure criteria (to prevent linking-up of failure lobes in the pillar foundations).

According to best evidence - numerical modelling as well as borehole observations and hydrofracturing data - the last of these criteria will only be satisfied if the average pillar stress (APS) were not allowed to build up above about 300-400 MPa in typical deep mining conditions. This in turn would require pillar widths of the order of 70 m at least to prevent linking-up of the foundation failure lobes.

Yet there are scores of instances of much narrower pillars (many with the statutory width of only 18 m) which have successfully resisted water intrushes over long periods of time. There is one case on record of a pillar, only 10 m wide in places, which withstood a water head of up to 600 m (6 MPa) over a period of many years. The reason why these relatively narrow pillars seem to be able to resist water incursion, even though their foundations are undoubtedly heavily crushed, is probably the effect

of the very high vertical stresses which act to tightly close up any potential *sub-horizontal* water-bearing channels. Indeed, the most likely hazard which could breach the integrity of a water-barrier pillar could well be the presence of a sub-vertical plane of weakness (fault or dyke) which transects the pillar, particularly if seismic activity (rupture along the plane of weakness) is a possibility. The following **practical design guidelines for water-barrier pillars** are therefore suggested:

- (1) The *width:height ratio* should not be less than about 15:1. This implies that for a stopewidth of 1 m, the statutory width (18 m) of a boundary pillar would be basically adequate. On the other hand, for a stope width of say 3 m, a width of at least 45 m would be more appropriate.
- (2) Where *planes of weakness* are present, these should be protected by *bracket pillars* of appropriate width and extent from the barrier (these dimensions being guided by means of ESS analyses, but certainly not less than the width of the barrier itself).
- (3) Cases of *multiple-reef mining* need to be addressed with particular care. For example, a flooded excavation closer than a few hundred metres above current workings could pose a significant hazard of water inrush.

3.3.2 Stabilizing Pillars

Stabilizing pillars exploit the concept of *partial extraction* in order to provide efficient regional support. Figure 3.3.1 illustrates the massive theoretical impact of stabilizing pillars on ERR levels in a deep (3000 m) heavily mined environment.

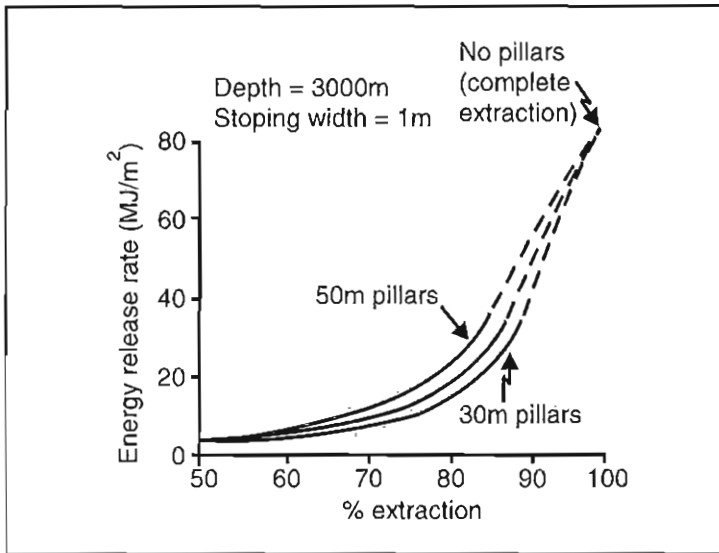


Figure 3.3.1 Theoretical effect of partial extraction (stabilizing pillars) on ERR

In the absence of any pillars (100 % extraction), the average face ERR is simply given by

$$ERR_1 = q_v S_w$$

where q_v is the virgin vertical stress (about 81 MPa at 3000 m), and S_w is the (effective) stope width (taken as 1 m in Figure 3.3.1, with resulting $ERR_1 = 81 \text{ MJ/m}^2$). It

is worth noting that the limiting ERR in complete-extraction situations is proportional both to *depth* and to *effective stope width*.

