Safety in Mines Research Advisory Committee

Final Project Report

A methodology for the definition of geotechnical areas within the South African gold and platinum stoping horizons

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Executive Summary

Variable rock mass behaviour is present in Witwatersrand and Bushveld mining districts. Geotechnical classification of the stoping environments is therefore essential. However, a sound methodology that classifies these environments had not been established before 1997. This is the major focus of this investigation. A detailed literature review, in addition, reveals that current rock mass classification systems do not adequately address the basic requirements for defining geotechnical areas in South African gold and platinum stoping horizons.

It was found during initial surveys carried out under this project that various definitions were assigned to the term "geotechnical area" across the South African gold and platinum mining industry. To provide a common basis of understanding across the entire industry, this investigation refers to a geotechnical area as an area exhibiting a specific rock mass behaviour and associated hazards in response to mining activities, where the same rock engineering strategies can be applied.

Generally, mine design strategies are adopted on both regional and local scales. The regional mine design strategies, such as mining method/layout and sequence, and regional support, are decided upon approximate values for a few critical parameters, which control the overall rock mass behaviour. Once the regional strategies are determined it is necessary to identify local strata control strategies, both providing a safe and productive mine. Local strategies depend upon a more detailed knowledge of the rock mass response, and are controlled by combinations of a wider range of parameters than the regional strategies. It is thus logical that the geotechnical subdivision of a mine has to be carried out at two levels. These are referred to as Regional Geotechnical Areas, or Mining Environments and Ground Control Districts (GCDs) in this investigation to correspond with the two levels of strategic decision making.

Regional geotechnical areas are ideally delineated before the onset of mining. The input parameters that are required for delineating these geotechnical areas are derived from orebody information, depth, stratigraphy, major geological discontinuities, and regional hydrology. By combining this information, appropriate mining methods can be selected, for different areas the likely rock mass behaviour can be ascertained. Areas of similar rock mass behaviour will be considered as a particular Regional Geotechnical Area, where a specific regional support strategy can be defined.

Once mining commences more detailed information concerning the rock mass response and the contributing parameters will become available. GCDs that have different rock mass responses can then be delineated. The procedure to define the GCDs will include identifying, quantifying, and classifying the critical geotechnical parameters, establishing the rock mass response and demarcating areas of similar behaviour, and implementing appropriate rock engineering strategies.

The methodology was developed in close interaction with representatives from the mining industry. The various criteria that form part of this methodology were tested and evaluated during several underground investigations. It is found that the newly established methodology adequately defines a large variety of geotechnical settings. Selected practitioners should however test it further. This will facilitate any required modifications and implementation.

1

Problem definition and literature survey

Chapters 1 and 2, Appendix A

Major Outcomes:

- problems are defined,
- determination of aims and objectives,
- establishment of research methodology,
- current rock mass classification systems are unsuitable,
- different rock mass classification systems are required for Witwatersrand Basin and Bushveld Complex.

2

Mine survey

Chapter 3, Appendix B

Major Outcomes:

- determination of mine personnel's understanding and expectations,
- establishment of rock mass classification systems requirements,
- evaluation of current systems via case studies.

3

Analysis of workshops on rock mass classification systems

Chapter 4

Major Outcomes:

- identification and documentation of mining environments to be addressed,
- establishment of rock mass classification systems requirements,
- establishment of level of user friendliness expected from system.

4

Underground visits

Appendices C and D

Major Outcomes:

- identification of critical criteria,
- documentation of critical geotechnical parameters
- instrumentation techniques and monitoring requirements,
- suggestion on data collection and analysis.

5

Establishment of methodology

Chapter 5

Major Outcomes:

- methodology applicable to Witwatersrand and Bushveld mining environments is developed,
- definition of individual criteria,
- establishment of data collection procedure for regional and local scales,
- establishment of data analysis procedure.

6

Methodology: Review and evaluation

Chapter 6 and Appendix E

Major Outcomes:

- evaluation of measurements and observations from underground monitoring sites against the methodology.

7

Underground visits

Appendix F

Major Outcomes:

- verification of established methodology in differing geotechnical areas,
- fine tuning of the methodology.

8

Discussion and conclusions, and recommendations for future work

Chapters 7 and 8

Major Outcomes:

- methodology defining Witwatersrand and Bushveld mining environments has been successfully developed and applied,
- methodology should be further tested and implemented at selected mines before regional implementation.

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Mr. P Murton kindly took minutes at the two workshops.

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1 Introduction

In order to implement correct design criteria for regional and local support systems, a detailed knowledge of the rock mass environment needs to be established. The significant variability of the rock mass characteristics across mining districts requires an ability to classify the stoping environment into regions of like rock mass behaviour. Implementation of mine layouts and support systems specifically designed for the various conditions will, ultimately, result in a significant reduction in the incidence of rock related accidents. For this reason, in 1997, SIMRAC contracted the CSIR: Mining Technology to define a methodology for the definition of geotechnical areas within the South African gold and platinum stoping horizons.

At the beginning of this project confusion about the term “Geotechnical Area” existed. Some defined an “Area” by employing geological parameters, whereas others utilised mining-related criteria. However, the derivation of a successful methodology for the definition of geotechnical areas will greatly enhance the ability of the mining industry to implement suitable rock engineering strategies for the prevention of rock related injuries and fatalities.

Various definitions have been assigned to the term “geotechnical area” across the South African gold and platinum mining industry. One such definition appears in the “Guideline for the compilation of a mandatory code of practice to combat rockfall and rockburst accidents in metalliferous mines and mines other than coal”. This guideline is issued in terms of the Mine Health and Safety Act, 1996 (Act No. 29), Section 9 (3) by the Chief Inspector of Mines and refers to a geotechnical area “as a portion of a mine where similar geological conditions exist which give rise to a unique set of identifiable rock related hazards for which a common set of strategies can be employed to minimise the risk resulting from mining”.

Some rock mechanics practitioners propose that the number of geotechnical areas on a particular mine should be tailored to the number of different support strategies applicable to that mine. There are many varied opinions and clearly a standard definition is necessary to ensure and provide a common basis of understanding across the entire industry. This also facilitates identification of the most appropriate mining and support strategies.

As a result of the confusion, it was necessary to re-define a geotechnical area as follows: a geotechnical area is an area exhibiting a specific rock mass behaviour and associated hazards in response to mining activities, where the same rock engineering strategies may be applied. The basis of the rock mechanics strategy will generally be a function of the anticipated rock mass behaviour and thus a geotechnical area encompasses areas of comparable rock mass responses to mining. However, it is clear that rock engineering strategies encompass both regional strategies and local tactics. It was, therefore, necessary in this project to develop a methodology to distinguish areas where different regional strategies should be applied as well as a methodology for delineating areas where local tactical solutions are required. The regional strategies comprise what regional support should be used, what mining method should be employed, and type of mining layout, whereas the tactical issues involve strata control procedures, and stope support.

Thus, the mine rock mechanic engineers ideally have to assess the rock mass response via the identification of all the critical geotechnical parameters. These parameters include the geological major and subordinate discontinuities, the stratigraphy of the rock mass in the vicinity of the stoping horizon, the associated rock engineering properties, the stress environment and the potential for the incidence of seismicity.

Information gathered from the past in similar geological environments provides a key to what is expected in unmined ground. This forms the basis of a potential classification system which, when applied, provides initial information about the likely behaviour of the rock mass surrounding the proposed excavation. A review of previous studies reveals that, currently no single index can adequately describe this behaviour, and that this can best be achieved by a combination of various factors which have all been observed to influence the stability of underground excavations. Primary attention is given by existing rock mass classification systems to characterising rock masses in core logging, tunnelling, chambers, foundations and slope stability rather than stoping horizons. As such, and for other practical reasons, none of the systems in their current form are deemed suitable for the delineation of geotechnical areas. It was, therefore considered important, at the initial stage of the project, to identify basic requirements for a rock mass classification system that would assist in defining geotechnical areas for South African platinum and gold stoping environments.

1.1 Aims and objectives

The project had two main enabling objectives for the development of a methodology for the definition and delineation of geotechnical areas on both regional and local scales:

- firstly, to identify critical geotechnical parameters to be considered in developing a methodology for the definition of geotechnical areas within the South African gold and platinum mines,

- secondly, to review current rock mass classification systems and establish basic requirements of a rock mass classification system for South African gold and platinum stoping horizons.

1.2 Research methodology

In order to achieve the aims and objectives, the following methodology was adopted (see Fig. 1.1):

- definition of the problem, aims and objectives (Chapter 1),

- survey, review and analysis of existing rock mass classification systems with available case studies for South African gold and platinum mines (Chapter 2 and Appendix A),

- a definition and application of geotechnical areas survey of mines to establish the understanding and expectations of mine personnel with regard to current rock engineering design strategies (Chapter 3 and Appendix B),

- compilation of a checklist of possible parameters to be included in a rock mass classification system and the requirements for a successful rock mass classification system (Chapter 4),

- establishment of a methodology for the definition of geotechnical areas (Chapter 5, Appendices C and D),

- underground investigations at various gold and platinum mines to evaluate critical geotechnical parameters (Appendix C and Appendix F),

- review, application, evaluation, and verification of the newly established geotechnical area methodology (Chapter 6 and Appendix E),

- summary and conclusion of the project (Chapter 7), and

- recommendations for future work (Chapter 8).

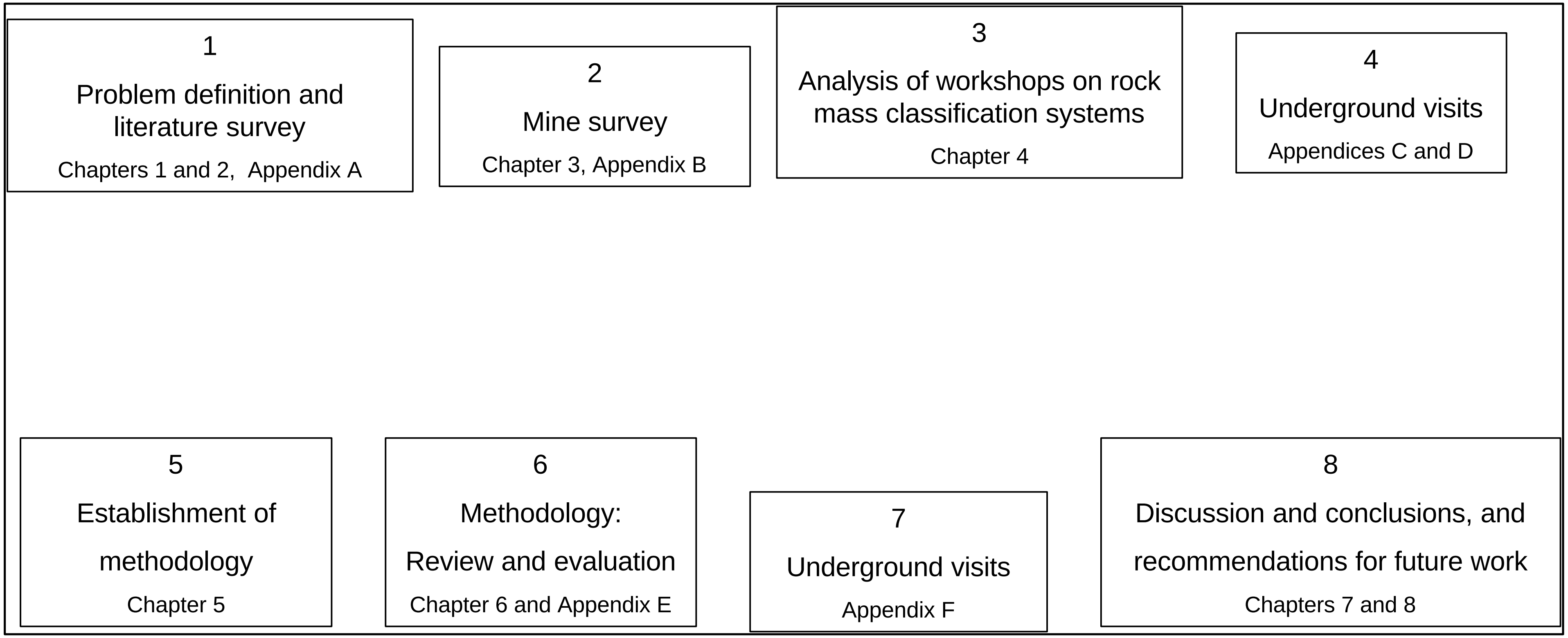

Figure 1.1. Methodology considered during the course of the investigation.

2 Literature review: most commonly used rock mass classification systems

One of the issues strongly debated at the beginning of the project was whether a specially developed rock mass classification scheme would, on its own, meet the requirements for defining geotechnical areas. The following sections discuss the investigations into this proposal.

2.1 Most commonly used rock mass classification systems

The complex nature of rock masses surrounding underground excavations makes it difficult to accurately predict rock mass behaviour. Even in cases where excavations have been previously constructed in the same rock type, the intensity of jointing and the general distribution of other geological features might be different. However, information gathered from past experiences in similar geological environments provide a key to what might be expected in the new environment. This calls for some form of classification system, which could be applied to provide some initial information about the likely behaviour of the rock mass surrounding the proposed excavation. Initial attempts established a means of describing the behaviour of rock masses by a single parameter such as the rock quality index (RQD). Later studies revealed that no single index can adequately describe this behaviour and that the mining environment can best be described and quantified by a combination of various factors that influence the stability of underground excavations.

Underground excavations are usually designed at the beginning of a project when little information is known about the properties of the rock mass. Such design procedures are normally based on previous experiences encountered in similar geological environments. However, in cases where construction works are to be undertaken in formations where such prior information is lacking or sparse, some form of classification system which enables regional correlation is essential. The ultimate aim of such a classification system is to describe the average strength of the rock mass. This information, when compared to the expected resultant stress due to the mining activities, facilitates the recommendation of appropriate support systems.

Some of the most commonly used rock mass classification systems, which have found various applications in mining but particularly for civil engineering structures and slope stability problems, are reviewed. A number of rock mass classification systems, starting from Terzaghi's (1946) rock load factor, have been proposed over the years. The commonly used systems which have found various applications in mining are the Rock Structure Rating (RSR), Bieniawski's rock mass rating system (RMR), the NGI Q system, the mining rock mass rating (MRMR), the Modified Basic RMR (MBR), and the rock mass index (RMi). These systems are detailed and reviewed in Appendix A.

2.2 Case studies for South African gold and platinum mines

In order to provide a rock mass rating system for the Bushveld Complex, mainly aiming at determining safe panel spans, Watson and Noble (1997) applied several classification schemes at different sites. Site details with regard to geological and mining aspects are given in Table 2.1.

Watson and Noble (1997) employed four different rock mass classification systems, namely the CSIR Rock Mass Rating (RMR), the Tunnelling Quality Index (Q system) of the Norwegian

Geotechnical Institute (NGI), the Impala adaptation of the NGI Tunnelling Quality Index, and the Amandelbult adaptation of the CSIR classification system. Table 2.2 compares the different rock mass classification systems for the analysis sites.

Table 2.1. List of analysed sites (Watson and Noble, 1997).

Mine	Depth/Span	Major joint orientation	Observed panel conditions
Union	1200 m/30 m	Perpendicular to face	Ultimately dangerous mining conditions with many falls of ground and a high closure rate
Amandelbult 1	500 m/35 m	sub parallel to face	Good mining conditions- no falls of ground.
Lebowa Plats	250 m/38 m	45 degrees to face	Reasonable mining conditions with a few small falls of grounds well behind the face and a sudden panel collapse at 53 m Face advance
Amandelbult 2	170 m/50 m	40 degrees to face	Apparently good mining conditions with a panel collapse
Amandelbult 3	500 m/30 m	Perpendicular to face	Dangerous mining conditions with dome structures and curved joints parallel to the face

Table 2.2. Comparison of classification schemes (from Watson and Noble, 1997).

Scheme	**Union**	**Amandelbult 1**	**Lebowa Plats**	**Amandelbult 2**	**Amandelbult 3**
CSIR (RMR)	Fair	Good	Good	Very good	Fair
NGI	Extremely poor	Poor	Poor	Fair	Very poor
Modified NGI	Very poor	Fair	Very poor	Fair	Very poor
Modified RMR	Class 3	Class 3	Class 4	Class 3	Class 4

2.3 Discussion and conclusions

The six classification systems under consideration are not able to adequately describe the complex rock mass behaviour associated with South African gold and platinum mines. Combinations of several factors have, however, been found to give a fair representation of the expected behaviour of the rock mass. Four main factors have been found to form the basis for any classification system. These include the RQD, joint alteration number, joint set number and joint roughness number. The main differences between the various classification systems appear to result from the degree of detail, with regards to data collection and manipulation, needed to arrive at the overall rock mass rating.

The CSIR classification system proposed by Bieniawski (1973) puts slightly greater emphasis on the orientation and inclination of the structural features in the rock mass, while taking no account of the rock stress. The NGI classification does not include a joint orientation term, but the properties of the most unfavourable joint sets are considered in the assessment of the joint roughness and the joint alteration numbers, both of which represent the shear strength of the rock mass. The classification system, which has found a more general application in mining, is the mining rock mass rating proposed by Laubscher (1975), an extension of the rock mass rating by Bieniawski (1973). In using the system to assess how the rock mass will behave in a mining environment, the rock mass ratings are adjusted for weathering, mining-induced stresses, joint orientation, and blasting effects. In order to avoid overestimation of the strength of the in situ rock mass which might be due to ignorance of the influence of the weakest zones, Laubscher (1975) suggested that narrow and weak geological features that are continuous within and beyond the stope or pillar must be identified and rated separately.

Using RMi to determine rock support requirements differs from other existing classification systems. While previous methods combine all the selected parameters to directly arrive at a quality or rating for the ground conditions, the RMi method applies an index (RMi) to characterise the material, i.e. the rock mass. This index is then applied as input to determine the ground quality. The application of the RMi in rock support involves a more systematic collection and application of the geological input data. RMi also makes use of a clearer definition of the different types of ground. It probably covers a wider range of ground conditions and includes more variables than the two main support classification systems, the RMR and the Q-systems.

A comparison of the various rock mass classification systems by Watson and Noble (1997, see Tables 2.1 and 2.2) shows that the Impala Platinum modification of the "NGI Tunnelling Quality Index" system provides the most accurate description of actual observed conditions. It is noted that this tool does not predict the influence of stress conditions or discontinuity orientation.

Of interest is the fact that none of the rock mass classification systems investigated in the study identified a potential problem at the Amandelbult 2 site, where the stope actually collapsed. When the questionnaire was applied (to the limited data available), a distinct reason for the failure of the stope can be seen - that is the presence of both a fault and parting-planes (see Table 2.3). Note also the high degree of similarity in the geological setting of the Amandelbult 2 and Union sites despite the vastly different ratings. Obviously this result needs to be confirmed with more examples from existing work as well as through underground investigations.

Watson and Noble (1997) noted that laboratory tests and underground observations indicate that the behaviour of the Bushveld Complex rock mass does not behave in the same way as that of the Witwatersrand Supergroup.

However, it is re-emphasised that none of the above systems adequately define the complex geotechnical environment as encountered at Witwatersrand and Bushveld gold and platinum mines.

Table 2.3. Critical geotechnical parameters as identified by questionnaire. Important aspects are highlighted and underlined (from Watson and Noble, 1997).

Site	**Union**	**Amandelbult 1**	**Lebowa Plats**	**Amandelbult 2**	**Amandelbult 3**
Rating position	5th	2nd	3rd	1st	4th
Depth (m below surface)	1200	500	250	170	500
Geological features	dyke	no	no	**fault**	no
Discontinuity Strike	**face perpendicular**	face parallel	45° to face	40° to face	**face perpendicular**
Discontinuity Dip	**low angle**	steep	low angle	steep	**low angle**
Parting planes	yes	no	minor	**yes**	no
Other features	pothole	no	no	no	domes

3 Mine survey

In order to develop a scientific and engineering methodology for the definition of geotechnical areas that would be widely acceptable and applicable to the mining industry, over 20 different gold and platinum mines from the various mining districts were surveyed (see Table 3.1).

Table 3.1: List of various mines visited.

	Region	**Mines visited**
1	Klerksdorp	Vaal Reefs
2	Carletonville	Western Deep Levels (South, East and West Shafts), Elandsrand, Deelkraal, Driefontein
3	West Rand	Leeudoorn, Kloof, Libanon, Randfontein Estates Doornkop, Randfontein Estates Cooke Sections
4	Free State	Freddies, Western Holdings, Saaiplaas, President Steyn, Freegold, Beatrix, Oryx, St. Helena, Unisel
5	Bushveld	Lonhro, RPM Rustenburg, Union Section, Amandelbult, Impala Platinum

In each case discussions were held with the rock mechanics practitioners (and where possible the geologists) about the methodologies that are employed on each mine for the definition of geotechnical areas. Where no well-defined methodology was currently being employed, a questionnaire (see Appendix B) was used to facilitate the gathering of data on the critical geotechnical parameters for the mine.

3.1 Analysis of the questionnaire

The questionnaire, which was developed to aid in the establishment of a methodology for the definition of geotechnical areas, was often modified after a visit, so as to include further characteristics needed to define geotechnical areas.

The major outputs from the Geotechnical Questionnaire can be summarised as follows.

Firstly, the expectations of the mine personnel are that a methodology for the definition of geotechnical areas:

- must be simple to use,
- must not conflict with existing strategies,
- must allow clear distinction between different geotechnical areas,
- must result in a reasonable number of geotechnical areas per mine.

In addition to this it was realised that any definition of a geotechnical area should be:

- scientific/engineering based,
- generic (applicable to a wide range of situations),
- non-prescriptive - rather advisory or guidelines.

