APPLICATION OF THE Q-SYSTEM TO AUSTRALIAN UNDERGROUND METAL MINES

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ABSTRACT

The applicability of the Q-system [Barton et al. 1974] to Australian underground metal mines is discussed with reference to two common design issues: ground support for horizontal mine development, and assessing the stability of bored raises.

Installed ground support in mine development is compared to empirical estimates using the Q-system and associated support capacity calculations. Data are graphically presented from 59 specially selected sites at 15 contributing mines.

The actual performance of large-diameter raise-bored shafts is also compared to empirical stability assessments using a modified version of Q (Qr, after McCracken and Stacey [1989]). Lower-bound Qr values are plotted against raise diameter for 47 selected sites at 23 mines in Australia and Papua New Guinea.

The influence on Q and Qr of some geotechnical aspects of the Australian landscape, the dynamic nature of mines (compared to civil construction), and occupational health and safety regulations are discussed.

Stability and support assessments that are based just on Q or Qr are not always conclusive. It is often necessary to consider other rock mass parameters, the regulatory environment, and risk issues.

These results are interim; further data collection and analysis are required with regard to comparing actual performance versus empirical assessments.

INTRODUCTION

This study examines the applicability of two aspects of the Q-system [Barton et al. 1974] to Australian underground metal mines:

- For horizontal development, actual installed ground support versus empirical predictions using the Q-system and associated support pressure calculations;
- For raise-bored shafts, actual performance versus empirical predictions of stability using the modified version of Q published by McCracken and Stacey [1989].

The data presented in this paper, with one exception, have been supplied by mine management with the understanding that individual mines would not be identified in any report or published paper. Individual cases are discussed in a way that would not permit their locations to be identified.

AUSTRALIAN GEOTECHNICAL ENVIRONMENT

Several geotechnical aspects of mines and the Australian occupational health and safety (OH&S) environment have a significant influence on the stability of openings and ground support requirements.

Deep Weathering

Australia is an old, stable, continental mass. It has undergone numerous climate changes without appreciable erosion or glacial removal of weathered rocks. Weak weathered rocks often extend to depths of up to 90 m below the surface.

Extremely weathered near-surface rocks can be stronger due to the deposition of silica, carbonates, and iron oxides to form “caps” of stronger materials. Below the harder caps, weathered rocks often form a variety of rock-like saprolites through to weak clays. Soil mechanics rather than rock mechanics principles are more applicable to some of these weaker materials.

While the present groundwater level is often near the base of complete oxidation, this may not be the same as the base of weathering.

With regard to shafts, weak near-surface rocks are often the biggest challenge for site investigations, design, and construction. Failures have been relatively common due to a generally poor understanding of their behavior, their highly variable nature, and risk-taking during construction. In some cases, high rock quality designation (RQD) ratings have been erroneously assigned to extremely weathered rocks that were neither hard nor sound and therefore should have been assigned a nominal RQD rating of 10%. Groundwater or wet materials are also often implicated in these failures.

Alteration and Weak Sheared Contacts

Altered rocks and thick weak sheared contacts are a common feature of some ore body styles, e.g., volcanogenic lead-zinc deposits and hydrothermal copper or gold deposits. It is often necessary to mine access development

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and stopes along these weak contacts. Some of the common joint infill minerals include quartz, carbonates, chlorites, sericite, talc, zeolites, clays (both swelling and nonswelling), and gypsum.

In the case of porphyry copper deposits, the associated alteration is typically more pervasive and associated with intense fracturing. The entire rock mass is often either silicified, carbonated, or sericitized, and weak joint infill materials are common. Normally, strong igneous rock can be decomposed to clay at depths of hundreds of meters below the ground surface, while the surrounding jointed rock may be recemented by the gypsum released in the hydrothermal alteration process. Graphite is a common (low-friction) joint infill material in carbonate-hosted base metal deposits.

Weak Ultramafic Host Rocks

Soft, weak ultramafic rocks are a feature of Australia's (Achaean) nickel mines. Their geologic history is varied and complex, as is their behavior, which is often significantly time-dependent.

A wide variety of soft, weak talc-rich ultramafic lavas are present due to serpentinization very soon after eruption, variable grades of metamorphism, possible carbonation or potassium metasomatism, and finally deep weathering. Some rocks contain the very water-sensitive mineral brucite (magnesium hydroxide).

Ground conditions are very challenging, especially as soft, weak ultramafics can abut much stiffer and stronger rocks. Ground conditions and ground support are often extreme. Significant squeezing has occurred in ultramafics as shallow as 250 m, but as the ores can have very high values, mining has, thus far, reached 1,400 m below surface in one mine.

Groundwater

Although high groundwater pressures are not common in mines, groundwater is often acidic or saline. Both influence the longevity of installed support, which typically must have an effective life of 10+ years. Point corrosion or rusting of bolts, possibly leading to premature failure, is also assisted by the tendency of mine rock masses to crack and loosen with time, especially when they are adjacent to stoping areas.

The effective life of support can be extended by using galvanized elements, fully grouted bolts (resin or cement), or plastic sheaves or (very expensive) low-grade stainless steel support elements. However, experience has shown that none of these measures guarantees the long-term integrity and effectiveness of support. There is also, presently, no foolproof method of testing or monitoring the adequacy of acid- or salt-challenged bolts with time.

High Horizontal Stresses

In contrast to stresses in other tectonic plates, in situ measurements of premining rock stresses in Australian mines have demonstrated large variations in principal stress magnitudes with depth (Figure 1). While the major principal stress is often horizontal, its orientation can also vary widely between local regions [Lee et al. 2006].

