METHODS OF ASSESSMENT AND MONITORING OF THE EFFECTS OF GAS PRESSURES ON STABILITY OF ROCK CUTS DUE TO BLASTING IN THE NEAR-FIELD

GEO REPORT No. 100

Blastronics Pty Ltd

GEOTECHNICAL ENGINEERING OFFICE
CIVIL ENGINEERING DEPARTMENT
THE GOVERNMENT OF THE HONG KONG
SPECIAL ADMINISTRATIVE REGION

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PREFACE

In keeping with our policy of releasing information which may be of general interest to the geotechnical profession and the public, we make available selected internal reports in a series of publications termed the GEO Report series. A charge is made to cover the cost of printing.

The Geotechnical Engineering Office also publishes guidance documents as GEO Publications. These publications and the GEO Reports may be obtained from the Government's Information Services Department. Information on how to purchase these documents is given on the last page of this report.

R.K.S. Chan
Head, Geotechnical Engineering Office
March 2000

FOREWORD

This GEO Report presents a succinct review of the current state of knowledge on gas pressure effects on the stability of rock cuts due to blasting. The review was carried out by Dr Cameron K McKenzie of Blastronics Pty Ltd for the Geotechnical Engineering Office. Mr B N Leung and Mr S C Leung of the Special Projects Division administered the consultancy and reviewed the Report.

The main objectives of this study are: (1) to summarise the state-of-the-art methods for assessment and monitoring of the effects of gas pressure on stability of rock cuts and excavations due to blasting within close distances, (2) to determine, if possible, the extent of no-blast zones to be imposed, and (3) to recommend practical measures to be carried out, for eliminating adverse blasting effects on nearby rock cuts and excavation under the Hong Kong geological conditions and urban environment.

The study has included a comprehensive search and review of relevant papers published in the international literature dealing with gas pressure from blasting and a detailed examination of five cases of field gas pressure measurement. It was found that none of the published papers addresses the effects of gas pressure on rock slope stability. The report concludes that specification of safe no-blast zones around excavations will continue to have to be largely based on engineering judgement and experience, due to the shortage of reliable field data and lack of reliable methodology for assessment. The information presented in this Report could serve as a useful reference.

P.L.R. Pang
Chief Geotechnical Engineer/Special Projects

Methods of Assessment and Monitoring of the Effects of Gas Pressures on Stability of Rock Cuts Due to Blasting in the Near-field

EXECUTIVE SUMMARY

BACKGROUND

Following a blast-induced rock failure along Sau Mau Ping Road, current procedures for controlling slope stability during rock excavation are being reviewed. Back analysis of the incident suggests a possible new mechanism of failure through the action of high pressure gases from the blastholes themselves.

This review has been undertaken to establish the current level of international recognition of the phenomenon of gas penetration through rock mass fractures, and its influence on slope stability. The study has included a review of published technical literature, as well as personal contact with a range of research groups around the world.

CONCLUSIONS

Clear evidence of explosion gas penetration through joints and fractures in rock masses has been established by a small number of blasting research groups around the world. Field studies have been limited to Australia and Sweden.

Fracture dilation, possibly caused simply by vibration and resulting in negative pressures, has also been commonly observed behind blast patterns, and may be as significant as, or more significant than, positive gas pressures associated with gas flows.

Gas penetration is characterised by vertical swell in benches around blasthole patterns. It would appear that gas flow through rock joints is a result of bench swell, rather than the cause of the swell. The initial cause of fracture dilation appears to be the level of vibration induced by the explosive charges.

No evidence has been found in the literature of research into the effects of the gas penetration on block stability, though many researchers refer to the potential for reduction in slope stability as a direct result of the gas action. Further, no geotechnical groups anywhere in the world are known to have experience in controlling or predicting stability of blocks under the influence of gas pressure.

Gas pressures measured behind blasthole patterns can be as high as 3 MPa and are commonly in the range of 10 kPa to 100 kPa for distances less than 10 metres. Simple static limiting equilibrium analysis suggests that pressures of this magnitude can render most blocks unstable, even where the factor of safety in the absence of gas pressure is very high.

Little effort has been made to quantify the rate of attenuation of gas pressure (positive and negative) with distance behind blast patterns. One study presented a negative exponential rate of dissipation, determined through simple regression techniques rather than theoretical considerations. The results were expected to be highly site-specific.

Gas penetration can be minimised, if not eliminated, by allowing rapid venting of explosion gases after explosive detonation. This, however, is likely to promote flyrock and be an unacceptable risk for civil development projects in Hong Kong. Elimination of ground dilation (and negative pressures) is considerably more difficult.

Gas flow into dilated fractures and joints is promoted by conditions of high charge confinement, including excessive stemming lengths, and very low powder factors.

The velocity of gas propagation has been measured by several researchers, and again appears to be site specific. Velocities in excess of 200 m/s were common, with some velocities exceeding the sonic velocity of air (approx 340 m/s).

Gas has been observed to flow through both natural joints, as well as blast-induced fractures, for distances up to 20 metres behind blasthole patterns. The distance to which gas flow can be detected behind blast patterns appears to be controlled largely by the original intensity of jointing. Negative pressures are recorded at greater distances behind blasts than positive pressures.

It may be more appropriate to control the effects of gas penetration and fracture dilation than to try and control the process itself. Temporary artificial support can be utilised to control joint dilation and to apply additional restraint against block mobilisation. Risk assessment procedures need to be developed specifically for the gas pressure influence, together with modelling and predictive tools.

Specification of safe no-blast zones around excavations will largely be based on engineering judgement, due to the shortage of reliable field data, and lack of knowledge.

The Hong Kong procedures for assessing slope stability appear to be more advanced than procedures found in any other country. The procedure developed by Wong and Pang (1992) provides Hong Kong geotechnical engineers with a tool for assessing the impact of blast vibrations on slope stability. To the author's knowledge, all other countries utilise conventional static limiting equilibrium analysis only, ignoring the influence of blasting on the stability of slopes.

Modification of this risk assessment framework should be undertaken in collaboration with dynamic block modelling studies, and personnel with a strong background in blasting.

CONTENTS

Methods of Assessment and Monitoring of the Effects of Gas Pressures on Stability of Rock Cuts Due to Blasting in the Near-field

1. INTRODUCTION

Rock blasting is an essential process in the formation of most construction platforms in Hong Kong. The Hong Kong topography generally features steep hills in hard, granitic rock types. Large infrastructure projects frequently require the formation of expansive areas of level ground into which the foundations for large structures are cast. Similarly, roadway construction and expansion frequently involves the formation of deep, steep sided cuttings in hard rock.

Frequently, construction blasting is conducted in close proximity to other industrial and domestic activities which can not be interrupted by the blasting process. In these instances, blasting activities will occur in close proximity to normal everyday activities of the community, including domestic activities, office activities, schooling, shopping, and commuting.

The risk of personal, or structural, damage caused by the blasting processes must be maintained at a level commensurate with normal activities. Hong Kong has already developed procedures for blasting which generally provide an extremely high level of protection against any form of damage resulting from blasting activities. The success of these protective measures is reflected in the very low accident rate, while at the same time, permitting cost effective rock excavation in situations where blasting would not be permitted in many other countries. The report "Assessment of Stability of Slopes Subjected to Blasting Vibration" prepared in 1992 by the GEO, is an example of technical innovation developed for the specific problems encountered in Hong Kong during rock excavation by blasting.

In December 1997, an incident occurred during a large Hong Kong Housing Authority site formation contract which has highlighted the need for a review of the controls associated with blasting near rock slopes. In this incident, a rock failure was initiated by a small scale blast. The failed rock slope was approximately 25 m high, involving approximately 1000 m^3 of rock which originated mainly from the upper part of the slope, immediately behind the small blast.

The size of the largest rock was approximately 150 m^3, and the rock debris slid down the slope onto the Sau Mau Ping Road. As in normal practice, the road been closed to vehicular and pedestrian traffic at the time. Protective rock catchment fences proved inadequate to retain the material, and although no personal injuries were sustained, the road was closed for a period of investigation of the incident. Concern is felt regarding future potential incidents associated with site formations and road widening projects where it will not be possible to stop vehicular traffic flow. Under these conditions, blasting activities must not unduly increase hazardous risk to passing motorists, and even small scale failures can not be tolerated.

Back analysis of the Sau Mau Ping failure, by the GEO geotechnical engineers, suggests that it was unlikely to have been caused by vibration induced by the blasting. Although the major part of the failure appears to have been associated with well defined joint planes, the low angle of this joint plane (25^0 to 30^0) suggests that vibration could not have induced the failure.

The literature review in this report was initiated by the GEO to investigate the possibility that the Sau Mau Ping failure was caused by the influence of explosion gas pressures. Currently, there are no controls associated with gas pressure effects from blasting. If appropriate, additional controls will be implemented to eliminate the possibility of further failures like the Sau Mau Ping failure.

2. SCOPE OF WORK

The objectives of this report are as follows, based on the original GEO brief:

- carry out a comprehensive review of international literature and a critical appraisal of such literature to summarise the state-of-the-art information on generation of gas pressures in rock masses due to blasting, the attenuation characteristics of such gas pressures with distance, depth and geology, and their effects on the stability of rock cuts and excavations within close distances of the blast;

- identify approaches to modelling and prediction of the extent of gas pressure penetration into rock discontinuities and blast-induced fractures and to recommend guidance on the assessment of its effect on the stability of nearby rock cuts and excavations in Hong Kong (including the need for site-specific monitoring) taking into account the range of local geological conditions;

- review word-wide practices and, where possible, advise on the extent of no-blast zones to be imposed for eliminating adverse blasting effects on nearby rock cuts and excavations, and to give recommendations in this respect for rock cuts and excavations in different geological conditions under the Hong Kong urban environment;

- give guidance on when the effects of blasting would need to be considered as near-field and monitored and evaluated as such prior to blasting near a rock cut or excavation.

3. INTERNATIONAL LITERATURE

Although there have been many papers published in the international literature dealing with gas pressure from blasting, very few of these address measurement, and none address, in any quantitative manner, the effects on slope stability. The literature review therefore can address only the current state of knowledge of gas pressure.

3.1 Gas Pressures Generated by Blasting

There is general agreement within the explosives industry concerning the pressures generated by commercial, ammonium nitrate-based explosives. The peak gas pressure generated within a blasthole is generally taken to be around one half of the detonation pressure, and can be calculated from the equation:

$$P_b = 0.12\, f_c^n\, \rho_{exp}\, VOD^2 \qquad (1)$$

where P_b is the gas pressure (Pa), VOD is the velocity of detonation of the explosive (m/s), ρ_{exp} is the density of the explosive (kg/m^3), f_c is the coupling factor, defined as the ratio of the volume of the explosive to the volume of the blasthole (excluding the stemming column), and n is the coupling factor exponent generally taken to be between 1.2 and 1.3 for dry holes and 0.9 for holes filled with water.