The expectations of the mine personnel are discussed in further detail in the following section.

3.2 Analysis of mine personnel's understanding and expectations

Personnel on the shaft realised that it is essential to define the various geotechnical areas associated with a shaft so as to have a more scientific basis for the development of the support and/or mining strategy. Critical parameters for the recognition of geotechnical areas are seen to correspond with the critical design parameters. An example of design parameters would be the areal coverage and support resistance for local support of the hangingwall. Thus, if a methodology was able to highlight the differences (such as height of fall of ground, spacing of discontinuities etc., for this example) it would be practically useful.

During the discussions, several additional needs were identified. These include the fact that any methodology for the definition of geotechnical areas should be easy to use, it should not conflict with existing strategies but at the same time it should allow a clear distinction between various geotechnical areas. Concern was also raised that a particular methodology or scheme would be too detailed, precluding its use as a practical engineering tool.

Thus a scheme which is generic (allowing its use in widely varying geotechnical areas), not overwhelmingly complex and yet is able to highlight the differences involved is needed. This is shown by the geotechnical area questionnaire in Appendix B. An important aspect of this questionnaire is that it is specifically designed to address the problems of the South African gold and platinum mining industry in terms of geotechnical areas.

The questionnaire is thus a synthesis of the perceptions of the rock mechanics in the majority of the gold and platinum mines. It is not designed as a rating scheme but rather as a tool to identify critical geotechnical parameters that should be taken into account when designing a rock engineering strategy or support design.

During the mine visits, discussions were held with the rock engineering and geological personnel with regard to their existing definitions of geotechnical areas. A wide spectrum of situations was encountered from those who had a complete geotechnical model of their mine and were using it to plan and develop strategies through to those mines that had not even considered the concept.

In general it was found that the platinum mines had more mature strategies for the definition of geotechnical areas than the gold mines. This is possibly because many of the gold mines only use a single support strategy per shaft or per mine. Thus they did not see the need to subdivide it into a series of geotechnical areas, and define what the different geotechnical areas might be. However, following discussions, the mine personnel realised that their support strategy could be refined and improved with the correct identification of geotechnical areas.

The suggestions and ideas of the various mines visited were incorporated into the questionnaire, which thus became a strategic document upon which the methodology could be based. As such the individual responses of the mines are not included in this final report, as it is felt that the methodology described in this document incorporates all relevant approaches (many of which were common to several mines and mining districts).

It should also be noted that this survey was undertaken in the early part of 1997, as part of the initial stages of the project. At that time many of the mines had only commenced defining geotechnical areas and it is likely that by now the vast majority of the mines have actually implemented their own methodology.

At the beginning of the project it was envisaged that employing a suitable rock mass classification system would be one of the vital steps in delineating geotechnical areas. Detailed review of the existing classification systems as described previously, however, revealed that the current schemes are not directly applicable for classifying the rock mass in the South African stoping horizon. It was later confirmed during the visits highlighted in the preceding paragraphs, that the existing schemes would have to undergo several modifications to enhance their applicability on these mines. Most of the platinum mines have indeed modified various schemes to suit their individual needs. In combining these findings it became necessary to compile the requirements for establishing a suitable rock mass classification that will effectively classify the different conditions encountered on the gold and platinum mines. For this reason, various workshops were organised.

4 Requirements for establishing an appropriate rock mass classification system

Workshops were held with the objective of identifying and reaching consensus on the variables, which should be taken into account in producing a rock mass quality index which could be used in discriminating geotechnical areas. Once these variables had been identified, an attempt would be made at the workshop to quantify and rank the importance of each of those variables, using the knowledge and experience of all persons present. Thus, the intention was to group the parameters into three categories: important, possibly useful, and unimportant. This latter objective was not achieved.

4.1 Workshop results

4.1.1 Workshop held at Impala Platinum Mine

The first workshop was held on 4th June 1998 at Impala Platinum Mines to discuss rock mass quality measuring systems. In the ensuing discussion:

1- Several members expressed reservations at the development of a rock mass classification index. The general feeling was that this would inevitably become an industry standard and that the industry would be forced to use it. This sequence of events had already occurred with the support guidelines, which were regarded, at least in a legal forum, as a highly authoritative document "to be ignored at one's peril". It was also suggested that, with the development of a rock mass classification index, there was a danger of the industry "creating a rod for its own back". After all, in the event of an inquest or inquiry into an accident, the rock mechanics practitioner would certainly be asked whether the rock mass classification index was in use. If not, the rock mechanics practitioner would immediately be placed on the defensive, even if that particular index was unsuitable for the particular area of the mine in question.

2- It was quoted that rock mass classification indices had been used by the mining industry for many years and, as far as known, little or no success had been achieved in developing a "generic" index, or one which could easily be modified. Indeed, most rock mass classification indices had been developed for the Merensky Reef, but they would not be used for the UG2 Reef on the same mine. It was stated that experience had shown that they were of little or no value on the UG2, mainly because the weightings given to the various variables were not suitable, and needed adjustment.

3- It was mentioned that rock mass classification indices were of only limited use to the mining industry. At best, they could be used to distinguish between "good" and "bad" and possibly "medium" conditions. There was little point in developing a complicated index with a large number of parameters. Moreover, there was little correlation between the accident rate and the quality of the rock, which raised questions over the purpose of, and utility of, a rock mass classification index. This latter statement, however could be interpreted to mean that good and bad conditions are in fact identified informally by the designers of support and stope personnel such that appropriate support is designed and that in relatively bad conditions the awareness of the more obvious hazards leads to more caution and better implementation of good strata control practices. With the reverse often being the case in good conditions, a correlation between rock mass quality and accident rate would therefore not be expected. The statement therefore, by implication, supports the concept of rock mass classification.

The discussion on the various parameters to be used in a rock mass classification index is summarised as follow:

1- "The entire issue of which parameters were important to measure, and which were not, was essentially a red herring, as it was misleading." The important issue was to understand the mechanism of failure, and to take this into account when developing a rock mass classification index.

2- Impala Platinum Mines had developed a rock mass classification rating system, which was in use. "Frequently, the system gave a rating of "bad", yet no accidents occurred; the inevitable outcome was that the rating tool lost its credibility." This underlined the need to understand the mechanism of failure, rather than become sidetracked into measuring large numbers of parameters, and trying to rank them in order of importance.

3- "Areas on Impala that were graded as "good" with their rock mass classification rating system often experienced accidents for no apparent cause. This suggested that a rock mass classification index was of limited or no value on that mine and that a measuring system, which examined the mechanism of failure, would be of much more value."

4- "It was felt that there was no correlation between fatality rates and the condition of the rock. In poor rock conditions, the workforce tended to be very careful, and adhered strictly to all standard procedures, especially with regard to support. In good rock conditions, the production people tended to become careless or complacent, and procedures were not always followed. As a result, accidents often occurred where conditions were good. It was important to recognise that most accidents were caused by failure of the mines to implement the prescribed system properly, and that failure of support was far less common."

5- It was agreed that the propensity for keyblock formation is a very important parameter in measuring rock mass quality, as keyblock failure was an important cause of rockfalls. "The orientation and inclination of the joints were key factors, especially the orientation relative to the face, as opposed to each other." Also, the presence of laminations and flat jointing was possibly even more important than fracturing, as these partings, in particular, tended to be far more continuous, over a wider area, than simple jointing.

6- "The presence of water was an important factor in assessing a keyblock's propensity to fail, as water obviously had a lubricating effect and lowered the coefficient of friction and reduced the clamping stresses."

7- It was agreed that most collapses were associated with geological discontinuities. "It was also discussed that any rock mass classification index would need to take into account the time factor, as this had a critical effect on the behaviour of joints; for example, a face advancing very rapidly would affect joint behaviour very differently from a face mined slowly."

8- A modified rock mass classification system was used at one of the mine, as a part of routine stope risk assessments. It was considered to be successful and an essential input to the risk assessment.

4.1.2 Workshop held at CSIR: Mining Technology

The second workshop was held at CSIR: Mining Technology with the objective of being able to identify possible applications of a rock mass quality index.

The rock mass quality index which was developed and used at Western Areas takes into account only those parameters which, based on previous experience at the mine, together with general intuition, are regarded as being the most important. Using previous experience, weighting has been attached to each parameter, and the respondent is merely required to answer yes or no, with no allowance being made for intermediate choices. For example, one of the questions is “Is there a brow present?” . In the case of yes, the answer scores +3 and in the case of no, the answer scores -3. The scores for each parameter are then totalled, to arrive at a figure indicating the rock mass quality. In response, speaking in his capacity as a labour representative, Mr More O’Ferrall expressed concern that the rock mass quality systems have been developed by rock engineers, and are almost exclusively used by rock engineers, yet the results from these systems never seem to affect operations. For example, a rock mass quality system might indicate extremely poor rock quality, yet this warning signal never appeared to have any effect on the operations of the mine. Preventative measures usually appeared to be lacking, which was probably indicative of an attitude problem throughout the mine. Thus, even at the stope face, those workers most affected by the dangerous rock conditions would state that “we work underground and can read the behaviour of rock and do not need a rock engineer to tell us how to operate”.

One important potential role of the rock mass quality systems is in hazard assessment, as the first step towards risk assessment. A rock mass quality system is a very useful hazard assessment tool, although existing tools had not been developed to the stage of being useful in prediction. There was some doubt over the question of whether a rock mass quality tool could ever be developed to that stage, while still remaining useful from a practical point of view.

4.1.3 Possible applications and scenarios

At the second workshop, possible applications or scenarios for a rock mass quality index, in addition to its application in defining geotechnical areas, were identified. The following discussion took place:

Service excavations: It was felt that there was a definite need for rock mass quality indices in this environment.

Pillar design: Likewise, there was a definite need for rock mass quality indices in this task.

Determine stable span between pillars: This was definitely a situation in which a rock mass quality index would be useful.

Stope support design: The main role of the rock mass quality index would be in establishing spacing of support.

Shaft pillar extraction: There was probably a role for a rock mass quality index, although it was far from essential.

Shaft sinking: A rock mass quality index could probably be useful in certain elements of shaft design such as design of the lining or temporary support.

Assist in defining geotechnical areas: The rock mass quality index would certainly be able to play a role in this activity

4.2 Requirements for establishing an appropriate rock mass classification system

The requirements for a successful rock mass quality measurement system as implied from the workshops are summarised as follows:

1- the system should be simple. At least 25 different parameters could be measured, but the success of any index depended upon identifying the three or possibly four parameters, which were of importance to the area under consideration. Any system employing five or more parameters would probably run into resistance.

2- the system should be quick to apply; all mines were reducing staff, and the remaining staff were often overburdened, with no time to apply complex rock mass quality indices. For example, fracturing was obviously a very important parameter but it was considered impractical to use in a rock mass quality index on a daily basis, if the data were time consuming to collect. Indeed, any rock mass quality index, which entailed a large amount of routine data collection, was certain to fail.

3- staff with minimal levels of skill should be able to apply the rock mass quality index and arrive at a valid and reproducible reading; it was pointless to develop an index which could only be used by a trained geologist.

4- the system should be of value to rock engineers, and others, in applications such as support design.

5- the rock mass quality index should be of value in legal proceedings. For example, if a rockfall incident led to legal proceedings, then the index should allow the defendant to support his claim that he was acting reasonably and/or in accordance with accepted standards. However, it was difficult to reconcile the need for a simple, foolproof system with the need for a system which would withstand the rigors of the legal process; ideally, it should be possible to define a compromise index, meeting these conflicting requirements.

4.3 Discussion and conclusions

The major outcomes from the first workshop involving personnel from platinum mines, who were surprisingly negative, was that a developed rock mass quality index would inevitably become an industry standard and that the industry would be forced to use it. The main outcomes are summarised below:

1- There is a danger of the industry "creating a rod for its own back" with the development of a rock mass quality index in the event of an inquest or inquiry into an accident.

2- The weightings given to various parameters in a rock mass quality index would need some adjustment which varies from reef to reef.

3- The potential usage of a rock mass quality index is very limited in the mining industry. There is little correlation between the accident rate and the quality ascribed by the rock mass classification indices used on the mines.

4- The primary issue in developing a rock mass quality index is to understand the mechanism of failure rather than measuring large numbers of parameters.

5- The primary parameters in measuring rock mass quality are orientation, inclination of joints with their relative position to the face together with bedding. The presence of water is an additional, important factor in assessing propensity of a keyblock to fail.

The outcomes from the second workshop held at CSIR: Mining Technology, predominantly addressing the Witwatersrand gold deposits, were more constructive and positive. Although the results of rock mass quality indices never seem to affect operations, one important potential role of the rock mass quality index is in hazard assessment, as the first step towards risk assessment. The general feeling was that a rock quality system is a very important hazard assessment tool. A list of possible applications or scenarios for a rock mass quality index are given as follows:

a- service excavations,

b- pillar design,

c- determine stable span between pillars,

d- stope support design,

e- shaft pillar extraction,

f- shaft sinking, and

g- assist in defining geotechnical areas.

Available rock mass classification systems are probably suitable for deep mine tunnels with only minor modifications to cater for the intense stress fracturing, dilation, stress changes and dynamic loading. Requirements for a successful rock mass quality index for tabular hard rock mine stopes are summarised below:

1- The success of any rock mass quality index will depend upon the number of parameters that will be employed. For application in gold and platinum stopes, it was suggested that a maximum of four parameters have to be measured on a regular basis in order to keep it as simple as possible.

2- The rock mass quality index to be developed should be able to be applied quickly. It will fail if it requires collection and examination of many different parameters.

3- It should be usable by semi-skilled persons.

4- It should have a common meaningful value for everyone across various applications.

5- The rock mass quality index should be of value in legal proceedings, for instance those concerning rockfall incidents.

6- It should cater for stress fracturing and the persistence of the structures.

It is concluded that because of the many and varied parameters (see Chapter 5) that are involved in the definition of geotechnical areas, and as only a few of these are significant and need to be considered for a particular geotechnical area, it is not feasible to formulate an all-embracing rock mass classification system that is practical for defining geotechnical areas. Rather general guidelines that preclude the assignment of weighting values to these parameters to achieve a rock mass rating, but suggest a logical assessment of the rock mass behaviour, are considered to be essential. These guidelines form the basis for establishing the methodology for defining geotechnical as described in the following chapter.

5 Methodology for the definition of geotechnical areas

The definition of a geotechnical area adopted in this study refers to an area exhibiting a specific rock mass behaviour and associated hazards in response to mining activities, where the same rock engineering strategies may be applied.

In the course of the many discussions held with mine rock mechanics personnel several pressing questions were raised. These include questions like "**What should the extent of a geotechnical area be and how should such an area be demarcated**"?

The extent of a geotechnical area should be determined by the relevant geological characteristics of the rock mass, which in turn dictate the expected response of the rock mass to mining. Thus the crucial factor for delineating such areas would be the expected response of the rock mass to mining operations. Once this response is ascertained the ultimate aim would be to adopt appropriate rock engineering strategies to minimise potential rock related hazards.

It became clear during this investigation that as rock engineering strategies address either regional issues or local problems, so the delineation of geotechnical areas should reflect these differences and disparate requirements. The methodology developed for delineating geotechnical areas is thus divided into two parts, - firstly the subdivision of a mine into Regional Geotechnical Areas or Mining Environments and secondly the further subdivision into Ground Control Districts (GCD).

The regional strategies include such issues as mining method, mining layout, mining sequences, and regional support. Decisions as to what to apply for each of these fundamental aspects of mine design and operation are dependent upon approximate values for only a few critical parameters, controlling the overall rock mass behaviour. Having decided what major strategies to implement it is then necessary to consider what secondary strategies to apply to complement them in providing a safe and productive mine. These second level strategies are concerned with support of excavations and other strata control measures. They depend upon a more detailed knowledge of rock mass response, which is controlled by combinations of a wider range of parameters than is necessary for defining the regional strategies.

It is thus logical that the subdivision of a mine based on geotechnical conditions should be carried out at two levels of detail, to correspond to the requirements for the definition of the two categories of strategic decision making. The names given to the two types of geotechnical areas are inconsequential but the following are put forward for consideration.

1- Geotechnical Areas or Mining Environments for regional subdivisions leading to major strategic decisions,

2- Local geotechnical areas, also termed Ground Control Districts (GCD), for distinguishing areas requiring either different strata control procedures and or stope support designs.

Figure 5.1 below is provided to clarify the two levels of geotechnical area demarcation.

5.1 Regional geotechnical areas

In accordance with Figure 5.1, it is proposed here that the regional geotechnical areas should ideally be delineated before the onset of mining activities. This facilitates strategic decision making. The delineation of geotechnical areas is a relatively simple process involving a few important parameters such as:

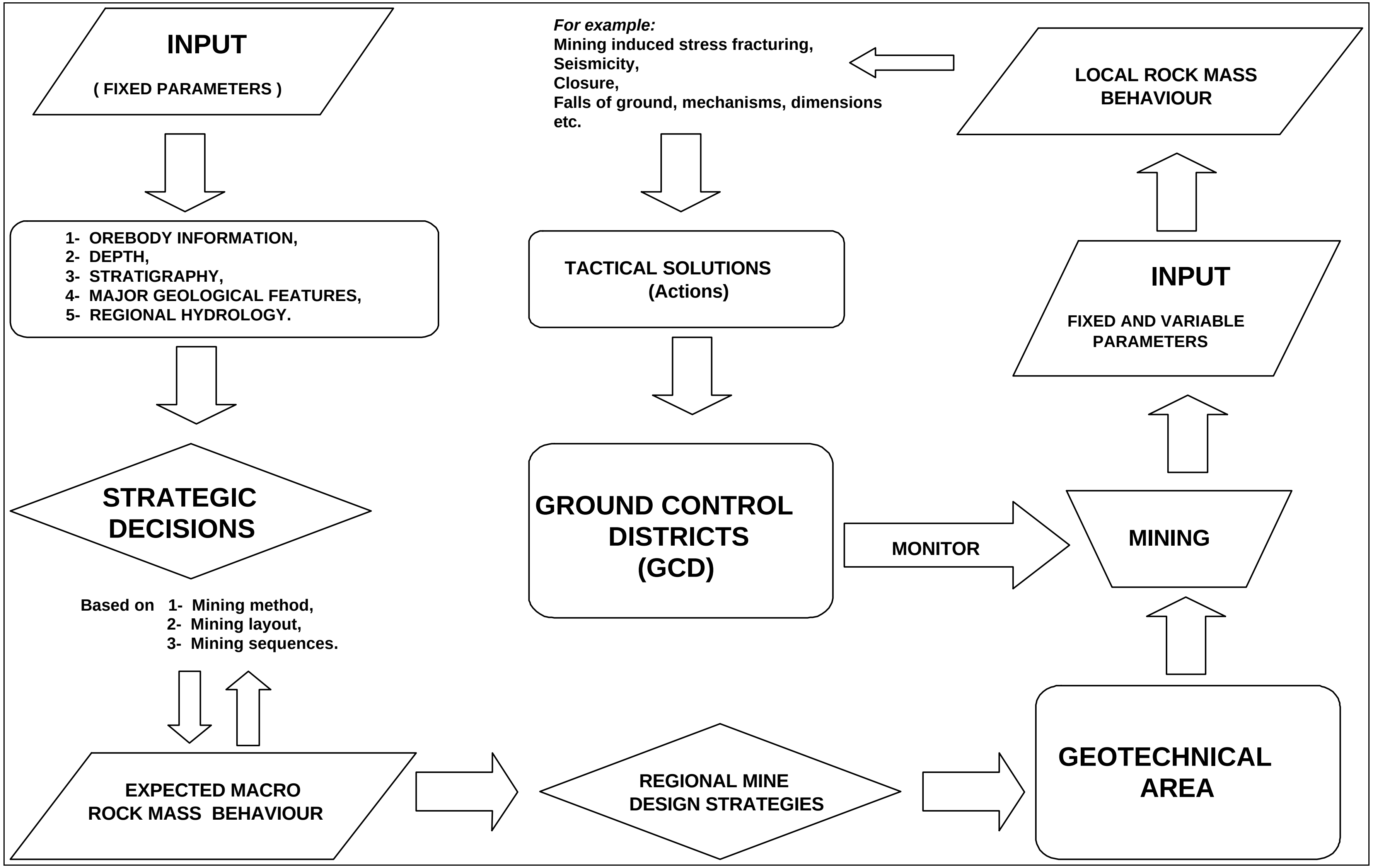

Figure 5.1. Flow-chart outlining the procedure for delineating regional and local (Ground Control Districts) geotechnical areas.

1- Orebody information

- dip of the orebody,
- reef thickness,
- grade distribution,
- middling thickness.

2- Depth (indicating potential for stress regime, seismicity and rock fracturing)

3- Stratigraphy

- rock types and their characteristic strengths,
- presence of poor rock formations for example tuffaceous lava and Khaki shale
- presence of regionally persistent partings such as Bastard Reef and Green Bar.

4- Major geological features

- faults,
- dykes.

5- Regional hydrology

The above valuable geotechnical parameters are commonly gathered during the pre-production stages of a mining operation. It is assumed that they have been determined by the application of geophysical techniques, borehole data and historical records from known areas of similar geological settings. These parameters could be identified and quantified from these sources, and could then be used as input data to arrive at fundamental strategic decisions. Suggestions on instrumentation and techniques applicable for data collection on a regional scale are presented in Appendix C.