When regular mature rib stabilizing pillars are present, whether arranged on strike or on dip, the formula for the average face ERR is more complex [and is valid only where there is no total closure between the pillars], but is nevertheless able to yield important insights:

$$ERR_c = [4(1-\nu^2)/\pi] [q_v^2 L/E] [(\ln \sec e\pi/2)/e],$$

where ν is the Poisson's ratio of the strata (typically 0.2), q_v is the virgin vertical stress, L is the half-centres spacing of the pillars, E is the effective strata Young's modulus, and e is the extraction ratio (for 85 % extraction, $e = 0.85$). This formula not only demonstrates the potential massive reduction in ERR due to the presence of stabilizing pillars (Figure 3.3.1), but brings to light the following significant points:

- (i) With regular stabilizing pillars and with no total closure between the pillars, the *stope width* S_w , no longer features in the equation. Thus in high stope width environments (in which both conventional backfills or stiffer cemented fills suffer reduced effectiveness), stabilizing pillars retain their effectiveness and become increasingly the most appropriate method of providing regional support.
- (ii) ERR_c values increase rapidly with extraction ratio e (Figure 3.3.1); thus, lower extraction ratios become increasingly necessary in ultra-déep mining scenarios.
- (iii) ERR_c values increase directly with half-centres spacing L between the pillars. Thus, use of small spans between pillars is also favoured at great depths of mining (section 3.4.5).

The use of stabilizing pillars not only decreases face ERR levels with corresponding expected improvements in general mining conditions, but also results in a diminution of seismicity-inducing ESS lobes which sweep through the rock in front of the advancing mining faces – Figure 3.3.2. An unprotected heavily-mined longwall sees an indefinitely long lobe of positive ESS just in front of its advancing face and can be (and historically, has been) subject to face-parallel bursts of massive magnitude. On the other hand, a stope protected by stabilizing pillars will advance into areas where the ESS is broken up into smaller isolated lobes (the so-called 'buttress pillar effect'), where seismic events will be more localized and will have relatively small expected magnitude.

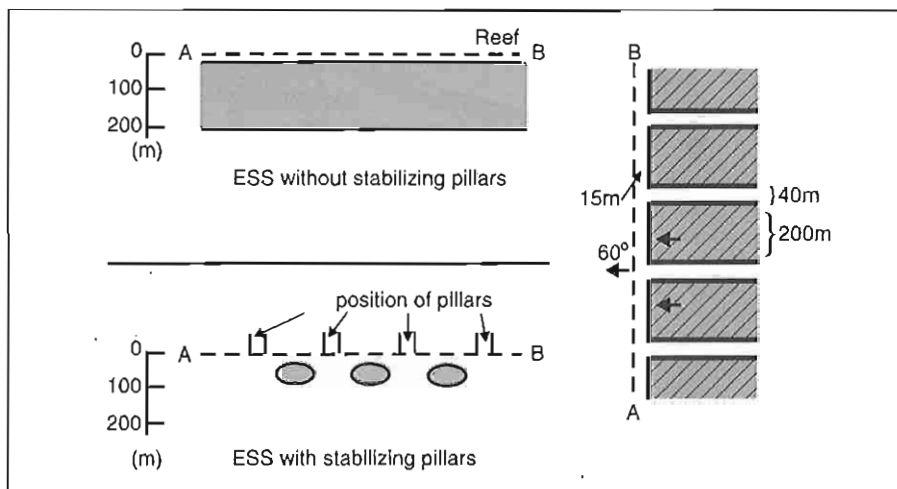


Figure 3.3.2 Effect of presence of stabilizing pillars on ESS in front of stope faces

Stabilizing pillars have featured in deep South African mining operations since the mid 1960s. Operational results have for the most part been favourable, though not without significant problems arising in practice:

- (a) Occasionally serious 'foundation failure' type seismic events have occurred, damaging footwall follow-behind haulages.
- (b) Mining difficulties have arisen associated with general inflexibility of layouts (notably, difficulties with negotiating geological structures), and hazards of late cutting of ventilation and crosscut-protection slots in established pillars.
- (c) Severe strata control problems have tended to arise in the areas immediately up-dip of pillars (see section 3.4, Figure 3.4.17).

On balance, though, stabilizing pillars would seem to have succeeded in their primary aim of reducing the incidence of seismic events and of rockbursting at the working stope faces – Figure 3.3.3.