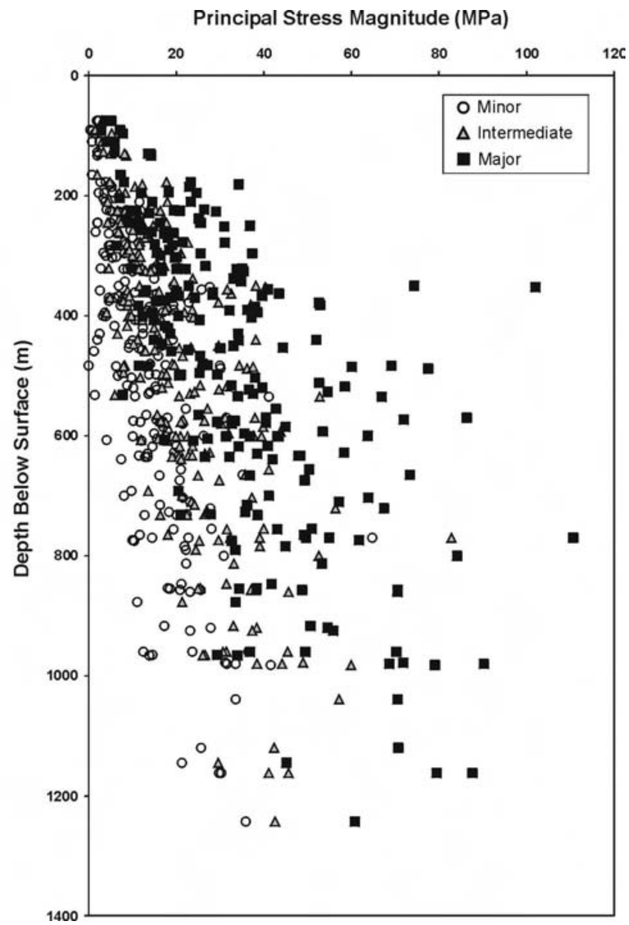

Figure 1.—Australian principal stress magnitudes versus depth.

Compared to similar mining provinces in eastern Australia and in other tectonic plates, anomalously high and deviatoric horizontal stresses are a feature of the Achaean Yilgarn Craton in the southwestern corner of Western Australia. This area hosts numerous gold, nickel, and copper-zinc mines, which currently stope to depths of up to 1,500 m. Mine openings typically have very high tangential stresses in development backs and shaft walls, often close to the strength of the rock mass [Lee et al. 2001].

Mining-induced seismicity is common in some of the deeper mines in strong, often jointed, stiff rocks due to both violent fracture through intact rock and shearing on structures. In stark contrast, some of the softer and weaker talc-rich rocks and schist tend to squeeze, even at shallow depths.

Mining-induced Stress Changes

Stress changes around openings are not usually an issue in civil engineering projects, but they are an important feature of mining. Due to nearby stoping, an opening might first be subjected to high or excessive abutment stresses, then low confining stresses as the opening is shielded by stoping. Both can encourage local shearing on structures, with associated cracking of the intact rock, and dilation plus loosening of the rock mass.

The installed support must be able to accommodate all of the associated movements and loosening, yet still adequately support the rock mass. Areas affected by stoping can therefore often seem to be oversupported.

Mining-induced Seismicity and Blast Damage

For deep mines or those considered to be prone to seismic damage, significantly more support is often installed. It usually comprises the following:

- Fibercrete (minimum 50 mm) + rebars (backs) and friction anchors (walls); then:
- Mesh + friction anchors (backs + walls); and then maybe:
- An extra 50 mm of fibercrete.

The Q-system does not presently have a facility to assess ground behavior in potentially seismically active areas. Ground support design in such areas usually considers the toughness or energy absorption (kilojoules per square meter) capacity of the support system, rather than the support capacity of bolts in tonnes per square meter.

Vibration damage from large-stope production blasts is sometimes addressed in the same manner as seismicity. In addition to the support of credible or worst-case wedges defined by structures, it is usually sufficient to install mesh with friction anchor bolts in backs and walls that might be exposed to blast damage.

OH&S Regulations

OH&S regulations and company policy often dictate the minimum support that must be installed, irrespective of ground conditions.

Under the "general duty of care" provisions of the Western Australian Mines Safety and Inspection Act 1994:

- "An employer must, as far as practicable, provide a work environment in which employees are not exposed to hazards and provide information, instruction, training and supervision;
- Employees must take reasonable care for their own safety and health, and that of others, at work; and
- Self-employed persons must, as far as practicable, ensure their work does not adversely affect the safety and health of others."

In terms of human exposure to possible falls of ground, guidelines relating to the above provisions imply that "nobody is allowed to work beneath unsupported ground," no matter how competent the ground may seem. Mining companies and the governments of the other Australian states have generally adopted this policy.

In terms of ground support requirements, guidelines relating to the above provisions imply that "all development backs must be scaled or adequately supported down to a height of 3.5 m, unless a report by a competent person justifies otherwise." After firing, development backs in Australian mines are now either routinely meshed or sprayed with a minimum 50-mm thickness of fibercrete, then bolted.

The above regulations and policies also influence the choice of excavation support ratio (ESR) when determining ground support using the Q-system chart. There is a growing awareness in Australian mines that the appropriate minimum ESR value is 1.3 for all human-access development, whether it is permanent or just a temporary stope access, because miners must travel and work in both. The only difference to the installed support in permanent, versus temporary, openings might be the use of galvanized support elements and fully grouted bolts to improve their longevity. Recognition of the limited life of ground support is not embodied into the existing Q-system chart.

HORIZONTAL DEVELOPMENT

Prior to the early 1980s, development openings in most Australian mines were typically small, up to 4.5 by 4.5 m, and often manually mined and supported. Backs were also often flat, and only a few bolt types were available and used, mostly 2.4-m mechanical point-anchored bolts. Openings were routinely check-scaled, and mesh was only used in exceptional circumstances.

When decline access and large trackless equipment became popular, development was mined and supported using multiboom jumbos and their size increased to be nominally 5.5 by 5.5 m. But this small increase in development width implied a large increase in the required support capacity (approximately 50%), which was partly offset by arching the development backs. Friction anchor bolts (often referred to as Split Sets) also became popular because they are cheaper (per unit), easily installed using the jumbo, and they are an excellent bolt to pin mesh tightly to irregular development backs and walls. However, they have a much lower end-anchorage capacity than point-anchored solid bar bolts.

Unless short friction anchors are used to just pin mesh, by inserting them in previously installed longer friction anchors, the minimum standard bolt length is typically 2.4 m. Ground support typically comprises a mix of friction anchors, solid rebar bolts, and cable dowels.

Falls of ground became more common. OH&S regulations were reviewed. Mines also began to focus on the design and adequacy of ground support.