Having identified the relevant geotechnical parameters, the information should be plotted on mine plans as follows:

a- A base map which shows significant depth contours subdividing mines into shallow, intermediate or deep regions must be created. A boundary line between shallow and steeply dipping areas must be defined. The location of major geological features, and rock types must also be plotted. Geological features which are noted to have past seismic history and/or are prone to seismicity must be highlighted. These could give indications of seismicity and appropriate regional support strategies.

b- Areas of high or low grade must be identified and highlighted, as this will influence the mining sequence and layout.

c- An isopach map should be established showing significant variations in reef thickness where different stoping methods would be required. This information could also be deduced from borehole core logging during the pre production stages of the mine. This would facilitate the adoption of appropriate mining methods that would eventually improve support performance.

Differences in rock mass behaviour have been noted within specific depth ranges. These have led to a rough classification of mining horizons into shallow, intermediate and deep mines. This categorisation is principally based on depth. Mining operations at depths down to 1000 m are considered to be shallow. At depths between 1000 m to 2250 m, these are regarded as intermediate. Between 2250 m and 3000 m, the activities take place at deep levels. Mining operations beyond 3000 m are considered to be ultra-deep. Characteristic rock mass behaviours within these depth ranges have been presented in the Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts, 1988 edition.

Superimposing these plans on one another can then facilitate strategic decision making. The first decision is the choice of a mining method. Combining the pre-existing ground conditions and the chosen mining method, it should be possible to estimate the likely regional rock mass behaviour. This could take the form of, for instance, stope backbreaks, foundation failure of pillars, or regional seismicity or a combination of these.

Having established the likely response of the rock mass within a particular locality, the second decision centres on the appropriate regional support strategies. Previous research work has identified characteristic rock mass behaviour within certain depth ranges and correspondingly effective design strategies. This decision can therefore be easily made if the depth of mining is estimated from the borehole data. As an example, in cases where the depth is less than 1000 m, the regional support strategies could include the cutting of regional pillars to reduce the probability of beam failure, stope collapse and stope backbreaks. Strategies that have been found to work effectively in the intermediate and deep mining environments have been suggested in the Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts, 1988 edition.

5.2 Ground Control Districts (GCD)

Once the regional strategies have been adopted and mining commences, it is possible to obtain data on the lesser geotechnical features which control the more local rock mass behaviour including the strata control issues of for example panel collapses, falls of ground or rockbursts. It is at this stage that the mechanisms leading to these instabilities and failures need to be determined. During this process those few parameters, out of potentially twenty or thirty, that significantly contribute to the instabilities are identified. These few parameters then form the basis for the definition and the delineation of local geotechnical areas or Ground Control Districts. However, it is important to note that the process of defining the Ground Control Districts would best be done if certain essential rock mechanics procedures were followed sequentially. These include:

- identification and quantification of critical geotechnical parameters,
- classification of geotechnical parameters,
- establishment of local rock mass responses,
- demarcation of Ground Control Districts, and
- implementation of rock engineering strategies.

Since the above steps constitute a summarised operational sequence in rock mechanics practice, they have been adopted as guidelines that could be used for defining Ground Control Districts across the South African gold and platinum mines. Detailed discussions of each of the above steps are presented in the following subsections.

5.2.1 Identification and quantification of critical geotechnical parameters

The first stage in the definition of Ground Control Districts is the identification and quantification of the relevant parameters. Depending on the prevailing conditions in a particular mine, these are identified from the categories listed below, but would also include anomalous conditions such as domes, potholes or channels.

a) orebody information,
b) stratigraphy
c) discontinuities,
d) stress environment and,
e) production parameters.

Suggestions on instrumentation techniques and monitoring requirements for data collection on a local scale are presented in Appendix C.

Orebody and production parameters originate from the plane of mining. The remaining three parameters are taken to be important within 50 m of the excavation. However, this distance may be different for platinum and gold orebodies or for individual orebodies within the Bushveld Complex or the Witwatersrand Basin. These categories may be further subdivided as follows:

1- Orebody information

- orientation of the orebody,
- stoping width
- grade distribution, and
- middling thickness.

2- Stratigraphy

- rock types,
- rock engineering properties.
  - Uniaxial Compressive Strength (UCS),
  - Young's Modulus,
  - cohesion,
  - friction angle, and
  - Poisson's ratio.

3- Discontinuities

- type (geological, mining induced),
- orientation,
- spacing,
- infill and discontinuity/host contact relationship,
- persistence/shape,
- displacement,
- zone thickness, and
- water conditions.

4- Stress environment

- k-ratio, and
- orientation of Φ_1 and Φ_3.

5- Production parameters

- mine layout,
- mining direction,
- span, and
- face advance rate.

5.2.2 Classification of geotechnical parameters

Within a particular stoping horizon, many geotechnical parameters occur. Significant changes in local rock mass behaviour are expected due to substantial changes in some of these geotechnical parameters. It is therefore essential that these critical geotechnical parameters be quantified to deduce different or similar rock mass behaviour. The effects of these parameters on rock mass behaviour and their appropriate categories are presented below.

5.2.2.1 Orebody information

The majority of platinum and gold bearing reefs currently being mined within South Africa are tabular inclined bodies. Within a Ground Control District the orebody will have essentially the same dip angle, stoping width and middling between the stoping horizons. There are however, regions within the general shape of the orebody where factors such as the dip orientation and angle change sufficiently to cause a change in, for example, stope support. It is important to recognise these as distinct Ground Control Districts as the rock mass will behave differently within them and consequently the support strategies will have to be modified.

For example, as the mining progresses over a roll within the VCR, the dip steepens and the potential for hazardous rock mass behaviour increases. It has been noted by Roberts *et al.* (1997) that mining across a roll is complicated and fatalities are often associated with these features. Bedding parallel faulting does not usually follow the reef hangingwall contact across a roll and pilloid load casts are often present in the lava resulting in poor hangingwall conditions (see Fig. 5.2). Within the Bushveld Complex, potholes within the reef horizons also cause a sharp change in the dip direction and angle of the reef horizon and result in a deterioration of rock mass conditions. It is re-emphasised that a change in the orebody orientation changes the relative intersection of the various discontinuities (e.g. joints) and thus alters the rock mass behaviour.

The areal extent of such regions should be noted and plotted on plans. These areas can thus be recognised for their particular geotechnical condition and thus as a specific Ground Control District requiring a common rock engineering strategy even though they may be separated in space.

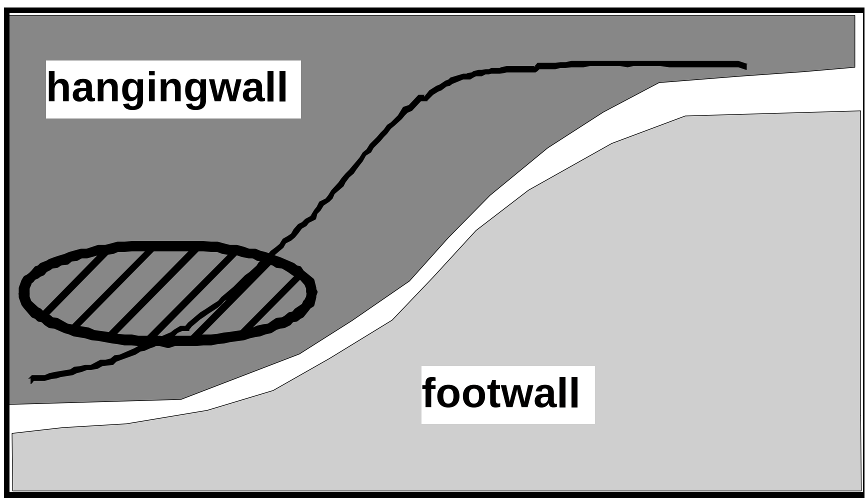

Figure 5.2. Block diagram showing some of the hazards associated with rolls (steepening up of dip angle) within the VCR including pilloid occurrence (within hatched area) and change in attitude of bedding parallel faulting (after Roberts et al. 1997).

The stoping width and middling thickness between reefs define the mining geometries in the vertical dimension. When stoping width changes greatly, not only does the type of in-panel support change but also the dimensions of larger pillars. The stoping width can be categorised as shown in Table 5.1.

Table 5.1. Stoping width classification.

Stoping width	Description
Less than 1 m.	Narrow
Between 1 m and 1,8 m.	Intermediate
Between 1,8 m and 8 m.	Wide
Greater than 8 m.	Massive

These thicknesses are considered significant as far as support design and strata control practices are concerned under typical stoping conditions. Contours of estimated stoping widths corresponding to the above limits, or what are locally considered significant, should be plotted on the mine plan and would then likely be limits for GCDs.

When the middling thickness decreases sufficiently for the rock mass behaviour of one stoping horizon to affect the other, then a separate Ground Control District needs to be recognised. This distance varies due to factors such as percentage mined and the rock mass properties of the reefs and the middling, but is usually in the region of 50 m. The contour of this thickness or what is locally considered significant should also be contoured on the mine plan and would form another limit of different GCDs.

5.2.2.2 Discontinuities

If on a particular mine, minor geological discontinuities have been recognised as the most important features for defining and effecting Ground Control Districts, then the quality, orientation and intensity of such features should be studied in detail. These must be quantified to an appropriate level and significant differences in these properties determined in order to separate areas expected to have different rock mass behaviour and hence GCDs. The rock mass behaviour around small-scale joints, faults and dykes is often different from the general rock mass behaviour in the vicinity of for example, large-scale faulting. On a local scale the characteristics of small-scale discontinuities can be used to define the support type and spacing. It is therefore imperative to correctly identify the relevant discontinuities and their characteristics. Type, orientation, spacing, infill, persistence, displacement, zone thickness, water conditions and host-rock contact relationships are the critical features to be examined for each discontinuity or set of discontinuities. Possible variations and relationships which should be identified for each of these features are indicated in Table 5.2 below:

Table 5.2. Type of discontinuities and their characteristics.

Type of discontinuity						
	Structural		Compositional			
	Fault	Joint	Dyke	Sill	Sedimentary	Metamorphic
Orientation	Variable systems	Variable sets	Variable	Orebody parallel	Mostly hangingwall parallel	Variable
Spacing						
Roughness						
Infill	None, fault rock	Various; quartz, calcite etc.	Basalt-rhyolite; pegmatoid etc	As dyke	Variable; heavy mineral, chlorite, none, etc.	No
Persistence	High	Low to moderate	High	Moderate	High to intermediate	As associated structure
Displacement	Minor to major	Nil	Nil to major	± Minor	No to limited	No
Thickness	Millimetres to tens of metres	Millimetres to centimetres	Millimetres to tens of metres	Millimetres to tens of metres	Millimetres to centimetres	Millimetres to tens of metres
Water	Possible	No	Possible	Possible	Unlikely	No
Host rock contact	Sharp	Sharp	Sharp	Sharp	Sharp to transitional	Transitional

Intensity and orientation of discontinuities

Small-scale discontinuities may intersect to form unfavourable ground conditions. The greater the variations in the orientations of the discontinuities the more potentially unstable the rock mass is. Thus, if an estimate of the number of intersecting discontinuities is made, then an idea of the potential behaviour of the rock mass can be obtained. Five cases of the number of discontinuities are identified as follows:

1- No intersecting discontinuities,
2- One set of intersecting discontinuities,
3- Two sets of intersecting discontinuities,
4- Three sets of intersecting discontinuities,
5- Four or more sets of intersecting discontinuities or sets of discontinuities.

On the basis of dip directions, the five cases can be classified into five categories as shown in Table 5.3. Category I is described as very good and Category V is a bad hangingwall condition. A three dimensional illustration and the theoretical basis for defining the categories are presented in Appendix C.

In delineating GCDs, these categories will indicate lower and higher degrees of potential instabilities respectively. Bedding planes stratification, partings, and joints are special cases of great significance requiring special attention. It is suggested that J-Block analysis be performed to determine effective support spacing for these cases.

Depending on the degree of inclination of the discontinuity to the horizontal plane, the dip angle can also be classified as shown in Table 5.4 below.

Table 5.3. Classification of discontinuities based on the discontinuity sets.

	Category 1	Category II	Category III	Category IV	Category V
	No intersecting discontinuities	One set of discontinuities	Two intersecting sets of discontinuities	Three intersecting sets of discontinuities	Four or more intersecting sets of discontinuities
Description	Very good	Good	Fair	Poor	Bad

Table 5.4. Classification of discontinuities on the basis of their inclination.

Dip (degrees)	Less 30	Between 30 and 60	Above 60
Description	Shallow dipping	Moderate dipping	Steep

Discontinuity spacing

As a general statement, the closer discontinuities are to each other the worse the ground conditions. The spacing of discontinuities also affects the support density because the closer the discontinuities, the higher the support density needs to be (if all other factors remain constant). If the minimum spacing between discontinuities is measured, it facilitates the estimation of support spacing. Spacing values are classified into four categories in Table 5.5.

Table 5.5. Classification of discontinuity spacing.

Category	Description
1	Less than 100 mm apart
2	Between 100 mm and 0,5 m apart
3	Between 0,5 m to 1 m apart
4	Between 1 m to 3 m apart
5	Greater than 3 m apart

Infill and discontinuity/host rock contact relationship

Material on the surface of a discontinuity varies in strength and frictional properties and thus affects the cohesion and slip potential of the plane, hence the shear strength of the discontinuity. For example, the presence of a weak low friction material may facilitate sliding of the plane. Three different types of infill are defined. The precise chemical composition of the infill is not important for the definition of different Ground Control Districts but rather the mechanical behaviour of the surfaces coated by the infill. The three general types of infill are given in Table 5.6.

Table 5.6. Classification of discontinuity infill material.

Category	Description	Slip or parting potential
1	No filling	High
2	Serpentinised or argillaceous	Medium
3	Strong and welded	Low

The roughness and aperture of the discontinuity also needs to be considered. It is suggested that the same procedure as outlined by the ISRM's Rock Characterisation Testing and Monitoring be followed.

Persistence/Shape

The persistence rating of a discontinuity is a measure of its continuity. The longer the length of a discontinuity the greater the likelihood of intersecting other discontinuities and thus creating a potentially unstable block. Discontinuity length can be considered in the categories given in Table 5.7.

Table 5.7. Classification of discontinuity persistence.

Category	Description	Implication
1	Less than 1 m	Less problematic
2	1 m to 10 m	Keyblock formation
3	10 m to 30 m	Panel collapse
4	30 m to 150 m	Massive areas of fall

The discontinuity shape also impacts on rock mass behaviour. Planar or straight discontinuities could slip more easily, whereas undulating and irregular discontinuities result in features such as hangingwall domes and local falls of ground, see Table 5.8.

Table 5.8. Classification of discontinuity shape.

Category	Description	Implications for slipping
1	Planar	High
2	Undulating	Moderate
3	Irregular	Low

<u>Zone thickness</u>

This is the thickness of the area affected by the discontinuity. It may for example be the width of igneous material within a dyke or the width of an anatomising shear-zone. It is the area where rock mass behaviour changes (see Fig. 5.3) and hence the mining strategy may change (e.g. fault loss and duplication). Suggestions for zone thickness ranges are given in Table 5.9 below.

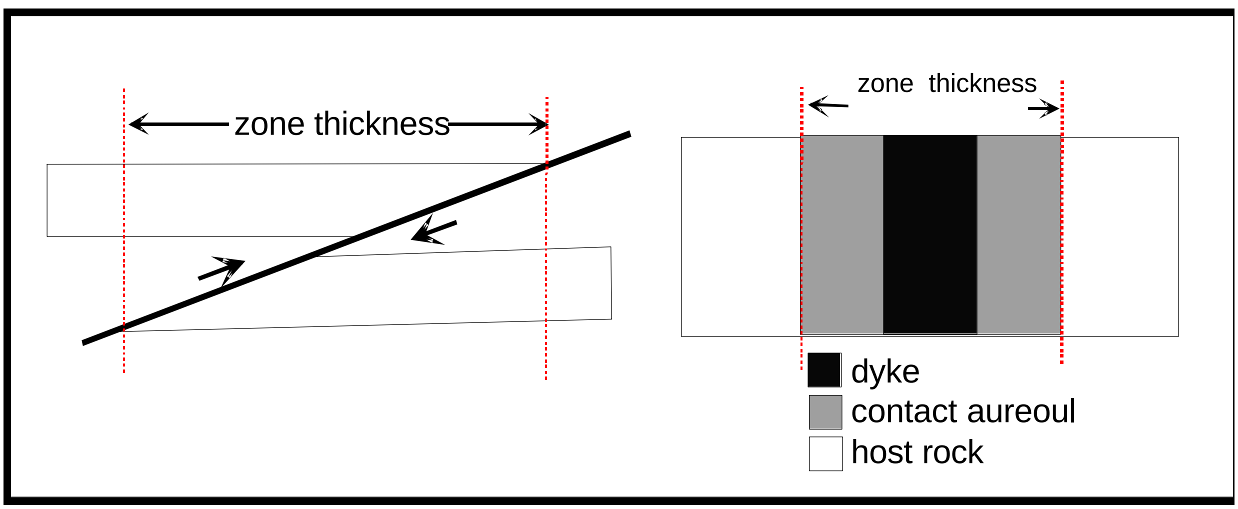

Figure 5.3. Block diagrams showing two examples of zone-thickness.

Table 5.9. Classification of zone thickness.

Category	Description	Significance
1	Less than 1 m	Less significant
2	1 m to 10 m	Potholes usually have this dimension
3	10 m to 100 m	Usually common could form the basis for a separate GCD
4	100 m to 1 km	Could form the basis for defining a separate Geotechnical Area

Water conditions

The amount of water associated with a discontinuity (or set of discontinuities) can vary from zero (as is often the case for mining induced fractures around deep level stopes) to large volumes such as associated with fissures in a near-surface environment. The categorisation given in Table 5.10 is the same as developed in the Rock Mass Rating (RMR) classification scheme. In this case however the inflow per 10 m of tunnel length is left out as this methodology has been developed for stoping horizons. It is important to note that the consideration of water is important due to its ability to act as a lubricant on the discontinuity surface thereby lowering the clamping force on the discontinuity.

Table 5.10. Groundwater conditions (after Bieniawski, 1976).

Category	Description	Lubricating effect on discontinuity surface
1	Completely dry	Very low
2	Damp	Low
3	Wet	Moderate
4	Dripping	High
5	Flowing	Very high

It is of importance to note that not all the characteristics will apply equally to all the discontinuities within a certain Ground Control District. However, it is important to consider each discontinuity or set of discontinuities in detail initially so as to ensure all the relevant characteristics are recognised. Thus the categories should serve more as a checklist than as a rating system.

5.2.2.3 Stratigraphy

This category is aimed at assessing representative stratigraphic profiles so as to differentiate between weak and strong rocks on the basis of their respective properties. The properties of the rock mass indicate its strength characteristics. These may include Uniaxial Compressive Strength (UCS), cohesion, friction angle, Young's Modulus and Poisson's ratio. A comparison of the strength of the rock mass with the stress environment could also be used as a relative measure of the stability of underground excavations.

In the definition of Ground Control Districts, the UCS is seen as an index that could be used to differentiate weak rock mass from strong. This distinction is very important considering the fact that the rock mass behaviour is likely to be different in both cases. A typical example is the lava overlying the Ventersdorp Contact Reef (VCR). Laboratory test results showed large disparities in UCS values for what was initially identified as the same rock type. Further analysis to outline the reasons for the significant differences revealed that, for the same rock type, the mineral composition is different. Therefore, under similar mining conditions the rock mass behaviour could also be different. This could necessitate modifications in adopted rock engineering strategies. A practical example is the soft and tuffaceous lava overlying the VCR in places. The unique behaviour of the tuff necessitates different mining strategies to efficiently exploit the VCR. Another example that could be sited is the case of bedded deposits or regions where strong strata are underlain by a weaker material. This example is typical of a situation that is recognised in the Bushveld Complex. Both the hangingwall and footwall are made up of pyroxenite. Cracks and heaving of the footwall are common features between pillars. In certain areas in the footwall one or more clay layers are sandwiched between the pyroxenite layers. This serves to emphasise the need for both the mineralogical composition and stratigraphy to be considered in assessing the rock mass strength and behaviour. In order to account for these significant variations in rock mass strength, the categories given in Table 5.11 are proposed. However, future studies may reconsider the definition of soft/medium/hard thresholds for individual orebodies, and also bedding quality and spacing, middling thickness between reef and weak rock, for example, Khaki shale.

Table 5.11. Classification of the UCS of intact rocks.

Category	UCS (MPa)	Description
1	Less than 120	Soft
2	Between 120 and 200	Medium
3	Above 200	Hard

5.2.2.4 Stress environment

At very shallow depths the k-ratio can be high and in the Bushveld Complex very variable. In contrast this ratio tends to approach 0,5 in deep mines. In the underground environment, the virgin stress situation may be much more complicated. The tectonic history of the area is a decisive factor in the prevailing k-ratio. Gay (1986) found that, in the vicinity of some major faults and dykes, abnormal k-ratios might be encountered. It is therefore essential that, where major geological structures exist, the prevailing k-ratio be established. In addition, the orientation of the major and minor principal stresses should be determined. Depending on the magnitude of the k-ratio, different Ground Control Districts and appropriate rock engineering strategies may have to be identified and employed. In a high stress environment, which is typical of deep level mining, stope closure rates are very high. The intensity of fracturing is also very high and this helps to generate additional, horizontal compressive stresses that act to knit the immediate hangingwall into a coherent beam. Thus, in a geologically undisturbed region, the compressive stresses generated as a result of fracturing help the rock mass to be self-supporting. Contrary to this, a low stress environment is characterised by low stope closure rates, less fracturing and the presence of tensile stresses in the hangingwall. Such an environment is similar to an area where there has been extensive overstoping.

Ideally the values provided in Table 5.12 have been observed to prevail. However, due to the factors mentioned previously, they will have to be determined in geologically disturbed regions. Significant deviations from these values could form the basis for delineating Ground Control Districts.