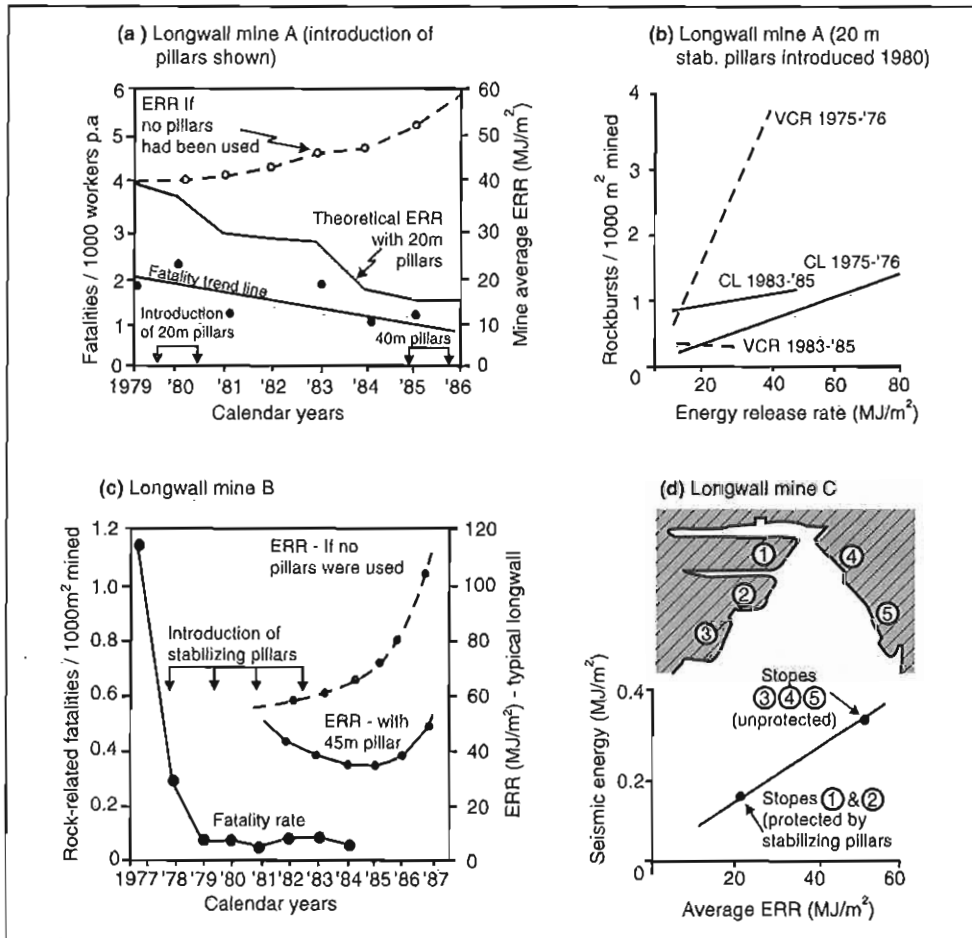


Figure 3.3.3 Observed effect of stabilizing pillars on seismicity/rockbursting. Figures a and c show the impact of the introduction of stabilizing pillars on the average ERR levels in two deep mines, and the concomitant significant reduction in the incidence of serious rockbursts.

Figure b suggests important differences in the response of two geotechnically-distinct reefs to the introduction of 20 m stabilizing pillars. Figure d, based on data from the first mine to introduce stabilizing pillars, helped motivate their more general use in deep longwall mining.

The design of stabilizing pillar layouts is governed by the precepts of section 3.2.

(i) *ERR control.* A target average face ERR level needs to be set, based on experience of the particular mining environment in question. A typical value is 30 MJ/m^2 , but values as low as 10 MJ/m^2 or as high as perhaps 40 MJ/m^2 could be appropriate for different environments. [ERR estimation was discussed in section 3.2.1.]

(ii) *Pillar width.* In a 1 m stope width environment, pillar widths of 20 m were at one time considered to be 'indestructible', that is, not be susceptible to through-going crushing. Nevertheless, in a high-stress environment (3000 m depth, 86 % extraction), such pillars were observed to sustain fracturing right through to the core with resulting partial loss of integrity – Figure 3.3.4. A possible explanation for this is the presence of gullies and heavy pillar-edge fracturing and mobile parting planes which, particularly in an environment of continual seismic shaking, could produce an 'effective' pillar height of up to 6 m and a resulting 'effective' width:height ratio of less than 4:1.

