The application of the Q-Tunnelling Quality Index to rock mass assessment at Impala Platinum Mine

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Synopsis

Publications on hard rock tunnel support design in the South African mining industry have originated primarily from the gold mines during the past 50 years. Little has been published to date on the platinum and chrome mines of the Bushveld Complex. This paper covers the search for a suitable design methodology for off-reef tunnels at Impala Platinum Mine, situated on the Western Lobe of the Bushveld Complex. The fall of ground accident statistics for off-reef excavations of the mine are presented and the available rock mass quality evaluation systems are reviewed. The Q-Tunnelling Quality Index (or Q-Index) is selected because it assesses the important driving factors behind falls of ground at Impala. Two off-reef excavations are evaluated using the Q-Index, and it is shown that minor modifications are required for implementation at Impala. It will take some time before support design throughout the mine is based on the outcome of proper geotechnical investigations based on the Q-Index. Widespread implementation of such support designs should help to solve the fall of ground problem in off-reef excavations while at the same time reducing support costs.

Introduction

The current mining industry regulations and guidelines for South Africa call for systematic underground support that is capable of resisting 95% of all potential falls of ground as determined by statistical analysis. Data obtained from all fall of ground injuries at Impala Platinum Mine for the past 12 years show that most falls occurred on the reef horizon, with a smaller proportion in off-reef excavations. A proper engineering approach to support design is already in place for stoping, but support in off-reef excavations is still designed using the 33 kN/m^2 -support resistance criterion without a prior geotechnical investigation. This practice suffers from the disadvantage that geotechnical conditions, which affect the potential for rockfalls, are ignored. Hence, some off-reef excavations are likely to be over-supported while others are under-supported.

Consultants to the mine suggested several rock mass classification systems to take account of geotechnical conditions in support design, but it soon became apparent that every available rock mass classification system has its origins in the application of qualitative means to quantify rock mass characteristics, generally in one number. This is often not specific enough, which could lead to erroneous evaluations, and reducing potential applicability on the mine. It became apparent during this period that in order to ensure safe and cost-effective support in off-reef excavations at Impala Platinum, a suitable rock mass classification system would have to be found.

All rock mass classification systems strive for simplicity while at the same time addressing the complexity and diversity of natural rock masses. It appears from this study and the experience of others, for example Jakubec and Laubscher[1], that rock mass classification schemes can give only guidelines, but that the geologist or engineer must interpret the finer details. Interpretations are often subjective, and their quality will depend on the engineer's 'local knowledge'. Here, 'local knowledge' is defined as that knowledge acquired by a person who has developed over time a familiarity, and therefore an intuitive feel for a situation. Thus, one engineer, who has used a rock mass classification system for many years in a particular mining environment may always reliably interpret the geotechnical data, correctly taking account the finer points which the system is not in itself, capable of identifying. Thus, rock mass classification systems generally need a further undefined element for correct application, and that is the 'local knowledge' of the engineer.

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This paper describes the approach used to select and implement the Q-Tunnelling Quality Index[2] (or Q-Index) in the off-reef excavation support design process at Impala Platinum. First, an accident analysis is presented in which the nature of the fall of ground problem in off-reef development is defined. Then, currently available rock mass evaluation systems are described to highlight the strengths and weaknesses of each. This forms the background to the choice of the Q-Index, and why it is suitable for Impala. To confirm its applicability, the Q-Index is applied to two tunnels in different geotechnical conditions. The first case study covers a tunnel, which has stood virtually unsupported in a very good quality rock mass for twenty years, while the second covers an incline shaft presently being developed in a fair to poor quality rock mass. The methodology developed to assess these tunnels is described and the support designs appropriate for these tunnels is presented.

The case studies show that the Q-Tunnelling Quality Index predicts larger stable unsupported spans than those observed by experience on the mine. Thus, the system needs some minor adjustment to fit with the 'local knowledge'. The results were used to adjust the relationship between the equivalent dimension D_e and the rock mass quality index Q (Barton *et al.*[2]). A simple and reliable rock mass evaluation system based on the Q-Tunnelling Quality Index incorporating local experience for off-reef development is now in place on Impala Platinum Mine.

Mine location and geological setting

Impala Platinum Mine is situated 23 km north of Rustenburg on the western edge of the western lobe of the Bushveld Complex. The mining area extends approximately 25 km from the south to the north (see Figure 1). The mine exploits the Merensky Reef and the UG2 Reef for platinum group metals and several other by-products.

The Bushveld Complex is a layered igneous intrusion consisting of mafic rocks such as chromitite, pyroxenite, norite, gabbro, gabbronorite, harzburgite, and anorthosite in the Lower, Critical, and Main zones[3]. The Upper Zone, which forms the cap of the layered intrusion, consists of gabbro, anorthosite, and gabbronorite towards its base, with diorites at the top. The various Bushveld Granites succeeded these mafic intrusions. There are seven overlapping mafic intrusions, which are shaped like inverted cones spreading out laterally toward the surface. Each can be likened to an inverted wine glass. The layers around the edge of each intrusion (equivalent to the wine glass base) dip towards the centre at an average of 9 to 10 degrees. The dip increases towards the centre where the vertical feeder is situated. The Bushveld Complex is large, stretching 370 km east–west across North West Province, northern Gauteng, and Mpumalanga.

The most important platinum group element carriers in the Bushveld Complex are the Merensky Reef and the UG2 Reef, which occur parallel to the major layering as distinct layers usually less than 2 metres thick. The Merensky Reef lies at the base of the Main Zone, and consists of pyroxenite and pegmatoid units, while the UG2 Chromitite Seam is located between 20 to 300 metres below the Merensky Reef in the Critical Zone. Both carry economic concentrations of the platinum group elements, namely platinum, palladium, ruthenium, rhodium, iridium, and osmium as well as the base metals nickel, copper and cobalt, and finally gold, in minor concentrations. At Impala, the vertical separation between the two reefs increases from 60 metres in the north to 130 metres in the south. The general strike of both orebodies is north-northwest to south-southeast, dipping gently towards the east at about 10 degrees. Both reefs outcrop at surface along the western boundary of the Impala lease[4]. The geological succession for the Impala Platinum Mine is summarized in Table I.