Table 5.12. Classification of the virgin stress environment.

Category	Depth (m)	k-ratio	Description
1	Less 1000	Greater than 0,74	Shallow
2	Between 1000 and 2250	Between 0,53 and 0,74	Intermediate
3	Between 2250 and 3000	Less than 0,53	Deep
4	Below 3000	Research is ongoing	Ultra-deep

The existing rock engineering strategies adopted prior to identifying the Ground Control Districts must be noted. This information can be combined with a knowledge of the pre-existing small-scale discontinuities to identify fixed and variable factors. Fixed factors will include the characteristics of small-scale geological discontinuities, the virgin stress environment, rock strength and orebody geometry. To a large extent they dictate the behaviour of the rock mass and support requirements. The variable factors include mining induced discontinuities and stresses, and production parameters (see Fig. 5.4). It is a combination of these two categories that influences the behaviour of the rock mass. The fixed parameters may dictate the response of the rock mass but the variable factors could help to aggravate or ameliorate the potential hazard. Knowing which variable factors to adjust or modify provides the key to achieving a safe and efficient mining layout.

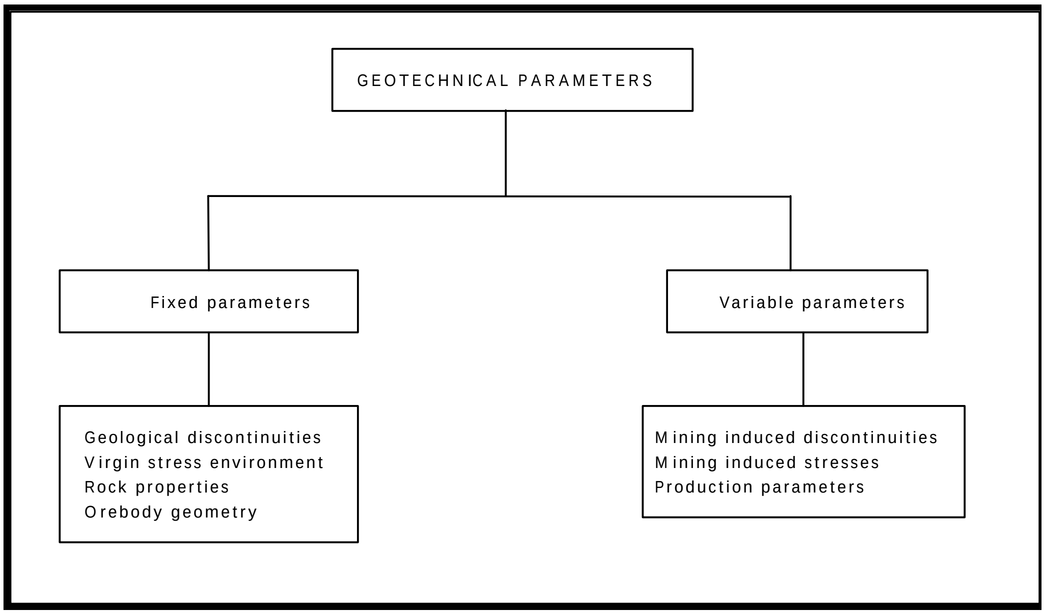

Figure 5.4. Categorisation of the geotechnical parameters.

5.2.3 Establishment of local rock mass responses

The discussions presented above have highlighted some of the effects of the various parameters on the local rock mass behaviour. Underground observations have also shown that the response of the rock mass due to various combinations of the geotechnical parameters imposed on it may be either one or more of the following or other less common consequences:

1 Fall of ground and beam failure,
2 Hangingwall punching, foundation failure and footwall heave,
3 Stress build-up in dykes and slip along geological discontinuities leading to seismicity,
4 Fracturing,
5 Closure and stress concentration at stope face.

In order to understand the mechanisms responsible for these occurrences, the above mentioned responses are explained below:

Fall of ground and beam failure

In shallow mines potential failures are defined by relatively widely spaced tensile fractures or opened-up geological discontinuities and can take the form of block failure, hangingwall beam failure or stope backbreaks. These can occur either rapidly or slowly and can involve a great thickness of hangingwall strata. At depths at which stress fracturing occurs, additional compressive stresses develop in the stope hangingwall, as a result of dilation of the rock in the fracture zone ahead of the face. Moderately high compressive horizontal stresses probably occur in the immediate hangingwall of intermediate and deep stopes; large spans of broken hangingwall rock are held in place by these stresses and thus become to some extent essentially self-supporting. These effects can be disrupted when fractures or geological weaknesses in the immediate hangingwall are unfavourably oriented and intersect, dividing it into a set of blocks of which some are poorly knit and only weakly self-supporting. Thus rockfalls in deep mines are as a result of unfavourable orientation and intersection of mining induced fractures and geological discontinuities.

From the explanation given in the preceding paragraph, it can be summarised that the factors contributing to falls of ground in underground stopes are a combination of discontinuities and the stress environment.

Hangingwall punching, foundation failure and footwall heave

The discussion of pillar design using the Tributary Area Method assumes implicitly that a pillar's support capacity for the country rock is determined by the strength of the ore. Where hangingwall and footwall rocks are weak relative to the orebody, a pillar support system may fail by punching of pillars into the hangingwall or footwall. The mode of failure is analogous to the bearing capacity of a foundation, and may be analysed in a similar way. Heave of floor rock adjacent to the pillar lines, or extensive frittering and collapse of roof rock around a pillar will accompany this type of local response.

Stress build-up in dykes and slip along geological discontinuities leading to seismicity

Dykes are noted to have the ability to store high strain energy. This however depends on the strength of the dyke. Very low strength dykes usually fail at low stress levels when loaded and are classified as non burst-prone. Burst-prone dykes have a UCS of at least 260 MPa and burst violently when the stress on the dyke exceeds this strength. The level of stress build-up in dykes depends on the mining strategy. Earlier researchers (Gay, 1986) measured the state of stress in a dyke with a history of rockbursting. After eliminating the mining induced stresses, Gay (1986) concluded that the pre-mining stress state was lithostatic, i.e. both the vertical and horizontal stresses were approximately equal to the weight of the overburden. He further noted that the horizontal stresses in the dyke were much higher than the equivalent horizontal stresses in the adjacent quartzites. This was thought to be due, perhaps, to the presence of residual tectonic and thermal stresses within the dyke. Such an imbalance in horizontal stress may well be associated with the unstable response of the dyke to the additional induced stresses due to mining.

Conditions for slip on major pervasive features such as faults and joints or for the sliding of individual blocks from the boundaries of excavations are governed by the shear strengths that can be developed by the discontinuities concerned. In addition, the shear and normal stiffness of discontinuities can exert a controlling influence on the distribution of stresses and displacements within a discontinuous rock mass.

Discontinuities may contain infilling materials such as gouge in faults, silt in bedding planes, low-friction materials such as chlorite, graphite and serpentine in joints, and stronger materials such as quartz or calcite in veins or healed joints. Clearly, the presence of these materials will influence the shear behaviour of discontinuities. The presence of gouge or clay seams can decrease both stiffness and shear strength. Low-friction materials such as chlorite, graphite and serpentine can markedly decrease friction angles, while vein materials such as quartz can serve to increase shear strengths (see Brady and Brown, 1985).

All these factors together with mining depth, therefore need to be considered with respect to the likelihood of seismicity and hence the distinction of Ground Control Districts.

Fracturing

Rock fractures when the maximum principal stress exceeds the triaxial strength of the rock at the existing confining stress, or when the confining stress is reduced below that required to hold the rock intact at a particular maximum principal stress. Rock may also fail in tension, the tensile strength being of the order of 10 % of the uniaxial compressive strength. Several categories of mining-induced fractures have been recognised in stopes underground. There are two types namely extension and shear fractures. They all strike parallel, within $\pm 10^{o}$, to the orientation of the stope face and are distinguished by their dip and surface characteristics. Gay and Jager (1986) have given detailed descriptions of each of these categories. It is evident that the type and orientation of fracturing around stopes are controlled by the geometry of the excavation and the magnitude and orientation of the virgin stresses. Under homogeneous geological conditions, the fracturing is remarkably symmetric around the stope. Geological inhomogeneities, both stratigraphic and structural, do, however, have a significant influence on the pattern of fracturing.

In summary, fracturing in the surroundings of an underground stope depends on the rock properties, stress environment and production parameters.

Closure and stress concentration at stope faces

Hawkes and Kwitowski (1979), in an attempt to simplify the observed characteristics of the rock mass behaviour surrounding underground excavations, explained that when rock is excavated in underground mining operations, the static stress distribution around the excavation is redistributed. The vertical stresses on the roof and the floor and the lateral stresses on the side (pillar) walls are removed and, to maintain equilibrium, the rock subsides, the floor heaves, and the sidewalls bulge out. These movements, according to Hawkes and Kwitowski, are usually very small and take place during the actual excavation, so they are rarely, if ever measured. They further allege that subsequent movements of the excavation are brought about by a combination of two factors, i.e. workings in adjacent strata, which causes a further redistribution of the stresses, and creep, associated with fracturing of the rock in the roof, floor and sides (pillars) of the excavation.

Subsequent contributions by Gurtunca *et al.* (1989) stated that the convergence measured underground has three components. These are elastic convergence, inelastic bed separations and closure due to dilation. They described the elastic convergence as the elastic relaxation of the rock mass in response to mining. The inelastic bed separations take place very close to the face where the first few layers in the hangingwall displace downwards due to their own mass. The contribution due to dilation in the rock mass ahead of the face, as proposed by Brummer (1987), is due to shear movements and extension fracturing, which generate horizontal compressive forces, which in turn cause the strata to deform into the stope. Synthesising the above observations, it is deduced that closure depends on the stress environment, rock properties, production parameters and discontinuities (see Fig. 5.5). Excessive amounts of closure in the back areas results in transfer of stress to the stope abutments and may result in violent failure of the rock mass.

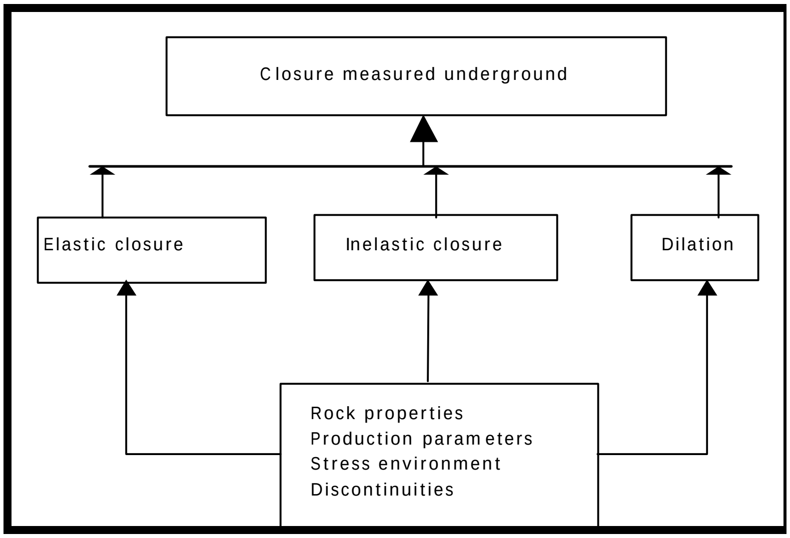

Figure 5.5. Parameters contributing to closure.

The brief explanations offered above concerning the various responses of the rock mass, on a local scale, to mining are only intended to provide an insight into the contributing factors. It is worth noting from those explanations that several combinations of parameters give rise to each of the responses described above. Understanding these responses is crucial to successful delineation of Ground Control Districts. However, the literature survey and review of previous research findings indicate that very little is known to facilitate a thorough description of these responses. Nevertheless, the basic parameters influencing the local rock mass behaviour are summarised in Table 5.13.

Table 5.13 is proposed as a framework that could be used as a tool in identifying any of the rock mass responses described above. No rating values have been assigned to any of the parameters. The direction of the arrows is only intended to highlight the effective range of a parameter on increasing the likelihood of a local rock mass response. Subdivisions exist for each major parameter and these could be used to estimate the probability of occurrence of a rock mass response in that category. To serve as an example, it would be expected that the probability of stope back break occurring at shallow depths would be higher than at greater depths. In each individual mine, experience might suggest the consideration of additional parameters or exclusion of others. In all cases the onus is on the rock mechanics practitioner to ensure the correct identification of the relevant parameters and the corresponding, probable rock mass response. Experience gained by the project team from interaction with mine rock mechanics officers and geologists, and underground observations revealed that several combinations of these parameters could give rise to different degrees of rock mass response to mining. Inadequate information and the unique ground conditions observed on some mines in the Witwatersrand and the Bushveld Complex make it very difficult to account fully for all the parameters as presented in Table 5.13. Detailed discussions of these observations are presented in Chapter 6 and Appendices E and F. Thus some degree of latitude that encourages the practitioners to apply their experience and engineering judgement is deemed necessary.

5.2.4 Demarcation of Ground Control Districts

The next stage in the methodology is the demarcation of Ground Control Districts. This will have to be based on the expected response of the rock mass and the mechanisms responsible for the observed behaviour. In some cases, this response could easily be confirmed from underground observations. The mechanisms involved could be investigated by numerical analysis. It is however important that the aims of this analysis be clearly defined before proceeding with the investigation. The outcomes of SIMRAC Project GAP 415 could assist in applying the correct modelling technique. Once the factors contributing to these mechanisms are determined, similar areas of rock mass response are demarcated on mine plans. It is important to ensure that only Ground Control Districts that depict significant differences in rock mass behaviour are plotted. Different colour coding can be used to highlight different areas. Another way of defining GCDs exists for the mines where large distances of unmined ground separate mining areas. Here the ground conditions and relevant parameters are quantified for each mining area and results plotted in matrix form, area versus parameter. In this way different and similar GCDs can be identified. The danger of this method is that subdivisions within a longwall could be missed. The reasons for considering areas as similar or different Ground Control Districts must be well documented. Included in these descriptions should be the critical parameters used in the distinction of the GCD and the major potential hazards in that GCD. In addition any peculiarities of the GCD such as the effect of mining direction on hangingwall stability, likelihood of potholes, domes, channels, effect of face advance rate on closure, etc.

5.2.5 Implementation of rock engineering strategies

Once the probable response of the rock mass is established and the responsible mechanisms have been identified, the contributing parameters can then be re-evaluated to determine which ones are to be modified to improve the support strategy. Except in situations where height of fall of ground data might warrant re-evaluation of the support resistance, the modifications to be made fall within the variable parameter category. In general terms, this includes consideration of factors such as the mining layout, the direction of mining, rate of face advance, orientation and inclination of mining induced stress fractures, mining span, etc. If the relevant factors are recognised and appropriate measures are effected, the support strategy will be improved.

An example is an underground stope with high closure. Based on the closure rate, areas with similar responses could be demarcated. The important consideration for the demarcation is an understanding of the mechanisms that are responsible for the observed behaviour. This might enable the relative contributions of the elastic and inelastic components of the closure to be determined. In the case of the inelastic component, extensometer results give an indication of the opening of parting planes in both the hangingwall and footwall. Knowing the exact horizons at which separations are taking place in the hangingwall, facilitates the adjustment of the support resistance. Hence an improvement in the support strategy is achieved. Another example is the height of fall of ground and ejected block sizes from the hangingwall. In the presence of well-defined parting planes in the hangingwall, such as the chromitite stringers in platinum mines, this could be taken as the possible height of falls of ground in support resistance calculations. However, in situations where such partings are not clearly defined and jointing predominates as is the case with the soft and hard lavas making up the hangingwall of the VCR, the height of falls of ground will to some extent be controlled by the choice of appropriate mining strategies. The choice of the mining strategies under these conditions has been found to include the mining direction (refer to GAP 102) and possibly the blasting technique.

The responses of the rock mass to mining are so diverse, and in some cases localised, that it is almost impractical to prescribe solutions to specific problems. Nevertheless it is recommended that the current edition of the Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts (1988) be consulted to provide the needed assistance.

Table 5.13. Guidelines for determining rock mass response.

Parameter	Fall of ground	Closure	Punching	Seismicity/ Rockburst
Depth (m)				
0 – 1000	↕	↓	↓	↓
1000 – 2250				
2250 – 3000				
>3000				
k-ratio				
>1	↓	⇡		↓
0,5 – 1				
<0,5				
Reef dip (degrees)				
0-15	↑	⇡		
15 – 25				
25 – 40				
>40				
Middling (m)				
<50	↑			↑
>50				
Orebody strength (UCS)				
<120			↓	
120-200				
>200				
Groundwater condition				
Completely dry	↓			↓
Damp				
Wet				
Dripping				
Flowing				
Hangingwall beam (m)				
0 – 0,5	↑		↑	
0,5 – 2				
2 – 3				
>3				
Hangingwall strength (MPa)				
<120	↕	↑	↑	
120 – 200				
>200				
Footwall strength (MPa)				
<120		↑	↑	
120 – 200				
>200				
Footwall beam (m)				
0 – 0,5			↑	
0,5 – 2				
2 – 3				
>3				

Table 5.13. Guidelines for determining rock mass response (continued).

Parameter	Fall of ground	Closure	Punching	Seismicity/ Rockburst
Number of discontinuities				
Category I	↓	↓	↓	
Category II				
Category III				
Category IV				
Category V				
Dip of discontinuity				
0 – 20 degrees	↑			
20 – 50 degrees				
Above 50 degrees				
Discontinuity spacing				
<100 mm	↑	↑	↑	
100 mm – 1 m				
1 m – 3 m				
3 m – 10 m				
>10 m				
Filling in discontinuity				
No filling	↑			↑
Talcaceous				
Strong but sharp contacts				
Strong but welded				
Discontinuity surface				
Rough	↓		↓	↓
Smooth				
Discontinuity shape				
Planar	↑			↑
Undulating				
Irregular				
Persistence of discontinuity				
1 m to 10 m	↓			↓
10 m to 100 m				
100 m to 1 km				
Several kilometres				

5.3 Summary

Geotechnical Areas ideally should be defined before the onset of mining. This will involve the consideration of parameters such as depth, rock types and their respective strengths, characteristics and abundance of major geological discontinuities, dip and channel width of the orebody, and regional hydrology. Strategic decisions including the choice of mining method, mining layout, mining sequence and regional support strategies must then be made for each regional area. Expected rock mass behaviour which should include stope backbreaks, foundation failure and regional seismicity must be checked against the adopted strategies. Where unfavourable ground conditions are likely to be encountered, the adopted strategic decisions must be modified. The criterion for the adoption and review of the strategic decisions must be documented. Different colour codes must be used to differentiate between distinct Geotechnical Areas on mine plans. The number of Geotechnical Areas must however be restricted to a manageable size; generally, not more than four.

As mining progresses detailed assessment of the ground conditions within different Geotechnical Areas must be conducted to identify distinct, local rock mass behaviour. These local responses which may include closure, fall of ground, hangingwall and footwall punching, and seismicity or rockburst potential must be continuously monitored as mining progresses. In each of these areas, significant differences in the geotechnical parameters that necessitate a modification in the adopted support strategies must be classified into Ground Control Districts. These areas must be plotted within their respective Geotechnical Areas on the base plans.

5.4 Discussion and conclusions

Due to varying opinion on the meaning of a Geotechnical Area across the mining industry, a standard definition has been adopted. The definition describes a geotechnical area as an area having a specific rock mass response and associated hazards to mining activities where the same rock engineering strategies may be applied.

Geotechnical Areas need to be delineated on two scales, which require different degrees of detail. Names suggested for regional geotechnical areas are either Geotechnical Areas or Mining Environments and for local areas, Ground Control Districts.

A methodology for the definition of Geotechnical Areas and Ground Control Districts on the South African gold and platinum mines has been established. Contrary to other geotechnical classification schemes that aim at assigning numerical values to describe various rock mass characteristics, this methodology is seen as a guideline that enables the delineation of both Geotechnical Areas and Ground Control Districts.

6 Newly defined methodology: Review and evaluation

The identification of critical parameters has been highlighted as a crucial initial step in identifying and delineating geotechnical areas. Theoretical analyses of fundamental rock mechanics principles coupled with various views expressed by mine practitioners, and the findings of past research work, suggest that there could be a variable degree of influence of these parameters on rock mass behaviour. The parameters as imbedded in the methodology were investigated during underground visits at selected mines.

Monitoring sites in the Bushveld Complex were established at Impala and Eastern Platinum Mines. The investigations conducted were geared towards identifying and assessing the degree of relevance of the critical geotechnical parameters associated with the two main reefs, i.e. UG2 and Merensky. A summary of the strategies adopted, results obtained and the observations made (see Appendices E and F), coupled with the impact on rock mass behaviour are discussed below. The approach adopted for the gold mines was slightly different and mainly centred on intensive visits to distinct geotechnical areas at Beatrix, St. Helena, and Oryx gold mines in the Free State. In addition, observations made during underground visits to some selected sites at Kloof gold mine are discussed. The major deductions arising from these visits and past research findings are also presented in this section. Photographs, to substantiate the observations made are included in Appendix F. The results of this work have validated many of the parameters included in the methodology as significant factors in distinguishing different geotechnical areas and ground control districts.