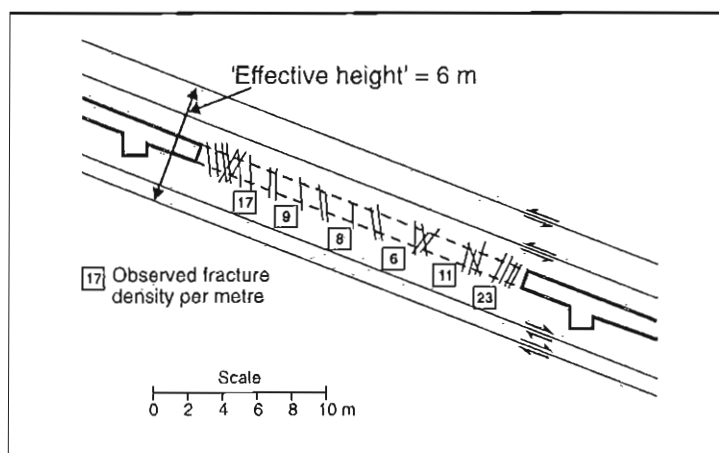


Figure 3.3.4 Observed fracturing through a 20 m wide stabilizing pillar at depth

Thus modern practice in deep mining environments favours the use of stabilizing pillars *at least 30-40 m wide*. [For example, in a mature 45 m pillar at 2200 m depth, fracturing was observed to extend less than 7 m into the pillar sidewalls. Wider pillars also seem to have intrinsically greater foundation stability.] Operational experience of stabilizing pillars in high stope width environments (say 2-3 m) is lacking, but the use of 40-50 m wide pillars would in all probability suffice here too.

(iii) *Extraction ratio e .* For a given pillar width, the wider the pillar spacing the better the extraction ratio e , but the higher the ultimate loads (APS levels) to be carried and the greater the chance of damaging 'foundation failures' taking place with partial loss of pillar load-bearing capacity and increase in face ERR levels. In section 3.2.7, use of the criterion $\text{APS} < 2.5\sigma_c$ was suggested, where σ_c is the uniaxial compressive strength of the foundation strata. For 200 MPa rock, this implies that APS levels should not be permitted to rise above about 500 MPa – i.e. extraction ratios of no more than about 85 % at a mining depth of 3000 m, 80 % at 4000 m, and 75 % at 5000 m. Lower extraction ratios, and hence lower APS levels, are indicated for weaker foundation rocks, or environments where very mobile partings are present.

At elevated stress levels, some degree of foundation failure in stabilizing pillars doubtless continues to occur on a regular basis. Laboratory simulations, together with numerical modelling studies, have suggested that the pillars' primary design objective (to reduce face ERRs) should not be significantly compromised if the $APS < 2.5 \sigma_c$ criterion is met. At the same time, in the computer modelling of deep layouts, the following modifications to standard simple procedures seem to be necessary: (i) the elastic *modulus* of the rock mass needs to be downgraded by around 10–20 %; (ii) the (effective modelled) *stope width* needs to be reduced by some 0.4 m (due to irrecoverable bulking in the hangingwall strata); and (iii) the pillars need to be modelled as effectively compressible structures suffering on the order of 5–10 % closure. These adjustments, necessary to capture the realities of deep-level mining environments, are still the subject of ongoing research.

In high-grade but ultra-deep or otherwise unfavourable environments, the use of **hybrid** regional support systems (those featuring the use of stiff backfills or concrete pillars, in conjunction with conventional stabilizing pillars - sections 3.3.4 and 3.3.5) should be evaluated in order to maintain extraction ratios at reasonably economic levels.

3.3.3 Bracket Pillars

Bracket pillars are strips of unmined ground left against hazardous geological structures to act as partial barriers against seismic activity. Especially in highly geologically-disturbed regions, the identification of *which* structures to protect seems best left to local experience supplemented with well-analysed quantitative seismic information. Whether or not bracket pillars are used, the basic principles of negotiating hazardous geological structures (mining *obliquely* at $> 35^\circ$ to, or preferably *away from*, the feature – Figure 3.4.8) need to be adhered to if at all possible.