Product	Density (kg/m^3)	Diameter (mm)	VOD (m/s)
ANFO	800	76	2840
ANFO	800	102	3240
emulsion cartridge	1150 - 1200	32	4000
emulsion cartridge	1150 - 1200	50	4800
emulsion cartridge	1150 - 1200	75	5000

Table 1. *Average density and VOD values for common explosives used in Hong Kong.*

Values for the velocity of detonation and density are product-dependent, but are generally quoted by the various manufacturers. The velocity of detonation of ammonium nitrate-based explosives is diameter-dependent, increasing with increasing diameter. A good explanation for this is presented by Brinkman (1990), and typical values for density and VOD, for different explosive products and diameters, are presented in Table 1 above.

Based on the average values represented in Table 1, and the use of equation 1, peak blasthole pressures can be shown to vary between 800 MPa and 3900 MPa for fully coupled explosives. Commonly in Hong Kong, cartridged explosives are used, so that there is a degree of decoupling between the explosive and the surrounding rock, and peak pressures are reduced relative to those for fully coupled explosives. Assuming normal degrees of decoupling, peak blasthole pressures resulting from the use of cartridged explosives are likely to be approximately 1000 MPa.

Where blastholes are filled with water, the degree of pressurisation increases markedly. Under these conditions, blasthole pressures with cartridged explosives and normal degrees of decoupling can be expected to be around 1500 MPa.

Although blasthole pressures as high as 4000 MPa can be generated during blasting, these pressures generally apply only for a very short time period during the initial stages of blasthole deformation. As the blasthole deforms, and cracks and fractures around the hole become dilated, gas streams out of the blasthole into an increasing volume, thereby reducing the peak gas pressures. At some stage, a path is formed for the gases to escape to the atmosphere, after which time, the gases perform no further useful work. This pressure is generally considered to be around 100 MPa (Persson *et al*, p 115). In situations where powder factors are high, and front row burdens are small, the rock burden will move sooner, and more quickly, so that gas will vent at higher pressures, and penetrate less into the surrounding rock. Conversely, where powder factors are low and burdens are large, gas will penetrate further into the surrounding rock mass, and venting pressures will be lower.

3.2 DISSIPATION OF GAS PRESSURE

High gas pressures resulting from the detonation of explosives are an inevitable outcome from all blasting operations. The gas pressures are useful in completing the fragmentation process, as well as for producing movement of the broken rock fragments. The movement of the fragmented rock is considered essential in order to introduce sufficient swell and void space that the broken material can be easily and efficiently excavated.

The extreme gas pressures generated by detonating explosives are dissipated as the blasthole diameter expands, thereby increasing the volume of the confined gases. Cracks, either natural or induced by the stress waves associated with the detonation of the explosive, become dilated, and the gas flows rapidly into the fracture matrix, effectively fluidising the fragmented material around the blasthole. Under normal bench blasting conditions, the broken rock burden continues to expand until there is eventually formed a free path for the gases to escape to atmosphere. Once the gases are free to escape to atmosphere, no further useful work is performed by the explosive, and the gas which permeated the fragmented rock volume slowly dissipates over a period of several seconds.

Several authors (Armstrong, 1987, LeJuge *et al*, 1994, Ouchterlony, 1995, Ouchterlony *et al*, 1996, Brent, 1996, Matheson, 1986, Matheson, 1992) refer to the process of bench dilation due to penetrating gases. The dilation has been measured at distances in excess of 20 metres behind blast patterns, in rocks varying from massive sandstone (Armstrong, 1987 and Brent, 1996) to fractured andesite (Armstrong, 1987), to biotite gneiss (Ouchterlony, 1995 & 1996) and granite gneiss (LeJuge, 1994). These rock types exhibit variable degrees of jointing - the sandstones of Brent's studies being massive, the biotite gneiss of Ouchterlony's being foliated and tightly jointed, and the andesite and granite gneiss of Armstrong (1987) and LeJuge (1994) being relatively highly jointed.

Factors which impede the dissipation of high pressure gases, and tend to promote a period of sustained pressurisation of the rock mass around blastholes, include the following:

- low powder factors, making movement of the rock burden material slow and difficult;
- large stemming columns, forcing the gas to penetrate into cracks and fissures which intersect the blasthole;
- constrained rock movement caused by the unavailability of a free vertical face to promote rock movement.

Blasting in Hong Kong commonly features all of the above features of blast design. Powder factors are maintained at low levels, and stemming lengths are maintained high, in order to control movement of broken rock and flyrock. This control is considered necessary due to the close proximity to other structures, and public safety.

In normal blasting circumstances, international practice would be to utilise powder factors in the range 0.5 kg/m^3 to 0.8 kg/m^3 in hard granitic material. This compares with typical Hong Kong powder factors in the range 0 .3 to 0.4 kg/m^3, and actual powder factors in the range of 0.15 to 0.25 kg/m^3 for the Sau Mau Ping site. Further, good blasting practice would consider optimum stemming lengths to be in the range 25 to 30 times the hole diameter, depending on the degree of control required over overpressure levels and flyrock. This compares with stemming lengths in the range 32 to 43 times the hole diameter used in the Sau Mau Ping site.

Adopting best practice blast design features promotes excellent fragmentation, excellent muckpile movement and looseness, and rapid dissipation of high pressure explosion gases. In this type of blasting, Johansson and Persson (1974), for example, report that gas pressure in the major part of an extended crack is far less than it is in the blasthole itself.

The above Sau Mau Ping design statistics, however, suggest very tight control over rock movement and ejection, but they also suggest extreme degrees of confinement over the high gas pressures generated by the cartridged explosives utilised. Under these conditions, the high pressure levels will be sustained for a long period, expected to be measured in seconds rather than the normal situation of fractions of a second. Dally *et al* (1975) show that for small contained explosions, (*i.e.* blasts without a free face, or heavily confined charges) there is time for the gas pressure in the fractures around a blasthole to come into pressure equilibrium with that in the blasthole itself. Although this equilibrium pressure is difficult to estimate, it is noted that the pressure widely considered to be the point beyond which no further work is performed by the expanding gases, and at which venting usually starts to occur, is around 100 MPa (Persson *et al*, p 115).

Brent & Smith (1996) conducted field studies of gas flow through cracks and fractures behind blasthole patterns and observed no gas flow behind normal free face blasts in massive sandstone. They also note, however, the absence of visible structures or discontinuities in the rock (as determined by borehole video inspection) before the blast. After the blast, many large open cracks could be seen, though no flow of gas was observed from pressure transducers located in the sealed holes. When the test was repeated using fully confined crater blasting, the same workers report significant gas flows over the full range of monitoring distance, up to 62 times the charge diameter. They conclude "the fully confined nature of the blast, together with the permeability of the rock mass, resulted in gas penetration to all the monitoring holes." Brent & Smith (1996) report the exponential gas pressure decay function for confined charges:

$$Press = 445.6\, e^{-0.535d} \tag{2}$$

where d is the distance from the blasthole, measured in metres (for fully coupled charges), and *Press* is the pressure measured in kPa.

Similar work conducted by Ouchterlony (1995, 1996), suggests that pre-splits may play a vital role in dissipating gas pressures in cracks and fissures behind blast patterns. They noted that gas flows were not observed behind pre-splits, though it must be assumed that this implies the formation of a well defined pre-split fracture through which gas can vent. Interestingly, however, they also report that the pre-split itself produces both fracture dilation and gas flows due to the confined nature of these blasts. This latter observation confirmed the earlier work by LeJuge *et al* (1994).

3.3 Field Measurement Studies

Comparatively little work has been conducted to measure gas pressures, or even to observe the presence of gas, in the rock mass fabric surrounding blastholes. Essentially, only five sources of field gas pressure measurement have been found from the literature review:

- Julius Kruttschnitt Mineral Research Centre (JKMRC), with work conducted either by the author, or under the direct supervision of the author in the period 1985 to 1987;
- Blastronics (formerly Australian Blasting Consultants), under the supervision of the author;
- Advanced Technical Development group of Conzinc Rio Tinto (CRA), in Perth, Western Australia (performed by JKMRC graduate);
- SveBefo, Sweden;
- ICI (currently Orica) Explosives Australia.

Importantly, there is a great deal of commonality in the field results observed by all workers from all of the above sources. In summary, the results of field measurements of gas pressures from blasting can be stated:

- both positive and negative gas pressures have been observed behind blast patterns (Figure 1);
- gas flows (positive and negative pressures) can be measured for large distances behind patterns (up to and exceeding 25 metres);
- positive pressures in the range of 10 kPa to 100 kPa are commonly measured at distances of around 10 metres behind blastholes (Figure 1);
- peak positive pressure levels are highest, and sustained for longer periods, behind highly confined explosive charges.

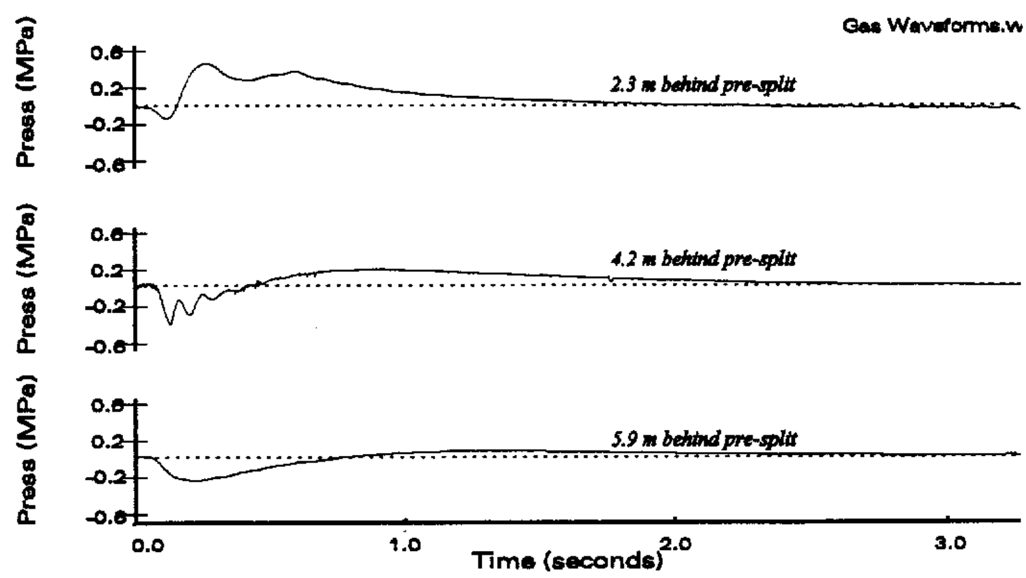

Figure 1 Example gas pressure waveforms measured by Blastronics at various distances behind pre-split charges.

The most recent work associated with the effects of gas pressures on rock mass stability, however, has focussed on the measurement of fracture dilation and bench dilation, since this is what is considered to have the primary influence on slope stability.