6.1 Platinum Mines

Unlike the deep gold mines, the behaviour of the rock mass at the platinum mines is typical of what is expected at shallow mining depths. Comparatively little work has been done to fully understand and quantify this behaviour. Nevertheless, some of these mines have variable stratigraphic assemblages, unfavourable jointing and unpredictable characteristics of the orebody geometry. These need careful consideration while defining geotechnical areas. In an attempt to also verify the influence these parameters exert on the rock mass behaviour as associated with the two most exploited reefs, i.e. Merensky and UG2, the following strategies were adopted:

a- underground visits to potential panels to select a fairly representative monitoring site,

b- installation of monitoring devices to quantify the vertical component of closure taking place around the excavation,

c- borehole petroscopy, and extensometers to traverse existing planes of weakness with the objective of quantifying the amount and the horizon where separation along discontinuities occurs,

d- joint surveys to determine how discontinuity orientation affects the stability of the immediate hangingwall in relation to the mining direction. Gathering of relevant data was undertaken and stereographic projection techniques were used to analyse the results.

Results and interpretations are discussed in the following section.

6.1.1 Impala Platinum Mine

The selected site is located on the 16/21 line, 1 Shaft at a depth of about 1000 m below the collar elevation. The Merensky Reef is being mined. The immediate hangingwall rock type is spotted leuconorite and the footwall is made up of spotted anorthosite. South 4 and 5 panels (see Fig. 6.1) were initially chosen as the most suitable sites that could be used to study the behaviour of the rock mass in this geological environment. Mining was progressing in these panels towards a mined out area.

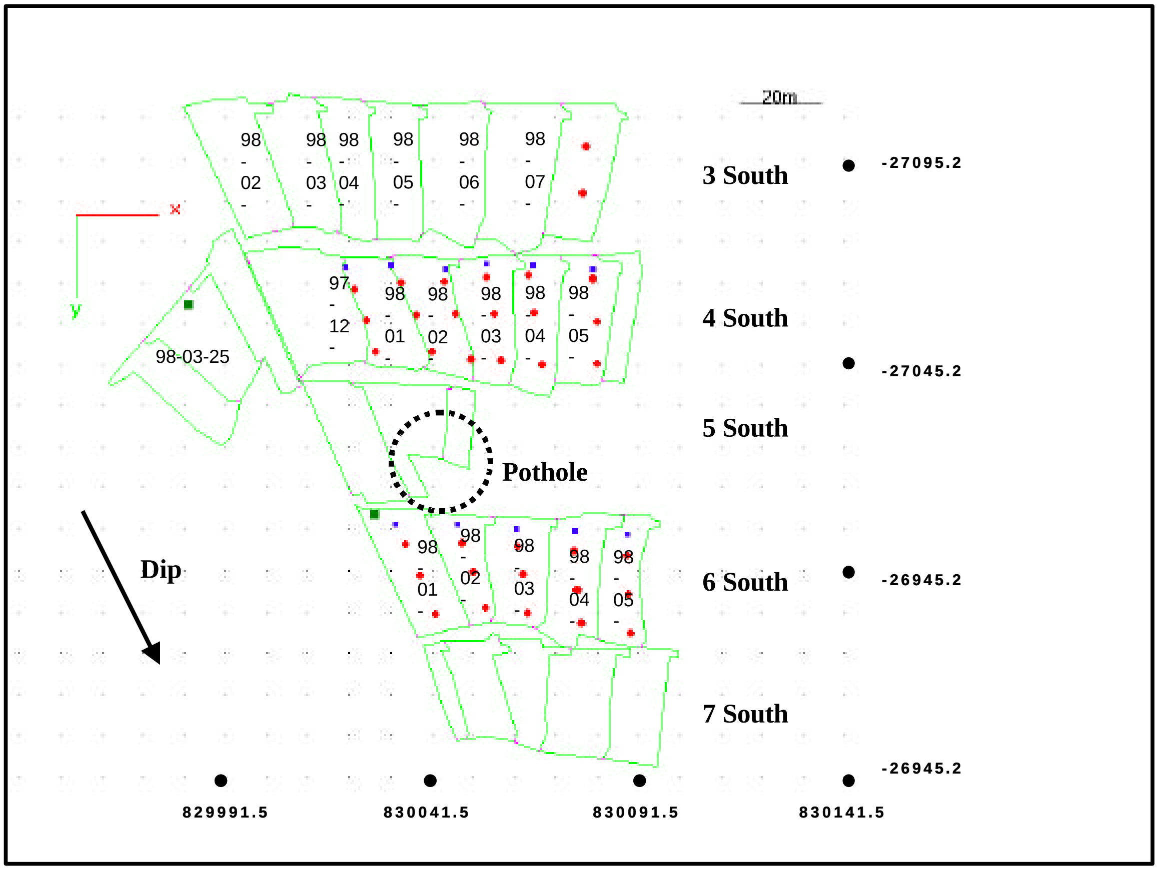

Figure 6.1. The study area at Impala Platinum Mine.

South 4 panel was the first stope to be equipped with closure monitoring devices. The next stope proposed was to be South 5 panel but, due to the presence of a pothole, South 6 was selected and instrumented. This decision was partially necessitated by the desire to determine the influence of the rate of face advance on closure.

In addition to the closure monitoring stations in the South 4 panel, extensometers were installed along the strike gully. Drilling of a borehole was discontinued due to poor hangingwall conditions. Alternatively, two boreholes were drilled, one each in the hangingwall and footwall, along the strike gully of South 6 panel. The hangingwall borehole extended up to about 16 m. The cores recovered from the drilling operation were logged and an RQD estimation suggests values in the region of 80 % to 90 %, giving an indication of competent strata. In contrast, the footwall material from the 8 m long borehole was weaker. Despite the draining of water from the borehole into the panel, the water level remained almost half full. This gave an indication that there could be a major discontinuity transecting the borehole. A petroscope was used to observe the location of fractures in both holes. Up to about 4 m into the hangingwall borehole, the results tallied with those obtained from the core logging. Beyond this height poor visibility rendered the results unreliable. Little information was obtained from the footwall borehole due to the presence of water. Combining the scanty information obtained from the petroscope and the core logging, anchors were placed at specific intervals in the boreholes. These were monitored periodically with the aim of deducing the heights and depths at which separations are taking place in the hangingwall and footwall.

Summary of results

Average closure rates of about 5,7 mm per metre of face advance were recorded at some of the monitoring stations in South 4 panel (see Fig. 6.2). These high closure rates resulted in most of the sticks, used as support units, failing at about 15 m behind the stope face (see Appendix E, Figs. E.15 and E16). Consequently loose blocks fell from the hangingwall whilst the neighbouring panels remained stable. This gave an indication that this area is a separate geotechnical district which needs additional and yielding support.

Figure 6.3 shows a comparison of convergence results obtained from the in situ monitoring and MINSIM-W modelling. It is indicated that, even if the elastic constant is downgraded by more than 50 % in the model to 35 GPa, the onset of a higher closure rate in South 4 panel is not simulated. The amount of inelastic deformation is five times that predicted by elastic theory. Hence the observed high closure could have been triggered by a significant, inelastic separation along a bedding-parallel discontinuity in the hangingwall. This was later confirmed when a fall of ground occurred along the centre gully and exposed a well-defined plane of weakness about 2 m into the hangingwall. However, detailed mapping of discontinuities in the area reveals that the presence of this structure is localised. This seems to account for the negligible effect of the potential partings on the comparatively lower closure rates in South 6 panel (refer to Appendix E, subsection E.1, for a detailed report on the joint mapping, closure rates and extensometer results).

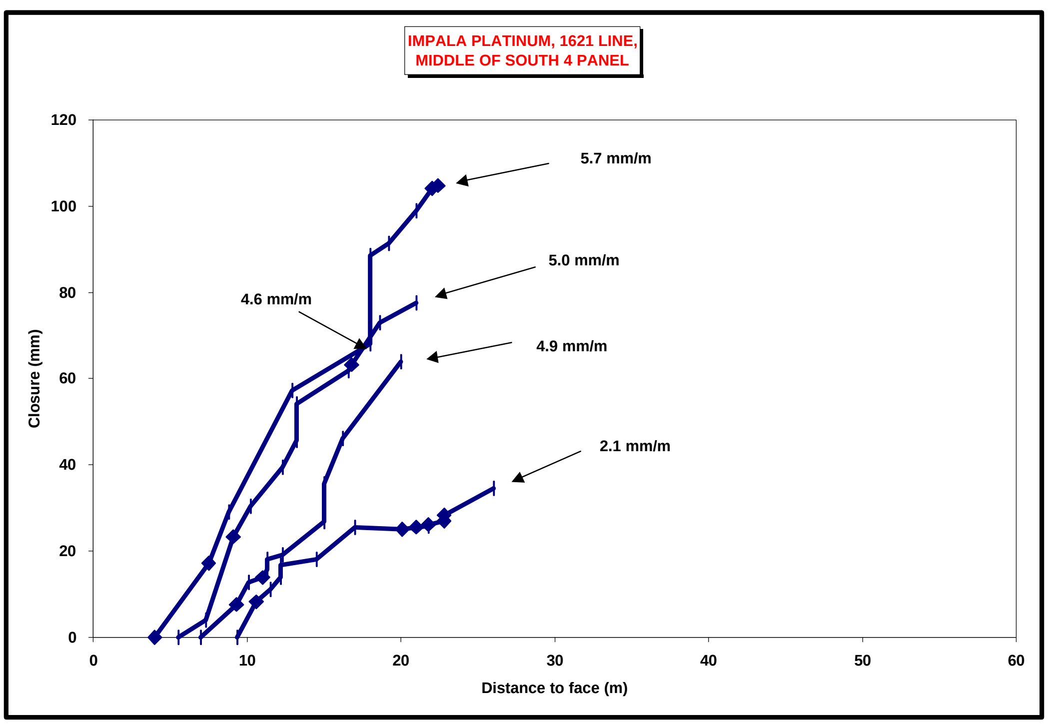

Figure 6.2. Closure results obtained at Impala Platinum Mine, along 16/21 Line, 1 Shaft, middle of South 4 panel, period January 1998 to June 1998.

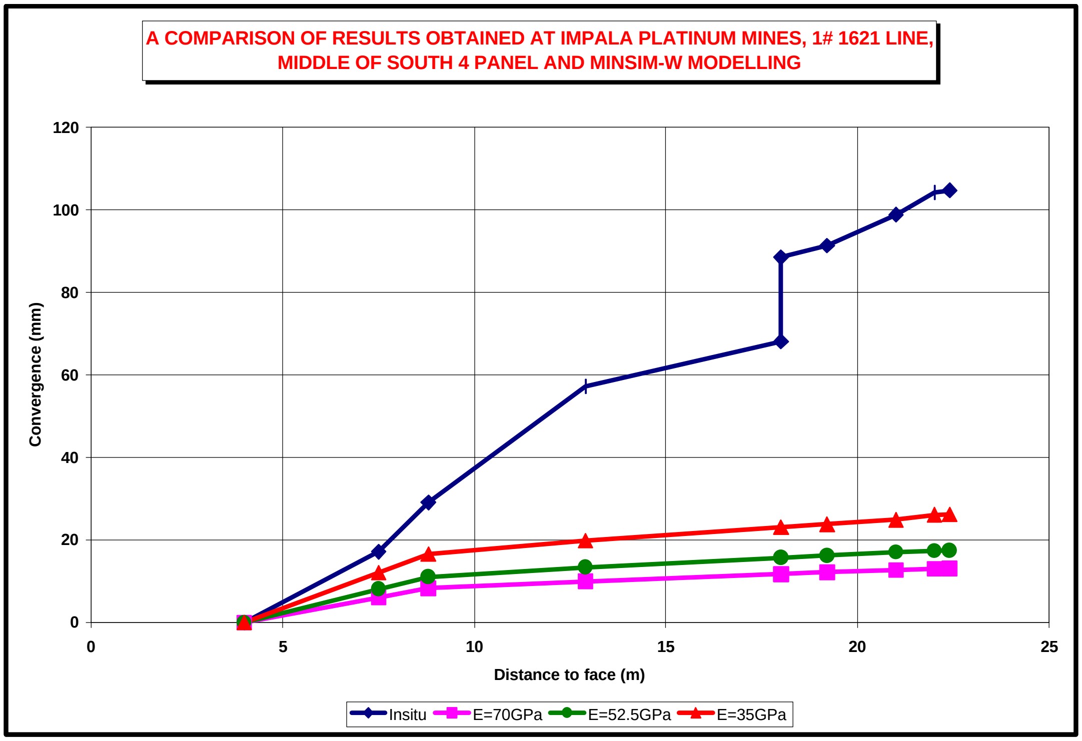

***Figure 6.3.	Comparison of results obtained at Impala Platinum Mine, 16/21 Line, 1 Shaft, middle of the South 4 panel, and MINSIM-W modelling.***

6.1.2 Eastern Platinum

The selected sites are located on 4 and 5 levels (see Fig. 6.4). The UG2 Reef is being mined. The footwall of the UG2 comprises norite or locally pyroxenitic norite. A 10-30 cm thick serpentinised pegmatoid locally occurs between the norite and the UG2. The hangingwall comprises up to 9 m of pyroxenite, which is locally feldspathic. A 1-3 cm thick chromitite seam occurs 70-90 cm above the UG2-pyroxenite contact. Three other chromitite seams (5-8 cm thick) occur 2,9-4,0 m above the UG2-pyroxenite contact. These chromitite seams divide the pyroxenite into an upper and lower unit. The pyroxenite is overlain by mottled anorthosites.

The UG2-pyroxenite contact is locally polished with a white, powdery infill indicating some horizontal movement. The contact of the first chromitite seam with the pyroxenite is in places also polished with either a grey gouge or white powdery infilling. This contact forms a pronounced parting along which rockfalls may occur.

On 4 level, 4W and 5W panels were initially instrumented with closure ride monitoring devices. Two boreholes, one in the hangingwall and the other directly below it in the footwall, were also drilled. Cores obtained from the drilling were logged. This information was combined with petroscope observations to determine the intervals at which the anchors of the extensometer should be placed.

No consistency exists in the obtained closure results (see Appendix E, subsection E.2.3). This is attributed to the shallow depth of mining. In contrast a considerable amount of separation was recorded by the extensometer along 5W panel (see Fig. 6.5). Detailed analysis of the results reveals that the highest anchor records the maximum separation. The results emphasise the importance of stratigraphic profiles in delineating geotechnical areas across the UG2, especially with regard to pronounced parting planes.

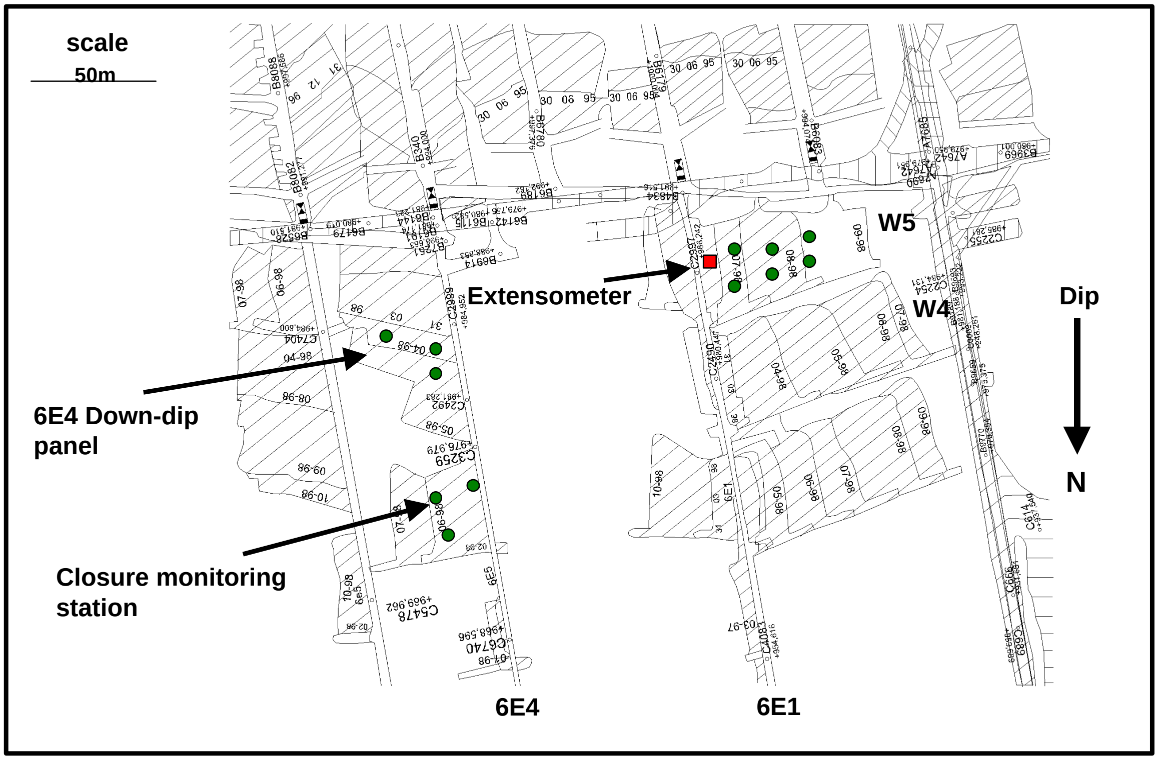

Figure 6.4. Study area at Eastern Platinum Mine.

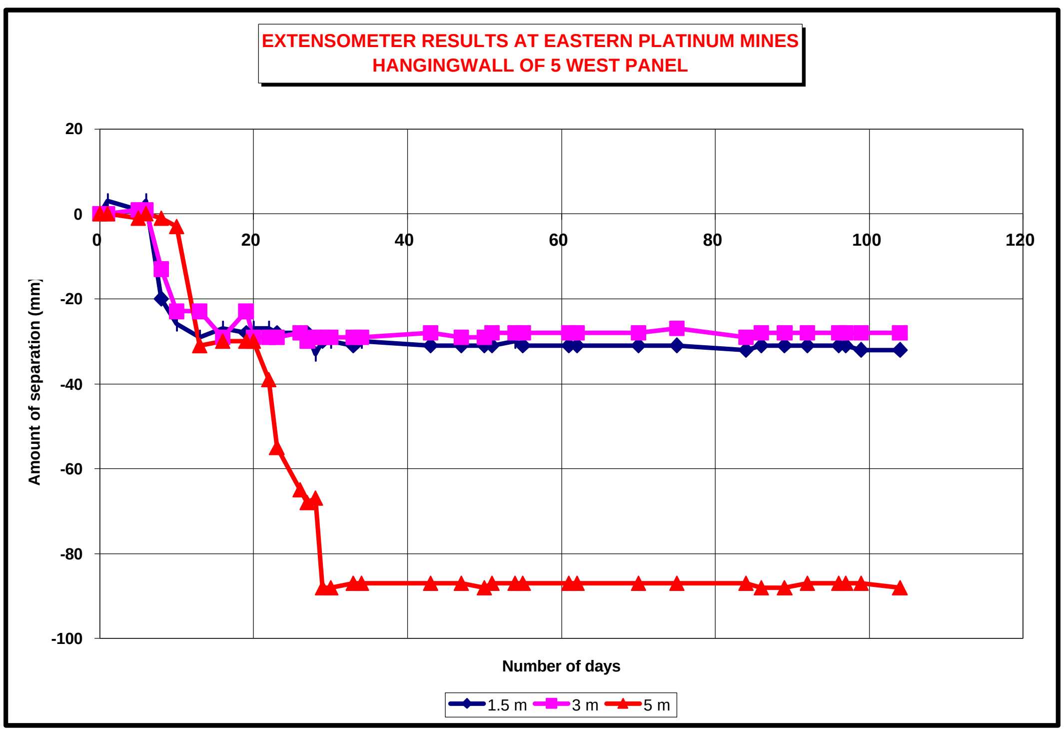

Figure 6.5. Extensometer results obtained at Eastern Platinum Mine, hangingwall along 5W panel.

6.2 Gold Mines

Geotechnical areas have in the past been delineated on the basis of different combinations of hangingwall and footwall lithologies. Detailed studies of the rock mass behaviour in these areas that are associated with the Ventersdorp Contact and Carbon Leader Reefs have been conducted. In order to improve the understanding gained from previous projects and to verify the relevance of other parameters, the project team visited several underground mines in the Free State and some in the West Rand. The visits reiterated the fact that other factors such as geological discontinuities, stratigraphic assemblages, orebody geometry and the state of the in situ stress will also influence the behaviour of the rock mass. However, the choice of correct mining and rock engineering strategies may help to alleviate the potential hazards. The following paragraphs outline some of the observations made during the visits.

6.2.1 Beatrix Gold Mine

The Beatrix Reef of the Kimberley succession is the primary orebody. The reef sub-outcrops against the Karoo Supergroup along the southern boundary of Beatrix Mine. It dips towards the north and northeast. Dips average between 5°-10° in the southern portion of the mine, becoming flatter in the centre, and steeper (15°) towards the north. The reef thickness varies rapidly from 0,2 - 5 m, due to well-developed channelling. The channels trend EW to NE-SW. The top contact of the reef is planar.

The reef occurs at the base of the Eldorado Formation and unconformably overlies argillaceous, occasionally gritty quartzites of the Kimberley successions (LF1 and 2). The angular unconformity is approximately 2°.

The VS4A forms the immediate hangingwall to the Beatrix Reef. It comprises dark grey to black fine grained quartzites with interbedded black, horizontally laminated shale. Bedding planes are characterised by thin shale partings, which are locally 10 cm thick. A smectite layer occurs locally in the footwall along a parting plane. This results in poor cohesion of the beds resulting in lateral movement along the parting plane.

<u>Description of sites</u>

Various sites around 1 and 2 Shafts were visited, and these are described in the following. The most relevant geotechnical parameters of the visited sites and their relationship to the established methodology for defining geotechnical areas are summarised in Table 6.1.