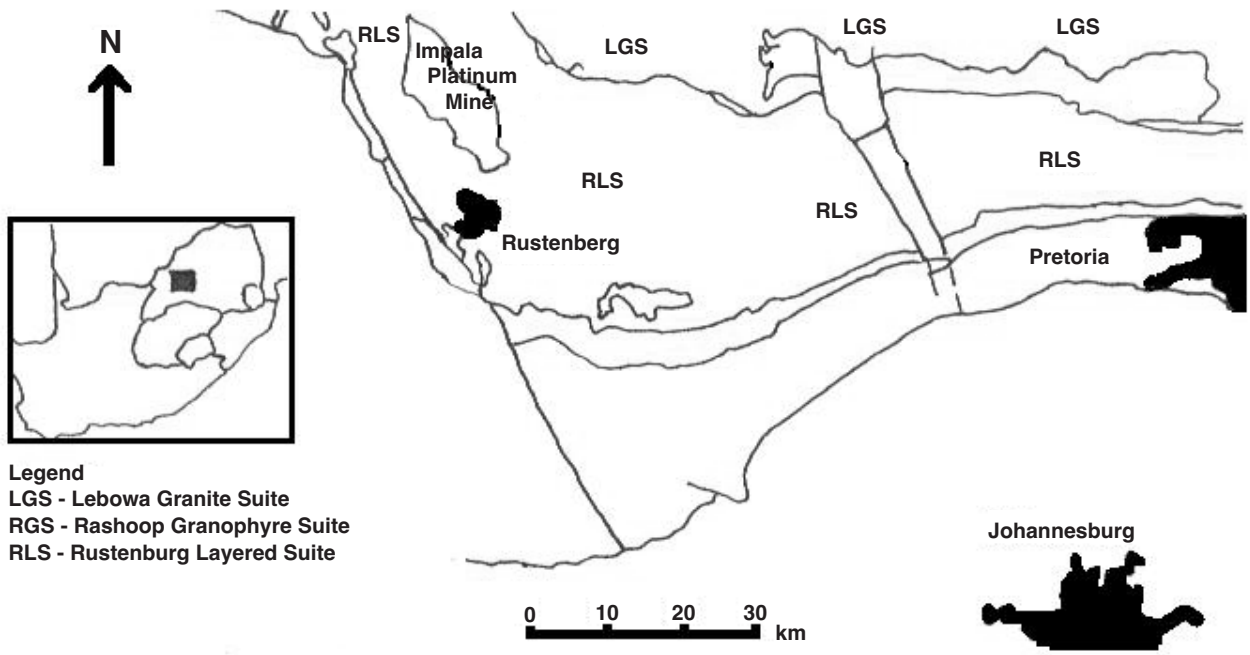

Figure 1—Location of Impala Platinum Mine

Table I

Generalized geological succession for Impala Platinum Mine

Average thickness (m)	Unit	Rock type
3–4	HW5	Mottled and spotted Anorthosite
3–6	HW4	Spotted Anorthosite
5–7	HW3	Mottled Anorthosite
1.5–3	HW2	Spotted Anorthositic Norite
2–6	HW1	Norite
2–3	Bastard Pyroxenite	Pyroxenite, Coarse Grained
2–3	M3	Mottled Anorthosite
3–7	M2	Spotted Anorthositic Norite
0.5	M1	Norite
1–1.5	Merensky Pyroxenite	Medium to Coarse grain Pyroxenite
0–2	Merensky Reef	Chromitite Layer – Pegmatoid
0.4	FW1	Spotted Anorthositic Norite
0.2	FW2	Cyclic Unit – Pyroxenite
3-5	FW3	Spotted Anorthositic Norite
0.1–0.3	FW4	Mottled Anorthosite
1–3	FW5	Spotted Anorthositic Norite
1–3	FW6	Cyclic Unit—Spotted and Mottled Anorthosite
1–3	FW7	Spotted Anorthositic Norite
0.8–1.2	FW8	Spotted Anorthosite
3–6	FW9	Mottled Anorthosite
3–5	FW10	Spotted Anorthositic Norite
12–15	FW11	Spotted Anorthosite
10–12	FW12	Mottled Anorthosite
5–7		UG2 Pyroxenite with Leader Chromitite Stringers
0,7	UG2 Reef	Chromitite
10–12	FW	UG2 Pegmatoid
5–7	FW13	Spotted Anorthositic Norite
6–8	FW14	Pyroxenite
1–3	UG1 (FW15)	Chromitite with anorthosite and pyroxenite lenses
>100	FW16	Anorthosite with chrome stringers

The problem

The off-reef development at Impala Mine is placed in the footwall of the Merensky and UG2 reefs, in rock masses that vary in quality from very poor to extremely good. Until recently, all support designs were based upon the 33 kN/m^2 resistance criterion that was applied universally without regard to the rock mass quality. This criterion had been determined from the 95th percentile on a cumulative fall of ground thickness curve based on fall of ground statistics on the mine.

Universal use of a single criterion such as this means that some excavations may be under-supported while others are over-supported. In addition to this, fall of ground accidents continue to occur in off-reef development. It is therefore clear that a more scientific approach based on well-established rock engineering principles is necessary. First, an analysis of fall of ground accidents at Impala is presented, followed by an assessment of different rock mass evaluation schemes. This forms the background to the choice of the Q-Index, which is then applied to mine conditions in an underground study described in a later section.

Fall of ground analysis

The fall of ground data used in the analysis covers a five-year period from 1992 to 1996, to ensure statistically meaningful results from a sufficiently large database. The data have been strictly limited to falls of ground in off-reef development to remove any bias that may arise if in-stope data were included. The following information was extracted from the accident reports for the analysis:

- reef type
- stope or off-reef development
- depth below surface
- distance from face
- excavation size
- origin of the fall of ground: face, hangingwall, sidewall, or footwall
- mechanism of fall of ground: buckling, shear, or dead weight
- size of fall of ground: small, medium, or large
- shape of fall block: dome, wedge, or scaling
- dimensions of fall of ground: height, width, length, area, volume, and weight
- rock type
- proximity of major geological features: faults, dykes, potholes or joints
- boundaries of the fall of ground: joints, faults, dykes, or partings between layers such as a chromitite seam

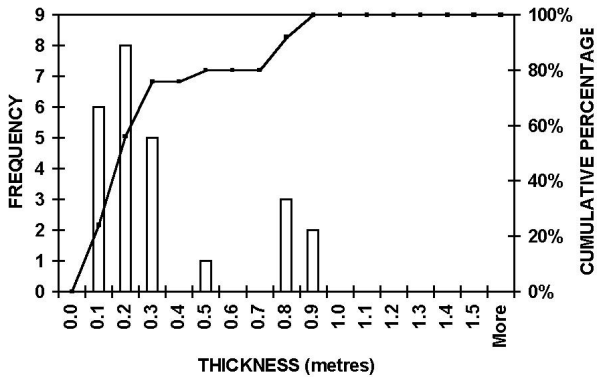

Figure 2—Cumulative frequency plot of maximum thickness of falls of ground

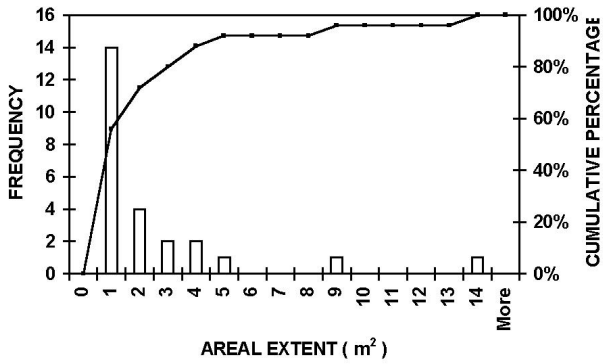

Figure 3—Cumulative frequency plot of areal extent of falls of ground

The database has been visualized by preparing a number of cumulative frequency plots and pie charts, which appear in Figures 2 to 5 and Figures 6 to 10 respectively. The 95th percentile dimensions, obtained from Figures 2 to 5 have been collected into Table II for easy reference.