Barton's Q database is dominated by civil engineering examples, not mining ones, where ground conditions can be more dynamic. It contains few, if any, cases from mining in Australia under the current legislative environment.

It is a significant challenge to provide adequate ground support for all of the diverse areas and situations in (underground metalliferous) mines all the time. Unlike most civil projects, ground conditions can change with time because of weathering and oxidation of minerals and the rock mass, variations in moisture content due to seasonal or ventilation changes, stress history, and/or damage to exposures due to nearby stope blasting, seismicity, etc.

Previous Australian Studies

Mikula and Lee [2003] considered that "applying Q to a mine is like importing a knowledge database to a mine. Because the knowledge was compiled elsewhere, it should be confirmed to ensure relevance and correct use in the new environment." They reported that Q is a suitable design tool for assisting ground support selection at the Mt. Charlotte gold mine, provided appropriate stress reduction factor (SRF) values are used and allowance is made for stress field anisotropy.

An unpublished 1999–2002 survey of 183 km of development headings with spans of 4.0–5.5 m in 20 Australian mine sites concluded that significantly more support was being used than is recommended by the Q-system tunnel reinforcement design chart (after Grimstad and Barton [1993]). Support usage was obtained from warehouse documentation, and Q values were averaged over several months of mining development headings. Q values ranged from 34 to 0.01.

The following examples indicate that significant amounts of support are being used at some Australian mines:

- Development in ultramafic rock at depths in excess of 1 km, with average Q = 1.0 and a minimum Q = 0.54. Support averaged 32 friction anchors (commonly called Split Sets) per meter advance plus 75 mm of fibercrete. This is about twice the number of friction anchors suggested by the Q chart for Q = 0.5, when the predicted number of solid bar bolts is converted to an equivalent number of friction anchors having the same support capacity.
- Development in folded and faulted Paleozoic volcanics at depths in excess of 900 m, with Q ranging between 6.0 and 16.2. Support averaged nine friction anchors per meter plus 50 mm of fibercrete. The Q chart suggests that minimal support is required, and to satisfy OH&S regulations the minimum back support is only about seven anchors plus mesh per meter advance.

A major shortcoming of the 1999–2002 survey was that actual support patterns were not correlated with specific ground conditions and actual Q values.

Applicability of Q in Horizontal Development

A significant concern with the way the Australian mining industry has used the Q versus the equivalent dimension design chart [Grimstad and Barton 1993] has been the assumption that "bolts" referred to in the design chart has included friction anchors. However, the bolts in the Q chart were nominal 20-mm-diam solid mild steel bolts, fully grouted using either cement or resin.

Fortunately, there is a growing awareness in the Australian mining industry that pattern bolting using only friction anchors often provides insufficient support for many situations, and supplementary solid steel bars are required.

Data Collection

The 1999–2002 survey mentioned above used averaged Q values. As this approach can skew the data, it has not been used below.

General mine-wide databases of ground condition versus minimum support standards have also been ignored. It is was considered that company policy and government OH&S requirements dictate the quantity of ground support installed. This is especially a concern in good-quality rock, which might not technically require support to ensure the stability of some openings.

This survey uses specific Q values determined by mapping at the same location where the actual installed support was also recorded. To maximize the impact of the local ground conditions for support selection, numerous sites in the poorer rock classes were included, especially where the following was true:

- The installed support was considered to be just sufficient for the ground conditions; or
- There had been a fall of ground and the support had been upgraded.

The database contains 59 data sets from 15 mine sites drawn from all six Australian states and the Northern Territory. Nearly all of the data apply to 5.5-m-wide and variably arched development.

Installed Support Capacity Versus Q

The end-anchored capacity (at yield) of installed support was calculated in tonnes per meter squared. The following assumptions were used for the three commonly installed support types:

- Friction anchors = 3 tonnes
- Solid bar bolts = 15 tonnes
- Cable dowels = 21 tonnes

No capacity allowance was made for mesh or fibercrete, as they rarely extend down the walls to invert level, often stopping about 3 m above floor level. Their main function is to retain loosened pieces of rock and transfer the weight of loosened blocks to the rock bolts. Except in areas prone to seismicity, both are often capable of supporting the deadweight of any loose material or small wedges that might develop between reasonably spaced bolt collars.

Q values are plotted against the installed support capacities in Figure 2. In four cases, two data points plot on top of each other.

While one might expect the installed support capacity to vary inversely with the Q values, there is significant scatter in Figure 2, particularly in the 30 data points at bolt capacities less than 3 t/m^2. Friction anchors were used at 27 of these 30 sites.

The almost Australia-wide requirement (government OH&S regulations and/or company policy) for backs to be screened and supported (with either mesh plus bolts or fibercrete plus bolts) implies a minimum bolting pattern of either 1.1 m by 1.1 m or 1.1 m by 1.4 m for either 2.4-m by 2.4-m or 2.4-m by 3.0-m mesh, allowing for overlap. Thus, if only friction anchors are used, the minimum end-anchored bolt capacity is in the range of 2–2.5 t/m^2.

The data in Figure 2 show that a bolt capacity of less than 3 t/m^2 was used in some Australian mines for Q values ranging from 90 to 0.3, and the Q-system predicted support capacity is up to several times the actual installed capacity. The extent to which this is the result of substituting friction anchors for the solid bar bolts intended by Grimstad and Barton's 1993 chart is unknown.

As only 3 sites in this group of 30 sites required rehabilitation, it is possible that local site experience has shown that some sites are sufficiently stable for mining purposes with less support than predicted by the Q-system. Alternatively, the Q values may have been optimistically estimated.

There is a large zone in Figure 2 where the installed support capacity is less than half the empirical predicted requirements. There is also a diagonal band where the ratio of actual to predicted support capacity ranges from 0.5 to <3.0. Finally, there is a zone where the ratio of the installed to predicted support capacity exceeds 3.0.

If Q values have been accurately assessed and if it can be assumed that the installed support is just adequate, the data in Figure 2 suggest that there is a tendency in the Australian mining environment for the Q-system to underestimate the support required for the "good" and "very good" rock classes, probably due to the OH&S considerations discussed above, and to overestimate support requirements for "poor" to "extremely poor" rock classes.