6.2.1.1 Beatrix Gold Mine, 1 Shaft

The site visited is the 17B59 stope, on 16 Level at a depth of 866 m below surface. The Beatrix Reef is mined at stoping widths of up to 5 m. The geotechnical parameters of the site are provided in Table 6.1. The 80 degrees west dipping N/S and the near vertical E/W joint sets, together with the hangingwall shale partings, define the fall of ground characteristics. The N/S joints are calcite coated. Therefore 3 m x 6 m pillars are left in place to provide the needed hangingwall support.

The area lies between very important service excavations. Directly below are the chairlift, material and conveyor declines. The 16 crosscut and RAW north to the shaft lie immediately above the area. The span between raise lines is about 120 m. Due to the high stoping widths, and previous occurrences of stope back breaks, 3 m x 6 m pillars are left between 15 m panels so as to limit the effective span, and to stabilise the hangingwall. This strategy had been adopted effectively in the back areas until mining progressed to the region between the service excavations where an increase in the stoping width occurred. This rendered the adoption of the same support strategy impractical, due to an increase in the pillar dimensions to 5 m x 8 m, also resulting in a stress build-up on the footwall service excavation. The mine therefore decided to use a combined support system of 1,8 m rockstuds, 150 cm x 150 cm composite timber packs and 20 ton Rocprops. The Rocprops are 6 m long and are placed at 3 m x 3 m, skin to skin (see Appendix F, Figs F.1 and F.2 where the hangingwall condition and the installed support units prior to a collapse are shown).

A massive fall of ground occurred about five months prior to the visit, affecting E6, W6 and E7 panels. The extent of fall covered an area of about 2365 m^2. The fall of ground occurred due to an unfavourable intersection of a fault, joints and a hangingwall bedding plane. The fault with a throw of about 10 m dips almost vertically to the west. It is up to about 10 cm thick, and contains quartz and fault gouge. A 15 cm thick dyke is associated with the fault (see Appendix F, Fig. F.3). The joints run along dip and are almost vertical. They strike in the E/W direction and have a clean average aperture of 1 mm (see Appendix F, Fig. F.4). The hangingwall bedding plane is argillaceous with ripple marks on its surface. The intersection of these structures isolated a large block, the weight of which could not be sustained by the existing support system. Figure F.5 (see Appendix F) shows the intersecting joint and the fault planes responsible for the collapse. The bedding plane that formed the top surface of the ejected block is not conspicuous in the picture. Figure F.6 (see Appendix F) shows the buckling of the Rocprop, cracks in the footwall and the punching of the Rocprops into the footwall.

E4 and E5 stopes were still standing, although the possibility of future collapses in some parts are anticipated. Precautionary measures such as increased support density and keeping to the right support recommendations are being taken by mine personnel to minimise the extent of damage. Figure F.7 (see Appendix F) shows variable beam thicknesses and the predominant joint directions.

The observations made in the 17B59 stope and presented in the preceding paragraphs demonstrate the importance of considering geological features in defining geotechnical areas. They also highlight the relevance of most of the characteristics of discontinuities such as dip, dip direction, throw and infilling, as listed in Chapter 5 and Appendix C. Furthermore, they reiterate the fact that cognisance must be taken of these parameters in adopting the correct mining strategies.

6.2.1.2 Beatrix Gold Mine, 2 Shaft

In deep level mines, seismicity is common and is often associated with geological structures such as faults and dykes. In shallow mines, geological features control the hangingwall instability. A practical example of how these features dictate the behaviour of the rock mass surrounding underground excavations at such depths was identified at Beatrix. The observations were made in E17 panel of 16I28 crosscut, on 25 Level. Figure 6.6 shows the extent of mining in the area, and an associated fall of ground. Figure 6.7 is a section along A – A^1. Refer to Table 6.1 for details on geotechnical parameters associated with this site. A combination of timber packs and props was used as support.

The fall of ground site has two sub-parallel east-west trending dykes. The northerly dyke is 20 cm thick, greyish and aphanitic with smooth, regular chilled margins. It dips 50° to the north. The contacts contain calcite. The southerly, waterbearing dyke is 50 cm thick, green-black, highly altered and soft. It dips 40° to the south (see Fig. 6.7). Lateral movement has occurred along the dyke - slickensided contacts and localised jointing within 45 cm of the dyke was observed.

***Table 6.1. Summary of underground observations at Beatrix Gold Mine and implication for geotechnical area definition*.**

<table>
<tr><th colspan="2" rowspan="2">Geotechnical parameters</th><th colspan="2">Site visited</th><th rowspan="2">Remarks</th><th rowspan="2">Implication for geotechnical area definition</th></tr>
<tr><td>1 Shaft
17B59 stope</td><td>2 shaft
E17, 16I28 stope</td></tr>
<tr><td colspan="2">Depth (m)</td><td>866</td><td>818</td><td rowspan="9">The difference in reef thickness in both the 17B59 and E17, 16I28 stopes resulted in different support strategies being adopted.

Local occurrences of more argillaceous quartzites in the footwall were seen to contain cracks in the 17B59 stope. Rocprops were observed to be punching into the footwall

Massive falls of ground observed at both sites. The discontinuities defining the falls of ground are similar. The levels at which the shale partings occur in the hangingwall differ.</td><td rowspan="9">Considering the general characteristics of the orebody, major geological discontinuities, stratigraphy and the depth of mining, the two sites would be classified under the same regional geotechnical area according to the methodology.

However, the local stratigraphic variations and the occurrence and difference of the characteristics of small-scale discontinuities which accounts for the local responses would require the demarcation of different ground control domains.

In the 17B59 stope, the important parameters to be considered would include:
a) The high stoping width,
b) The characteristics of the fault, dyke, joint and the bedding plane.
c) The strength of the footwall due to the cracks observed.

In the E17 panel, the area would be characterised by the two dykes, the N/S trending joints, and the well-defined shale parting.</td></tr>
<tr><td rowspan="2">Stratigraphy</td><td>Hangingwall</td><td>VS4A Quartzites</td><td>VS4A Quartzites</td></tr>
<tr><td>Footwall</td><td>LF1 Quartzites</td><td>LF1 Quartzites</td></tr>
<tr><td rowspan="3">Orebody</td><td>Reef</td><td>Beatrix</td><td>Beatrix</td></tr>
<tr><td>Reef thickness (m)</td><td>1.8 m –5 m</td><td>0.9 m</td></tr>
<tr><td>Dip</td><td>Shallow dipping
(less 15 degrees)</td><td>Shallow dipping
(less than 15 degrees)</td></tr>
<tr><td rowspan="2">Discontinuities</td><td>Hangingwall</td><td>Shale partings up to 10 cm thick. Bedding planes defined by large scale planar cross beds.

Joints trend in N/S and E/W direction. E/W set is near vertical and N/S set dips at 80 degrees to the west. Spacing of the E/W set is 50-150 cm. Local NW/SE joint direction.

Faults follow the N/S joint trend. Dykes are associated with faults.</td><td>Shale partings up to 10 cm thick and 1.3 m into hangingwall. Bedding planes defined by large scale planar cross beds.

Joints trend in N/S and E/W direction. E/W set is near vertical and NS set dips at 80 degrees to the west. Spacing of the E/W set is 50-150 cm.

Two E/W trending dykes. The northerly dyke dips at 50 degrees. Chilled margins observed. Calcite observed along contacts.</td></tr>
<tr><td>Footwall</td><td>Thin shaly partings. Clayey layer in footwall.</td><td>Footwall is more siliceous than at 17B59 stope.</td></tr>
</table>

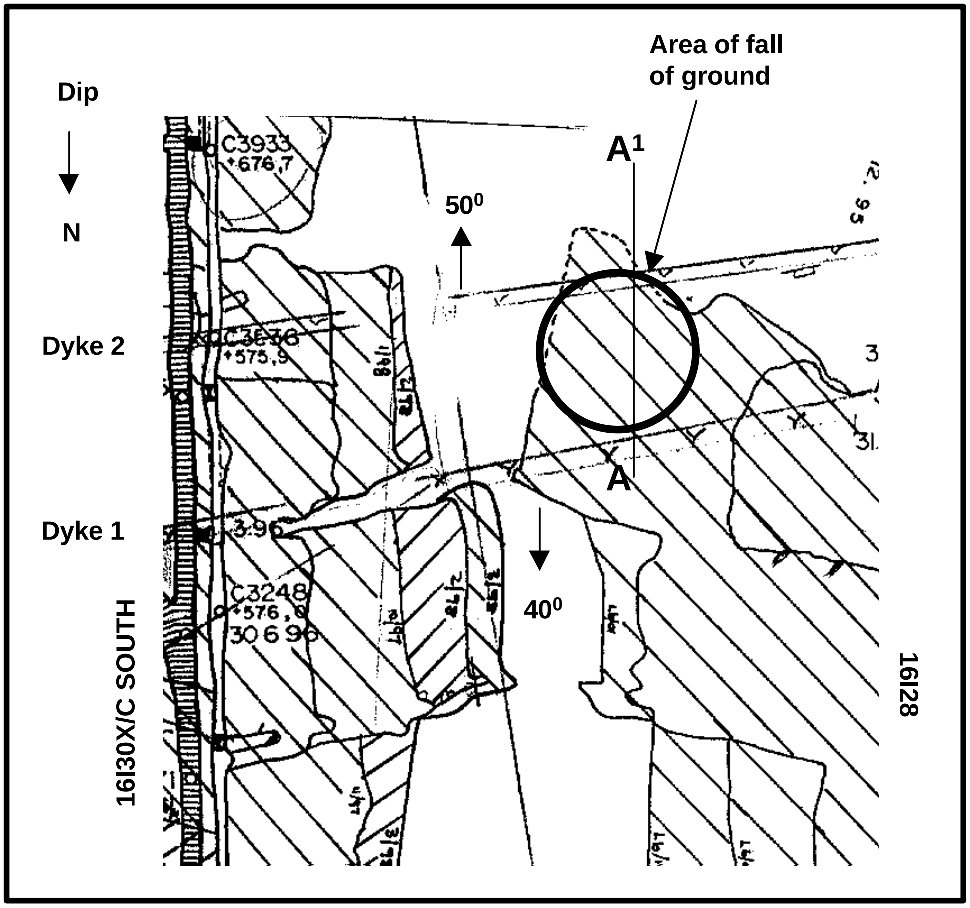

Figure 6.6. Area of fall of ground bounded by dykes at Beatrix 2 Shaft (Scale: 1:1000).

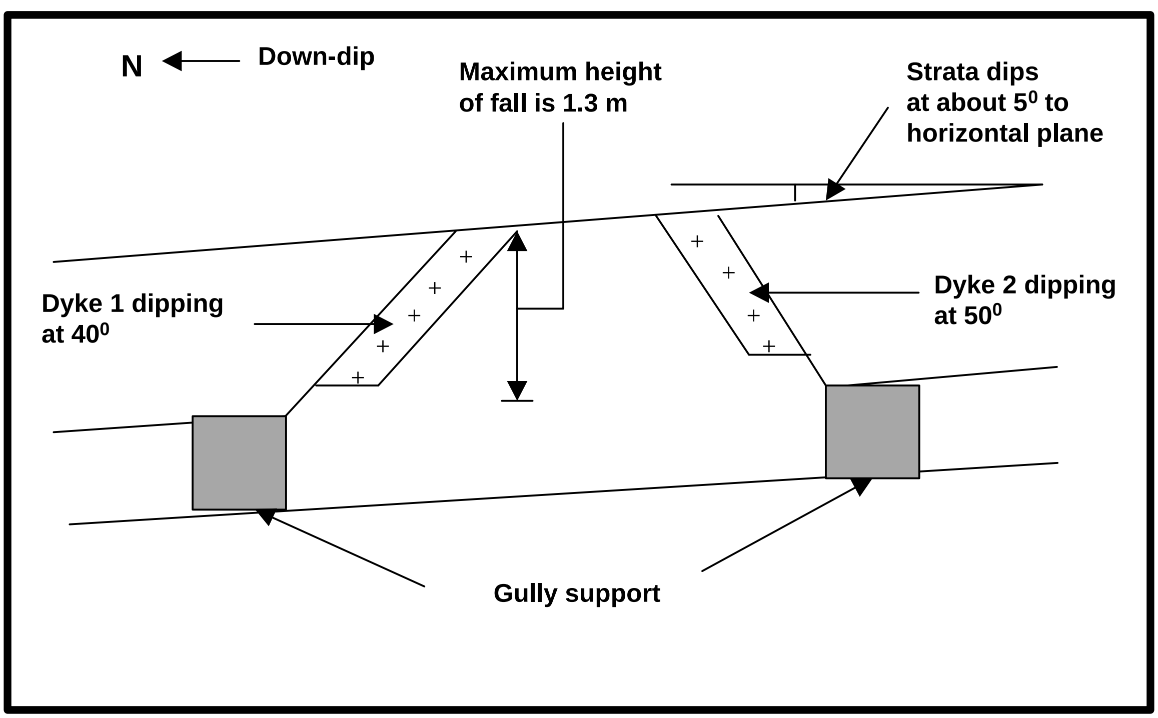

Figure 6.7. Schematic section along A – A^1, as shown in Figure 6.6.

The dykes bound the fall-out area (see Appendix F, Fig. F.8, and Fig. 6.6). N/S trending, near vertical, clean joints divide the block into smaller units. The upper surface is a well-defined shale parting. The height of the fall of ground is about 1,3 m. The fallout has occurred along the strike of the dyke almost to the face. Hangingwall conditions are acceptable for the rest of the panel.

Implications for defining geotechnical areas

The general observations presented in the preceding paragraphs support the need for factors like the stratigraphy, i.e. rock assemblage and discontinuities, to be considered, while defining geotechnical areas. A consideration of only the stratigraphy to determine the beam thickness on an entire mine will be erroneous in geologically disturbed areas.

With regard to the established methodology, it is noted that the characteristics of the geological discontinuities, differences in reef thickness and strength of the footwall are the major geotechnical criteria. These criteria form part of the methodology and are defined once mining has commenced.

6.2.2 St. Helena Gold Mine

It has been proposed that reef orientation including rolls is one of the critical geotechnical parameters, due to its importance when choosing a mining method. It is expected that in areas where different mining methods, amongst longwall and cut-and-fill, are adopted, significant changes in the reef orientation amidst other factors, like the competence of the surrounding strata, can influence the choice. The above is mainly driven by theoretical considerations. However, for practical purposes the methodology needs to be evaluated against case studies. A suitable example was identified at St. Helena.

General orebody description: St. Helena Gold Mine

The Basal Reef is a multi-facies conglomerate varying in thickness from 0,1 – 6,0 m. It occurs at the base of the Harmony Formation and lies with a small angular unconformity on the UF1 Member of the Welkom Formation. This is a fine to medium grained semi-siliceous quartzite, containing scattered angular fragments and rare interbedded grits. It is well bedded with thin argillite partings along the bedding planes. The variable thickness of the Basal Reef is due to channelling, which follows a north easterly to easterly trend. The mining cut usually considers the basal 50 cm of reef where the gold is concentrated. The reef dips towards the east and northeast varying from 25° to 50°. In the extreme west of the mine a southerly plunging syncline is present where dips reach 70°. The hangingwall consists of the Basal Reef quartzite, Top of Reef (TOR) quartzite, the Khaki Shale member and the middling quartzite. Refer to Table 6.2 for further geotechnical criteria as associated with the Basal Reef at St. Helena gold mine. The Leader Reef is the secondary orebody and lies some 8-20 m above the Basal Reef.

Description of sites

Various sites around 4 and 2 Shafts were inspected, and these are described below.

6.2.2.1 St. Helena Gold Mine, 4 Shaft

The stopes visited were 4C36 and 5C36, located at an approximate depth of 400 m below the collar elevation of 4 Shaft (see Table 6.2).

The 4C36 stope is characterised by a NW/SE trending, steeply SW dipping joint set. The Basal Reef dips at about 70°. The hangingwall and footwall lithologies are relatively competent with localised geological features impacting on stope stability. Micro-faulting trending along strike, results in ridging of the footwall along the faults. The hangingwall is more rugged in the vicinity of faults. The steep dip of the reef combined with the fairly competent nature of the surrounding rock mass makes shrinkage-mining the best method to exploit the deposit (see Appendix F, Fig. F.9). No support units were observed in the excavation.

The 5C36 stope has a similar joint set as well as a N/S trending set which dips at 45° to the east. The footwall and hangingwall rock types are similar to 4C36 stope, but the reef dips at about 45°. Up dip mining is used with pencil sticks placed at 2 m x 2 m spacing serving as in-stope support units (see Appendix F, Fig. F.10). Crush pillars 6 m x 3 m are cut along the gullies to carry the deadweight.

6.2.2.2 St. Helena Gold Mine, 2 Shaft

The site is located at 24/26 crosscut (see Table 6.2). At the time of the visit no mining was taking place. The Basal Reef has been dragged and sheared by a flat dipping fault. The stope dips to the west following the orientation of the fault. Ramp-like faults have their origin in the flat fault and extend into the hangingwall (see Appendix F, Fig. F.11). Normal faulting has resulted in the Leader Reef being locally exposed in the stope. The diamictite is a dark grey, argillaceous quartzite. It appears to be massive with planar bedding. It thus tends to fracture in any direction and causes serious strata control problems in the area. The ramp-like faults, the omni-directional fracturing and the drilling of blastholes into the hangingwall may serve to destabilise the hangingwall diamictite. A seismic event had occurred in the area prior to the visit. The diamictite was highly fragmented in the process and had fallen out from the hangingwall in between the timber packs (see Appendix F, Fig. F.12).

<u>Implications for defining geotechnical areas on St Helena Gold Mine</u>

The observations made at the 4C36 and 5C36 stopes have revealed that the reef orientation and the competence of the surrounding strata are critical parameters in choosing a mining method. On the steeper limb of the orebody, the stable nature of the host rock influenced the choice of the shrinkage mining method. No support units are required to keep the excavation open. On the contrary, pencil sticks had to be used to maintain the integrity of the hangingwall in the up dip stope. Hence the difference in the reef orientation and the strength of the rock mass results in these two areas being geotechnically different. On the basis of these parameters, the areas visited could be categorised into three different ground control districts as follows:

a) Steeping dipping, i.e. $>70^0$ and strong surrounding strata leading to the choice of shrinkage mining method.

b) Moderate dipping, i.e. $\pm 45^0$ leading to the choice of an up-dip mining configuration.

c) Shallow dipping, leading to the choice of scattered breast mining strategy.

These geotechnical differences are adequately addressed in the methodology.

Table 6.2. Summary of underground observations at St. Helena Gold Mine and implications for geotechnical area definition.

Geotechnical parameters		**Site visited**		**Remarks**	**Implication for geotechnical area definition**
		4 Shaft 4C36/5C36	2 Shaft 24/26 crosscut		
Depth (m)		400	1600	The variation of the reef dip from steep to moderate results in the adoption of two different mining strategies. These are shrinkage and up-dip mining strategies where reef dips 70 and 45 degrees respectively. The occurrence of a localised diamictite exacerbates the hangingwall conditions in the 24/26 X-cut. The hangingwall thus tends to fracture in any direction.	In applying the methodology, the 4C36 and 5C36 would be classified under the same regional geotechnical area. A steep dipping reef and competent surrounding strata could define this geotechnical area. Variation in the reef dip however, would result in the regional geotechnical area being subdivided into different ground control domains. The 24/26 X-cut could be classified as a ground control domain under a regional geotechnical area defined by a shallow dipping reef, depth and incompetent hangingwall lithology.
Stratigraphy	Hangingwall	Competent TOR quartzites	Middling quartzite (Harmony formation)		
	Footwall	Competent UF1 quartzites	Competent UF1 quartzites		
Orebody	Reef type	Basal	Basal		
	Reef thickness (m)				
	Dip	Variable dip between 30 and 70 degrees. In 4C36 it dips at 70 degrees and 45 degrees in 5C36.	Reef dragged along flat fault. Unusual stoping as stope follows reef dragged along flat fault.		
Discontinuities	Hangingwall	NW-SE trending joint steeply dipping to the SW. Micro-faulting trending along strike and perpendicular to the reef. These were noted at both sites but in addition, there is another NS trending joint dipping at 45 to the east in the 5C36 stope.	The hangingwall was highly fractured due to seismicity, blasting and ramp faults that branch off the flat fault.		
	Footwall				

6.2.3 Oryx Gold Mine

The primary orebody is the Kalkoenkrans Reef varying in thickness from 0,2-5 m. Maximum thickness is obtained in a down-dip channel, which is several hundreds of metres thick. The western part of the orebody is characterised by very steep dips due to a syncline. The reef occurs at the base of the Aandenk Formation and rests unconformably on the LMF2 quartzites. It is the only orebody in this area due to southward truncation of the succession.

The general stratigraphy of the rock mass surrounding the Kalkoenkrans Reef is illustrated in Figure 6.8. Immediately above the reef are the top of reef (TOR) quartzites, the strength of which varies between 240 and 270 MPa. A weak shale layer overlies the TOR. The Aandenk, with a variable UCS between 140 and 180 MPa is sandwiched between the shale and the much stronger Eldorado Formation. The parting between the shale and the overlying and underlying formations is weak. In areas where the TOR is unstable, the height of fall of ground has been observed to extend up to the contact between the Eldorado Formation and the Aandenk. It has also been noted from underground observations and core logging that this stratigraphic sequence varies laterally. The stratigraphic relationships of the various lithologies as viewed from NE to SW are shown in Figure 6.9. Lateral variations results in different combinations of hangingwall and footwall rock types in some places. Figures 6.10, 6.11 and 6.12 illustrate the typical hangingwall conditions associated with each of these combinations.

