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Overbreak and underbreak in underground openings Part 2: causes and implications

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Summary

The newly developed light sectioning method has been used to investigate some of the causes and costs of overbreak and underbreak. Investigations at the Aquamilpa Hydroelectric Project in Mexico have shown decreased overbreak and increased underbreak as a result of increased rock quality and decreased explosive energy. A new measure of explosive energy, the 'perimeter powder factor' (PPF), has been defined and shown to be useful in the context of tunnel-wall rock damage. Tentative results indicate that explosive energy (PPF) may be a more important factor in producing underbreak, whereas rock quality may be a greater factor in producing overbreak. A site-specific equation is given for predicting overbreak or underbreak as a function of rock quality and explosive energy, with an evaluation of the cost of underbreak and overbreak.

Keywords: overbreak, underbreak, digital image processing, tunnelling, blasting assessment, tunnel profiling, RMR, Q, powder factor, perimeter powder factor

Introduction

The light sectioning method of measuring overbreak and underbreak, presented in Part 1 of this paper, provides the opportunity to measure and study the causes and consequences of overbreak and underbreak.

The two primary causes of overbreak or underbreak are geological conditions and blasting factors. In this paper, their effects on overbreak and underbreak are discussed.

Review of factors influencing overbreak and underbreak

The factors influencing overbreak and underbreak can conveniently be grouped into two categories, the make-up and character of the ground (geological factors) and the nature of the excavation operation. Although the relationships between blasting, geological conditions, overbreak and underbreak are well known intuitively, there has been no systematic and quantitative investigation of them.

Geological factors

Jointing is a major factor influencing overbreak and underbreak in blasting operations. The orientation, spacing (block size), and filling (or alteration) of joints all contribute. Other geological factors include the intact strength of the rock, and *in-situ* ground-stress conditions.

Joint orientation. The orientation of joints relative to the perimeter of the excavation is one of the major factors influencing overbreak and underbreak in blasting operations.

Overbreak and underbreak are typically less where joints and faults strike nearly perpendicular to the tunnel axis, and greater when joints are parallel to the tunnel axis. When joints run closely parallel to a proposed line of tunnelling, the rock tends to break along the joints rather than along the lines intended by the designer. Overbreak can be expected to increase with the combination of two joint sets and increases dramatically with the intersection of three or more near orthogonal joint sets (Martna, 1986).

A large amount of overbreak usually occurs in tunnels when the rock is layered horizontally with vertical joints (Isaac and Bubb, 1981). Horizontal bedding in tunnels usually results in the crown of a tunnel becoming almost flat (Nelson, 1983; Ontario Hydro, 1983).

Inclined strata or schistosity have a tendency to change the profile of the opening, making an asymmetrical enlargement with a high production of overbreak and underbreak (Stini, 1950; Cecil, 1975).

Joint spacing (block size). Probably the most important characteristic of jointing with respect to overbreak and underbreak is the joint spacing (block size). Intensely jointed rock with small blocks is difficult to blast without excessive overbreak, whereas massive rock is much easier to excavate in neat lines. Rock stability also depends on how many sets (types) of weaknesses are present.

Clay fillings and alteration. Martna (1986) discusses the interaction of weak and weathered rock. Clay-filled joints and faults containing gouge are particularly likely to contribute to difficulties in wall control. The explosive energy required to break the adjacent strong intact rock will overwhelm the weaker filling materials. Even a properly designed blast is likely to dislodge wedges bracketed by these planes of extreme weakness. In one case, as much as 20 m of overbreak was caused by thin clay seams along shale bedding (Arnold *et al.*, 1972).

If a fault zone is wide and filled with weak gouge such as clay or graphite, much damage may be expected. If it is narrow, with stable filing materials – such as brecciated rock or quartz veining – the overbreak expected will depend on the orientation of the fault respective to the axis of the tunnel.

The potential swellability of clays is a factor in long-term stability, and its influence on tunnel instability is given by Brekke and Selmer-Olsen (1965).

Rock strength. Under some circumstances, rock strength can be a significant factor. In general, the higher the strength of the rock, the more likely that the breaks caused by blasting will propagate more or less along pre-existing joints along the path of maximum shear stress (Müller, 1959; Hagan, 1983). The contrast between intact rock and joint strengths is the determining factor.

Ground-stress effect. The preferred stress direction and the magnitude of the principle stress ratio are also important factors contributing to overbreak, especially when the stresses approach the strength of the jointed rock. An example of overbreak under the influence of rock stress is given by Martna (1972). In a 190 m^2 cross-section headrace tunnel, the horizontal stress was initially high, approaching the 200 MPa strength of the intact quartzite. A stress magnification of between 2 and 3 accompanies tunnel excavation. Even massive (unjointed) rock can develop 'breakout' along the principal stress direction under these conditions. This condition with the principal stress direction (10°–20° to the horizontal) associated with joints dipping steeply towards the tunnel axis produced a large amount of overbreak.

Blasting factors

Overbreak and underbreak can also be the result of poor blast design and/or execution. Poor design of the perimeter part of the blast is most likely to result in overbreak or underbreak, but the central part of the blast (the 'cut') may also cause perimeter damage, although usually to a lesser extent. Even a well-designed blast can give poor results if poorly implemented. Particularly important are the accurate marking out and drilling of blastholes. Much overbreak is caused by blastholes that diverge or converge, and holes that fail to detonate on time and in sequence.