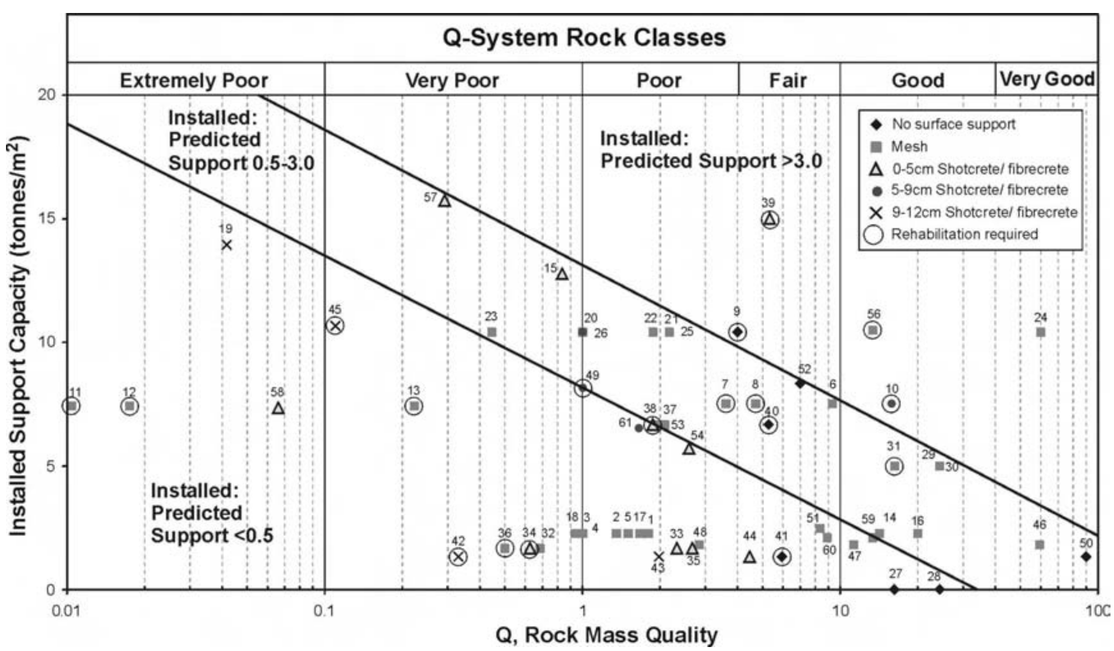

Figure 2.—Installed support capacity versus Q.

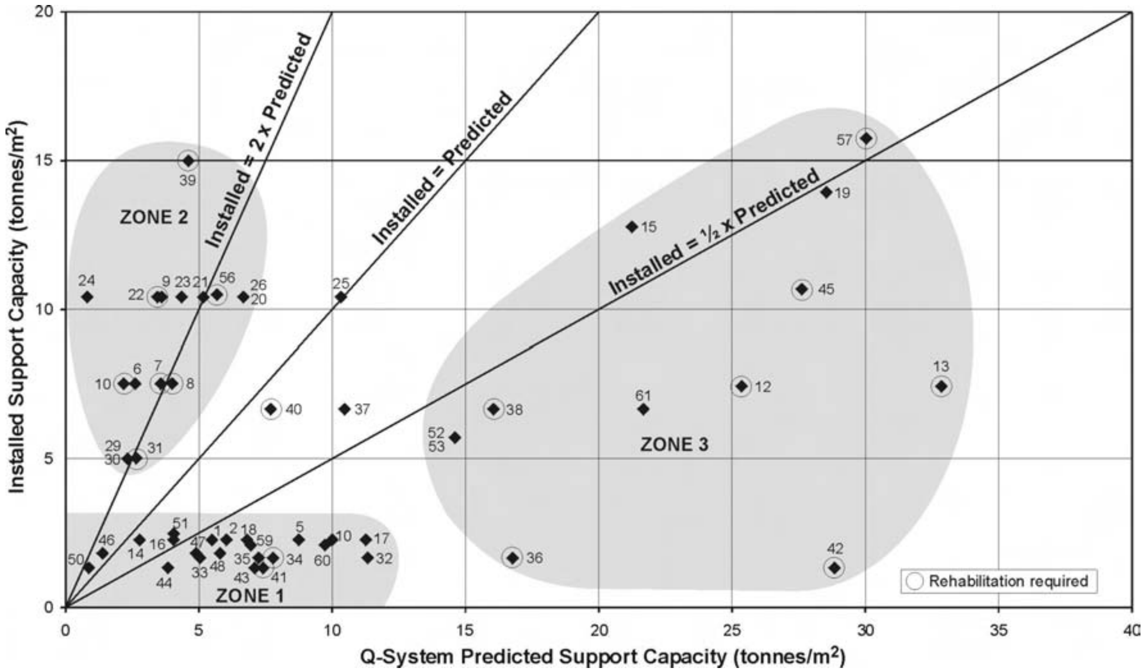

Figure 3.—Installed versus Q-predicted support capacities.

Installed Versus Predicted Support Capacity

Predicted support capacity, P, has been calculated in tonnes of required bolt capacity per square meter prior to bolt yield, using the following relationships involving Q, J_n, and J_r published by Barton et al. [1974]:

$$P = (20\ J_n^{1/2}\ Q^{-1/3})\ /\ 3\ J_r \text{ (for fewer than three joint sets, } J_n < 9) \quad (1)$$

$$P = (20\ Q^{-1/3})\ /\ J_r \text{ (for three or more joint sets, } J_n \geq 9) \quad (2)$$

Installed versus predicted support capacities are shown in 3 of the 15 surveyed mines (Figure 3). The cases where resupport was needed as part of a rehabilitation program have been highlighted.

It can be immediately recognized that only a few of the data points plot close to the line representing the condition where the predicted support requirements were matched by what the mine actually used. For ease of discussion, three general areas, or zones, have been delineated.

- Zone 1 contains those data points where the actual installed support capacity is less than 3 t/m^2 and the Q-system predicted support capacity was less than 12 t/m^2.
- Zone 2 contains data where the installed support is greater than predicted.
- Zone 3 contains data points where the installed support is less than predicted.