Operational experience with the leaving of bracket pillars has shown many unqualified successes: clearcut falloffs in damaging seismicity on specific troublesome features. There have, on the other hand, been numerous failures. Indeed, serious added hazards can arise if bracket pillars prove in practice to be *under-designed*. For example, a narrow pillar left against a fault can temporarily lock-up slip movement, but when the other side of the fault loss is stripped out, a damaging large event can take place. [The opinion is often expressed that either a bracket pillar should be large (40–50 m wide), or it should be omitted altogether and the feature simply mined through as expeditiously as possible.] It is thus imperative to estimate *worst-case conditions* when attempting to design bracket pillars (or any other form of pillar).

In the Far West Rand goldfield, because of the constraints imposed by longwalling, structures of small throw or which strike obliquely are often mined through; but 'remnant'-type precautions are used with an oblique/arrowhead face shape and limited spans in the panel negotiating the structure. Occasionally, problematic structures are left intact with a small 5 m wide 'buffer' of unmined reef; this has little effect on seismicity but serves to improve local strata control to a significant extent. Structures posing particular problems include shallow-dipping planes of weakness, fault-dyke intersections, and dykes which bulge outwards away from the reef horizon; these require either very substantial bracketing, or else are simply mined through with due caution. Elsewhere, bracket pillars are used to advantage, designed using the criteria summarized below.

In the heavily geologically-disturbed Klerksdorp goldfield, a great deal of work has been done in characterizing the more dangerous structures: these seem to be planar and young for the most part. Medium width dykes are particularly burst-prone. Small spans and oblique face orientation are used in mining out a structure, and 5 m buffer pillars also feature to some extent. Mining on retreat 'away from' structures is practiced wherever possible. 'Buttress pillars' (thin strips of unmined ground normal to a shallowly-dipping structure) have proved a successful innovation. Bracket pillars are widely used, but problematic structures include fault losses in which the other side has already been completely stripped out (symmetrical bracket pillars on either side of a feature seem to be most successful), extraction of remnants which butt up against geological weaknesses, and dyke-fault intersections.

In the OFS goldfield, massive slippages have occurred on large faults but otherwise rockbursting is comparatively rare. Bracket pillars are used quite frequently, with much the same experiences and reservations as elsewhere.

Two criteria have been found useful in the **design of bracket pillars**.

(1) For **bursting dykes**, the *stress:strength* ratio should be kept below about unity:

$$\sigma_1/\sigma_{cd} < 1$$

where σ_1 is the stress at the edge of the dyke when the face is stopped to leave the bracket pillar, and σ_{cd} is the UCS of the dyke material. Often, only strong young dykes with UCS > about 280 MPa are classified as burst-prone and so requiring bracket pillar protection.

(2) For **faults (or some dykes)** prone to shear-type seismic events, the *excess shear stress (ESS)* criterion – section 3.2.3 – may be invoked to assist in the design of bracket pillar widths and layouts. Here, the option of leaving pillars *normal* to the feature ('*buttress pillars*') should not be overlooked – by these means, the ESS lobes may be efficiently fragmented and the *magnitude* of possible seismic events thereby restrained. If a standard bracket pillar is to be left butting against the weak feature, it is also usually impractical to attempt the elimination of *all* positive ESSs. Rather, the width of the pillar should be chosen to limit the likely maximum *magnitude* of seismic activity to some accepted limit tolerable to the support systems in use. [In principle, ESS modelling is able to estimate the seismic moment and hence magnitude associated with slip on a given, appropriately quantified, plane of weakness.]

In ESS modelling exercises, the setting of certain critical parameters requires attention:

- (i) the k-ratio (often assumed as 0.5 in deep mining scenarios, but which can take on very different values in abnormal tectonic situations);
- (ii) the dynamic friction angle ϕ (usually assumed as 30°, but which on abnormally 'slippery' planes of weakness seems to take on values as low as 15-20°);
- (iii) the planarity and continuity of the weakness 'plane'; and
- (iv) the integrity of small remnants or pillars in the area of interest, which may have crushed or been damaged by previous seismicity, and which may need to be treated as being effectively non-existent (in general, numerical modelling needs always to focus on *worst-case* situations).

The selection of such 'abnormal' parameters can best be guided by the back-analysis of unexpectedly large events known to have taken place on particular hazardous structures. In more normal scenarios, use can be made of standard design charts (developed by CSIR-Miningtek/SIMRAC), of which an example is given in Figure