Explosive type and powder factor. In general, the greater the explosive energy in a blast, the greater the potential for overbreak. High detonation velocities, high explosive densities, high weight strength, and large cartridge diameters all lead to increased rock breakage and perimeter damage. Conversely, low velocities densities, strengths, and small cartridge sizes result in decreased fragmentation and may leave underbreak 'tights' (zones of underbreak).

The powder factor or specific charge (SC) is the explosive weight in ANFO equivalent divided by the volume of rock blasted (Kuzyk *et al.,* 1986). Generally, higher specific charges will tend to produce overbreak and lower ones underbreak. A greater SC is employed in small tunnels because the high degree of confinement limits the blasthole length and alignment, and therefore a larger amount of overbreak is expected (Langefors and Kihlström, 1967).

Charge concentration. Charge concentration or charge weight is the weight of explosive per length of blasthole. Different values of charge weight are employed in a blast depending on the location of the blastholes. Proper choice of charge concentration results in the appropriate blasthole pressure required for adequate rock breakage with the minimum vibrations and adequate throw (Thompson *et al.*, 1979; Clark, 1981). Usually, the last two rows of blastholes from the perimeter are producers of overbreak if the blastholes have been charged heavily (Holmberg, 1979).

The physical characteristics of loading, the 'stemming' or 'amount of collar' also modify the effect of the charge concentration. Too long a collar causes excessive ground vibrations (Otuonye *et al.*, 1983), and therefore potential overbreak or underbreak and poor fragmentation. Too short a collar results in premature loss of confinement and excessive airblast.

Delay timing. Delay is the time between the detonation of successive blastholes. Proper delay allows enough time for the rock to be broken, fragmented, and displaced. If delays are too short or improperly sequenced, then the rock is not displaced adequately, and poor breakage and underbreak result. Excessive delays result in vibration and potential overbreak. Delays used for the perimeter blastholes, which most affect the final profile, should be as small as possible to obtain a good smooth effect and a reduction in overbreak. This condition can best be obtained using the same delay number in all the perimeter blastholes (Rustan, 1990).

Perimeter blasthole pattern. The drilling pattern in the perimeter is designed in terms of spacing and burden. Swedish experience in wall control has defined ideal spacing-to-burden ratios for both smooth wall blasting and presplitting techniques (Holmberg, 1983). Too small a spacing and/or burden will create overbreak, too large will create underbreak (Hagan, 1983).

The amount of overbreak can locally be greater if the charge weight is heavy, and the spacing and/or burden in the bottom of deviated blastholes is smaller. Large amounts of underbreak may be found if the charge has been poorly packed and the spacing and/or burden in the bottom of deviated blastholes is larger than designed.

Drilling deviation. Drilling deviation may be considered as one of the most common sources of the overbreak, underbreak, and backbreak. There are two types of deviation: in collaring and in drilling direction.

Deviation in collaring, the incorrect positioning of the drill bit at start of the blasthole, is caused by carelessness, inadequate marking technique, or local irregularities on the rock surface (Thompson *et al.*, 1979). In tunnel blasting, the unavoidable deviation needed to provide space for drilling operation is called look-out (Olofsson, 1988). If the look-out is uniformly too large, overbreak is automatically produced; if too small, underbreak may be produced.

Avoidable deviation in drilling direction is caused either by carelessness, or because the drill bit is deflected by geological features. The deviation is generally non-uniform and gives a reduction of spacing and burden in some zones of the perimeter, and an increase in others. Underbreak may occur in the zones of large spacing and burden and overbreak where the holes are more closely spaced.

Blasthole length and diameter. Long blastholes (in the range 4.5 m to 5.5 m), may produce more overbreak because condition of greater confinement call for an increase in powder factor (Hagan, 1983). Long blastholes (especially the perimeter and cut holes) require a greater degree of drilling control to avoid larger deviations. An increase in blasthole diameter is associated with overbreak, poor fragmentation, and increased cost of loading. A reduction is associated with better tunnel profiles, but with increased costs for drilling and charging.

Large hole cut. The large hole cut is the arrangement of a series of parallel blastholes, around one or several empty larger diameter holes, in the center of the blast. The available empty volume is an important consideration for an adequate flow of blasted material, to reduce overbreak and ground vibrations, and allows an increase the length in the round of blasting (Abel, 1982). Satisfactory release must be provided so that the perimeter blastholes, when detonated, are able to expel rock towards a previously created free face.

Influence of rock quality and blast design

Introduction

Investigations relating overbreak and underbreak to rock mass quality and blast design were conducted in rhyolites and tuffs in the Diversion Tunnel No. 2 at the Aquamilpa Hydroelectric Project in Nayarit, Mexico (Fig. 1).

The tunnel was excavated in two stages; first a semi-circular top heading 16 m diameter and 8 m high, and second a rectangular 8 m by 16 m bench system. In poor ground where stability problems were encountered or expected, the semicircular top heading was driven at a half face. All overbreak and underbreak measurements and analyses were conducted on the top heading of the tunnel (Fig. 2). Of the 92 measured sections, 39 were at full face, and the remaining 53 at half face.