Zone 1

There are 23 data points within Zone 1, all of which have used friction anchors and 21 of which are sites shallower than 500 m depth. Only three sites plot close to the line, indicating that the predicted support capacity equals that actually used. Although the remaining 20 sites have predicted support capacity requirements up to five times the installed support capacity, only two of them required rehabilitation. This suggests that at depths of less than 500 m, local site experience should be used in conjunction with the Q-system to dimension ground support requirements for mines.

Zone 2

There are 16 points in Zone 2, 12 of which are located at depths >900 m; 3 of the 12 have required rehabilitation. SRF values used by the site technical staff are regarded as being low, but SRF values suggested by strength versus stress relationships do not move these data points much closer to the line where the installed support capacity equals that predicted by the Q-system. All 12 sites experienced high stresses, and some experienced seismic events.

The remaining four data points in Zone 2 occur at depths of 200–550 m, and all required rehabilitation. The SRF values at three of them have been underestimated, but only in one case is the revision sufficient to get the predicted support capacity within 50% of that actually used.

Zone 3

There are 12 data points in Zone 3 at depths ranging between 145 and 1,600 m, 7 of which required rehabilitation. This is not surprising at three sites where the installed support capacity was less than 25% of the predicted support capacity required.

Suggested Modifications to Q-system Investigations

In a number of the cases already discussed, both the predicted support requirement and the SRF seem to have been underestimated. While underestimation of the SRF does not seem to be the sole reason for underestimating support requirements, it is significant. Peck [2000] discusses the problem of determining the SRF of highly stressed, jointed rock. The authors strongly recommend that practitioners calculate the support requirement using Barton's 1974 equations (see Equations 1–2 above), particularly as friction anchors are not considered in the Grimstad and Barton 1993 chart.

RAISE-BORED SHAFTS

Working in vertical openings is recognized as being more hazardous than horizontal development. Thus, there has been a concerted effort in Australia to reduce miners' exposure to vertical openings [Minahan 1974].

If a raise-bored shaft can be successfully completed without damage to, or loss of, the in-hole equipment, the potential savings over conventional shaft sinking can be up to about 30%. Unfortunately, raise boring in Australia has not been universally successful, and failures can be very expensive in terms of lost equipment and delays to production. Not only have some raises collapsed during reaming, some of the 310-mm-diam pilot holes have also been lost. Methods of reliably assessing the unaided stability of raise-bored shafts are therefore required.

The McCracken and Stacey [1989] method of assessing geotechnical risk for large-diameter raise-bored shafts has been widely used in Australia. It was successfully applied in 1989 to the planned Airshaft No. 5 at Broken Hill. The method predicted severe instability if raise boring was attempted at the planned 6-m diameter. Some overbreak was even predicted at a raise-bore diameter of 1.8 m. The method gained significant credibility when the predicted overbreak occurred during reaming at 1.8 m diameter prior to enlargement to 6.7 m by V-moling [Bennet and de Bruin 1993].

McCracken and Stacey Method

McCracken and Stacey [1989] applied the principles of the Q-system [Barton et al. 1974] to the problem of assessing raise-bore stability following the collapse of a number of large-diameter shafts during raise boring. The Q-system had developed a relationship between Q and the Maximum Stable Unsupported Span (MSUS). Additionally, the Q-system had established the ESR to account for different degrees of allowable instability based on excavation service life and usage. McCracken and Stacey used these concepts to develop the relationship between Raise Rock Quality (Qr) and the maximum stable raise diameter (Figure 4). Qr is based on the Q value obtained using Kirsten's [1983] approach to determine the SRF, with further corrections to accommodate adversely oriented sets of discontinuities, weathering, and alteration.

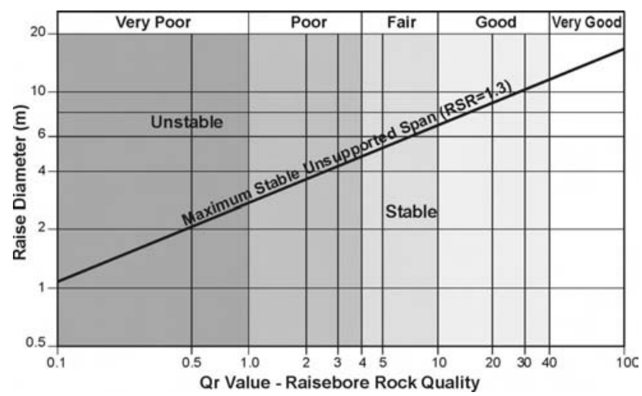

Figure 4.—Qr, raise diameter and stability (after McCracken and Stacey [1989]).

Kirsten [1983] developed a relationship between the SRF and the extent to which the rock is overstressed. His approach calculates an SRF value for the ground stress condition and another value for rock mass loosening and uses whichever SRF value is greater. Peck [2000] published similar SRF equations for Australian conditions.

Figure 4 demonstrates that the required Qr value for stability significantly increases with increasing raise diameter. While a 2-m raise is likely to be stable in poor-quality rock (Qr = 1 to 4), a 5-m raise requires fair-quality rock and a 6-m raise requires fair- to good-quality rock (Qr > 8). McCracken and Stacey also defined raise-bore rock quality in terms of block size (RQD/J_n) and low inter-block shear strength (J_r/J_a). Their paper drew attention to the fact that problems may be expected in large-diameter raises if the critical parameter values for RQD/J_n and J_r/J_a are poor or worse, using the guidelines they published.

Stability and Standup Time

Ideally, raises should be located in rock and sized so that they are permanently stable. Unfortunately, this is not always possible, and progressively larger raises are being bored as equipment is being improved.

While the Q database contains no shafts or raises, it does include data for walls of caverns. Bored shafts and raises with their circular cross-sections are inherently more stable than planar cavern walls. Different ESRs might also be more applicable to shaft walls (often progressively exposed, manually supported, and then permanently lined) than bored raises, which must often be permanently stable immediately after exposure.

Standup time is also important. Where a raise has intersected a marginally stable horizon, several weeks may elapse before it can be supported, e.g., by either installing a lining, manually bolting the raise walls, or remotely spraying fibercrete. In some cases, prereinforcement of marginally stable sections is also possible.

Unstable rock excavations without support will collapse in time, ranging from less than an hour to more than a year. The time-dependent behavior of unstable rock masses is complex and as yet poorly understood. The Q-system does not include any correlation between Q values and standup times.