In addition to the field measurement studies, much work has been done in the modelling of gas pressure effects, though this has been restricted primarily to the influence of gas pressure on fracture extension, and gas pressure effects from deeply buried nuclear charges. Some of the more relevant examples include Coursen (1979, 1985), Dally *et al* (1975), Haghighi *et al* (1988), Nilson *et al* (1985), Payne & Please (1994), Pitts & Brandt (1977), and Preece *et al* (1993). These references are included for the sake of providing a background to gas flow theory in jointed rock, though in the opinion of the author, it is unlikely that this information and detail will be useful for use in Hong Kong.

The earliest recorded work on gas pressure measurement behind full scale blasting patterns was outlined by Williamson & Armstrong (1986). These early workers recognised that gas pressures applied over the full surface area of a fissure would reduce the joint friction and induce instability of rock blocks. They further recognised that the effects of gas pressure in terms of damage and instability could be significant, even at rather low pressures, providing the pressure was applied for a relatively long period. However, these workers did not consider any formal analysis of gas pressure effects on block stability, though they did suggest that the primary influence of blast design was in controlling the rate of decay of gas pressure by controlling confinement.

3.3.1 VELOCITY OF GAS PENETRATION

Williamson & Armstrong (1986) measured gas penetration velocities in the range 5 to 10% of the p-wave velocity for both sedimentary overburden material and andesites. They suggest, however, that the velocity of penetration will be controlled by rock permeability, rather than by the initial blasthole pressure. Rock mass permeability was considered to be controlled by the *insitu* joint frequency. At this rate, expected velocities of propagation in massive granites could be expected to be at least 200 m/s. Maximum velocities deduced by Lymbery (1994) based on data recorded by Armstrong (1987) lie in the range240 m/s to 480 m/s, and measured velocities were reported by Sarma (1994) in the range 50 m/s to 350 m/s, with most measurements around 200 m/s.

Clearly, with these relatively high velocities of propagation, the explosion gases have the ability to penetrate relatively large distances, with the maximum distance being controlled largely by the time over which the pressures are sustained.

3.3.2 PEAK PRESSURE LEVELS

All reported studies of gas flows behind blastholes have included observations of negative pressures as well as positive pressures (except Bulow & Chapman, 1994). The early work conducted at the JKMRC was extended, and variously reported (Armstrong (1987), McKenzie, (1988, 1989, 1993), Lymbery (1994), Sarma (1994)), with considerable focus on the cause of negative pressures commonly observed, even in the absence of positive pressures. Eventually, the cause of this was attributed to vertical swelling of the bench around a blasthole pattern. The swelling was generally observed to occur prior to the gas flow, leading to the conclusion that gas flow was the result of dilation, not the cause of it.

The peak positive amplitude of gas pressure measured in the field studies conducted by Williamson and Armstrong was in the range 100 to 250 kPa, at distances up to 8 metres from blastholes with diameters in the range 270 mm to 311 mm. The workers concluded that the peak pressure was controlled by the degree of jointing, and also by the pressure in the blasthole. It should be noted, from equation 1, that the peak pressure generated by an explosive charge is essentially independent of the diameter of the charge though clearly the volume of the gases generated will be dependent on the hole charge.

LeJuge *et al* (1994) reports peak positive pressure levels of up to 80 kPa during site investigations at the Rössing Uranium Mine in Namibia. In this work, investigations were made behind both production blast patterns, and pre-split final limits patterns as part of a study of optimisation of limits blasting practices in large open pit mines. Although not reported in the technical paper, the work conducted by LeJuge at Rössing included measurements of peak positive pressures as high as 1000 kPa, at distances of only 2 metres behind pre-split blastholes. Pre-split blasting at Rössing, which initially incorporated 3 metres of stemming for greater effectiveness, was concluded to be entirely responsible for observed gas flows through the jointed, granitic rock mass. Little or no gas flow was attributed to trim blastholes which were fired to a free face, and left unstemmed.

In the work conducted at Rössing, bench dilation in excess of 50 mm was measured 20 metres behind blasthole patterns, and was recognised as a primary contributor to final wall stability. This work also identified the most critical impact of the gas pressure as being the time period over which it is acting on the rock mass, and the need for rapid venting of gases to minimise heaving of the bench, and joint dilation.

Of particular significance in the technical paper by LeJuge *et al* (1994), was the observation that peak positive pressures were sustained for periods in excess of 1 second. Further, it was noted that bench dilation could be detected over greater distances than gas flow. On these occasions, only the negative pressure pulses were recorded, and these were sustained for even longer periods of time, up to several seconds.

Sarma (1994) measured positive gas pressures in the range 10 kPa to 2000 kPa behind blastholes. He also measured negative pressures as low as -700 kPa, and concluded that the negative pressure pulses which he recorded were the result of a suction effect caused by the burden motion. In the absence of burden motion, Sarma's studies suggested that only a positive pressure would be observed, though this is not in agreement with the work conducted by LeJuge *et al* (1994), or

Ouchterlony (1995, 1996). However, Sarma also concluded that the presence of a pre-split was beneficial in acting to vent high pressure gases.

Ouchterlony (1995, 1996) reports peak positive pressures up to 1.77 atm (approximately 177 kPa), and peak negative pressures as low as -0.62 atm (approximately -62 kPa). Pressure duration times were reported to vary over the range 0.5 to 12.2 seconds. Ouchterlony, like LeJuge (1994) attributed the negative pressures to vertical movement of the bench, and showed that this conclusion was consistent with calculations of pressure based on the use of a polytropic gas law, and measured vertical bench displacements.

Brent & Smith (1996) report peak positive levels of gas pressure up to almost 300 kPa at distances around 1 metre from blastholes, decaying to around 6 kPa at a distance of 8 metres. The positive pressure levels are shown to be sustained for periods of around1 to 2 seconds. These workers quantified the rate of decay of positive pressure with distance, suggesting the exponential decay equation 2, on page 6. Brent & Smith report peak negative levels of vibration of around -80 kPa, with a duration of application of this pressure of around 2 seconds. Brent & Smith report no evidence of gas penetration (positive gas pressures) behind normal bench blasts, with gas flow apparently linked to the state of confinement of the charge. Negative pressures appear to be consistently associated with all monitoring events, and were again concluded to be caused by crack dilation, as supported by polytropic gas expansion calculations.

3.3.3 Distance of Penetration

Of all the researchers involved in measurement of gas pressures in joints around blastholes, only Brent & Smith (1996) presented a quantitative relationship between the pressure level and distance of propagation. It is expected, however, that this relationship will be very site specific, and influenced strongly by factors such as hole diameter, explosive type, rock permeability, and the state of charge confinement. The relationship was assumed to be exponential, equation 2, though no attempt was made to derive the form of this equation based on theoretical analysis. Brent & Smith (1996) report measurements of positive and negative gas pressures at distances up to 20 metres behind blastholes, implying both fracture dilation and gas flow over these distances. Even in holes where *insitu* jointing could not be observed before blasting (using a borehole video camera), large open cracks existed after blasting, permitting gas flows to distances of 105 blasthole diameters.

Ouchterlony (1995, 1996) reports positive pressures at distances up to 6.4 metres behind blastholes, but negative pressures up to distances in excess of 12 metres behind blastholes. In view of the rather high measured gas penetration velocities, and the period of application of the pressure pulse, the failure of gas to penetrate greater distances suggests that gas flow into dilated fractures is dependent on rock permeability.

McKenzie (1988) reported gas pressures around 150 kPa at distances in excess of 8 metres behind blastholes, even in massive sandstones relatively free from *insitu* jointing, and bedding.

LeJuge (1994) reports small positive pressures, and significant negative pressures at distances up to almost 10 metres behind pre-split holes , and strong positive and negative pressures at a distance of 5 metres. Independent measurements of bench dilation suggest that negative pressures would be recorded at distances of at least 30 metres behind blastholes. LeJuge specifically concluded that joint dilation occurs across a pre-split fracture, and far beyond the range of active gas penetration.

Bulow & Chapman (1994) report that gas penetration was measured at up to 21 metres behind blast patterns. It appears relevant that these large distances were measured behind large diameter blastholes up to 381 mm in diameter.

Sarma (1994) consistently measured strong negative pressures at 5 metres behind blastholes. In confined blasting conditions, he also measured significant positive pressures of up to 50 kPa at 5 metres behind blastholes. His work did not seek to measure gas flows at greater distances than 5 metres.

3.3.4 Measurement Instrumentation

The standard measurement procedure, adopted by all researchers involved in field measurement of gas pressures, utilised empty boreholes drilled at varying distances behind a blasthole pattern, and sealed at the top. Holes are sealed generally by the use of grout or cement, though techniques including polystyrene foam or inflated gas bags in combination with normal stemming material could also be utilised. Pressure transducers, of varying type and make, are inserted into the sealed hole, and connected either to an instrumentation tape recorder (Ouchterlony, 1995 & 1996), or to specialised digital, portable recording systems. Other than Ouchterlony, all other workers used multi-channel Blastronics digital recording systems.

Pressure gauges were selected according to the expected peak pressure levels, and to their response time, since the onset of gas flow can result in a fast rise time to peak positive or negative levels. Brent & Smith (1996) report identical response from both absolute and differential pressure gauges, and accuracy over the pressure range 0 kPa to 200 kPa, under dynamic loading conditions was established.

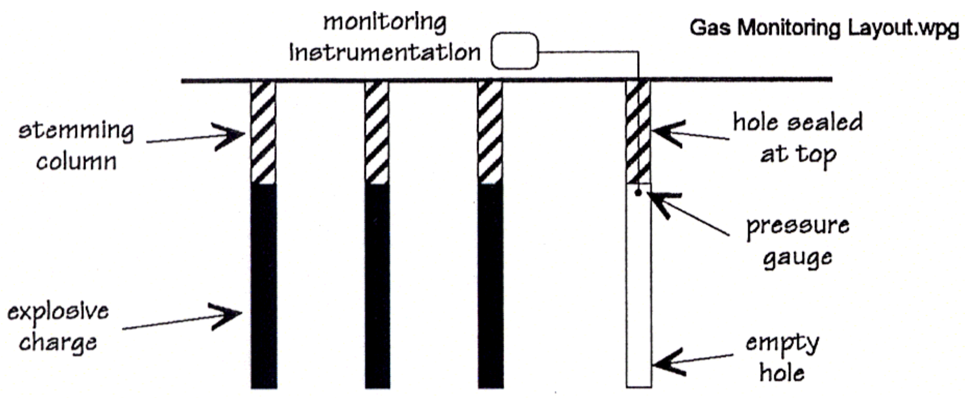

Figure 2*Gas pressure monitoring layout used in all reported field studies.*

Figure 2 above illustrates the monitoring arrangement reported by all field workers involved in gas pressure measurement. The empty monitoring holes are drilled to the same depth as the blastholes, though the length of the sealed section at the top of the hole may be shortened. The length of the sealed section should be sufficient to ensure that gas pressure can not escape, or enter, through fractures caused by previous sub-drilling. For this reason, the length of the sealing column is generally the same as the length of the stemming column in the blastholes. Further, the use of arrays of monitoring holes at varying distances from the pattern, or at different locations behind the pattern, is common. In this manner, some feel can be gained for the statistical variability of the observations. It is also common for the gas holes to be accompanied by extensometers to measure ground movement in three directions, as confirmation of fracture dilation.