The pie charts demonstrate aspects of falls of ground in off-reef tunnels e.g. location, shape, origin, rock type and boundaries that cannot be represented in cumulative frequency plots. These show that the majority of falls of ground are joint-bounded blocks originating in the hangingwall of the excavation. None of the statistics presented above suggests an insoluble problem. Instead, they suggest that appropriately designed local support would have been sufficient to prevent every occurrence listed.

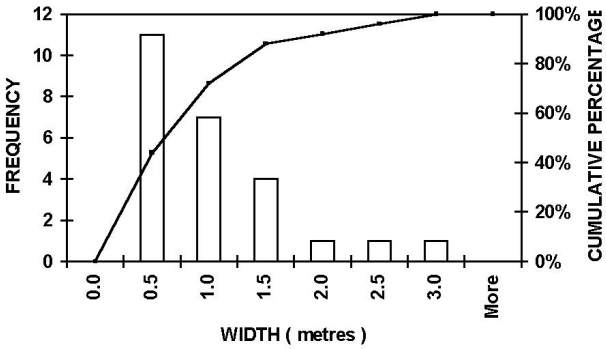

Figure 4—Cumulative frequency plot of width of falls of ground

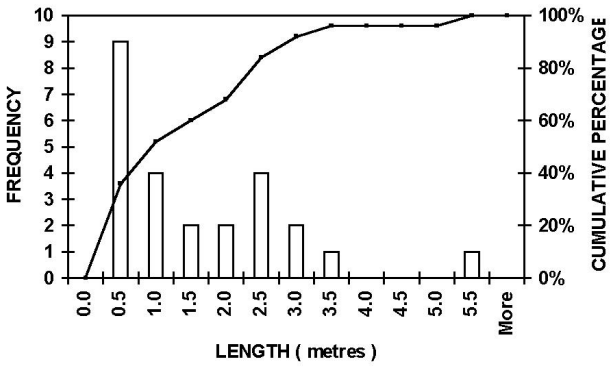

Figure 5—Cumulative frequency plot of length of falls of ground

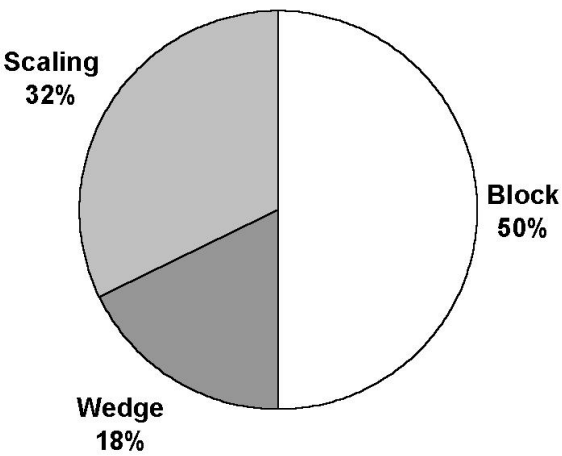

Figure 6—Shape of falls of ground

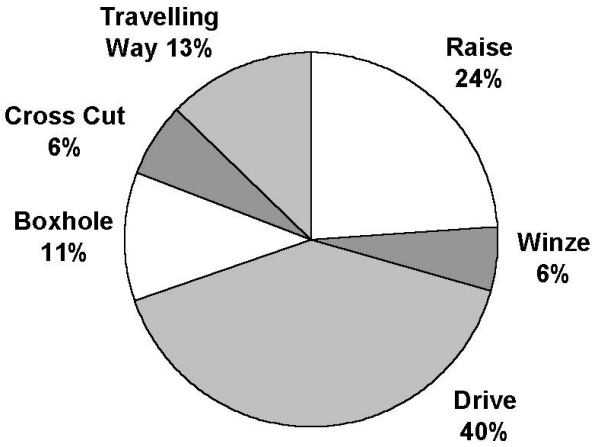

Figure 7—Location of falls of ground

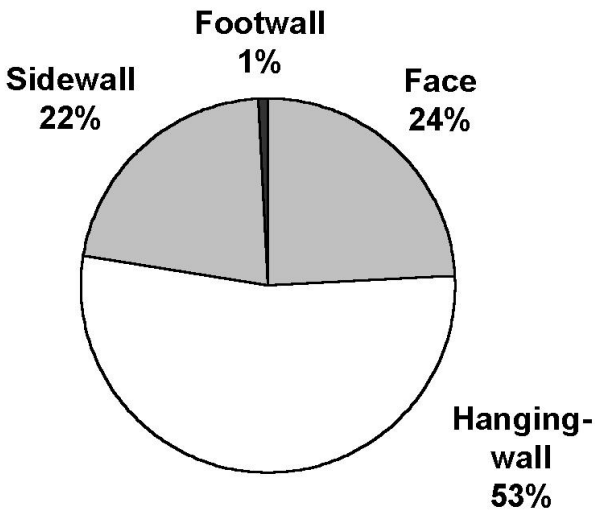

Figure 8—Origin of falls of ground

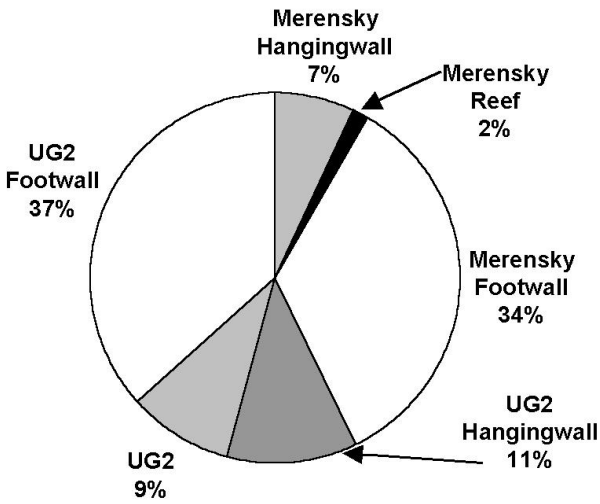

Figure 9—Rock type involved in falls of ground

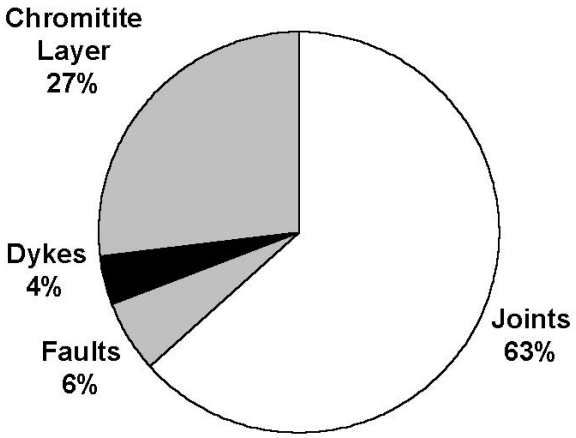

Figure 10—Boundaries of falls of ground

Table II

List of 95th percentile variables for falls of ground in off-reef tunnels

95th percentile variable	Dimension
Thickness (m)	0.9
Area (m^2)	9.0
Width (m)	2.5
Length (m)	3.5
Volume (derived from above variables, m^3)	8.0
Mass (derived from above variables, tons)	25.0

Description of rock mass classification schemes

A number of rock mass classification schemes have been applied in the field, and as far as is known, no authors have claimed that their classification can be applied in all situations. This survey shows that most classifications were developed from a particular basis, which tends to limit their range of applicability.