Rock quality at each section was measured in terms of three rock-mass classifications: rock quality designation (RQD) (Deere, 1964), rock mass rating (RMR) (Bieniawski, 1990), and the Q system (Barton *et al.*, 1974). The Q system gave the closest correlations with

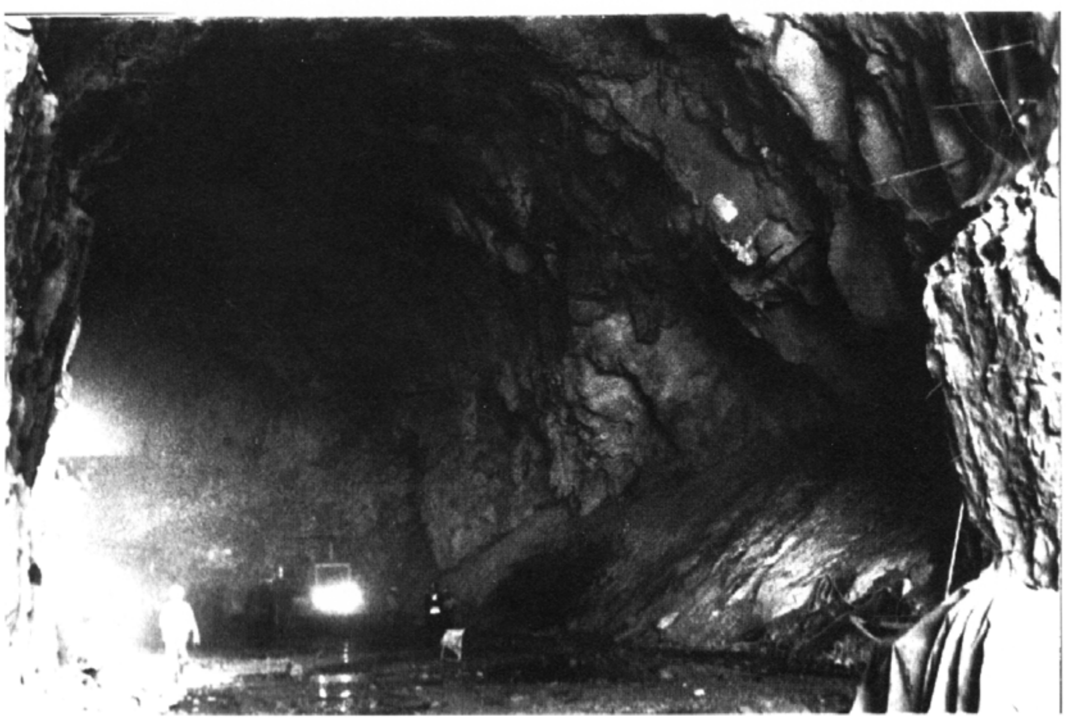

Fig. 1. Massive overbreak in the Aquamilpa diversion tunnel

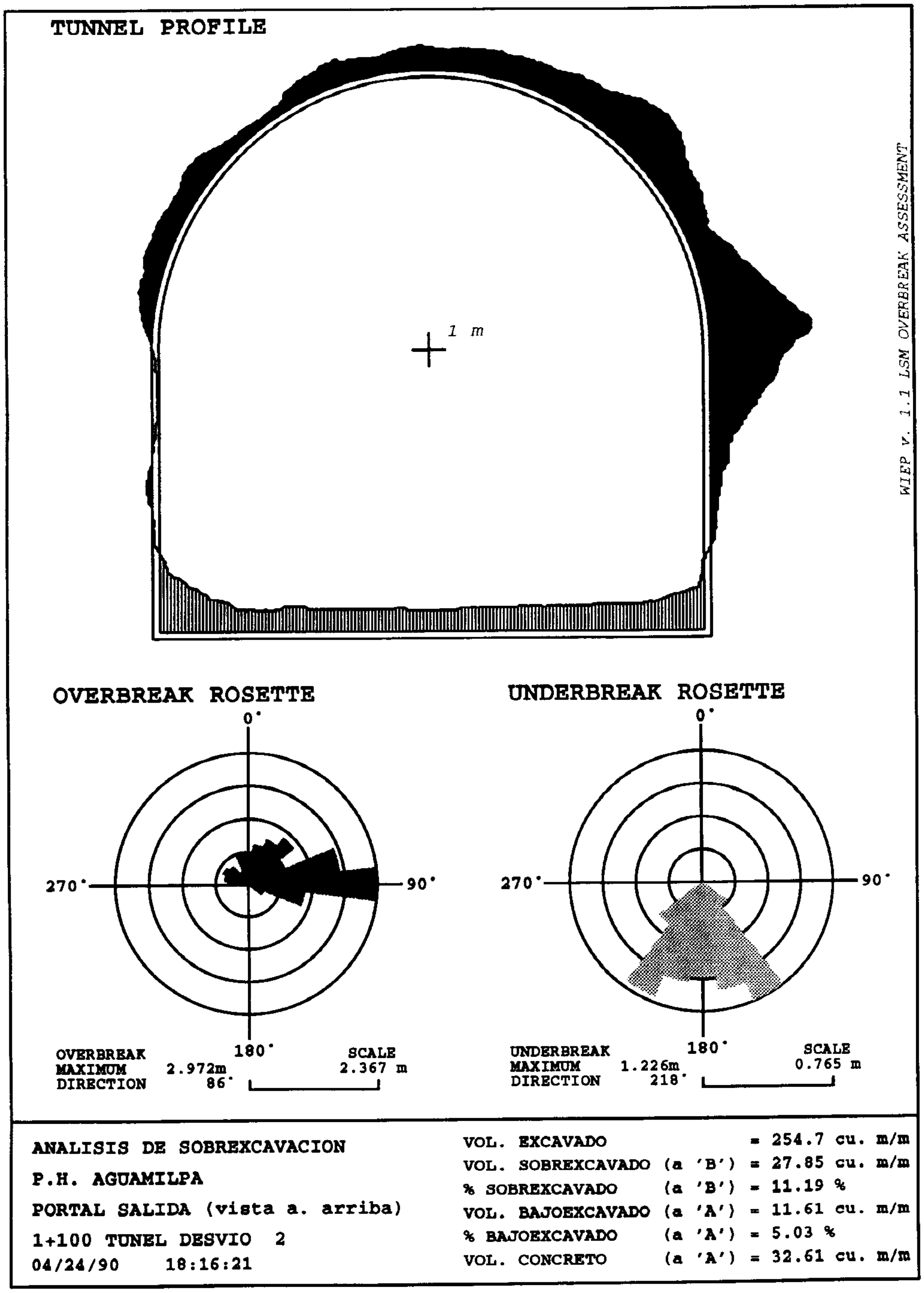

Fig. 2. Analysis of overbreak and underbreak in a section of the diversion tunnel, showing a vertical tunnel misalignment, and backbreak along jointing planes

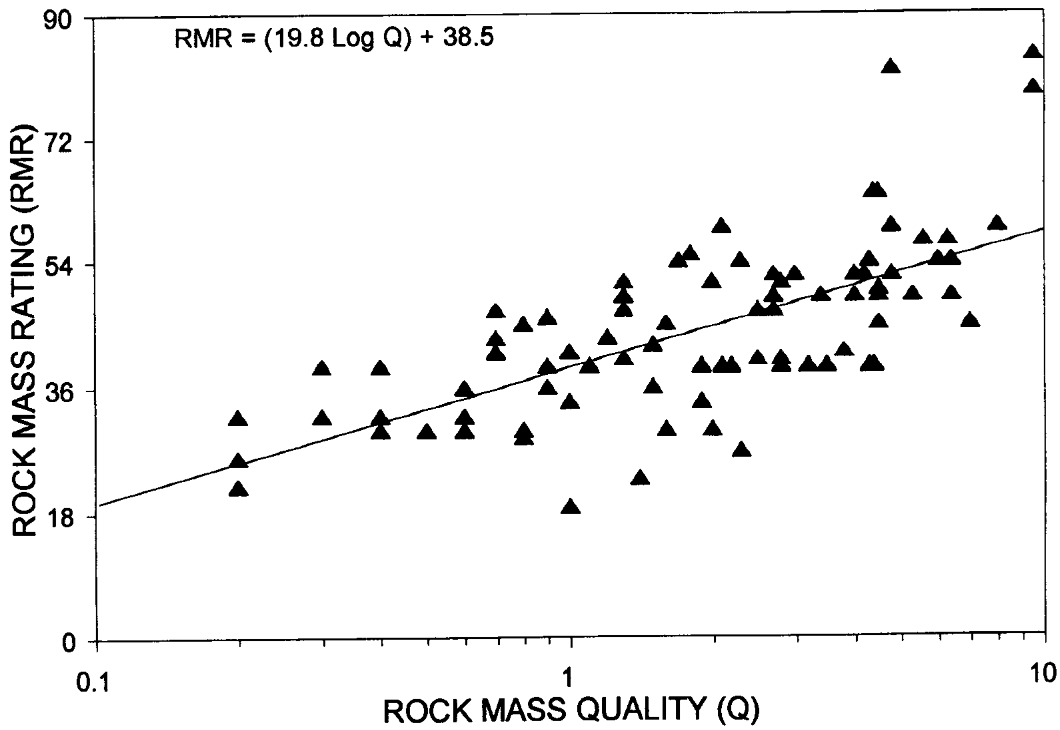

Fig. 3. Correlation between rock mass rating (RMR) and rock mass quality

overbreak and underbreak and only those results will be discussed in this paper. Figure 3 shows that RMR and Q can be correlated. This is in agreement with similar equations found by other researchers (Bieniawski, 1973; Rutledge, 1978; Kaiser *et al.*, 1986). The blasting design was quantified by a single parameter 'perimeter powder factor' (PPF) defined below.

Rock mass quality (Q)

The Q system determines the rock mass quality as a function of six geological parameters: rock quality designation (RQD), number of joint sets (J_n), joint roughness number (J_r), joint alteration number (J_a), water reduction factor (J_w), and stress reduction factor (SRF).

A value is measured or assigned for each parameter, and the rock mass quality Q is obtained from the following equation:

$$Q = \left(\frac{\mathrm{RQD}}{J_n}\right) \times \left(\frac{J_r}{J_a}\right) \times \left(\frac{J_w}{\mathrm{SRF}}\right) \quad (1)$$

RQD is defined as the percentage of drill core recovered in intact lengths of 100 mm or more (Deere, 1964). It was calculated from spacing measurements along tunnel walls by the application of the Priest and Hudson (1981) formulation:

$$\mathrm{RQD} = \left[100\ \mathrm{e}^{(-0.1\ \lambda)}\right] \times [0.1\ \lambda + 1] \quad (2)$$

where λ is the number of joints per metre.