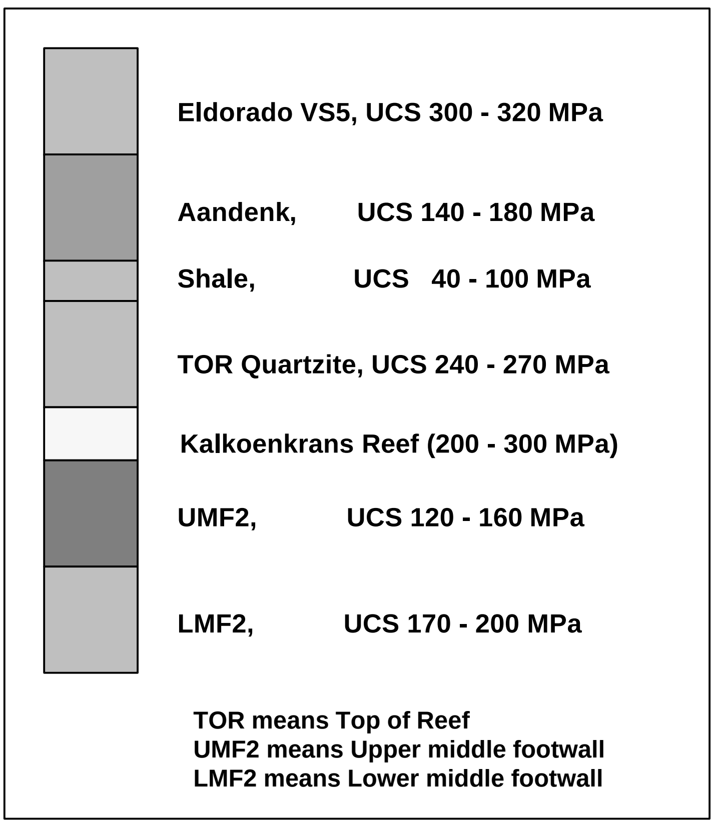

Figure 6.8. Typical stratigraphic profile, Oryx Gold Mine (Source: Oryx Mine, Rock Mechanics Department).

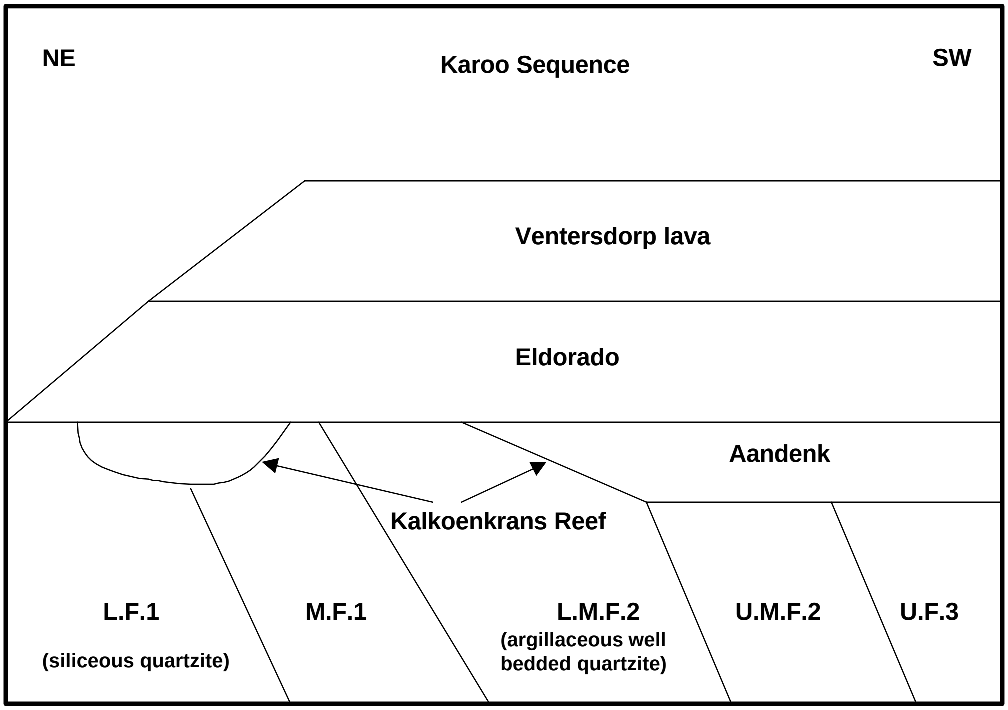

Figure 6.9. Stratigraphic relationship of various lithologies on Oryx mine as viewed from NE to SW (Source: Oryx Mine, Rock Mechanics Department).

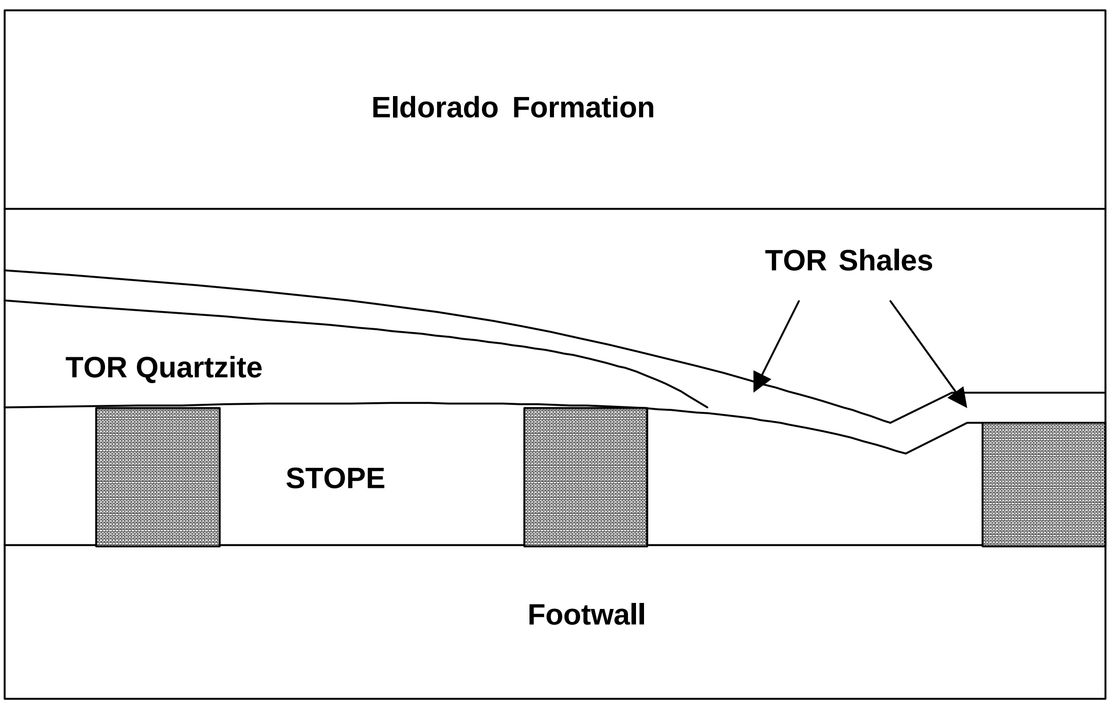

Figure 6.10. Typical hangingwall conditions in the TOR quartzites and shales (Source: Oryx Mine, Rock Mechanics Department).

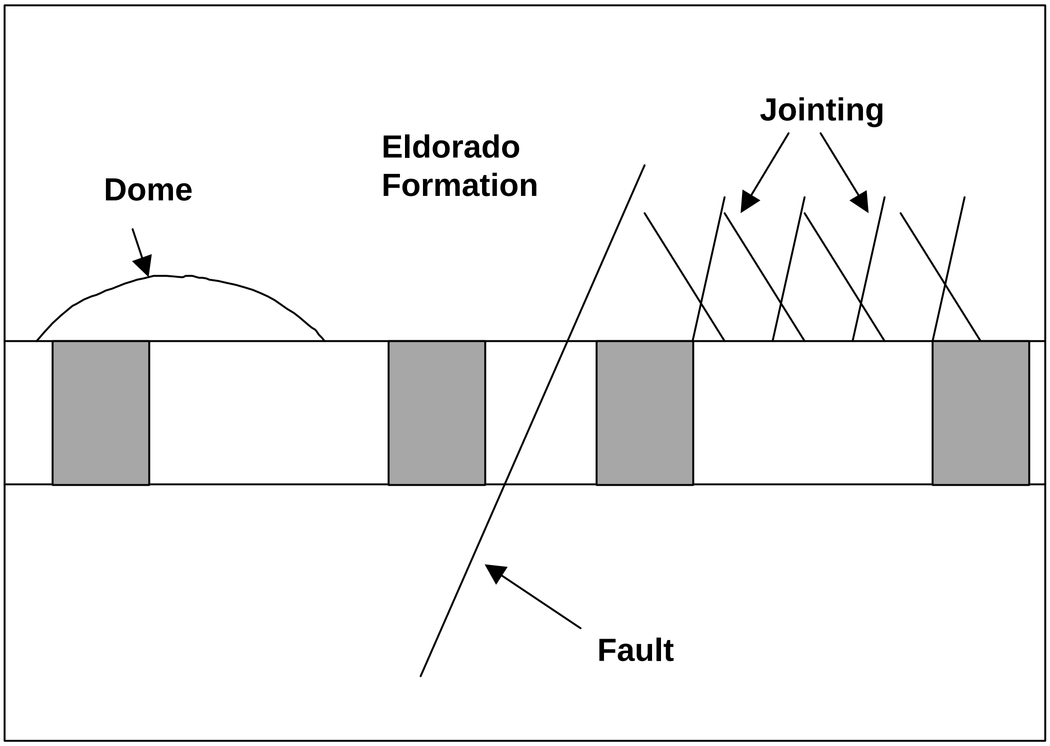

Figure 6.11. Typical hangingwall conditions in the Eldorado Formation (Source: Oryx Mine, Rock Mechanics Department).

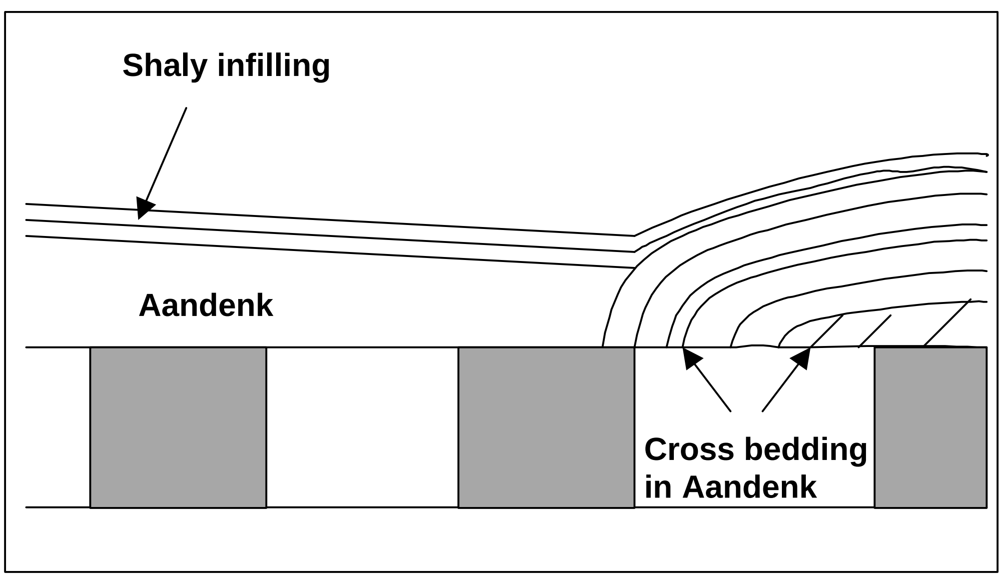

Figure 6.12. Typical hangingwall conditions in the Aandenk formations (Source: Oryx Mine, Rock Mechanics Department).

The footwall lithologies are the LMF2, UMF2 and the MF1. The LMF2 occurs in the western area of the mine. It is a siliceous fine to medium grained quartzite with scattered shale fragments. Argillite partings occur along the bedding planes. The UMF2 occurs in the central areas of the mine. It is a soft, argillaceous quartzite with shale fragments and well defined bedding planes. The MF1 occurs in the eastern, deepest portion of the mine.

Major faults are N/S trending and normal faulting, associated with thrust faulting, is present in the extreme west. Dykes generally trend E/W and usually have throws of up to 15 m.

<u>Description of the sites visited</u>

The sites visited were located at a depth of about 1950 m below the collar elevation of the shaft. Observations were made in 19N10 and 20N10 stopes. The Kalkoenkrans Reef is mined. The stoping width and the reef orientation vary widely. The area is geologically complex. The entire mine is divided into blocks with the boundaries defined by major geological structures. Discussions with the Rock Engineering Officer and the Chief Geologist revealed that the faulting in the area does not really pose any serious hangingwall instability. The blocks of ground are designated as Northern, A, B, and C from the western ends of the mine towards Beatrix Mine on the eastern edge. The 19N10 and 20N10 areas (see Table 6.3) fall within the northern block.

<u>19N10 Stope</u>

The 19N10 stope delineates a geotechnical area that is characterised by significant changes in the orebody geometry, especially its orientation, stratigraphy and discontinuities. This stope is characterised by an easterly dipping reef being steeply dragged up due to faulting, resulting in a synform. The exposed stratigraphy includes the LMF2 footwall quartzite, Kalkoenkrans Reef, TOR quartzite, TOR shale and Aandenk Quartzite (ADQ). Average reef thickness is 1,65 m and a higher thickness is present in the hinge of the synform. The axis of the synform strikes N/S. Dilation in the hinge zone of the synform along bedding and parting planes was observed. TOR quartzite is 1,4 m thick, and falls out exposing the TOR Shale. The TOR Shale is 0,9 m thick and well laminated. The contact between the quartzite and shale is polished and contains up to 3 cm of vein quartz and fault gouge. Fallout of the shale to expose the ADQ is also common. Channels of ADQ erode down to the Kalkoenkrans Reef, completely eliminating the shale. These channels trend N/S, are up to 2 m thick and extend along strike for up to 7 m. The hangingwall condition is exacerbated by the variation in thicknesses of the hangingwall lithologies. The mine personnel are compiling plans that delineate areas of equal beam thicknesses.

Initial breast mining was applied in areas where the dip averages between 20°–30°. The integrity of the immediate hangingwall was maintained until mining progressed to an area where the reef is overturned. The reef in this area is almost vertical and the stoping width increases significantly. The tectonic activity that might have caused the overturning of the reef also resulted in folding and altering the layering sequence of the strata. Due to the very low width to height ratio of the timber packs used as support in these stopes, buckling of some of the support units is observed. Mining has since been stopped and exploitation of the steeply dipping reef will be carried out by other means. The orientation and composition of the reef, coupled with the expected self-supporting nature of the surrounding strata, make the shrinkage-mining method potentially a better choice.

<u>20 N10 Stope</u>

Observations made in the 20N10 stope affirm the need to carefully analyse the combined influence of geological discontinuities, changes in reef orientation and the stratigraphy on the rock mass response. A combination of doming, rolling of the reef, complex stratigraphy and faulting constitute a complex geological setting in this area. The TOR quartzite is approximately 1,8 m thick, and TOR shale is represented by a sheared 5 cm thick zone.

Table 6.3. Summary of underground observations at Oryx Gold Mine and implications for geotechnical area definition.

Geotechnical parameters		**Site visited**		**Remarks**	**Implication for geotechnical area definition**
		19N10	20N10		
Depth (m)		1950	1950	North south trending ADQ channels may erode down to the reef as seen in 19N10. In 20N10, the VS5 erodes into the underlying lithologies. The variations in channel widths at the two sites lead to the adoption of different mining strategies. 20N11 which dips at 70 degrees to the west is mining the steeper limb of the synform. Although the hangingwall and footwall lithologies are similar to those in 19N10 and 20N10, the ground conditions were much better.	Regionally, factors such as depth, reef dip, and the stratigraphy would classify 19N10 and 20N10 under the same geotechnical area. Local variations in the hangingwall and footwall lithologies, and reef thickness would result in classifying these stopes as different ground control domains. Even though the stratigraphy and depth are similar to 19N10 and 20N10, the orientation of the reef in 20N11 would result in the definition of another regional geotechnical area.
Stratigraphy	Hangingwall	TOR quartzites, 1.4 m thick TOR shale, 0.9 m thick Aandenk quartzites (ADQ)	TOR quartzites, 1.8 m thick TOR shale, 0.5 m thick Aandenk quartzites (ADQ) is 1 m thick VS5 of the Eldorado formation		
	Footwall	LMF2 quartzites	LMF2 quartzites		
Orebody	Reef	Kalkoenkrans	Kalkoenkrans		
	Reef thickness (m)	Average thickness 1.6 m	Channel widths in excess of 2 m.		
	Dip	Average between 20 and 30 degrees. Western boundary of the stope lies along the hinge of the synform. Further west the reef steepens and overturns.	Average reef dip 15 degrees.		
Discontinuities	Hangingwall	Well defined partings between the TOR quartzite and the TOR shale, and between the shale and the ADQ.	Sheared parting along TOR shale and a well-defined parting at the base of VS5. Strike fault dipping at 25 degrees towards down-dip direction.		
	Footwall				

The ADQ is approximately 1 m thick. The stope is cut by numerous N/S trending normal faults with throws of less than 2 m.

A fall of ground had occurred along strike. The rock fall was bounded on the downdip side by a shallow dipping normal fault. The base of the VS5 was exposed. The size of the fall of ground was approximately 4 m x 7,5 m x 2,8 m. The hangingwall in the remaining stope was stable. The hangingwall becomes unstable when a critical span is exceeded. Information gathered was inadequate, and the critical span was not quantified. Fall out thicknesses varied widely. Double cutting is a mining strategy that has proved to be fairly safe in the area where the average reef dip is 15°.

20 N11 Stope

The down dip side of the overturned reef is exploited in 20N11 stope. A down dip mining configuration has been adopted and the excavation dips at about 60° to 75°. This area shows how one parameter could be used to delineate a geotechnical area, i.e. reef dip.

This stope is characterised by a steep dip of 70° to the west. Mining occurs at an apparent dip of 45° to strike in a southerly direction. Stope conditions are good and no faulting is present. Mining induced fracturing is parallel to the face and is steeply dipping towards the face. The footwall is LMF2 quartzites and the hangingwall is TOR quartzites.

Implications for defining geotechnical areas

The different structural features associated with the various hangingwall types and characteristics pose varying degrees of strata control problems. The problems observed and discussed in the preceding paragraphs give an indication of the degree to which complex geological settings dictate the rock mass behaviour. These areas can thus be seen as different geotechnical demarcations that need unique rock engineering strategies to exploit the reef safely and economically.

6.2.4 Kloof Gold Mine

During the GAP 102 SIMRAC Project, geotechnical areas were delineated on the basis of different combinations of hangingwall and footwall rock types. One of those areas was identified at Kloof Gold Mine where soft lava overlies and Elsburg quartzite underlies the Ventersdorp Contact Reef. Although the results obtained at the time confirmed the expected differences in rock mass behaviour, later observations revealed that other factors, if present, influence the general rock mass response. In addition to the observations that were made during the GAP 102 SIMRAC Project, underground visits to other areas on Kloof Gold Mine emphasise the importance of other parameters that might also influence the rock mass behaviour. These observations have been explained and compared below with information gathered in previous studies in areas that were demarcated on the basis of similar hangingwall and footwall rock types.

Kloof gold mine, Main Shaft, 32/34 longwall

The panels visited are located on 32/34 longwall to the northeast of Main Shaft at a depth of approximately 2700 m below surface (see Table 6.4). The Running dyke and the Spotted Dick dyke (both with a northerly trend) are situated to the west of the site. Extensive mining has occurred to the north and west of the site. The footwall is comprised of Elsburg quartzite. This is a light grey, very coarse-grained quartzite with rare khaki, argillaceous specks. Bedding is

15-40 cm apart and is frequently filled with white argillaceous material. The VCR comprises a very large pebble conglomerate with localised bifurcations and quartzites above and below the reef. The hangingwall is Westonaria Formation or "soft lava". The orebody strikes NE - SW and dips 30° to the SE. The observations made in the visited panels are discussed below.

2N Panel

This breast panel was temporarily stopped due to poor hangingwall conditions. The face length is approximately 6 m and the stope was supported by prestressed packs. The panel above was mined out in a southerly direction. The panel below was being mined northwards. The hangingwall is intensely fractured resulting in small fallout blocks, 5-30 cm in size. Fallout is occurring up to 2 m into the hangingwall.

3N Panel

Panel 3N is situated immediately below panel 2N and was mined towards the north. Face length is approximately 15 m and the dip is 31°. Support was provided by prestressed composite packs. The VCR is overlain by a 15 cm thick quartzite which is in turn overlain by a 4 cm thick bedding parallel fault filled with vein quartz and fault gouge. Very poor hangingwall conditions exist.

The hangingwall is intensely fractured resulting in a factory roof type geometry. Mining induced stress fracturing is face parallel and fracture spacing varies from 10-60 cm. Two dominant fracture sets occur, north dipping (towards the face) at 55°-70° and south dipping (towards the back area) at 50°-70° (see Appendix F, Fig. F.13). North dipping fractures are younger as they truncate the south dipping fractures. Fallout may occur up to 2,5 m into the hangingwall and are caused by the unfavourable intersection of mining induced stress fractures and joints (see Appendix F, Fig. F.14).