Terzaghi's rock mass classification

Terzaghi's[5] rock mass classification is applied to the design of tunnel support in which rock loads, carried by steel sets, are estimated on the basis of a descriptive classification. The scheme classifies rock as intact, stratified, moderately jointed, blocky and seamy, crushed (but chemically intact), squeezing and swelling. Where one or more of these characteristics dominate, Terzaghi proposed a particular steel set design. His classification scheme was applied to railroad tunnels in the Alps, and has been widely used in North America ever since the paper was published. It is generally not applicable to hard rock mines because steel sets are expensive and seldom used, but could still be applied to shallow workings in weathered rock where *in situ* stresses are low, and the rock mass therefore not well confined.

Rock Quality Designation (RQD)

Deere[6] developed the RQD index to provide a quantitative estimate of rock mass quality from drill core logs. The RQD is defined as the sum of the lengths of intact core pieces longer than 100 mm expressed as a percentage of the total length of core. The core should be at least 50 mm in diameter and should be drilled with double tube diamond drilling equipment. The RQD can be misleading in rock masses where discontinuities in the rock are widely spaced and contain either infilling or weathered material[7]. Such a situation may result in a blocky, unstable rock mass, despite a high measured RQD. The RQD is thus unsuitable as a rock mass evaluator on its own, but has proven to be valuable as a component of more sophisticated rockmass rating schemes (see some sections that follow).

Rock Structure Rating (RSR)

The RSR[8] is a quantitative method for describing the quality of a rock mass and for selecting appropriate support based on the classification. Most of the case histories, used in the development of this classification, were for relatively small, shallow tunnels supported by means of steel sets. The RSR was the first to refer to shotcrete as a means of support. It is unsuitable for hard rock mines because it confines itself to shallow tunnels and steel sets or shotcrete support.

Stini and Lauffer classifications

Stini[9] proposed a rock mass classification for tunnels and discussed many of the adverse conditions, which can be encountered in tunnelling. The original work is in German, and it attracted little attention in the English-speaking world[10]. He emphasized the importance of structural defects in the rock mass and stressed the need to avoid tunnelling parallel to the strike of steeply dipping discontinuities. While both Terzaghi and Stini had discussed time-dependent instability in tunnels, it was Lauffer[11] who proposed that the stand-up time for an unsupported span is related to the quality of the rock mass in which the span is excavated.

In a tunnel, the unsupported span is defined as the span of the tunnel or the distance between the face and the nearest support, if this is greater than the tunnel span. The significance of the stand-up time concept is that an increase in the span of the tunnel leads to a significant reduction in the stand-up time, i.e. the time available for the installation of support. Knowledge of stand-up time is useful in designing mining-support cycles for advancing tunnels, but provides no guide to a permanent support system. This classification may suffice as a part of a support design process for an underground mine, but would require further development for it to be universally applicable.

Checklist methodology for hazard identification in tunnels

The checklist approach was developed to assess the risk of falls of ground in a mine tunnel[12]. The hazard identification takes place in two phases: the user answers a series of questions which pertain to a given hazard or group of hazards and then, a ranking method is used to determine scores for each answer, thereby generating an overall hazard score with which to assess the relative importance of the hazard[12]. The disadvantage of this system is that the user tends to focus only on the points addressed by the questions, and if they do not highlight a particular hazard, it is likely to remain unnoticed. This method does not propose any support for a given hazard, but acts merely to alert the rock engineering practitioner to a certain group of hazards it is designed to identify. It is therefore unsuitable as a basis for objective support designs in mine tunnels.

Rockwall Condition Factor

The first classification scheme to be designed for use in deep level mine tunnels is that of Wiseman[13], who proposed evaluating the conditions of tunnels using *in situ* stress, and the uniaxial compressive strength of intact rock. Ryder[14] added the empirical rock condition factor F to Wiseman's[13] equation, and coined the name Rockwall Condition Factor (RCF) in 1987. The best available description of the RCF may be found in COMRO[15], pages 88 and 89. The formulation of the RCF is based on a simple comparison of the maximum induced tangential stress of an assumed circular excavation to the estimated rockmass strength. It is given by:

$$RCF = \frac{3\sigma_1 - \sigma_3}{F\sigma_c} \qquad [1]$$

In Equation [1], the major and minor principal stresses within the plane of the tunnel cross-section are σ_1 and σ_3 respectively, F is Ryder's[14] dimensionless empirical rock mass condition factor, and σ_c is the uniaxial compressive strength of the intact rock material.

Wiseman[13] logged some 20 kilometres of underground tunnels in the South African gold mining industry, and related the results obtained to the support systems installed in the tunnels. It therefore represents the first attempt to objectively determine support required given certain rock mass and stress conditions for underground mines. Although it is fairly widely used in the gold mines after Ryder's[14] addition of a rock mass condition factor, it is considered applicable to tunnels that are affected by both jointing and mining-induced fractures, and is therefore not entirely appropriate to evaluate the potential for discontinuity-bounded falls of ground, which are commonest at Impala Platinum Mine.

The CSIR geomechanics classification for jointed rock masses

Bieniawski[16] proposed a rock mass evaluation system that addresses some of the limitations outlined above by providing a more comprehensive description of the rock mass from which a quantitative result is obtained. This classification system is called the CSIR Geomechanics Classification for Jointed Rock Masses, or the Rock Mass Rating (RMR). It classifies a rock mass using the following six parameters:

- uniaxial compressive strength of the rock material
- Rock Quality Designation (RQD)
- spacing of discontinuities
- condition of discontinuities
- orientation of discontinuities
- groundwater conditions.

When applying this classification system, the rock mass should be divided into structural regions for separate classification. Structural features such as faults or dykes, or changes in rock type should define the boundaries of a structural region.

The RMR has been applied to civil engineering projects and coal mines at or near surface where the *in situ* stresses are generally not of any concern. This is the first scheme to consider geotechnical conditions in detail, but it is still generally unsuitable for underground mines because it omits the *in situ* stress from the classification.