Bieniawski [1993] published a correlation between the span of an opening, maximum standup time, and RMR value based on a study of a large number of mine openings and tunnels. Unfortunately, RMR values are not easily related to Q values, as not all of the same parameters are used to assess rock quality. For example, only Q gives a rating for the ground stress condition, while only RMR rates the persistence or continuity of the individual rock defects such as joints.

Bieniawski's chart suggests that to stand unsupported for 6 months, a 3-m span needs an RMR of at least 58 and a 6-m span requires an RMR of at least 64. As the RMR system rates good rock as having values between 61 and 80, fair to good rock is needed for 3- to 6-m spans to stand unsupported for 6 months.

Lower-bound Geotechnical Conditions

The lower-bound Qr value is a key geotechnical parameter in the McCracken and Stacey method of determining the maximum diameter at which a raise can be reamed without exceeding the acceptable probability of failure. Unfortunately, they did not define the logging intervals, which is perhaps relative to the intended raise diameter, over which the lower-bound Qr value should apply. For example, a 1-m-thick sheared or blocky zone might give a very low Qr value, but when it is included with 4 m of good-quality rock, the average may be greater than the lower-bound Qr. The orientation and thickness of the sheared or broken zone are also important. A 1-m-thick sheared zone may be of no consequence if it is shallow-dipping and confined between good-quality rock. Conversely, a thin, weak, and continuous steeply dipping structure within a poor zone may control significant unraveling.

Figure 5 shows two images from closed-circuit television monitoring of the walls of a recently completed and unlined raise. Figure 5A shows the relatively smooth raise walls in a section of the raise where no overbreak has occurred. Figure 5B shows the result of overbreak along joints.

Comparisons of preexcavation borehole logs with video camera inspections of completed raises has enabled estimates to be made of the minimum thickness of poor-quality rock in otherwise good-quality rock, which is needed to destabilize the walls of a raise.

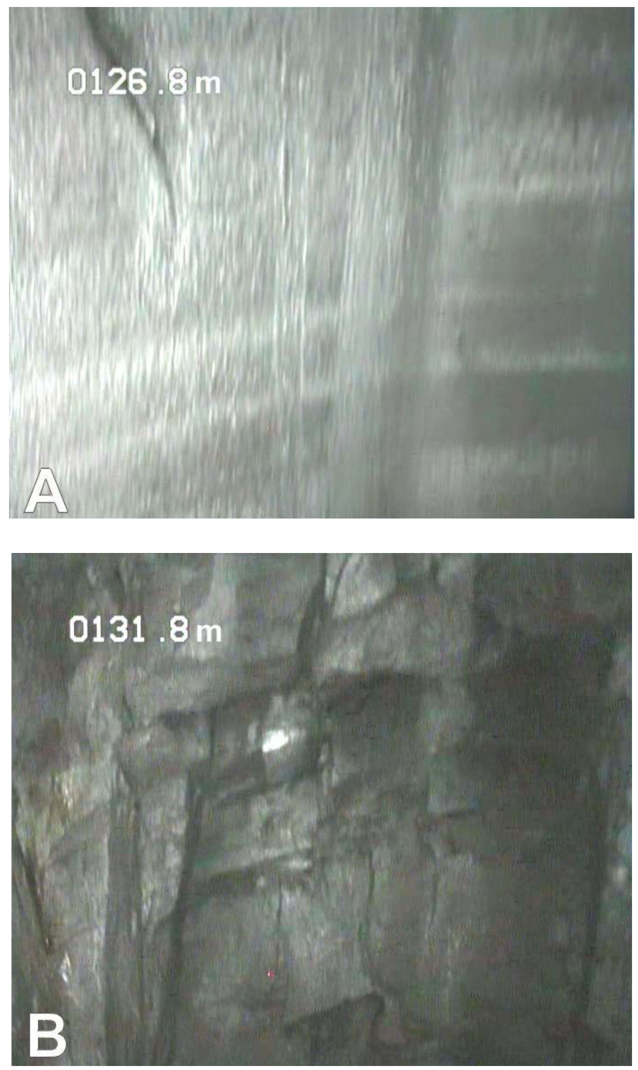

Figure 5.—No overbreak (A) versus overbreak (B) in a raise-bored shaft.

While thin shears and zones of blocky rock might produce some localized overbreak, zones of poor-quality rock need to be greater than 3 m to significantly impact the stability of raise walls. It is therefore recommended that core logging and analysis be done over lengths of about 1–1.5 m. Raise stability assessments should then use "rolling average" techniques to average rock quality over 3-m increments, i.e., to calculate lower-bound Qr values.

Stability Assessment

McCracken and Stacey stated that the preliminary geotechnical assessment should be aimed at determining the average and lower-bound geotechnical conditions: "The range and distribution of the raise-bore rock quality Qr, and the most important RQD/J_n and J_r/J_a parameters must be compared to the required minima for stability at the proposed shaft diameter." It is not sufficient to simply look at the variation of MSUS with depth.