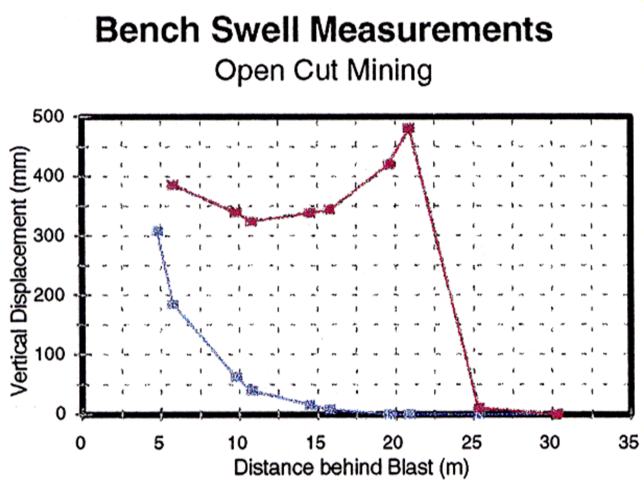

Figure 3*Example measurements of bench swell behind two different blasts.*

Figure 3 presents an example of bench swell measurements behind two different blasts in a large open cut mine. Measurements of bench swell are much easier to make than measurements of gas pressure, and are less affected by statistical variability. In many respects, bench swell is a more direct measurement of damage, providing measures which can be used in subsequent modelling studies.

4. PRELIMINARY MODELLING

It is disappointing that no field researchers have attempted to address the impact of gas flow, and associated bench dilation, on bench or block stability. The significance of dilation and gas flow is acknowledged by most researchers, but no attempts are made to predict the impact of dilation or gas flow on stability.

Because gas flow can be expected to be controlled very strongly by rock permeability, modelling and prediction will be difficult without good knowledge of joint conditions, and possibly even joint locations. Measurement can be expected to produce very variable results, especially in the more massive rock types such as could be frequently encountered in Hong Kong. However, it is probably these very rock masses, tending to form isolated, large rock blocks, which are the most important ones to understand and control.

gas damage.wpg

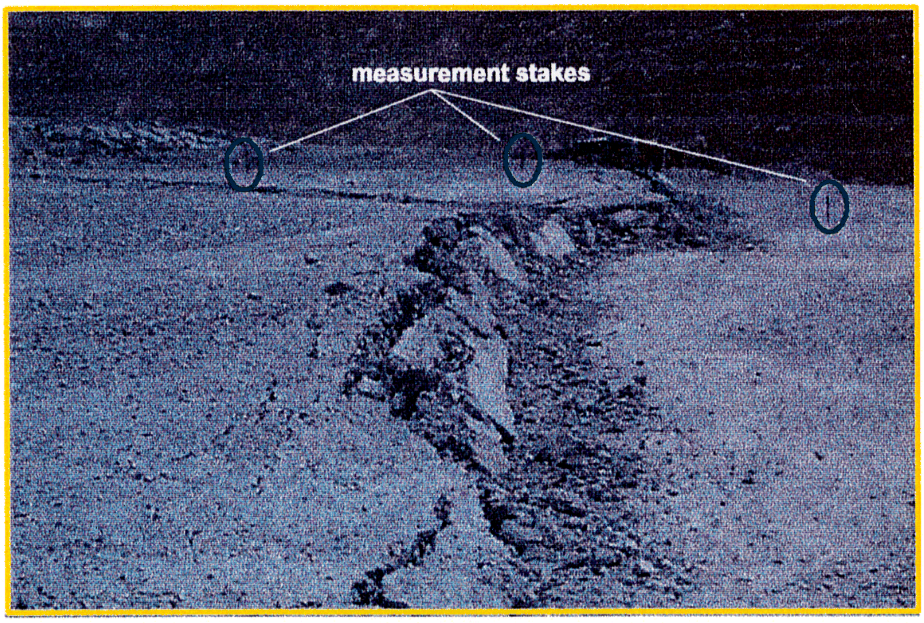

Figure 4*Influence of gas pressure promoting block movement along existing joint planes in a large open cut mine.*

That gas pressures can induce large block movements along existing joint planes is well demonstrated in the photograph of Figure 4, in which vertical movement of around 0.5 metres can be detected almost 30 metres behind a blasthole pattern. This photograph was obtained from a large open cut mine where gas penetration studies are currently being conducted by Blastronics, and a database is being developed to assist prediction and control of gas pressure influences.

In order to investigate the possible impact, the following simplistic modelling has been conducted, drawing on observations made from the literature review. Hoek & Bray (1981), present a general equation for estimating the factor of safety of a simple block under conditions of limiting equilibrium:

$$FOS_f = \frac{cA + (W\cos\psi - U + T\sin\beta)\,Tan\phi}{W\sin\psi + V - T\cos\beta} \qquad \textbf{(3)}$$

where FOS_f is the factor of safety of the block with pressurised fluid, c is the joint cohesion, A is the surface area of contact between the block and the joint plane, W is the weight of the rock block, U is the uplifting force due to water or other fluids in the joint, V is the water pressure force applied to the rear of the block, T is the tension load force from a rock bolt holding the block in place, β is the angle between the rock bolt and the joint plane, ϕ is the joint friction angle, and ψ is the angle of dip of the joint plane (Figure 5).

This equation has been used to examine the pressures required to reduce the factor of safety to unity due to the penetration of pressurised gases in the joint plane. The analysis, however, considers the simple case of a weathered joint with zero cohesion (assumed to be the case at Sau Mau Ping, based on observed weathering of the exposed surfaces), without artificial rock bolt support, and in the absence of pressure applied to the rear of the block. In reality, the force applied to the rear of the block at Sau Mau Ping may have been substantial, possibly representing the rock in contact with the back row of blastholes in the pattern which triggered the failure.

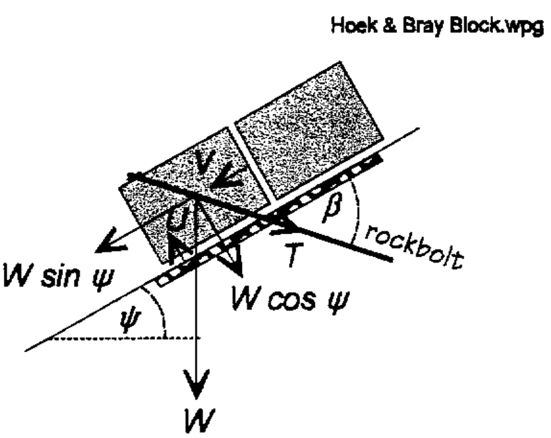

Figure 5Limiting equilibrium condition for simple sliding block stability analysis. (after Hoek & Bray. 1981.)

Under these conditions, the above equation reduces to:

$$FOS_f = \frac{(W Cos\psi - U)\ Tan\phi}{W Sin\psi} \tag{4}$$

where U is the *uplift force*, due to a fluid pressure u, applied over a contact area, A, and $U = uA$.

Rearranging these terms produces:

$$FOS_f = FOS^* - \frac{u\ Tan\phi}{g\ t_{rock}\ \rho_{rock}\ Sin\psi} \tag{5}$$

where t_{rock} is the thickness of the rock, g is the acceleration due to gravity, ρ_{rock} is the density of the rock, and the product of these three terms is the hydrostatic pressure of the block, acting normal to the basal area. FOS^* is the Factor of Safety in the absence of the pressurised gas.

Clearly, the presence of a pressurised gas in the joint will reduce the factor of safety. Once the factor of safety has been reduced to less than unity, the block is considered unstable.

This equation says that the pressure required to reduce the stability of the block is dependent on the thickness (and therefore weight) of the block, and that the resulting stability is dependent on the initial factor of safety prior to the application of the gas pressure. The effect of an applied gas pressure is shown graphically in Figure 6, for different conditions of applied pressure and joint plane angle, assuming a joint friction angle of 45^0, and a block thickness of 3 metres. The graph illustrates the low pressures required to reduce the factor of safety to unity for joint planes inclined at around 25^0 to 30^0 - approximately 30 kPa.

The analysis ignores the influence of friction along the sides of the block, though inspection of the Sau Mau Ping site indicated that the sides of the failed block were also likely to have been weathered joint planes. If these joint surfaces produced a slight wedge shape to the block, then the influence of friction along these surfaces may well have been insignificant.

The literature review suggests that pressures can exceed 30 kPa by as much as 2 orders of magnitude, if the basal joint plane intersects favourably with one or more of the blastholes in the pattern.

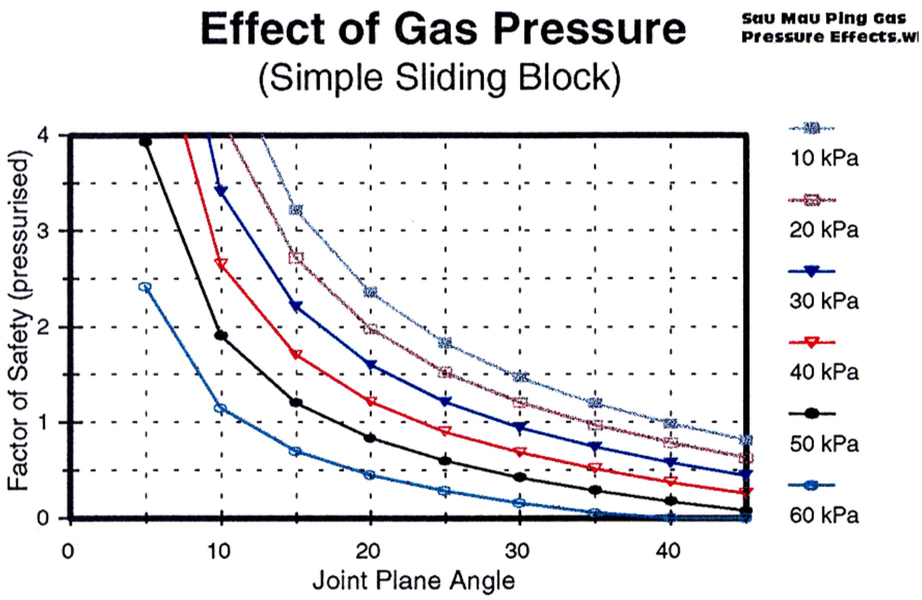

Figure 6*Effect of joint gas pressure on stability of simple sliding block under limiting equilibrium conditions.*

Figure 6 also suggests that the factor of safety can be reduced to zero, in which case there is no restraining force acting on the block at all. In this case, it would appear that the block can be "floated" by a uniform pressure applied to the base of the joint. Pressures in excess of 80 kPa will effectively cause the 3 m thick block in Figure 6 to "float", even though under normal conditions it may have a factor of safety of around 2.