The other parameters were selected using the Barton classification. These were:

the number of joint sets (J_n), ranged between 2 and 3 plus random joints, with assigned values between 4 and 12;

the joint roughness number (J_r) was estimated for the weakest joint set. Values ranged between 1 and 4, from smooth planar to rough undulating;

the joint alternation number (J_a) was also estimated for the weakest or most adversely orientated joint surface. Values ranged between 4 and 8, corresponding to clay fillings;

joint water reduction factor (J_w) was estimated as 1.0. This tunnel was almost completely dry;

the stress reduction factor (SRF) for competent rock depends on the ratio of the compressive strength of the rock (s_c) to the estimated maximum stress at the excavation depth (s_1). Values ranged between 2.5 and 5, indicating a massive rock. The (s_c) value was obtained at each analysed section by using the mean values of the Schmidt hammer rebound and rock density in the correlation chart of Miller's classification (1965). The Schmidt tests were performed according to the methodology described by the International Society of Rock Mechanics (1981).

The values of Q were calculated using a expert system called Classex developed at the University of Waterloo (Butler, 1990).

Analytical results. Analyses of the measurements reveal a relationship between underbreak and Q and between overbreak and Q (Fig. 4), both with statistically significant correlations. A linear inverse relationship was found between overbreak and log Q with a Pearson product-moment correlation coefficient of 0.831. A linear relationship was found between underbreak and log Q; it had more scatter, and a corresponding lower correlation coefficient of 0.594. These results confirm that as the rock quality improves, the amount of overbreak tends to decrease while that of underbreak tends to increase.

Perimeter powder factor (PPF)

The conventional 'powder factor' is an average measure of energy and is defined as the explosive weight divided by the volume of rock blasted – the higher the powder factor, the greater the damage to the rock. However, powder factor is not a very satisfactory parameter when evaluating wall rock damage, because it averages out the explosives used in the entire tunnel heading. Clearly, wall damage is more closely related to the explosives near the perimeter than to those near the center of the tunnel (Holmberg, 1979).

An alternative factor, 'perimeter powder factor' (PPF) is defined as the weight of explosives in the perimeter blastholes and the next row in, divided by the volume of rock within this annulus, ignoring the 'lifters' in the invert. The Aguamilpa studies (shown

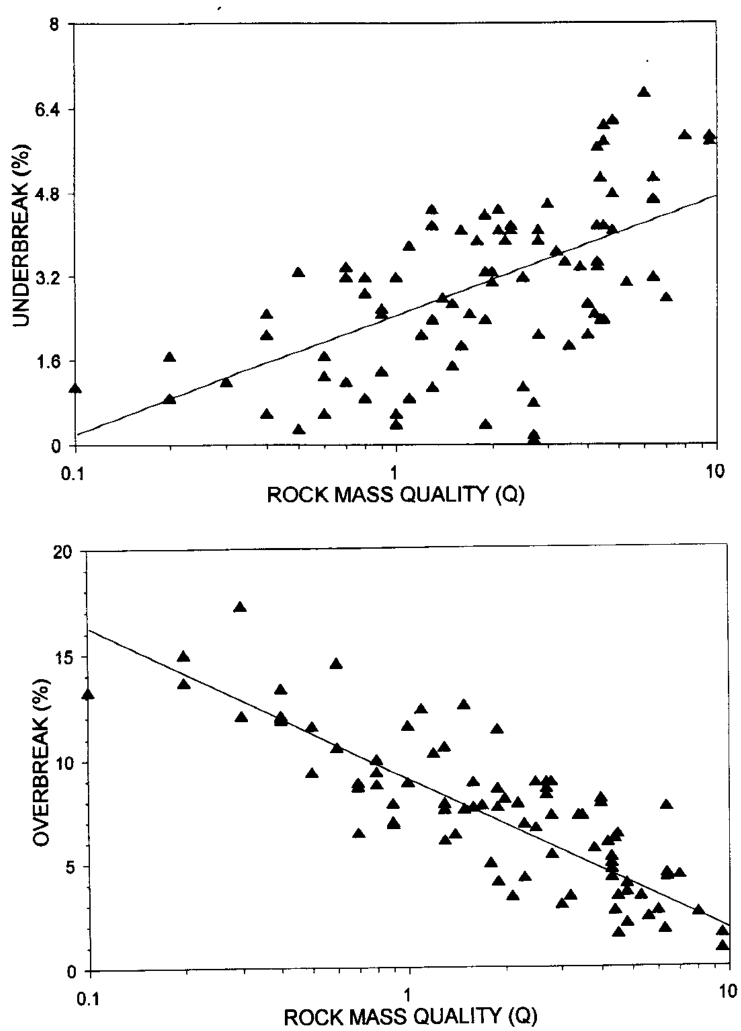

Fig. 4. (a) Underbreak and (b) overbreak in relation to rock mass quality (Q)

below) reveal that PPF is a sensitive measure of the quality of blast design in the context of wall damage. PPF was calculated from data recorded during drilling of blasthole length, diameter and spacing of perimeter holes and burden between the perimeter and the next inward row; and during loading of total explosive charge in the perimeter blastholes; explosive charge distribution along the perimeter holes and characteristics of the explosive used.