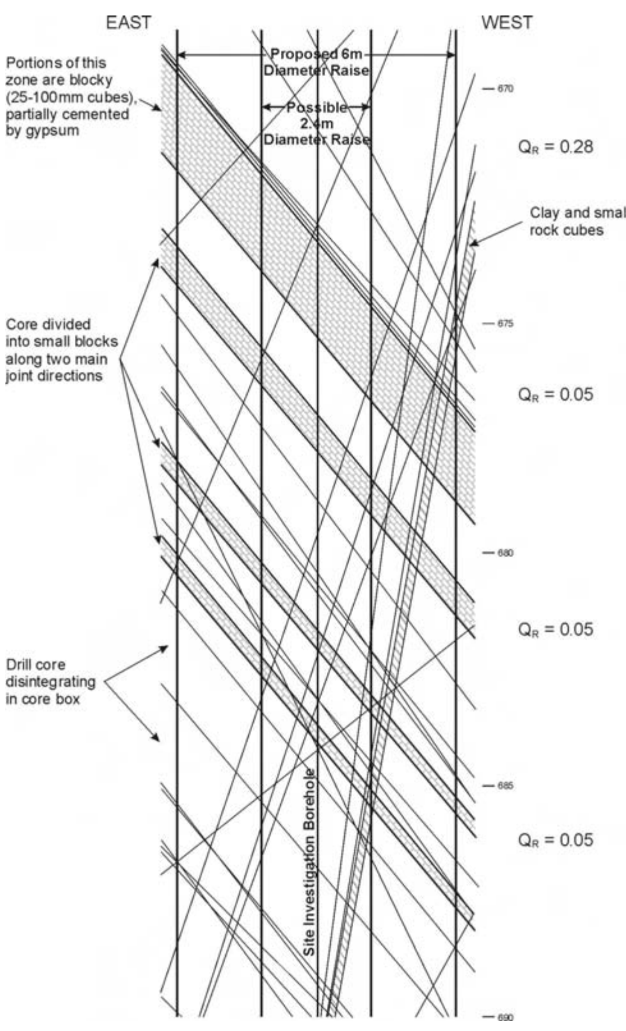

Figure 6.—Detailed stability analysis for a proposed raise.

"At the preliminary evaluation stage the risk should only be deemed 'acceptable' if the tunnelling quality is consistently indicated to significantly exceed (i.e., be in the next class up from), the required quality throughout its length." Their paper goes on to state that marginal cases occur where the indicated quality Qr either straddles the required value for stability or is not confidently known.

McCracken and Stacey also state: "In addition to simply assessing the range of predicted Qr values against those required, the rock mass properties and discontinuity orientations would be used as input to detailed stability analysis."

Figure 6 presents a detailed analysis for a proposed large-diameter raise where the apparent dip is shown for every discontinuity that had not been rehealed. The core was obtained from a diamond drill hole bored down the proposed alignment of the raise and was oriented using the metamorphic foliation, which was known to have a consistent orientation in that part of the mine.

This example is from the 1989 analysis for Airshaft No. 5 at Broken Hill. The small joint-block sizes within some structures, the numerous steeply dipping joints with low-friction coatings, such as talc, and the rapid deterioration of some of the core in the core boxes clearly indicated the potential to collapse if bored at 6 m diameter. Overbreak occurred in the interval shown in Figure 6 when it was raise-bored at a diameter of 1.8 m. However, this overbreak was not sufficient to cause general collapse prior to enlargement of the raise by V-moling and lining.

Presenting McCracken and Stacey Results

Figure 7 presents a typical plot of MSUS (expressed as a diameter) versus depth. Zones having small block size and low interblock shear strength have been identified. Two potentially unstable zones are identified, the first at the shaft collar (0–25 m) and the second between 60 and 70 m. Both were reinforced prior to raise boring. A 1-m-thick zone with an MSUS of only 2 m occurs at a depth of about 103 m. The Qr value in this zone is only 0.5, but the Qr values above and below it are of the order of 25. It was concluded that these zones would provide adequate stability for the thin weak layer. The raise was successfully bored at the planned 4-m diameter.

Highly Stressed Rock

In highly stressed rock, induced rock fracturing can occur in advance of the reaming head and in opposite walls of the completed raise. The generation of loose slabs at the cutting face can mean overbreak of large blocks into the cutters. Then high and irregular torque on the drilling string is possible during reaming.

Stacey and Harte [1989] described this failure mechanism for raises at depths in excess of 2 km in southern Africa. They analyzed spontaneous fracturing ahead of the

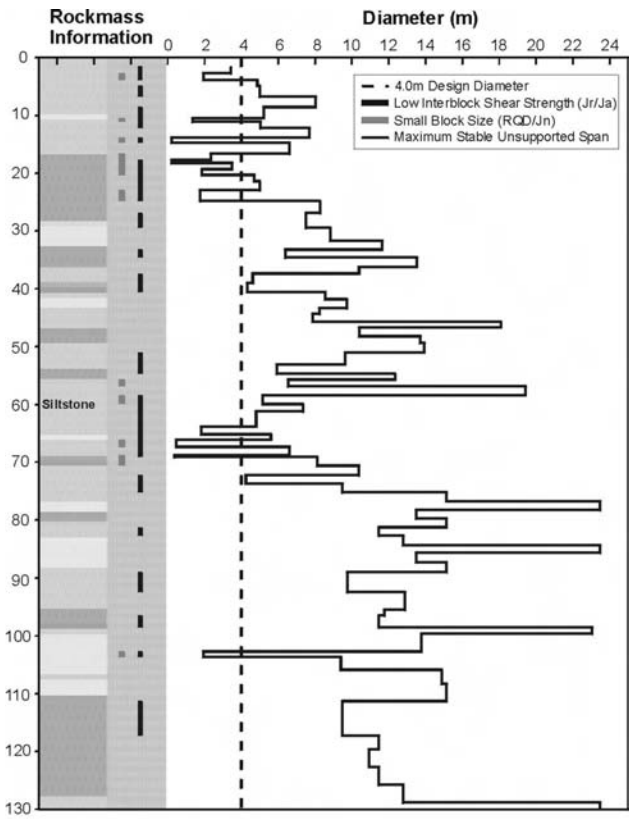

Figure 7.—Typical plot of MSUS (diameter) and critical rock quality.

face and derived a means of predicting its extent. O'Toole and Sidea [2005] demonstrated that significant fracturing was possible ahead of the raise-bore face at depths of 880 m in Australia.

High or deviatoric horizontal stresses can mean very high and low stresses around the raise wall. Both can assist loosening of rock masses by local shearing and dilation on joints. Deep high-stress fracturing and overbreak are possible where wall stresses exceed the strength of the rock mass. Rock mass strength can vary with rock type and blockiness, but it is typically half the strength of intact rock [Lee et al. 2001].

O'Toole and Sidea [2005] concluded: "[D]ue to the high risk of in-hole equipment damage and increased maintenance, the full costs associated with raise boring in highly stressed rock are likely to be significantly higher than reaming the equivalent strength rock in a low-stress environment."

The use of stress/strength-based SRF values in the McCracken and Stacey method and the consequential reduction in Qr values should alert geotechnical engineers to the existence of potential high-stress issues. It is now common for raise-boring contractors to torque-limit their machines. It may occasionally be necessary to lower the cutting head and remove large slabs of rock from the cutting head whenever high torque demand is reported.