However, it would appear that the presence of gas is not necessary to explain the instability of the block. Most of the field researchers report negative pressures caused by dilation of joints close to blastholes. In relatively massive rock types, dilation may be restricted to a single joint, thereby reducing, or removing altogether, the frictional forces along the basal plane. Where joint dilation exceeds the dimension of joint asperities on the basal plane, a large component of the restraining forces is removed from the block, and failure is likely to occur. In personal communication with Ouchterlony, it was revealed that the vibration and gas pressure study conducted at the Aitik Mine in Sweden, and reported by Ouchterlony (1995, 1996) showed that measurable swell of 5 to 10 mm resolution was related to a peak particle velocity of 1250 mm/s. Below this level of vibration, swell was too small to be significant using the simple monitoring system adopted in the study.

A more detailed approach to modelling, using an existing distinct element particle code coupled with a gas penetration algorithm which is linked to a 3 dimensional joint model has been prepared by Earth Technology, at the request and guidance of Blastronics. This approach is presented in Appendix A.

5. WORLD-WIDE PRACTICES

Geotechnical risk assessment procedures conducted in Hong Kong prior to development below potential natural terrain landslides have been outlined by Roberds *et al*, (1997). These procedures demonstrate how a detailed framework can be developed for making optimal decisions regarding excavation safety considering all relevant factors, not limited only to rock stability. The current framework in place in Hong Kong for evaluation of excavation safety is considered by the author to be as advanced and comprehensive as practised anywhere in the world. However, there would appear to be a compelling need for further development of this framework to include gas pressure effects for slopes to be excavated by blasting, or propellant action.

There is little information available in the published literature describing practices when blasting in sensitive situations like those encountered at Sau Mau Ping, or future situations such as the Tuen Mun Highway. Studies of gas penetration have only been conducted in two countries - Australia and Sweden, and none of these studies has considered the impact on stability, or the implications in applications such as blasting adjacent to structures, or roadways. Field studies identified in the literature review pertain only to mining environments, where higher risk factors are generally accepted.

However, contact has been made with several international groups to identify common approaches to ensuring the safety of nearby structures and people.

In the USA, for example, we are advised that normal practice is to clear the area of vehicular traffic, either by imposing road blocks, or by using police patrol cars to slow the flow of traffic in both directions, thereby introducing a traffic-free period of short, but sufficient, duration to fire the blast. The police patrol car method is used in simple situations where blasts are small, and can be quickly assessed after the event to determine safe conditions. In cases where more time is required to verify safe return to the area, traffic is stopped for periods of up to half an hour. It appears that geotechnical analysis is not conducted, and there is no indication from the literature that gas penetration is ever taken into consideration when assessing slope stability in the USA.

A similar situation exists in Australia, where road closure is standard practice when blasting is conducted within approximately 100 metres of carriageways - even major highways. To minimise these disruptions, which can clearly be of major significance, every effort will be made to use non-explosive methods of excavation, or alternatively to use detours, even if the detours require significant construction effort and cost. In the current extension of the Brisbane to Gold Coast highway, for example, in which the existing 4 lane highway is to be expanded to 8 lanes, several rock cuttings will be encountered. No special geotechnical investigations will be made of these cuttings, and slope design studies considered only the standard failure mechanisms, without any account being made of the effects of blasting, including seismic effects.

Ouchterlony (personal communication) advises that in Sweden, road alignment would be adjusted to avoid the situation where cuttings had to be blasted while traffic was still active.

Perhaps the only country, other than Hong Kong, in which detailed geotechnical analysis of road cuttings is made prior to undertaking road widening is Scotland, with a reasonably detailed methodology presented by McMillan & Matheson (1997). These authors describe a two staged process of assessment of the risk posed to the road user. Based on the values of a Hazard Index determined during very quick site inspection, more detailed analysis may be undertaken to determine the Hazard Rating for the cutting. It is this Hazard Rating which determines the priority given to remedial works on the cutting.

Since the Hazard Rating developed by McMillan & Matheson (1997) employs conventional deterministic failure models for calculating Factor of Safety in probabilistic analyses, it would seem appropriate to modify this process to allow for inclusion of the Wong and Pang Stability Method. Further, it may well be appropriate to modify Wong and Pang's method to include a gas pressure acting in a similar manner to water. In modifying the method, a more detailed structural mapping procedure such as that described by Earth Technology in Appendix A, may be a necessary step.

One of the factors influencing the Hazard Rating is the anticipated mechanism of failure. However, during personal communication with the Transport Research Laboratory (TRL), it is clear that previous work by this group has not considered the gas failure mechanism which is considered to have been largely responsible for the Sau Mau Ping failure. Perhaps the main reason for this, is that the TRL has developed the Hazard Rating to identify the level of hazard and risk associated with existing, frequently ageing, road cuttings. In its current form, it appears less suitable to evaluating the hazard and risk of failure during formation or excavation of the cutting. However, the proposed rating system appears to be an excellent foundation upon which further development can be made to address the concerns relating to road widening programs in Hong Kong.

The author has been advised of one instance in Sweden where, in order to control the flow of explosion gas pressures into rock joints, blastholes were lined with steel. Steel split sets were inserted into blastholes, and the explosives were loaded inside the split sets. The steel lining acted to prevent the flow of gases into joints intersecting the blastholes, in a manner very similar to the experiments reported by Brinkman (1990). The procedure was reported by Ouchterlony (personal communication), to be very successful, and easily implemented. Importantly, the presence of the steel liners is reported by Brinkman (1990) to have only a small effect on fragmentation.

Current Hong Kong design procedures for works in proximity to slopes, or associated with slopes, incorporate a wide range of risk assessment. These include conventional geomechanical, limiting equilibrium studies aimed to evaluate the potential for planar, wedge, circular, or toppling failures, as well as the sensitivity of the slopes to destabilisation by the effects of blast-induced vibrations. In this respect, Hong Kong practices are considered more advanced than other countries within the author's experience.

In order to further enhance control over slope failures, a basis for predicting the potential effects of gas pressure needs to be developed. In all likelihood, this will require investigation of the effects using particle codes such as UDEC to investigate the extent of the zone affected by gas pressure. Procedures can then be developed either to eliminate the flow of gas, or to manage and control the effects so that the level of risk can be reduced to an acceptable level.

6. PRACTICAL GUIDELINES

As stated earlier, it is considered that the most dangerous situation is in blocky ground, where large slabs or blocks of rock can be destabilised. In heavily jointed rock, permeability is expected to be higher, pressures will dissipate more quickly, and penetration distances will be reduced. Small blocks which are destabilised will likely be retained in place by other blocks which are not subjected to dilation.

In low permeability, lightly jointed ground, single joints which intersect a blasthole can be expected to provide efficient vents, activating large blocks in a very short time.

Again, it must be stated that it does not appear necessary to have gas flow in order to produce dilation of joints, and dramatic destabilisation of blocks through reduction of effective joint friction angles.

Based on the literature review, a number of blast design features can be highlighted as methods of minimising the influence of dilation and gas flow:
- minimisation of confinement through burden control;
- minimisation of vibration through charge diameter and weight control;
- pre-splitting to provide an effective gas vent.

6.1 MINIMISING CONFINEMENT

The literature identifies confinement as the primary factor controlling the extent of gas penetration, and the peak levels of positive pressure. If the period of containment of the high pressure explosion gases can be minimised, literature suggests that gas flow can be effectively eliminated. This suggests that although the gas penetration velocity may be relatively high, some time is required to dilate the fractures before the gas can flow into them.

Confinement is inextricably linked to *effective* powder factor. The term *effective powder factor* has been used in recognition of the frequent use of cartridged explosive in Hong Kong, where the explosive charge is decoupled from the rock by virtue of the difference in diameter between the blasthole and the explosive cartridge. This decoupling decreases the peak blasthole pressure, and also decreases the *effective powder factor*. The term can be described by the equation:

$$PF_{eff} = PF\left(\frac{Vol_{charge}}{Vol_{hole}}\right)^{1.2} \qquad \textbf{(6)}$$

where *PF* is the ratio of weight of explosive per blasthole to the volume of rock broken per blasthole (kg/m^3), Vol_{charge} is the volume of the explosive in the hole, and Vol_{hole} is the volume of the blasthole below the stemming column.

Using equation 6, if a 76 mm diameter blasthole is charged with 4 kg of 50 mm diameter cartridges of explosive, in a pattern of 1.8 m x 1.8 m, and a bench height of 4 metres, then the powder factor would normally be calculated to be approximately 0.31 kg/m^3. However, because the charges are decoupled, peak pressure in the blastholes will be reduced, and the ability to displace the rock and dissipate the gas pressures will also be reduced, so that the *effective powder factor* would be approximately 0.11 kg/m^3. In this situation, gas pressures generated by the blast would be in the vicinity of 900 MPa, and pressure decay would be slow, allowing gases to penetrate relatively large distances into fractures and joints.

At low effective powder factors, explosive charges become highly confined, unable to produce effective burden movement to dissipate the explosion gas pressures. Conversely, high effective powder factors, achieved through adoption of small burden dimensions, promote rapid breakage of burden rock, and rapid dissipation of gas pressures. Effective powder factors greater than 0.5 kg/m^3 are considered to promote rock movement and early gas dissipation in granitic rocks. Powder factors less than 0.5 kg/m^3 will progressively promote gas containment, and joint dilation.

Importantly, none of the field researchers identified high powder factors as a contributor to gas damage. Providing adequate delay intervals are utilised between blastholes, high powder factors will not produce increased damage from either vibration or gas damage mechanisms. In fact, modern best practice is to increase powder factor by reducing burden dimension in order to control rock damage.

When considering the state of confinement of a buried explosive charge, consideration must also be directed towards the depth of burial, and length of stemming. Excessive stemming column length leads to a depth of burial which can exceed the critical depth of burial, thereby promoting complete containment of the energy. The common approach to controlling confinement in mining applications is to eliminate stemming, and to use small burden dimensions to promote rapid movement of broken rock. In the context of charge confinement, stemming ejection and forward rock movement represent escape valves for high pressure gases in the blasthole, and may be beneficial in controlling gas related damage.

However, these approaches may produce an unacceptable risk with respect to damage from flyrock, and may be quite inappropriate control methods for the civil engineering applications in Hong Kong.

In the experience of the author, with effective powder factors in the range 0.4 to 0.5 kg/m^3, flyrock can be limited to distances less than 50 metres, and fracture dilation behind the blast pattern will be minimised. This assumes tight control over stemming ejection, achieved with a minimum stemming length equal to 30 times the blasthole diameter, and the use of screened aggregate stemming with a maximum size approximately equal to one sixth to one eighth of the blasthole diameter.

Where this amount of movement is still considered to pose an unacceptable risk, methods of amelioration may have to be directed at controlling block mobilisation, rather than controlling gas penetration. Block mobilisation can be controlled by the use of artificial support, including dowels, rock bolts, and cables.

6.2 MINIMISING VIBRATION

Minimisation of vibration will ensure that fracture dilation, even in the absence of gas flows, will be minimised. During transmission of vibrational energy across a fracture plane, differential movement of blocks on either side of the plane can occur, with the extent of differential movement increasing with increasing amplitude of vibration. In personal communication, Ouchterlony reveals that measurable dilation of between 5 mm and 10 mm was correlated with a vibration level of around 1250 mm/s in hard granitic material at the Aitik Copper Mine in Sweden.