Analytical results. Analyses of the measurements reveal a relationship between underbreak and PPF and between overbreak and PPF (Fig. 5), both with statistically significant correlations. A linear relationship was found between overbreak and PPF with a correlation coefficient of 0.637. A linear inverse relationship was found between underbreak and PPF; it had slightly less scatter, and a correspondingly higher correlation coefficient of 0.730. These results confirm that as PPF is increased (more explosive

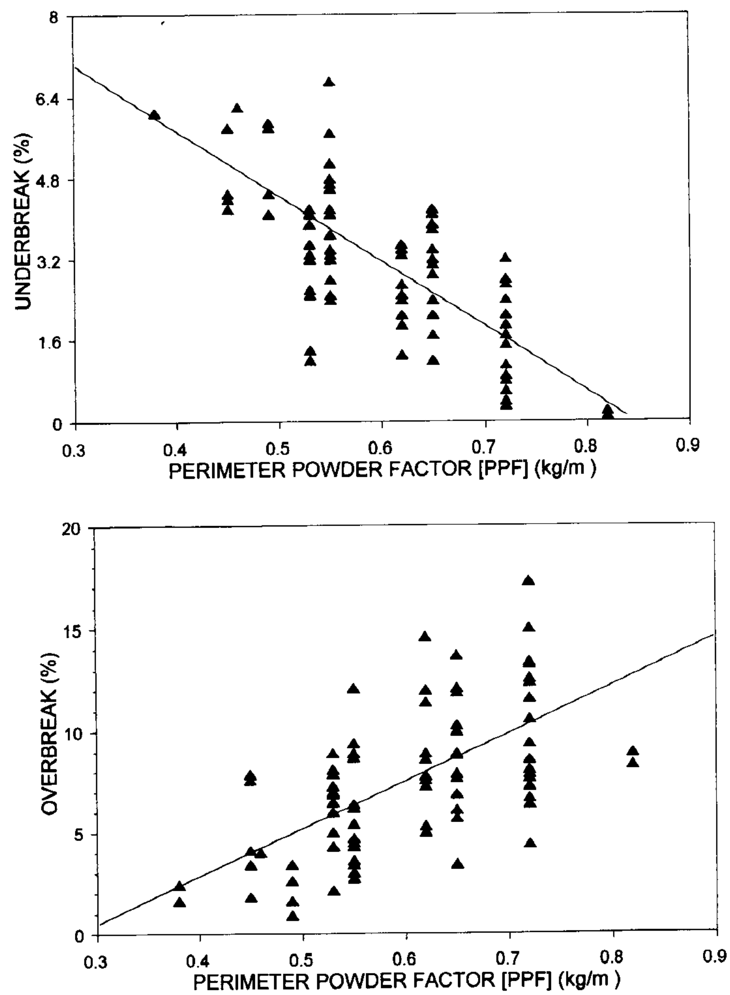

Fig. 5. (a) Underbreak and (b) overbreak in relation to explosive energy (PPF)

energy at the tunnel perimeter) the amount of underbreak tends to decrease, while that of overbreak tends to increase.

Rock mass quality (Q) and perimeter powder factor (PPF)

Because overbreak or underbreak has been shown to be a function of at least two independent variables (Q and PPF), a more advanced model can be formulated. Graphically, it can be shown, for example, that the relationship between overbreak and PPF changes when the values of Q are divided into five classes (Fig. 6). Statistically, this can be done with multiple regression analysis, which is more sophisticated, because it does not require one of the independent variables to be classified, and can be done for more than two independent variables. This can also be shown graphically (Fig. 7).