4S and 5S Panels

Discussions with the Sectional Rock Mechanics Officer revealed that there is a weak layer sandwiched between a soft lava hangingwall and the VCR (see Appendix F, Fig. F.15). The layer includes a black, medium grained, well bedded, immature quartzite up to 60 cm thick above the VCR (considered to be part of the VCR package). This material is called tuffaceous quartzite. The quartzite is interpreted to represent infilling of paleochannels, hence the variation in its thickness and occurrence. In some places, it stretches over several metres into the hangingwall. Bedding parallel faulting characterised by quartz vein and fault gouge occurs at the VCR-quartzite and the quartzite - lava contacts. The bedding parallel fault between the VCR-quartzite contact is seen to ramp up locally to the next contact. Previous tectonic activity has resulted in numerous polished tectonic surfaces in the quartzite providing suitable sites for dilation.

Jointing occurs as regular, north trending, almost vertical, vein quartz filled features. They are spaced 50 cm-150 cm apart, and are displaced by the VCR-quartzite bedding parallel fault. Rare northwest trending joints steeply dipping to the NE do also occur. Mining induced fracturing is more visible and pronounced in the lava than in the quartzite. Fracturing is face parallel and steeply dipping towards the face. The presence of the polished tectonic planes and bedding parallel faulting combined with the jointing and fracturing often results in the quartzite falling out.

The complex stratigraphy of the hangingwall, coupled with structural features like joints and dykes, poses an extremely difficult strata control problem in the area. Several attempts made to exploit the reef safely have resulted in a complex mining layout due to various combinations of breast and updip mining configurations while leaving crush pillars in some places. Two main problems are encountered when one of two activities are not performed. Firstly, the face must be blasted regularly. Irregular blasting operations usually result in an extensive area of fall.

Secondly, up-dip mining which is often used to re-establish collapsed panels, tends to pose serious strata control problems in the area. Backfilling is known to offer the best support strategy in the area. These observations were made in numbers 4 and 5 panels when the mining configurations were breast and updip respectively.

Kloof gold mine, 3 Shaft, 34/27 longwall

During the period 1994–1996, an intensive, GAP 102 SIMRAC Project, aimed at improving the understanding of the rock mass surrounding the VCR, was undertaken. Various strategies were adopted to accomplish this mission. This included the delineation of geotechnical areas on some gold mines, one of which was at Kloof Gold Mine, 3 Shaft, 34/27 longwall. A geotechnical area at the time referred to an area where there was a different assemblage of hangingwall and footwall rock types. At the site on Kloof Gold Mine, the hangingwall rock type was identified as soft lava and was located at about 2750 m below surface. The footwall material underlying the VCR is Elsburg quartzite (see Table 6.4). The rock mass behaviour was studied on both sides of the longwall.

Observations made during this study revealed that intense thrust faulting is associated with the soft lava (see Appendix F, Fig. F.16). Generally these are low dipping faults but occasionally, steeper ones are encountered. On the northern side of the longwall, falls of ground usually occurred as a result of unfavourable intersection of mining induced stress fractures, joints and thrust faults. Such falls exposed the slickensided surfaces of the thrust faults that tend to step up into the hangingwall. The geological features, particularly the thrust faults observed in the north panels, lacked exposure in the southern stopes. However, there was some evidence that they do exist. The few that were exposed and could easily be identified had similar features (i.e. striations, slickensides and calcite or quartz veins) to those noted in the north panels. The tuffaceous quartzite is absent in this area and a well-defined bedded quartzite layer that is quite common between the VCR and the soft lava is present. The average spacing between successive beds is 130 mm (see Appendix F, Fig. F.17).

Implications for defining geotechnical areas

The previous strategy adopted in delineating geotechnical areas would have geotechnically considered the above areas as the same due to the soft lava-quartzite rock assemblage in both cases. However, the observations made in both cases have shown a sharp contrast in rock mass behaviour. The quartzite layer between the VCR and the soft lava occurs along both longwalls but the degree of competence is different. So, as to reduce dilution when mining the VCR, it is necessary to undercut the quartzite. Along the 32/34 longwall, especially in the south stopes, it is intensely fractured and becomes highly unstable. The slightest disturbance, which normally results from seismic activities in that area, dislodges the tuffaceous quartzite from the hangingwall resulting in a fall out that is intensely fragmented. Apart from backfill, no other support type offers the needed performance in that area. In contrast, the bedded quartzite layer in the hangingwall, as observed along the 34/27 longwall, is much more competent. Fallout thicknesses, as and when they occur, are much greater but the integrity of the hangingwall is usually maintained by the use of composite timber packs.

On the basis of this methodology, the difference in the strength and composition of the immediate hangingwall material, and hence the different responses of rock mass, have necessitated different support strategies to be adopted. These observations depict different geotechnical conditions and emphasise the need for the strength and composition of the tuffaceous quartzite, quartzite and soft lava to be considered in the defining different ground control districts. Also, the thickness of the bedded strata and the intensity of mining induced stress fractures need to be considered in delineating such areas as is the case with the methodology.

Table 6.4. Summary of underground observations at Kloof Gold Mine and implications for geotechnical area definition.

Geotechnical parameters		**Site visited**		**Remarks**	**Implication for geotechnical area definition**
		1 Shaft 32/34 longwall (2N, 3N, 4S and 5S)	3 Shaft 34/27 longwall (North and South)		
Depth (m)		2700	2750	Different types of lithologies were observed between the VCR and the soft lava. These are tuffaceous quartzite and quartzite. The thicknesses of the tuffaceous quartzite and the quartzite vary locally. In places where the tuffaceous quartzite was observed, different mining strategies were adopted.	These panels fall under one regional geotechnical area on the basis of depth, hangingwall and footwall lithologies, reef thickness and orientation. The differences in the immediate hangingwall lithologies and the change in stoping widths would require the definition of different ground control domains.
Stratigraphy	Hangingwall	Soft lava	Soft lava		
	Footwall	Elsburg quartzite	Elsburg quartzite		
Orebody	Reef type	VCR	VCR		
	Reef thickness (m)	Average thickness is about 1 m	Average thickness is about 1 m		
	Dip	30 degrees towards south east	33 degrees towards south east		
Discontinuities	Hangingwall	Immature quartzites (tuffaceous) of variable thickness in the south panels. Well defined tectonised partings at the VCR quartzite contact and the quartzite lava contact. Mining induced fractures are more prevalent in the north panels than the south panels.	Bedded quartzite in the south panels. Shallow to steep dipping mining induced fractures in both north and south panels. Thrust faults exposed in north panels. Fractures dip towards the stope face in south panels. They dip away from the stope face in the north panels. The soft lava is jointed and had similar orientation in both north and south panels.		
	Footwall	15 – 40 cm spaced bedding planes	Steep mining induced fractures in the footwall.		

6.3. Discussion and Conclusions

The above observations have demonstrated the variability of the geological nature of the rock mass on the gold and platinum mines. The visits have also given an insight as to the relevance of the critical geotechnical parameters and the general applicability of the methodology as a whole. Stratigraphic profiles, stress regime, orebody geometry and particularly geological discontinuities were confirmed to be the crucial parameters in delineating geotechnical areas on the platinum mines. In addition to these, the propensity for seismicity is critical in the deeper gold mines. Furthermore, the visits to the different geological environments confirm that there could be a high variability of rock mass behaviour even in neighbouring panels. Thus, the dynamic nature of the mining operation usually exposes unexpected geological disturbances several metres ahead of the stope face. Most of these occurrences usually necessitate major or minor changes in the adopted support strategies.

In the light of these observations, the dynamism introduced into the methodology for defining geotechnical areas is justified. It is also gratifying that all the significant parameters, which are used to distinguish geotechnical areas, and were identified during the visits, appear in the checklist. There is currently no single rock mass classification scheme that can effectively account for all the diverse rock mass characteristics identified on these mines. However, some of the practitioners, especially on the platinum mines, claim that modified versions of existing rock mass classification schemes work effectively on their mines. The different systems being applied on these mines also attest to the fact that a single classification scheme may not be applicable in all cases. However, there is a potential for developing such a system in future if more experience is gained in delineating geotechnical areas which could possibly refine the list of potential important parameters. Before then the guidelines outlined in the methodology, coupled with the experience of rock mechanics officials on the mines, can be used to delineate geotechnical areas and associated geotechnical districts.

Moreover, it is concluded that the newly established methodology for delineating geotechnical areas caters for a wide range of mining conditions. It has successfully been used to relate different support and mining strategies to varying ground conditions. This has been done for both Witwatersrand gold and Bushveld platinum environments.

7 Summary and Conclusions

The objectives of this investigation were two-fold. Primarily, the objective was to identify critical geotechnical parameters and use these to develop a methodology for defining geotechnical areas across the South African gold and platinum stoping horizons. Secondly, it was necessary to establish whether a suitable rock mass classification scheme could be used to effect the definition of geotechnical areas. The most commonly applied schemes were therefore reviewed in detail. The expected enabling outputs and the major findings are summarised Table 7.1. The major findings are, however, discussed in more detail, in the following.

In identifying the critical parameters that are essential for delineating geotechnical areas, relevant literature was reviewed, a questionnaire was developed, underground visits and interaction with mine personnel were conducted, and various workshops were held. During the literature review, a back analysis was carried out to identify the critical parameters that contribute to strata control problems in gold and platinum stopes. Some of the typical problems that are noted include stope back break, foundation failure and hangingwall punching, excessive closure rates in the gold mines, seismicity, and varying degrees of rock falls. The back analysis identified many parameters which influence rock mass behaviour and hence geotechnical areas. The practical experience of the mine personnel was therefore sought, so as to narrow down the huge diversity of parameters. Consequently, a questionnaire was developed, which was completed with rock mechanics practitioners on various mines. Results from the questionnaire served two purposes. Firstly, it revealed similarity in some of the parameters contributing to typical strata control problems and indicated varied opinions on the meaning of a geotechnical area. This was later confirmed by underground visits. Secondly, the need for the methodology to be very simple, easy to use and a preference to guidelines rather than a prescriptive approach was emphasised. Combining the information obtained from the questionnaire, the interaction with mine personnel, and the underground visits, the parameters were grouped into five broad categories, namely: orebody information, discontinuities and their characteristics, stratigraphy, the stress environment, and production parameters. Once the list of critical parameters was finalised, efforts were concentrated on making use of this information to establish the methodology for defining geotechnical areas.

Firstly, the meaning of a geotechnical area was standardised so as to provide a common basis of understanding across the mining industry. A geotechnical area is defined as an area that exhibits a particular rock mass response and associated hazards to mining, where the same rock engineering strategies may be applied. Thus a geotechnical area classification groups areas of similar rock mass behaviour and their adopted rock engineering strategies into the same class. However, it became very clear that it was necessary to subdivide a mine based on inherent geological factors for two substantially different purposes. The first is to enable the selection of major, long term strategies for the mine, such as mining methods and layouts, regional support and other seismicity or rockburst control measures appropriate for different regional differences in the geological environment. This subdivision needs to be done early in the life of a mine, preferably even during feasibility studies. The process is quite straightforward, based on variations in a few fundamental parameters. This subdivision will seldom result in more than about three specific regions on the average mine but could be more where the geology is complex. The boundaries of these regions need to be reviewed or new subdivisions defined only when unexpected changes in the geological structure or stress regime are encountered. The second purpose of subdividing a mine, on principally geological differences, is to enable appropriate strata control measures and designs of stope support to be put in place that will maximise safety and profitability. This is a much more complex procedure involving the consideration of many parameters but the selection of only a few which are relevant to a particular mine. Critical ranges in values for each of these parameters need to be decided upon and then combinations of these for the parameter need to be identified which result in significantly different rock mass responses to mining. Areas on the mine which have these different combinations of parameters would thus comprise different ground control districts.

Table 7.1. The expected enabling outputs and the major findings.

Expected enabling output	Reference chapter	Major findings and conclusions
Survey and review of rock mass classification systems (literature, mine visits) against design requirements	Chapter two	Existing rock mass classification systems were developed for other purposes rather than studying the behaviour of rock masses surrounding underground stopes. None of the existing schemes can fully account for the diverse parameters and associated mechanisms that control the stability of these excavations. Since these are essential issues that must be considered in the design of stopes and appropriate support systems, none of the existing schemes can fully meet these requirements.
Analysis of classification systems against available case studies for South African gold and platinum mines.	Chapter two	Two of the most widely used classification systems namely the CSIR Rock Mass Rating (RMR) and the Tunnelling Quality Index (Q), and various modifications of these two have been applied in several mines in the Bushveld Igneous Complex. The studies revealed discrepancies in the weighting schemes and also affirmed the need for other parameters to be considered. Furthermore it reiterated the fact that the existing classification schemes can not fully account for the complex nature of the rock mass surrounding stopes.
Review of critical rock mass parameters and behavioural characteristics to be determined for definition of geotechnical areas for different rock mechanics strategies.	Chapter three	By means of a questionnaire, back-analysing of typical strata control problems and interaction with mine personnel, a detailed listing of the rock mass parameters that are believed to impact on rock mass behaviour have been compiled.
Design of a rock mass classification system specifically for the definition of geotechnical areas and associated behaviour, applicable to stoping environments.	Chapter four	Various discussions with mine personnel, review of existing classification systems and case studies, deductions made from several workshops, and the enormous parameters that influence rock mass behaviour do not make it feasible to formulate a rock mass classification that is practical for defining geotechnical areas. General guidelines to provide a logical assessment of the rock mass behaviour are rather recommended.
Define methodology for the determination of geotechnical areas based on classification system and rock mass response.	Chapter five	It is recommended that geotechnical areas be defined on regional and local scales. These have been referred to as geotechnical areas and ground control districts respectively. In general the methodology involves five steps: 1). Identification and quantification of critical geotechnical parameters. A checklist has been established to facilitate the process. 2). Classification of critical geotechnical parameters. 3). Establishment of probable rock mass behaviour. 4). Demarcation of geotechnical areas on the basis of similar rock mass response. 5). Implementation of appropriate rock engineering strategies. Details on each of these steps have been presented in the report. By applying these steps with differing degree of detail, it should be possible to define the geotechnical areas and associated ground control districts.
Implement and monitor verification sites.	Chapter six	Monitoring sites were implemented at Impala and Eastern Platinum Mines to verify the critical geotechnical parameters on these mines. Underground visits were also undertaken to many gold mines in the Free State and West Rand for the same purpose. Deductions made from the results obtained indicate the need to consider variations in stratigraphic profiles, characteristics of geological discontinuities, orebody geometry, and the stress regime in defining geotechnical areas and associated ground control districts on the platinum mines. In addition to these the propensity for seismicity on the gold mines are critical issues that must be considered.
Review and evaluation of application and verification sites of geotechnical area methodology.	Chapter six	It is gratifying that all the significant parameters that are used to distinguish geotechnical areas were identified during the visits in the checklist. Thus the established methodology caters for a wide range of mining conditions.

Five basic steps that form the core of the methodology were followed:

- -Identification and quantification of critical geotechnical parameters
- -Classification of critical geotechnical parameters
- -Establishment of probable areas of similar rock mass behaviour
- -Demarcation of geotechnical areas on the basis of similar rock mass response
- -Implementation of appropriate rock engineering strategies

When applying the methodology, the first step is the identification of critical geotechnical parameters. For this purpose a checklist is provided. Although in some cases it might become necessary to consider additional parameters, the list provided incorporates diverse views of practitioners on the gold and platinum mines and so contains almost all the critical parameters. Hence, depending on the ground conditions prevailing on a mine, the critical geotechnical parameters can be identified from this list. It is important to re-emphasise that in any particular instance a host of factors could be identified but only the critical parameters that control the rock mass behaviour need to be considered. Once these factors are identified, they need to be quantified. Suggestions on techniques available for specific purposes have been outlined in Appendix C.

After the identification and quantification of the relevant geotechnical parameters, they must be referred to a suitable classification scheme that will group the parameters into different categories. A detailed review of the most commonly used rock mass classification schemes as applied on the mines was undertaken. This study revealed that none of the available schemes is applicable for the diversity of ground conditions encountered in the stoping horizons of gold and platinum mines. Various reasons account for their inapplicability and have been described in the report.

The views of practitioners were that systems which others claim to work well would not yield reliable results on their mines. Most of the mines in the Bushveld have modified one or other of the existing schemes and attest to their effectiveness on their mines. Thus every mine has tailored its classification scheme to suit its own conditions.

It is unlikely one classification scheme will therefore be applicable in all the mines. Putting these findings together, a generalised scheme that will highlight significant changes in the magnitude of these parameters was considered to be more appropriate. This resulted in adopting, modifying and categorising some of the useful aspects of the existing schemes. These categories are only presented as guidelines and could form the basis for developing a new rock mass classification scheme for the stoping horizon. Furthermore, the production parameters, which are non-existent in the existing schemes, are worth considering. Hence, it has been proposed that all these parameters must be classified into fixed and variable parameters. The fixed parameters are geological and the variable parameters include those that are induced on the rock mass by the mining activities. The relevance for grouping these parameters into fixed and variable parameters lies in the fact that the latter provide the key for improving or aggravating the ground conditions.

Having identified and quantified the critical parameters, these must be combined to deduce and predict the probable rock mass behaviour in virgin areas. To facilitate the process, typical strata control problems have been reviewed and the respective influencing parameters and mechanisms are summarised. Some of these results have been presented in a tabular form in the text, as a guideline. It must be re-emphasised that this is only seen as a guideline and not a rule. Although several parameters might be contributing to the mining problems, only the key factors have been highlighted in that table. Thus there is some degree of flexibility, allowing the practitioners to apply their experience and engineering judgement when the need arises. It is essential to ensure that the adopted procedure to deduce the probable rock mass behaviour is logical and scientific. The approach adopted in developing Table 5.14 includes back analysing the factors that could influence those occurrences. Even in some cases the probable rock mass behaviour could be predicted from those experienced in areas of similar geological setting.

Once areas expected to exhibit similar behaviour are established, these could be grouped into the same class and their locations demarcated on mine plans by using some form of identifiers. Colour-coded contours on mine plans to differentiate between areas of different rock mass behaviour could be utilised. In some cases, the mechanisms responsible for the identified behaviours will have to be investigated by using numerical modelling techniques. This, when applied correctly, will also assess whether the difference in behaviour is significant enough to warrant a different class assignment. Areas that are expected to exhibit a similar response to mining can then be assigned a common set of appropriate rock engineering strategies.

After establishing the methodology, further discussions and workshops were held with the rock mechanics practitioners on the various mines. Although most of them accepted the methodology as a simple and useful guideline, others raised concern about when and how should geotechnical areas be defined. It is proposed that geotechnical areas be defined on two levels. Firstly, such areas should be delineated before the onset of mining operations, thus on a regional level. The information needed to delineate such areas and their sources have been discussed in the text. The ultimate aim of defining geotechnical areas on this scale is to facilitate the making of strategic decisions. These will include the choice of a mining method, mining layout, and regional support strategies. When mining starts and more information about the characteristics of the rock mass becomes available, these will be reviewed. The local rock mass behaviour, such as intensity of fracturing, closure rates, and fall of ground, will be known. These will be used to implement tactical solutions, which include support spacing and beam thickness to be supported. Areas that behave similarly on a local scale are referred to as ground control districts. These are subsets of the larger geotechnical areas. In order to clarify the involved procedure, a flow chart was developed (see Fig 5.23).

So as to verify the relevance of the geotechnical parameters provided in the checklist, and the general applicability of the methodology, underground monitoring sites and visits were undertaken on the platinum and gold mines. Results obtained from the monitoring work and observations made during the visits gave a strong indication that all the parameters included in the checklist are relevant for the different conditions. They further highlighted the unique strata control problems on the platinum and deeper gold mines. It became obvious that one classification scheme can not fully account for the diverse mechanisms responsible for the rock mass behaviour on these mines. However, the flexibility incorporated into the methodology and its logical expression enables it to account for these issues.

In conclusion, the critical geotechnical parameters required to establish the methodology have been identified. In the methodology, it is suggested that geotechnical areas be delineated prior to the onset of mining. The input parameters to be considered must include characteristics of major geological structures, depth, stratigraphic profiles, and regional hydrology. The critical parameters that describe these factors can be identified from various sources such as borehole core logging and historic data from areas of similar geological setting. The framework provided in Table 5.14 acts as a guideline in deducing the expected rock mass response on a regional scale. Regional support strategies can be deduced from the current edition of the Industry Guide to Methods of Ameliorating the Hazards of Rockfalls and Rockbursts (1988). As mining progresses and detailed information about the rock mass characteristics becomes available, the guidelines provided in Section 5.2 can be followed to identify GCD.

8 Recommendations for future work

At the beginning of the project, confusion existed regarding the definition of a geotechnical area. During this study, a methodology has been established and successfully tested. It is able to geotechnically classify a great variety of Witwatersrand and Bushveld mining environments, also establishing the associated rock mass response to mining. This study, for the first time, includes both fixed (geological) and variable (mining) parameters when defining geotechnical environments. However, it is proposed that the following activities could enhance the geotechnical classification of mining environments.

a) The newly defined methodology has been tested employing selected underground examples. The methodology could be handed to 'selected' practitioners for further mine testing. This will facilitate improvements and subsequent implementation of the methodology.

b) The methodology is currently geared towards both, Witwatersrand and Bushveld mining operations. Orebody-specific methodologies could be considered in the future.

c) Further to b): the methodology could be simplified when geared towards individual orebodies or regions. Underground studies during the course of this investigation have shown that the importance of parameters varies with orebody and region.

d) Parameters contained within the new methodology were not assigned weighting factors, due to the reasons listed under c). Future investigations could consider orebody and/or region specific parameters with weighting factors.

e) This investigation has highlighted the problems that are associated with a potential, universally applicable rock mass classification system. Steps b) to d) may define parts of the methodology that transforms the established methodology into a rock mass classification system that is applicable to individual, South African orebodies/regions for application in stopes.

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