Factors controlling the intensity of vibration around an explosive charge include:

- the diameter of the charge;
- the density of the charge;
- the ratio of charge diameter to hole diameter;
- the length and weight of the charge.

The equation relating these parameters was developed by Holmberg and Persson (1979):

$$PPV = k\ \gamma^{\alpha} \left[\int_{0}^{H} \frac{dh}{\left[R_0^2 + \left(R_0\ Tan\,\phi - h \right)^2 \right]^{\frac{\beta}{2\alpha}}} \right]^{\alpha} \tag{7}$$

where γ is the linear charge concentration (kg/m), k, α, β are fitted, site specific vibration attenuation parameters, and the other terms are identified in the schematic of Figure 7.

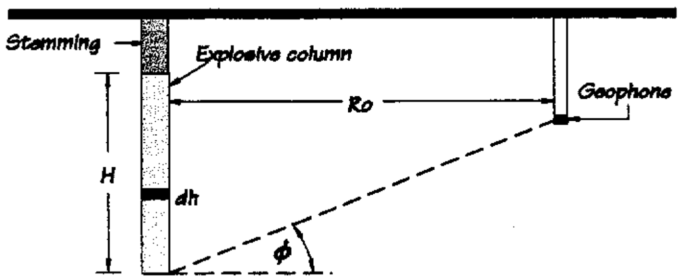

Figure 7 *Integration over charge length for near-field vibration calculation.*

Small diameter, short, and light charges produce significantly lower vibration levels than heavy, large diameter charges, especially when used in holes where the hole diameter exceeds the charge diameter.

The ideal configuration would appear to be the smallest practical hole diameter, with cartridge explosive to achieve a degree of decoupling, a small bench height, and a small burden. Such a configuration, however, will clearly involve additional drilling, and increased cost of excavation.

6.3 The Use of Pre-Splitting

Many of the field researchers make specific mention of the benefits of pre-splits to aid in the dissipation of explosion gases. Matheson (1992) refers to blast induced damage from bulk blasting being primarily in terms of dilated, natural discontinuities, in contrast with pre-split faces which display little or no change from the pre-existing natural discontinuities.

It may be quite appropriate that all blasts within a critical proximity of potentially unstable blocks should utilise pre-splitting to control the extent of possible gas action. However, when designing and implementing the pre-splits, care must be taken to ensure that these blasts do not trigger the failures, since Ouchterlony (1995, 1996) and LeJuge *et al* (1994) both observed significant gas flows behind pre-split holes. Pre-splits fired to protect sensitive structures should not be stemmed, so that rapid dissipation of the gases can occur, and very light charges and close hole spacings should effectively limit the effects of explosion gases.

6.4 ESTABLISHING NO-BLAST ZONES

One method to prevent the re-occurrence of the Sau Mau Ping failure is to establish a no-blast zone within critical distances of sensitive blocks. This requires, however, methods to determine both the critical proximity distance, and the sensitivity of blocks. With very limited data relating explosion gas penetration to either blast design or geological conditions, it is difficult to make recommendations concerning the zone of influence of penetrating gas pressures. A more structured approach is likely to involve the use of statistical models to assist in the prediction and control of the process.

The factors of critical distance and block sensitivity must ultimately be related to joint condition - length, and factor of safety. For short and weathered joints, *i.e.* heavily structured ground, rock permeability is high, and gas has been observed to travel larger distances than in less structured ground. Under these conditions, many small blocks may be mobilised by the permeating gases. For long joint lengths, *i.e.* in blocky ground, high pressure gases may activate large blocks which, if not buttressed by other secure blocks or artificially supported, are likely to become mobilised.

Joints which are weathered, and without cohesion, clearly offer the most dangerous conditions. Joint weathering, joint inclination, and joint length would appear to be important parameters requiring careful examination prior to blasting.

In terms of limiting the extent of gas penetration, few researchers reported gas flows at distances in excess of 10 metres behind blastholes. When coupled with good blast design, it would appear from available literature that a no-blast zone of 15 to 20 metres is likely to be effective in preventing mobilisation of blocks. Figure 8 presents all available gas pressure measurements, covering a range of rock types and blasting conditions. Also shown on this figure is an outline which represents an envelope within which these measurements lie. The envelope has been formed using the simple exponential decay equation shown below. This same form of pressure decay equation was used by Brent (1996).

$$Press = 700\, e^{-0.2\, dist} \qquad \textbf{(8)}$$

where *Press* is the pressure measured in kPa, and *dist* is the distance behind the blastholes, measured in metres.

Equation 8 predicts pressures of around 10 kPa at a distance of 21 metres behind blast patterns.

When deciding how to apply this trend to determine a safe no-blasting zone for Hong Kong, the following important issues must be remembered:

- blasthole pressure is essentially independent of hole diameter and bench height, (except for the influence of diameter on velocity of detonation), as per equation 1;
- the distance to which gas can penetrate will be strongly controlled by the volume of gas available, and the volume of open fractures;
- the factor having the greatest influence on gas penetration is expected to be the time over which the pressures are allowed to act.

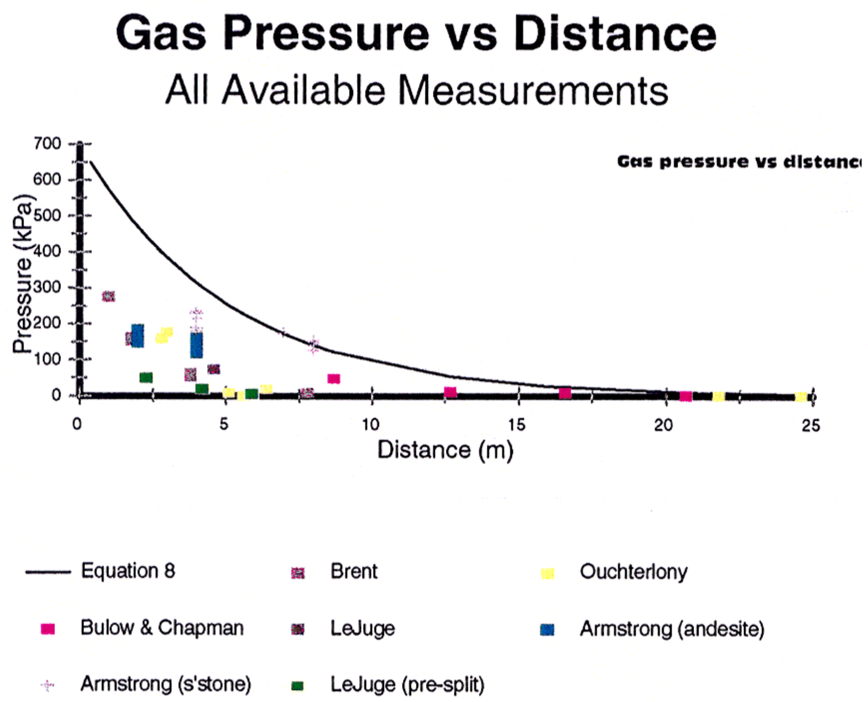

Figure 8*Gas pressure data obtained from literature review.*

It is proposed that scaling can not take into account the state of confinement, since this is an undefined term encompassing the influences of rock structure, rock strength, explosive strength, degree of coupling, and many other influences. Further, the author is not aware of equations which will quantify the effect of exposure time on the distance of gas penetration. It is clear that the ability to sustain a given pressure over an extended time will be dependent on the volume of high pressure gas in the blasthole, and the volume of joints and fractures into which the gas is free to expand. High pressures and volumes may not be necessary where only a single, open joint is available to dissipate the gases.

Consider the case where a decoupled explosive (typical of Hong Kong blasting conditions), produces a blasthole pressure of around 300 MPa. If the pressure is applied for a sufficient period of time (*i.e.* a high state of charge confinement), the combined volume of dilated fractures and blasthole can increase by a factor of approximately 1000 (assuming a non-ideal gas law of $PV^{1.4} = const$), before the pressure decays to a value less than 30 kPa. If the original hole was 76 mm diameter, and the charge length was 2 metres, this corresponds to a joint surface area of around 1000 m^2, for an average crack aperture of 10 mm. Under typical Hong Kong blasting conditions, it therefore seems quite feasible that pressure levels of 30 kPa can be generated at distances up to 15 or 20 metres behind blastholes.

For fully coupled charges, gas volumes and pressures both increase significantly, and the volume of the combined blasthole and dilated fractures can increase by around 4000 times before the pressures decay to less than 30 kPa.

Based on this simple analysis, it may seem that gas penetration behind decoupled blastholes could be reduced to around one half of the penetration which might occur around a fully coupled charge. However, this argument fails to account for the difference in the time over which the pressure is applied. Ouchterlony (1995, 1996) observed that under conditions where explosion gases are allowed to dissipate quickly, gas penetration does not occur to any significant distance behind even fully coupled blastholes.

It would therefore appear inappropriate, without more experimental data, to propose a method of scaling which would permit calculation of safe no-blast distances behind blastholes of differing explosive loading configurations. Depending on blasthole explosive loading configurations, and local structural geology, it is considered that gas could penetrate distances up to 15 metres behind small blasts in Hong Kong, producing joint pressures of around 30 kPa.

6.5 Alternative Controls

It is likely that, due to the need to control factors such as flyrock and general rock movement, it will be impractical or impossible to achieve effective control over gas penetration from blastholes in Hong Kong. As an alternative, we can consider methods of controlling the effects of gas penetration and mobilisation of rock blocks.

One obvious method of controlling block mobilisation from gas penetration is to utilise artificial reinforcement. The use of fully grouted cables, when blasting approaches within a critical proximity, could be expected to be effective in controlling block mobilisation. Such support would serve two purposes - to limit the extent of joint dilation, and to provide additional resistance to motion.

When considering artificial support, a detailed geotechnical study must be undertaken to ensure that the bolts or cables are of adequate length. Three dimensional joint mapping and block modelling is likely to be required, and the proposal by Earth Technology (Appendix A) addresses this requirement.

Although the most dangerous conditions are likely to be associated with blocky rock masses, it may also be that these are readily amenable to stabilisation using appropriate artificial support. Closely spaced, fully grouted dowels of adequate length will permit later fragmentation of portions of the blocks without disturbing those other portions of the blocks which lie outside the volume to be blasted.

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APPENDIX A

Submission by Earth Technology Pty Ltd
on Gas Pressure Modelling
in Jointed Rock

Prepared by: Dr. Alex Kavetsky
Managing Director, Earth Technology

DEVELOPMENT PROPOSAL

Modelling the Effects of Near-field Blast Gas Pressure on Stability of Rock Cuts

Introduction

This proposal is being prepared at the suggestion of Cameron McKenzie of Blastronics Pty Ltd. It will form part of a larger report and proposal being prepared by Blastronics for the Geotechnical Engineering Office of the Hong Kong Civil Engineering Department under Agreement No GEO 4/98.