Laubscher's mining modification to Bieniawski's RMR

In 1973, D.H. Laubscher met with Z.T. Bieniawski to discuss the RMR, which was then being developed at the CSIR[17]. At the meeting, it became clear that the RMR was much better than that under development in Zimbabwe by Heslop and Laubscher[18]. Although Bieniawski's scheme was better, it did not possess the flexibility to adjust to different mining situations, and this led to the development of the Modified Rock Mass Classification for Jointed Rock Masses[17] based on Bieniawski's RMR. Originally, block caving in asbestos mines in Africa formed the basis for the modifications, but subsequently other case histories from around the world have been added to the database[19–21]. For design purposes, the modified classification scheme uses the RMR obtained from Bieniawski's scheme, and adjusts it to account for weathering, joint orientation, induced stress, blasting, and water according to the following equation (Laubscher and Jakubec, p. 479[17]):

$$MRMR = RMR \times adjustment\ factors \qquad [2]$$

The classification has grown in complexity to the current procedure shown as a flowchart in Figure 11. This classification scheme is fine for design and detailed design, but the complexity of the scheme as a whole makes it too tedious to apply to the support design procedure on a day-to-day basis. Furthermore, there is a considerable amount of subjectivity in the choice of adjustments for water, blasting, weathering, and so on. Jakubec and Laubscher[1] discuss this issue in their introduction and to repeat in their own words '…there is a growing concern in the mining community

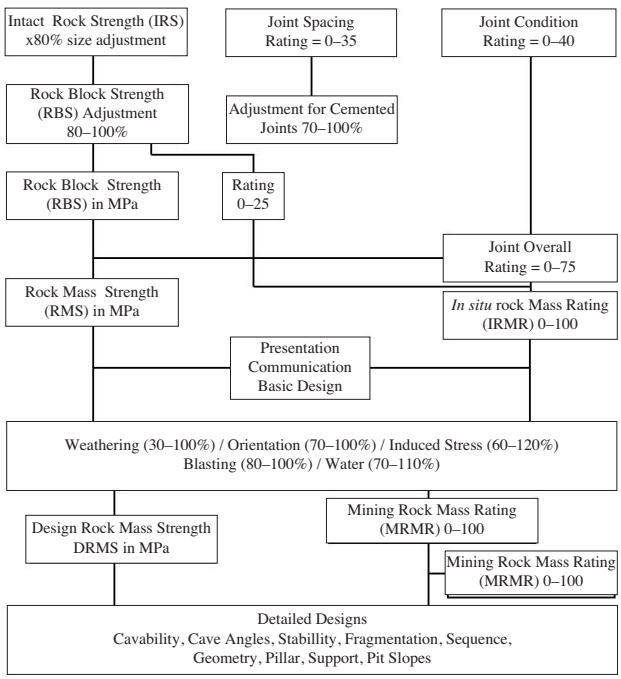

Figure 11—Flow sheet of the mining rock mass rating (MRMR) procedure with recent modifications (after Laubscher and Jakubec[17])

about their appropriateness and usefulness as a mine design tool. Some of the concerns are based on misunderstandings and misuse of the classification systems. It must be understood that a classification system can give the guidelines, but the geologist or engineer must interpret the finer details. The most important pitfall to avoid is the belief that the method is a rigorous analysis'. These concerns are applicable to every scheme that is available: engineering judgment is of paramount importance when assessing rock mass conditions, and much of this good judgement will come from the undefinable element of 'local knowledge'.

The authors decided to choose a scheme that could adequately address the variability of geotechnical conditions underground at Impala Platinum Mine, while at the same time being simple enough to make objective rock mass assessments daily. In addition, the scheme should be easily adjustable for the conditions, so that the 'local knowledge' element could be minimized, making the results obtained from the classification system a little more user-independent.

Rock Tunnelling Quality Index, Q

Barton *et al.*[2] proposed a Tunnelling Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirements. The numerical value of the index Q varies on a logarithmic scale from 0.001 to a maximum of 1,000 and is defined by:

$$Q = (RQD/J_n) * (J_r/J_a) * (J_w/SRF) \quad [3]$$

Where	RQD is the rock quality designation
J_n	is the joint set number
J_r	is the joint roughness number
J_a	is the joint alteration number
J_w	is the joint water reduction factor
SRF	is the stress reduction factor

The first quotient (RQD/J_n), represents the discontinuous structure of the rock mass, and can be considered a crude measure of the block or particle size. The second quotient (J_r/J_a) represents the roughness and frictional characteristics of the discontinuity surfaces and filling materials. This quotient is weighted in favour of rough, unaltered joints with opposing surfaces in direct contact with each other, while its value becomes small when rock joints have thin clay mineral coatings and fillings. The third quotient (J_w/SRF) consists of two stress parameters. The SRF is a measure of loosening load in the case of an excavation through shear zones and clay-bearing rock; rock stress in competent rock; and squeezing loads in plastic incompetent rocks. Together with J_w, which is a measure of water pressure in joints, the quotient forms a complex empirical stress factor describing the active stresses in the rock mass.

In summary, the rock tunnelling quality Q is a measure of:

- block size (RQD/J_n)
- inter-block shear strength (J_r/J_a)
- active stress (J_w/SRF).

These three factors are important in assessing mine tunnel stability, and together with the support guidelines provided by the system, provide a simple yet flexible scheme to take geotechnical factors into account in tunnel support designs.

Choice of rockmass rating system applicable to Impala Platinum Mine

The tunnelling quality index Q proposed by Barton *et al.*[1] and the MRMR classification scheme developed and modified by Laubscher[17] are probably the two most commonly used rock mass classifications in mining rock mechanics. Both are designed to assess factors which influence the stability of underground excavations. The similarities between the MRMR and the Q-Index stem from the use of identical, or very similar, parameters in calculating the final rock mass quality rating. The differences between the two systems lie in the different weightings given to similar parameters and in the use of distinct parameters in one or the other scheme.

The MRMR uses the uniaxial compressive strength of intact rock directly while the Q-Index only considers strength as it relates to *in situ* stress in competent rock. Both schemes deal with the geology and geometry of the rock mass, but in slightly different ways. Both consider groundwater, and both consider *in situ* stress, although the MRMR approaches stress more quantitatively than the Q-Index. Like the MRMR, the Q-Index has evolved over the years; for example, it has been extended to estimate tunnel boring machine performance in rock masses[22]. Unlike the MRMR, which has been generalized for all mining, the Q-Index has remained focused on tunnelling, particularly in the field of civil engineering.

Since both systems consider the factors necessary to assess underground tunnel stability, neither supersedes the other as a suitable system for application to off-reef tunnels at Impala Platinum Mine. The choice of the Q-Tunnelling Quality Index was therefore made for the following reasons:

- its simplicity as a measure of block size, inter-block shear strength, and active stress, all of which are critical factors in the fall of ground problem, as determined from the fall of ground analysis
- the Q-Index is easy to use on a daily basis underground
- the simple relationship provided in the scheme to decide whether support is required or not.

The Q-Index has since been applied in two case studies at Impala Platinum Mine.

Case studies

Two excavations were chosen for the case studies, namely 10 Level Crosscut West and 23 Level Conveyor Decline. The first excavation is in a fair to extremely good rock mass, the second in a poor to very poor rock mass. The range of the two cases covers all conditions likely to be encountered at Impala Platinum Mine.