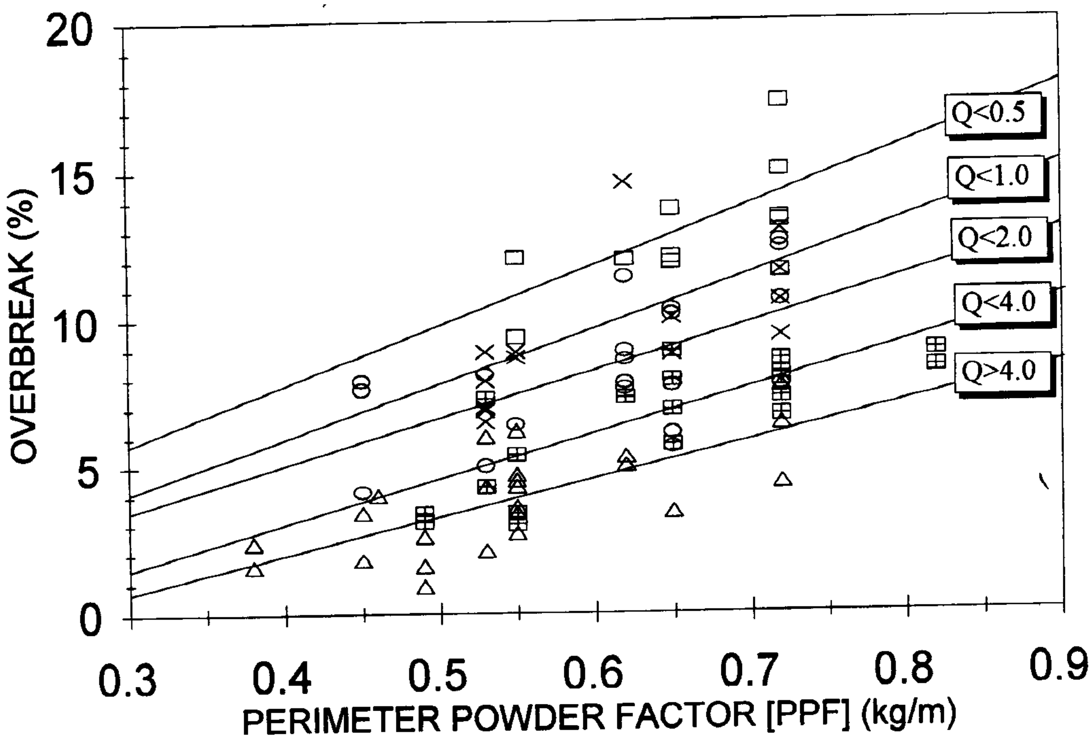

Fig. 6. Overbreak in relation to explosive energy (PPF) as a function of rock mass quality (Q)

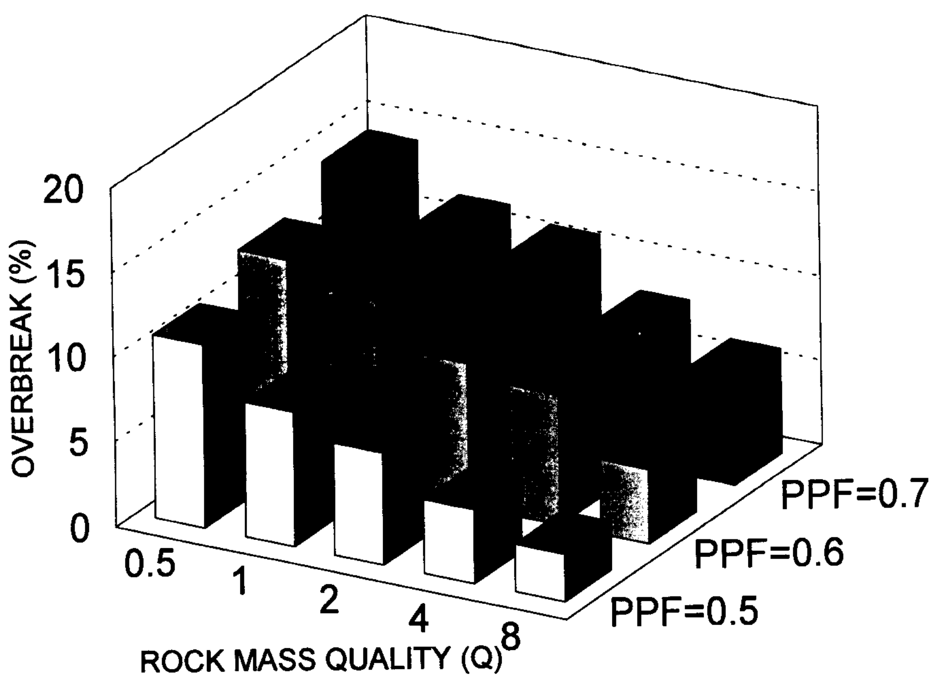

Fig. 7. Overbreak in relation to explosive energy (PPF) and rock mass quality (Q)

Multiple regression results. Analysis of the measurements reveal a relationship between underbreak, PPF and log Q, and between overbreak, PPF and log Q, both with statistically significant correlations. The relationship between overbreak, PPF and log Q was found to have a correlation coefficient of 0.913, and the relationship between underbreak, PPF and log Q had a correlation coefficient of 0.860.

Testing the regression coefficients showed that they were all highly significant. For the overbreak analysis, log Q was slightly more significant than PPF, while for the underbreak analysis, PPF was slightly more significant that log Q. The following two regression equations predict underbreak and overbreak under these particular circumstances.

$$\text{Overbreak}(\%) = -0.12 + 15.07 * \text{PPF} - 2.55 * \log(\text{Q}) \tag{3}$$

$$\text{Underbreak}(\%) = 9.33 - 11.14 * \text{PPF} + 0.72 * \log(\text{Q}) \tag{4}$$

Discussion. These results show that a greater degree of correlation is found when both PPF and log Q are considered simultaneously. They also indicate that rock quality may be slightly more influential in causing overbreak, and blasting parameters slightly more influential in causing underbreak. The predictive equations given are site specific, but can readily be calibrated to suit other projects, where the rock conditions differ from those at Aquamilpa. Trends such as those reported here are likely to be similar at other sites, although the individual coefficients may vary to some extent.

Cost of overbreak and underbreak

Introduction

Because overbreak and underbreak can be predicted in terms of rock mass quality (Q) and controlled to an extent by PPF, the next logical step is to optimize the blast design. The design may be adjusted in response to changes in rock conditions, to prevent unnecessary wall damage, to avoid underbreak to minimize costs, and to maintain the greatest possible rate of advance.

Ideally, both underbreak and overbreak would be eliminated; but in reality reducing overbreak will in all likelihood result in greater underbreak, and vice versa. The two must be optimized. The best method of quantifying optimal amounts of overbreak and underbreak is to evaluate the cost of both. In addition the cost of support can be added.