The proposal is motivated by recent concern regarding stability of rock slopes undergoing reshaping by blasting adjacent to roads in Hong Kong. In a recent blast-induced failure of a rock slope along Sau Mau Ping Road it is believed that gas pressures generated by the blasting contributed significantly to the failure. Consequently it is desired to develop a predictive approach to assessment of the risk of slope failure due to blast-induced gases.

This proposal describes an approach to developing such a risk assessment technique. The proposed approach is based on blast gas propagation modelling developed by Earth Technology as part of the QED Blast Simulator (Austin Powder Company). This in turn is based on theory described in a University of Queensland (Julius Kruttschnitt Mineral Research Centre) PhD thesis [1]-[3] and related publications.

The proposed development requires more detailed modelling of rock structure than that used in muckpile formation. For this it is proposed to adapt and use the model of rock structure developed in another University of Queensland (JK Centre) PhD thesis and related publications [4]--[8]

Executive Summary

The proposed approach to risk-assessment of slope failure due to blast gas is three-dimensional and will combine two relevant models to calculate rock joint properties and blast gas pressure:

- A three-dimensional model of blast gas propagation through rock, used in the QED Blast Simulator (Austin Powder Co) and arising from the doctoral thesis work of Yang [1]-[3].
- A three-dimensional model of rock jointing, from the doctoral thesis work of Villaescusa [4]-[8].

These models will be used to calculate the rock joint geometry and gas pressures for the blast and rock slope being assessed for risk. These calculated quantities will then be used in a stability analysis developed as part of the project, to assess the risk of failure of the rock slope due to blast gas pressure.

The proposed calculation will begin with measurements of rock jointing using line sampling of rock joints. It will apply the Villaescusa Model to calculate the statistical properties of the rock mass. These properties are joint surface density, in-situ block size distribution and distributions of joint orientation, joint size and joint location.

These statistical properties will then be combined with the blast design for further calculations. These calculations will use the Blast Gas Pressure Model from QED to calculate gas pressure in the block joints.

It is proposed that both these calculations will be presented in a computer program allowing users to go step-by-step through the data input and calculations. The three-dimensional gas pressure calculation is quite computer-intensive, and may take some hours to produce a result.

The calculated gas pressure and rock structure can then be used by engineers in a stability analysis, to assess the risk of failure for the given slope, rock structure and blast design. This analysis may be similar in style to the Wong and Pang stability analysis for blast vibrations currently used by GEO [9]. Development of this risk assessment technique forms part of the proposed work. Some initial ideas for an approach to this are presented later in this proposal.

Model calibration and validation is proposed through measurement of gas pressure in boreholes drilled near test blasts. Such measurements do not measure the gas pressure in rock joints directly, but rather the pressure changes in the boreholes due to various processes which occur during the blast. Pressure in such boreholes can even decrease during the blast gas propagation [10], [11]. Nevertheless these techniques provide the most direct method available to tie the predicted pressures to measurements.

The proposed development of gas calculation will consist of three stages:

- Initial development of model calculation
- Measurements of gas pressure in boreholes near blasts
- Model calibration to reproduce calculated pressures

Once the calibrated model has been developed, it can be used for assessment of slope failure risk. The risk assessment method will be developed in parallel with the gas pressure and rock structure calculations.

Modelling Blast Gas Propagation and Muckpile Formation

This section describes the theory used for the Muckpile Model calculation in the QED Blast Simulator. Subsequent sections indicate how this calculation is proposed to be adapted to calculate gas pressure for use in slope stability assessment.

General Description

The explosive gas pressure is an external force, pushing the rock overburden and penetrating the rock joints. In the blasting process, the gas pressure and rock displacement are coupled. As the fragmented overburden begins to move under the action of explosive forces, explosive gases stream into the spaces between the blocks. Consequently, these gases increase in volume and decrease in pressure. These changes both affect and depend on the movement of the rock blocks.

The gas pressure calculation is carried out in discrete time steps and is coupled with the block motion, i.e. the gas pressure influences the movement of the blocks, which in turn affects the gas pressure. This closely simulates the actual blasting process. The calculation is based on the theory of gas flow through porous media. It is conceptually described by Yang [3].

The overburden is modelled as a porous medium, the porosity of which depends on the rock structure before the blast and the displacement of the rock blocks during blasting. The gas pressure is calculated at convenient points throughout this overburden.

At each time step, the porosity of the overburden is calculated from the volume of rock blocks within the gas envelope. The gas pressure is then calculated from the porosity and the adiabatic equation of state. This pressure is used to calculated forces on the rock blocks, which are then advanced one more step in the "time marching" integration. Changes in the gas envelope shape over the time step are calculated from the radial gas penetration velocity, which depends on the pressure and porosity. The process may then be repeated for the next time step.

Gas Penetration into the Overburden

After detonation, the borehole gases penetrate into the overburden. The degree of penetration at a given time defines the gas envelope, as illustrated in Figure 1. This gas envelope contains a mixture of explosive gas and fragmented rock blocks. Its volume is thus given by:

$$V_{env} = V_{gas} + V_{rock} \tag{1}$$

where V_{rock} and V_{gas} are respectively the volumes of rock blocks and voids within the gas envelope. The voids are assumed to be fully taken up by gas.

The gas envelope is calculated from the gas penetration velocity, which depends on the gas pressure and rock porosity according to Darcy's Law [3] written here as:

$$c_{gas} = - \mu \lambda \nabla P \, . \tag{2}$$

Here μ is a coefficient, taken to be a model parameter, ∇P (grad P) is the pressure gradient vector and λ is the porosity of the overburden, defined by the equation:

$$\lambda = \frac{V_{gas}}{V_{env}} \, . \tag{3}$$

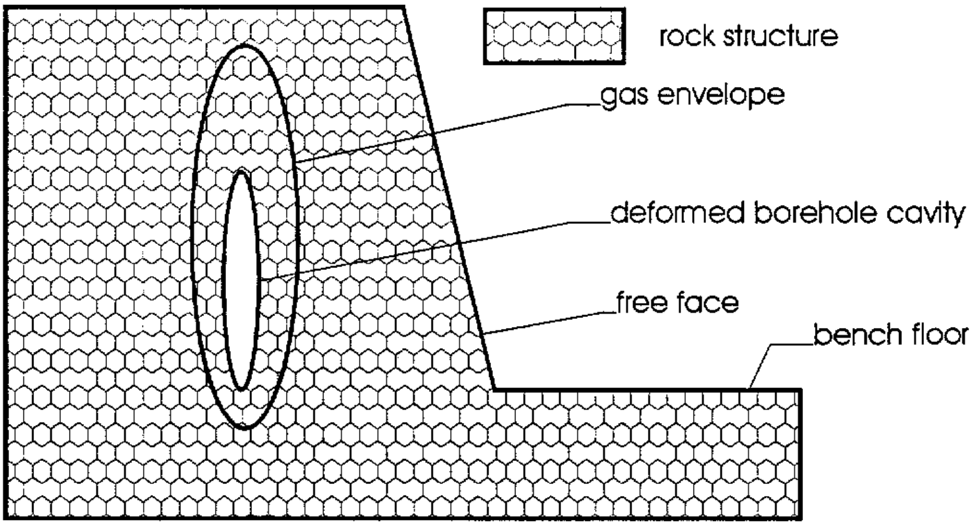


Figure 1 Gas Penetration Coupled with Rock Displacement

The product $\mu\lambda$ is the permeability of the rock for explosive gas penetration.

Substituting for V_{gas} from equation (1) into equation (3) gives for the overburden porosity:

$$\lambda = 1.0 - \frac{V_{rock}}{V_{env}}. \tag{4}$$

Using Average Gas Pressure, P_{av}

Assuming adiabatic gas expansion, the governing equation of the explosive gas is:

$$P_{av} V_{gas}^{\gamma} = R \tag{5}$$

where *g* is a constant for a given gas (the ratio of specific heats) and *R* is the universal gas constant.

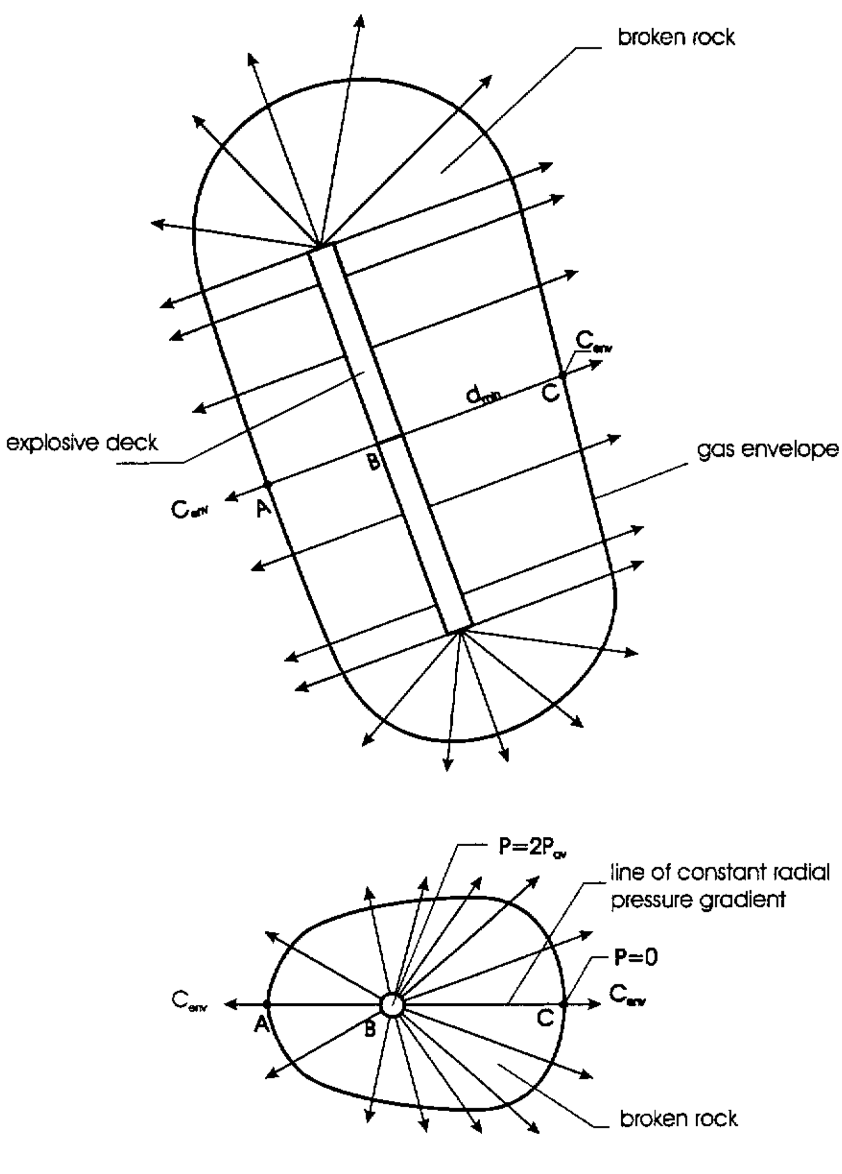
For the purpose of calculating the radial gas penetration velocity at the edge of the gas envelope, the radial pressure gradient at a point due to a single explosive deck is simplified to be

$$\nabla P_{rad} = \frac{2 P_{av}}{d_{\min}} \tag{6}$$

where d_{min} is the minimum distance from the corresponding point on the gas envelope to the explosive column contributing to the gas pressure. The geometry is illustrated in Figure 2. Here the magnitude of the pressure gradient is assumed to be constant throughout the gas envelope and directed from the point to the explosive column. The pressure is assumed to fall to atmospheric pressure (essentially zero) at the edge of the gas envelope. P_{av} can then be interpreted as an average gas pressure along the line of constant radial pressure gradient.