The 10 Level Crosscut is located 640 metres below surface at No. 9 Shaft where both the Merensky and UG2 Reefs are being exploited. It lies above both reefs at the shaft, crosscutting down through the strata in a westerly direction until it intersects the Merensky Reef 120 m west of the shaft, and the UG2 Reef about 640 m west of the shaft. Mining has been restricted from taking place either directly above or below the crosscut, which has limited mining-induced stress changes in the crosscut itself to small values. The vertical virgin stress along the entire length of the crosscut is estimated to be approximately 20 MPa, and

mining-induced stress changes remain insignificant. The crosscut intersects the entire stratigraphic succession given in Table I.

The average width of the crosscut is 3 m with sections that widen out to 5.2 metres in places. Systematic support was installed in the crosscut at the wider sections and spot bolts were installed in one other area. The tunnel was developed in 1981, and has remained open without any significant falls of ground along its entire length ever since. The Q-Index was estimated for 10 m intervals over a distance of 770 metres in this tunnel.

The 23 Level Conveyor Decline is located 1058 metres below surface at No. 14 Shaft, which is currently mining the Merensky Reef. The vertical virgin stress at No. 14 Shaft is approximately 32.3 MPa at 23 Level. The decline is currently being developed in Footwall 16 anorthosite below the UG2 Reef, at an average span of of 5.6 metres. Blasting has resulted in an average of 30 centimetres overbreak over the whole excavation length.

The support installed in the decline roof and sidewalls consists of 3 metre-long, 16 millimetre diameter, full column grouted shepherd's crooks placed one metre apart in both dip and strike directions. The rock mass shows hangingwall and sidewall instability, with falls of ground up to two metres high taking place almost immediately after the blast.

Methodology used to obtain the Q-Tunnelling Quality Index

The Q-Index obtained is used to assess the stability of an excavation, and to provide guidelines for the excavation support. The methodology used to estimate the input parameters for the Q-Index is discussed briefly below.

Estimating the RQD from scan line measurements

The RQD for a rock mass can be calculated from scan line measurements taken underground. A scan line is defined as a line, usually a tape, set on the surface of the rock mass, and the survey consists of counting the number of joints which intersect this line along its length. Three mutually perpendicular scan lines were set up in the tunnel; the first, parallel to the axis of the tunnel; the second, spanning the tunnel width at the midpoint of the axial scan line; and the third, vertical, also located at the midpoint of the axial scan line. The scans were repeated every ten metres along the axis of the tunnel. The scan line length along the axis of the tunnel was chosen to be ten metres in order to capture joint sets with spacing of up to ten metres.

Joint densities (number of joints per linear metre) were found by counting the number of joints intersecting the scan line and dividing by its length. In this way a linear joint density for the tunnel axis direction D, for its span S, and its height, V were obtained and inserted into equations [4] and [5]:

$$J_D = D + S + V \qquad [4]$$

$$RQD = 115 - 3.3J_D \qquad [5]$$

If the RQD obtained from Equation [5] is less than 10%, then a value of 10% is entered into Equation [2]. For values greater than 100% obtained using Equation [5], a value of 100% is used for the RQD in Equation [2].

Estimating the other variables

Estimates for J_n, J_r, J_a, J_w, and the SRF are obtained as described by Barton *et al.*[1]. Where the excavation is expected to be subject to a stress change, a stress analysis is carried out using MINSIM-W[24] to help estimate a stress–rock strength relationship.

Q-Index analysis of 10 Level Crosscut and 23 Level Conveyor Decline

A total of 89 tunnelling quality index measurements were made, 77 in 10 Level Crosscut and 12 in 23 Level Conveyor Decline. This represents 890 metres of tunnel covered using this method. Figure 12 provides statistics of the spread of Q-Index values obtained in the two excavations.

Figures 13 and 14 are log-linear scatter plots of Equivalent Dimension (D_e) versus Q-Index and Excavation Span versus Q-Index respectively. Two populations of points appear in the plots, the diamond symbols for unsupported sections, and the squares for supported sections. The type of support is ignored in these plots, so that excavations with support ranging from spot bolts to heavy support using several types of support elements are considered supported. In both plots, Barton's[2] 'no support' line straddles both unsupported and supported excavation data, which suggests that the current decision line position is not suitable for

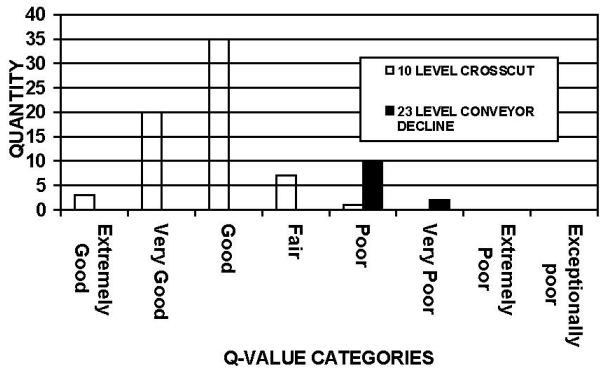

Figure 12—Distribution of Q-Index values obtained from 10 Level Crosscut and 23 Level Conveyor Decline

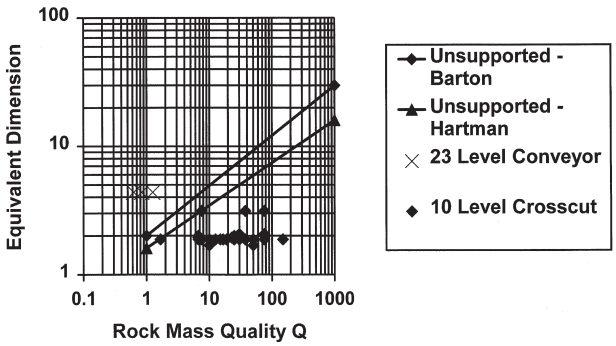

Figure 13—Scatter plot of equivalent dimension vs. Q-Index for an excavation support ratio of 1.6 (permanent mine openings)

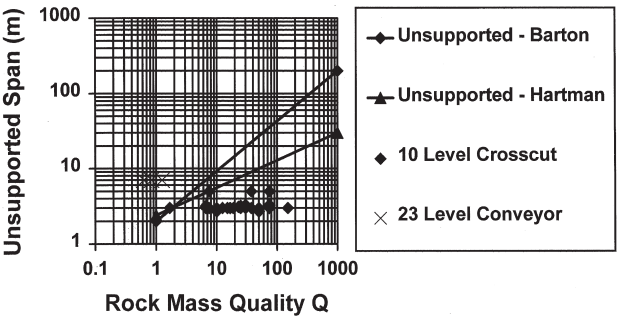

Figure 14—Scatter plot of excavation span vs. Q-Index

Impala. If this line were used as a guideline, there would be occasions when no support would be installed in an excavation that needs support, which could lead to a fall of ground. The 'no support' line only straddled data from 10 Level Crosscut, which is located in a good to very good rockmass. All data for the 23 Level Conveyor Decline plotted well to the left of the 'no support' line, since this excavation yielded relatively low Q-Index values.