Methodology

The cost of overbreak at Aquamilpa was taken as the cost of concrete needed to replace the overexcavated rock. The cost of underbreak was taken as the cost of the additional time, labour, and explosives needed to remove underbreak, plus the additional cost of secondary overbreak caused by secondary blasting. The cost of support was taken as the additional rock reinforcement and shotcreting. A exponential curve fit model was applied to all three sets of data, and the total cost was the sum of the individual costs (Fig. 8).

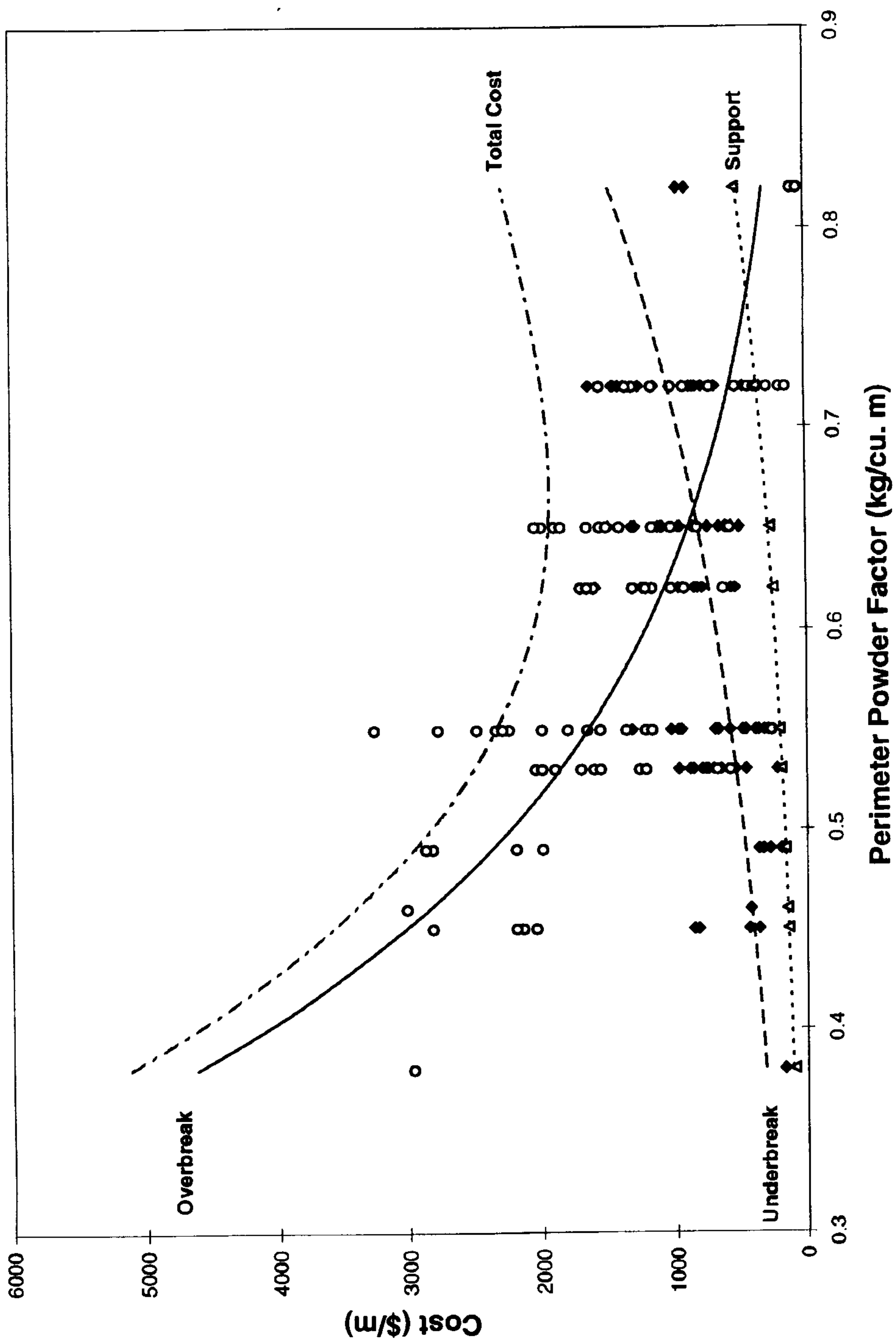

Fig. 8. Optimization of explosive energy (PPF) in terms of the cost of overbreak, underbreak, and support

Results

The minimum cost of underbreak and overbreak for the Aquamilpa project was found at a PPF of between 0.65 and 0.70 kg/m^3 (Fig. 8). This result is clearly site specific, depending on the variation of rock quality along the length of this excavation. In other projects, a broader range of optimum PPF values may apply, depending on rock conditions and the cost of labour.

Conclusions

The following conclusions can be drawn from a study of the Aquamilpa Hydroelectric Project, Diversion Tunnel No. 2. The technique of tunnel profiling using the light sectioning method has been tested and found to work well.

The perimeter powder factor (PPF) was defined and shown to be a useful predictor of tunnel-wall rock damage.

Increasing the explosive energy (PPF) reduces underbreak and increases overbreak.

As rock quality deteriorates, overbreak increases and underbreak decreases.

Predictive equations for overbreak or underbreak are improved when PPF and Q are considered simultaneously.

There is tentative evidence that rock quality may be slightly more influential in causing overbreak, and that explosive energy might be slightly more influential in causing underbreak.

Costs can be optimized by selecting an appropriate PPF.

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