Postconstruction videos of raise walls are also becoming increasing common in Australia to compare predicted and actual performance. Selected wall support may be necessary and can be provided by remotely spraying fibercrete.

Contrasting Face Conditions

When the raise-bore reaming head encounters a steeply dipping interface between rocks of contrasting strength, such as weak siltstone overlain by a steeply dipping strong sandstone, the head attempts to remain in the weaker material. This generates unbalanced forces that cause the head to tilt and the raise-bore drill rods to bend. Mechanical failure of the head or drill rods is possible, sometimes with the head plus rods falling to the bottom of the raise.

The McCracken and Stacey method does not provide any warning of this possibility. Other geotechnical investigations are required to complement Q-based analyses.

Australian Raise Performance Versus Predicted Qr

A database of Australia raise-boring experience has been compiled and is plotted in Figure 8. It comprises 47 data points of raise diameter, actual performance and component Q values for lower-bound Qr situations from 23 mine sites in Australia and Papua New Guinea. All of the raises plotted are known to at least one of the authors who also had access to the site investigation reports and borehole logs. For consistency, the lower-bound Qr values data presented in Figure 8 were determined by the authors using a "rolling average" of 3-m increments, as described above.

The following trends are illustrated in Figure 8:

- For lower-bound Qr values of less than 0.10, there is a high chance (9 in 10) of raise collapse or significant overbreak, irrespective of the proposed raise diameter.
- For raise diameters between 3 and 6 m and if the lower-bound Qr value is between 0.1 and 1.0, raise-bore performance ranges from stable to collapsed. A detailed stability analysis is recommended using the McCracken and Stacey method.
- For a raise diameter of less than 5 m and a lower-bound Qr value greater than 1.0, there is an excellent chance of constructing a stable raise (10 stable and 3 stable with support, out of 13). A detailed stability analysis should still be carried out if the rock structure rating (RSR) is greater than 1.3 for the desired raise diameter.

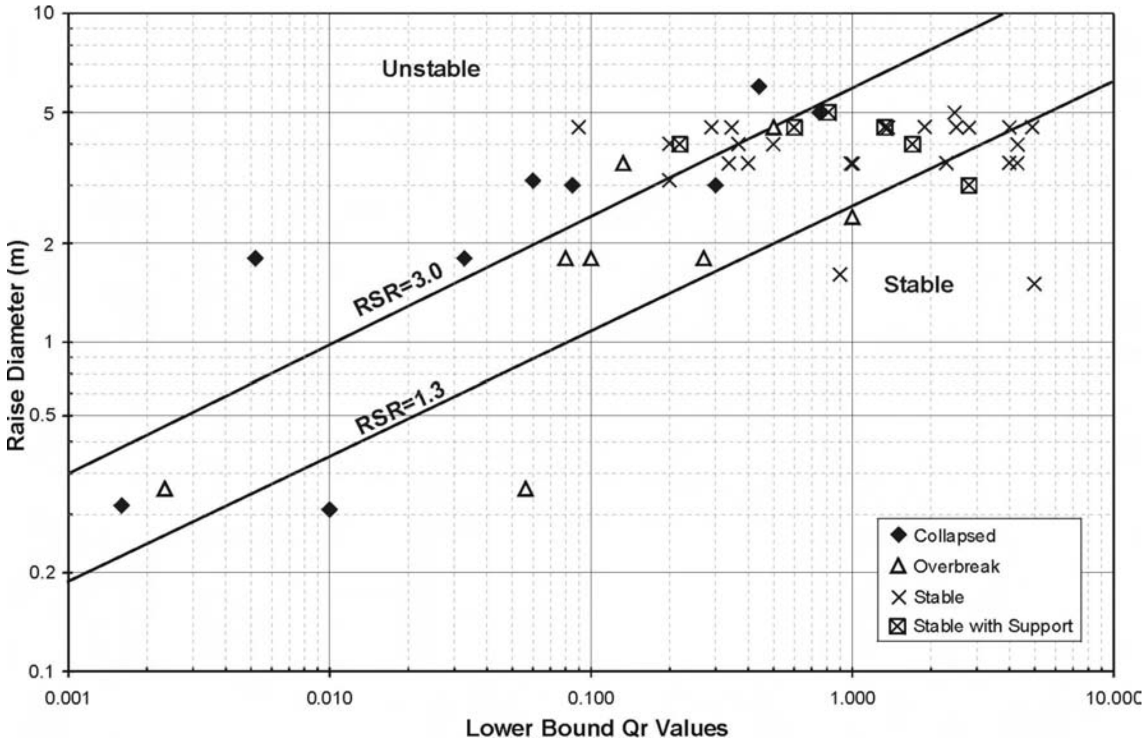

Figure 8.—Raise-bore diameter versus lower-bound Qr values.

- There are 11 data points on the unstable side of RSR = 3.0, including 5 collapsed raises. McCracken and Stacey considered there was a probability of failure of 1 in 4 for an RSR = 3.0.

The intermingling of collapsed and stable raises for Qr values between 0.05 and 1.0 and RSR values ≥2.0 demonstrates the need to acquire and closely consider additional geotechnical data for these cases. This was recommended by McCracken and Stacey where the proposed raise plotted on the unstable side of RSR = 1.3.

CONCLUSIONS

Despite the harsh geotechnical environment in many Australian underground metal mines, there is a reasonable correlation between actual ground performance and Q and Qr values. However, Q and Qr are not always conclusive if considered in isolation from other rock mass parameters. While significantly less support is being used in some areas than is indicated by the Q-system, there are other areas where much more support is required than the Q-system would indicate, e.g., deeper than 900 m below surface.

The calculated MSUS does not always indicate actual raise stability and additional investigation is required, particularly in cases of marginal stability. For these situations, greater emphasis should be given to the ratios RQD/J_n, J_r/J_a, and σ_c/σ_1 and the possibility of time-dependent behavior of some joint infillings, such as gypsum, chlorite, sericite, and talc.

Although there are several deficiencies for special cases, it is concluded that the Q-system is a suitable method of assessing rock mass conditions. It can be used to assess the likely stability of openings and the selection of ground support requirements, provided appropriate SRF values are used and other geotechnical parameters are considered in conjunction with Q and Qr. Local site experience is a valuable component of the process.

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