Barton's 'no support' line is given by the following formula for the Equivalent Dimension — Q-Index relationship:

$$D_e = 2Q^{0.4} \qquad [6]$$

D_e is the Equivalent Dimension, defined by excavation height or excavation span divided by the Excavation Support Ratio, which is given by Barton[1] as 1.6 for permanent mine openings. In considering the relationship for excavation span, Barton[24] considered both natural and man-made excavations, from which he deduced that:

$$Span = 2Q^{0.66} \qquad [7]$$

In order to estimate where the 'no support' line should plot for Impala data, it is assumed that conditions demanded support wherever it was installed, and that unsupported sections really do not need support. Since geotechnical parameters have not until recently been taken into account in the decision to support or not, it is also assumed that the relevant supervisor decided where support was necessary on the strength of a visual check. This decision was probably based on a fall of ground, either shortly after the development blast, or sometime later in the excavation's life.

There is no evidence to suggest that the relationship should not be the power law already given by Barton[1], hence this will be retained in the following general form:

$$Span = a \times ESR \times Q^b \qquad [8]$$

Where it remains to determine a and b to suit the data obtained for Impala, assuming in all cases that ESR = 1.6 for permanent mine openings. Two points on the 'no support' line need to be defined in order to determine the parameters for the relationship. Adjusting this line does not invalidate the large database from which it was determined in the first place—it merely shows that 'local knowledge' can introduce refinements to the system for a particular location. Van der Merwe[23] had to make similar adjustments to the coal strength factor k in Salamon's[24] pillar strength formula for the Vaal Basin to account for the unexpected pillar failures observed there. It is therefore argued that, since the formula for the support—no support line is empirical, there is no harm in adjusting it to suit a particular situation, and that it may be prudent to review it in all situations where the Q-Index is applied.

The smallest dimension measured in the fall of ground data is 0.5 m (see Figures 3, 4, and 5). It is therefore assumed that a fall of ground this size could occur in an excavation with a span of 0.5 m. Secondly, it appears from Figures 13 and 14 that all excavations with Q-Index values of 1.0 or less require support. The first point for the 'no support' line in Figure 14 would thus be (1.0, 0.5). It also appears from the data in Figures 13 and 14 that excavations with Q-Index values of 100 or more never need support. Since this was only ever observed in 10 Level Crosscut, a span of 3 m is assumed, and the second point is thus (100.0, 3.0).

Using the two points to solve for a and b in Equation [8] and assuming the ESR = 1.6, we obtain:

$$D_E = 0.3125Q^{0.3891} \qquad [9]$$

$$Span = 0.3125 \times ESR \times Q^{0.3891} \qquad [10]$$

These lines appear in Figures 13 and 14, and it can be seen that they separate the unsupported data from the supported data more effectively than Barton's[2,25] lines. The result of the changed support lines is that tunnels of smaller span will require support than that required by the original analyses of Barton[2] and Barton[25]. The reasons for this smaller span may lie in the hardness and brittleness of the rock material, together with the fact that it is blasted. If the tunnels were mechanically excavated, it is expected that larger spans would be stable in the same conditions.

This indicates that the Q-Tunnelling Quality Index is general, but in its generality needs refinement for application in specific situations. In an attempt to eliminate 'local knowledge' in the decision to support or not, the classification system has been adjusted slightly to provide a more reliable guideline on when to install support or not at Impala Platinum Mine.

Bolt length design for permanent mine openings

The Q-Index analysis first provides a guideline on whether the tunnel needs support or not, using the revised 'no support' line. If support is required, then the support design must consider the bolt type in use, the 95th percentile fallout height, and fall of ground dimensions. An example design, which results in a bolt length of 1.2 metres spaced 1 m apart, appears in Appendix I. Note that this design should not be applied everywhere on the mine. Rather, geotechnical conditions at the site should be reckoned into the support design. A comparable design using Barton's[1] formula follows in Appendix I. There is an opportunity to modify this formula as well for conditions at Impala Platinum Mine.

The work in implementing the use of geotechnical parameters in tunnel support design is still not complete. Geotechnical mapping should be carried out far more widely on the mine, and should include excavations with larger spans than those measured in this paper. This will help to fix the 'no support' line with better confidence. Short-life excavations should also be included in the mapping. With this increased database it will be possible to determine a set of excavation support designs for all geotechnical conditions and all excavation spans at Impala, so that support design could be simplified to a reference to a standard support table for the mine based on the Q-Index. With time it will be possible to refine the support designs to increase their cost-effectiveness, while at the same time eliminating falls of ground in off-reef excavations.

Conclusions

Support designs for off-reef excavations at Impala previously met the 33-kN/m^2 support resistance criterion without any consideration of the geotechnical conditions. This approach is no longer acceptable because it does not prevent fall of ground accidents. There are also many instances where such support may not be necessary.

A fall of ground accident analysis for off-reef tunnels at Impala revealed that:

- 95% of all falls of ground were 0.9 metres thick or less
- nearly all were controlled by discontinuities in the rock mass
- large falls of ground are rare
- conventional support designs already in use could have prevented all falls of ground if the circumstances leading to them had been identified beforehand.

The Q-Tunnelling Quality Index is sufficiently flexible to identify the potential fall of ground hazard if it is properly used at Impala, and it can be used as a guideline to more cost-effective support. Since testing the viability of the Q-Index as a rock mass classification scheme underground, it has shown that:

- the Q-Index addresses all the critical parameters that control falls of ground in tunnels at Impala
- it is easy to apply underground, and geotechnical data gathering using the system is quick
- Barton's[2] 'no support' line needs minor adjustment to improve its reliability as a guideline for Impala
- the changes to the parameters for the 'no support' line represent a first estimate, which should be refined as more geotechnical data is gathered
- routine mapping of the tunnels should reveal all potential fall of ground hazards
- Impala Platinum Mine should be able to take remedial action against potential falls of ground before they occur.

Implementing effective support designs based on geotechnical investigations should help solve the fall of ground problem in off-reef excavations, while underground support designs for off-reef excavations could be simplified once a comprehensive geotechnical database has been built up.

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Appendix I

Calculation of required bolt length for 3 metre span tunnels

Fall of ground data suggest that 56% of the falls are 1 m^2 in area or less, 44% of falls are 0.5 m or less wide, and 36% are 0.5 m or less long. To prevent these from occurring, a maximum bolt spacing of 1 by 1 m is necessary. This does not mean that bolt spacings everywhere on the mine should be 1 x 1 metre. Geotechnical mapping using the Q-Index will be far more effective in deducing maximum bolt spacings than fall of ground accident data, and should be used for this purpose wherever possible.

Each bolt must suspend a potential fall of ground. Hence, it should be bonded to solid ground above the 95th percentile potential fall-out height of 0.9 m. Assume that grouted rebars will be used. If the 95th percentile fall of ground thickness is 0.9 m, then the rebar must be long enough to suspend 0.9 m of rock. The volume of rock to be supported by each rebar is thus 0.9 m^3, amounting to a deadweight of 27.4 kN, if the rock density is assumed 3000 kg/m^3. The strength of a grout bond with the support element is given by:

$$bond\ strength = \frac{pull - out\ force}{\pi \times rebar\ diameter \times bond\ length} \quad [11]$$

Underground pull tests have revealed an average 4 MPa bond strength for cement grouts that have cured for 1 hour. Rearranging Equation [12]:

$$bond\ length = \frac{pull - out\ force}{\pi \times rebar\ diameter \times bond\ strength} \quad [12]$$

Assuming a 16 mm rebar diameter, a bond length of 0.14 m is required. Using a factor of safety of 2.0, the minimum bond length should be 0.28 m. The rebar length should thus be 0.9 + 0.28 = 1.18 metres, say 1.2 metres.

Comparing this with a typical bolt design from Barton *et al.*[2]:

$$L = 2.0 + \frac{0.15B}{ESR} \quad [13]$$

Where L is bolt length, B is the excavation width, and the ESR is the Excavation Support Ratio—a value related to the intended use of the excavation and the degree of security. For permanent mine openings assume the ESR = 1.6. For Impala, a permanent 3.0 m wide excavation will thus require a bolt 2.28 m long. This design length is some 90% longer than the design using the fall of ground data and measured pull-out forces. It is therefore considered inappropriate at present, but in time, there may be sufficient geotechnical data to change this formula to suit conditions at Impala Platinum Mine. ◆

SA to host major ferroalloy conference*

South Africa will be benefiting from another major international conference during 2004, when it once more plays host to international ferroalloy experts for the INFACON 10 conference. (It hosted INFACON 4 in 1992).

The conference, which will be co-organized by Mintek and the South African Institute of Mining and Metallurgy, and supported by the FerroAlloy Producers Association, will be held in Cape Town.

Dr Nic Barcza, General Manager: Commercialization at Mintek, and current Chairman of ICFA, the International Committee on FerroAlloys, says, suitable topics for discussion at the conference would include the following:

- ore/raw material sources and geology/mineralogy, pre-treatment process (e.g. pre-heating and pre-reduction)
- melting and smelting processes
- refining, casting and granulation
- unit operations (e.g. furnaces, kilns, fluidized bed reactors)
- process flowsheet development
- environmental, health and safety aspects
- stainless steel process technology and alloy development
- quality of ferroalloys and stainless steel
- modelling, simulation and control and economic considerations, including the potential benefits of new technology.

Dr Barcza stresses that industry and government institutions need to be encouraged to give more support to R&D projects in this important area.

For more details, please contact Dr Nic Barcza at Mintek on (011) 709-4680 or Tom Curr on (011) 709-4642. ◆

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ISEC 2002—International Solvent Extraction Conference*

ISEC, the triennial International Solvent Extraction Conference, is the premier meeting of solvent extraction (SX) experts. Hosted for the first time in Africa. ISEC 2002 took place in Cape Town from 17 to 21 March 2002. One of the most successful meetings of recent years, the conference was attended by over 350 delegates from some 40 countries and six continents, and included representation from academia, research institutions, suppliers, engineering contractors, and plant operations.

The ISEC 2002 technical programme emphasized applied aspects of solvent extraction, particularly less traditional and emerging applications, but theory and fundamentals were not neglected. The conference was opened by Barry Davison, Chair of the Chamber of Mines of South Africa, who provided a rare insight into the global platinum market. Five excellent plenary addresses tackled issues that ranged from nanotechnology to plant operations. Representing industry were Gord Bacon (INCO, Canada), who spoke on SX as an enabling technology in the nickel industry, and Peter Charlesworth (Anglo Platinum, South Africa) who discussed SX developments in the refining of platinum-group metals. Marc Halpern (PTC Organics, USA) related commercial experience in another area to traditional SX in his presentation 'Achieving high performance low cost processes for the manufacture of organic chemicals using phase transfer catalysis.' Alan Hatton of MIT discussed 'Tailored solvents for green processing of chemicals and biochemicals' and Robin Rogers (U Alabama) introduced the concept of room temperature ionic liquids as alternatives to traditional organic solvents in SX, both providing intriguing glimpses of the cutting edges of SX research.

The main technical sessions each kicked off with a Keynote address by a renowned expert in the field, and were followed by a series of oral and poster presentations. Some 230 papers were presented. Fundamental studies covered coalescence and interfacial phenomena, advances in SX theory, kinetics and mass transfer, extractions, and novel developments. Sessions also focused on analytical, biotechnology, pharmaceutical and nutriceutical applications, membranes, petrochemical, nuclear and hydrometallurgical applications, and on process engineering and design.

The Carl Hanson Medal, sponsored by Dechema and the Society of Chemical Industry, is presented every three years for excellence in the field of SX. Selected by the International Committee on Solvent Extraction, the award this year was made to Gordon Ritcey of Canada.

ISEC 2002 also provided a showcase for reagent and equipment vendors, with a small exhibition accompanying the conference. The main conference sponsors were Anglo American Platinum Corporation, Anglo American Research Laboratories, and Cytec. Other generous sponsors and exhibitors included Avecia, Bateman Engineering, Bayer, Cognis Engen Chemicals, ExxonMobil, Marcel Dekker, Mintek, Rhodia, Sasol, Schümann Sasol, Shell Chemicals, Spintek, and Technical Center Halle.

As antidotes to the more serious business of the week, several social events provided opportunities for delegates to make new friends as well as experience some of the beauty that The Mother City has to offer. A cable car trip up Table Mountain yielded spectacular views and a relaxing afternoon. The conference banquet was held at Nederberg, an internationally renowned wine estate. Accompanied by a glorious sunset over the Paarl mountains, traditional African entertainment, wine tasting, and excellent food and music, it was certainly an evening to remember.

Two post-conference technical tours visited base metal, precious metal, industrial and fine chemical SX plants in South Africa, and also provided an opportunity for delegates to experience the unique African bush and wildlife of the Pilanesberg Game Reserve. Visits to Anglo Platinum and Impala Platinum provided interesting contrasts of the base and precious metals' refining operations of the world's two largest platinum producers. Tours also visited Sasol, the largest synthetic fuels facility in the world and a leading supplier of liquid fuels and other petrochemical byproducts, and CSIR's BioChemtek Division which specializes in applications of solvent extraction in the food and biotechnology industries.

Proceedings of the International Solvent Extraction Conference ISEC 2002, K.C. Sole, P.M. Cole, J.S. Preston and D.J. Robinson (eds.), South African Institute of Mining and Metallurgy, Johannesburg, 2002, 1446 pp. Available as a two-volume publication (ISBN 1-919783-24-5) (R2000 + R300 postage) or on CD-ROM (ISBN 1-919783-25-3) (R500 + R50 postage). Abstracts of the papers are available on the conference website at www.isec2002.org.za) Proceedings can be ordered from: South African Institute of Mining and Metallurgy, PO Box 61127, Marshalltown, 2107, South Africa, Tel: +27 11 834-1273, Fax: +27 11 838-5923 or Fax +27 11 833-8156, E-mail: sam@saimm.co.za. ◆