The envelope spreading velocity C_{env} is then found by substituting equation (6) into equation (2) to give

$$c_{env} = -2\mu\lambda \frac{P_{av}}{d_{min}} \tag{7}$$



Plan view of section through ABC perpendicular to deck

Figure 2 Calculation of radial gas penetration velocity and the gas envelope

Initial Conditions

Porosity of overburden

The initial porosity of the overburden depends on the rock mass properties, and in particular the degree of jointing and voids within the intact overburden. It is calculated from:

$$\lambda_o = \frac{Void(pre\text{-}blast)}{Total\ Volume} \tag{8}$$

Here "*Total Volume*" is the volume of the overburden including rock and voids.

Gas pressure

The peak borehole pressure is given by:

$$P_{max} = 0.11 f_c^{1.2} \rho_{exp} VOD^2 \tag{9}$$

where f_c is the decoupling ratio and the other symbols have their usual meanings.

The initial pressure used in the calculation of muckpile formation is an average pressure defined here as the "heave pressure", P_o. This pressure is lower than the peak borehole pressure, i.e. the borehole pressure drops to the heave pressure due to borehole expansion and gas penetration into the overburden during the initial fragmentation process. This is assumed to occur before any significant overburden movement takes place. The initial state of an explosive deck and nearby rock at its heave pressure is illustrated in Figure 3.

The heave pressure is calculated from the peak borehole pressure. Assume that when the gas is at its heave pressure, it has penetrated an average distance d_o into the rock of porosity λ_o. Let $V_{o\ gas}$ be the volume occupied by the gas when at its heave pressure. This may be written as:

$$V_{o\ gas} = V_{o\ deck} + \lambda_o V_{d_o overburden} \tag{10}$$

where $V_{o\ deck}$ is volume of the (deformed) section of borehole containing the explosive deck and $V_{do\ \mathrm{overburden}}$ is the volume of overburden into which the gas has penetrated.

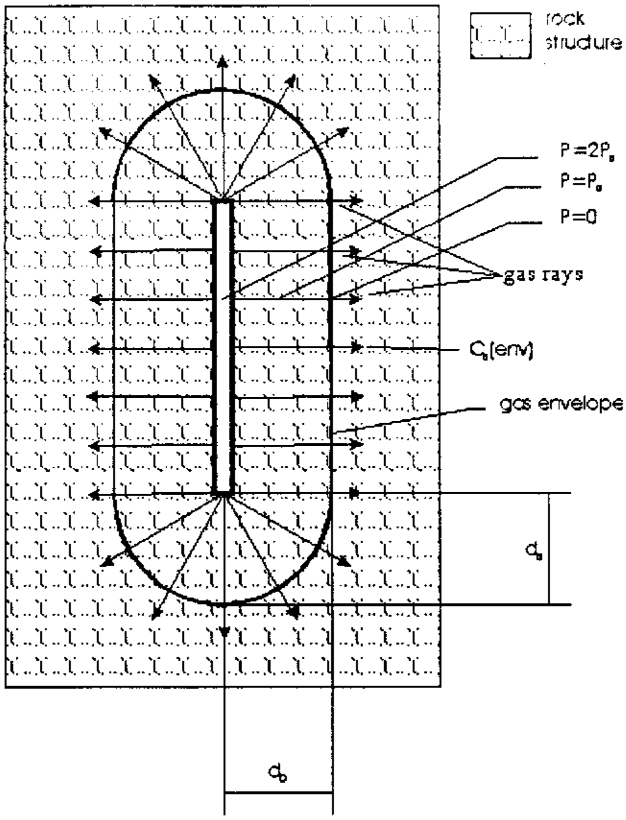


Figure 3 Initial gas penetration and heave pressure P_0

Applying equation (5) to the gas states at peak borehole pressure and heave pressure, gives:

$$P_o V_{o\,gas}^{\gamma} = P_{\max} V_{deck}^{\gamma} \tag{11}$$

where V_{deck} is the volume of the explosive deck at peak borehole pressure. Then substituting equations (9) and (10) into (11) gives for the initial heave pressure:

$$P_o = 0.11\, f_c^{1.2} \rho_{\exp} VOD^2 \left[\frac{V_{deck}}{V_{o\,deck} + \lambda_0 V_{d,overburden}}\right]^{\gamma}. \tag{12}$$

Velocity of gas penetration

The initial velocity of gas envelope penetration into the overburden from equations (2) and (6) is given by:

$$c_{o\,env} = -2\,\mu\,\lambda_0 \frac{P_o}{d_o}. \tag{13}$$

Steps In Calculation Of Burden Movement And Muckpile Shape

The calculation naturally divides into the following sections:

- initialize;

- update gas envelope;

- update gas pressure;

- calculate pressure forces;
- collide blocks: calculate interaction forces;
- move blocks: update positions and velocities one time step;
- land blocks: find landed blocks, spread, update terrain;
- repeat for unlanded blocks.

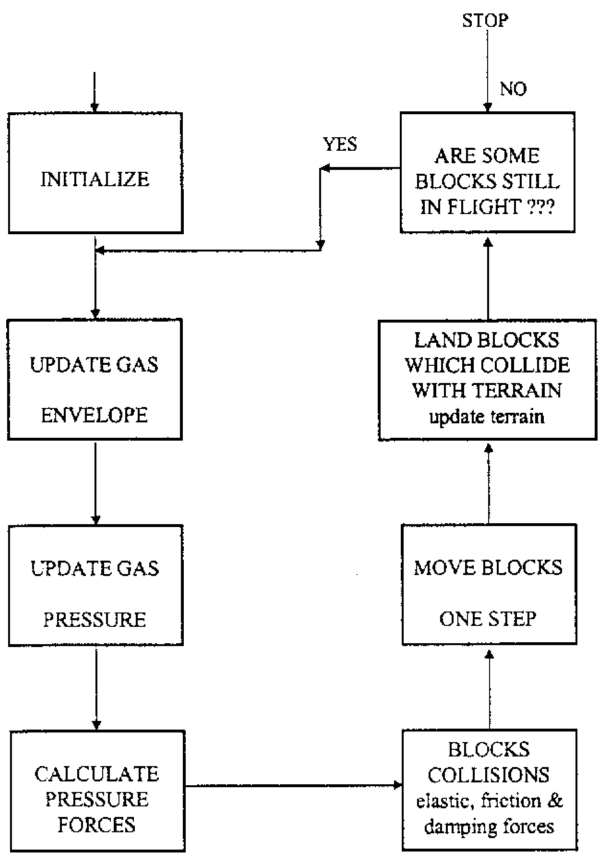


Figure 4 Overall calculation scheme for muckpile model

Figure 4 shows how these sections fit into an overall calculation scheme for the muckpile model.

Initialize

In this section the overburden is divided into computational blocks. Detonation times are calculated for each explosive deck according to established algorithms. Heave pressures for each deck are calculated using equation (12).

Update Gas Envelope

The gas envelope boundary for each detonated deck is recalculated using the envelope velocity calculated from equation (7) and (initially) equation (13) above. The average pressure P_{av} used in equation (7) is updated on each time step as described below.

In the calculation of c_{env} for each point on the envelope, a different porosity λ is used. This porosity is calculated using equation (4). V_{rock} is estimated for each point as an average from the explosive deck to the point, using current block positions and the radial geometry illustrated in Figure 3 above.

Update Gas Pressure

The average gas pressure inside the gas envelope is calculated at each time step from the volume of blocks inside the gas envelope. Once the volume of blocks inside the gas envelope and the volume of the gas envelope itself are known, the volume of gas inside the gas envelope V_{gas} may be calculated from equation (1).

Equation (5) may then be applied to the gas at its average heave pressure and new average pressure to give:

$$P_{av} = P_o \left[\frac{V_o}{V_{gas}}\right]^{\gamma} \tag{14}$$

Here P_o and V_o are the average gas heave pressure and total gas volume at this heave pressure respectively. They are given by equations (10) and (12) above.

Calculate Pressure Forces

The pressure forces are given by a ∇P terms in the equations each block, the gas pressure forces are calculated from the pressure gradient (∇P) at the block centre.

Firstly, the radial pressure gradient at a point due to an explosive deck is calculated from equation (6). Here d_{min} is interpreted as the minimum distance from the point to the explosive deck.

The radial pressure gradient is used to calculate a pressure on a three-dimensional grid of points. This calculated pressure is used to calculate a three-dimensional pressure gradient, that is then used in the calculation of gas pressure forces as above.

Collide Blocks

Block interaction is based on the Distinct Element Method [12]. Block collisions are detected using an efficient algorithm that localizes the blocks which must be tested for collision with each given block. This greatly speeds up this part of the calculation.

Once collisions are detected for a given block, the elastic, friction and damping forces acting are calculated. Blocks for which a cohesive bond has been established to simulate initial rock strength are also tested at this stage, using a somewhat less stringent definition of "collision" so they can be detected. Cohesive restoring forces are then added if necessary.

The calculated forces are then added to the total forces acting on each block involved in the collision. Care is taken to ensure that forces are never recalculated i.e. each collision between a given pair of blocks is only detected once.

Once all collisions are processed the total contact forces acting on each block are resolved along the coordinate directions for subsequent use in the equations of motion.

Move Blocks One Step

The equations of motion of each block may be integrated using a time-stepping integration method, to give block velocities and positions with time. The integration method used (the explicit Newmark algorithm) closely follows Bardet and Scott [13]. For this reason, and since it is quite lengthy, it will not be described here.

The effect of this integration at each time step, is to update the block positions and velocities over a time Δt, where Δt is the time step. These new positions and velocities are then used in the next time step to continue the integration.

Land Blocks

After updating the block positions one time step, each block is checked for collision with the terrain model. This model describes terrain surrounding the blast, including the terrain under the now-removed overburden. Each block whose position is below the current terrain surface has collided with the terrain. Its exact collision point may then be calculated, and the block's volume added to the terrain using the dump algorithm for Earth Technology Pty Ltd's 3d-Dig Excavation Editor.

The dump algorithm accurately simulates the actual rilling process undergone by material dumped on a surface. It calculates new values of terrain surface heights, taking into account the dumped volume, material repose angle and existing terrain shape. Calculations are performed iteratively, in order to accurately model the new terrain shape resulting from dumping on any topography.

Repeat for Unlanded Blocks

If after landing all colliding blocks, some blocks still remain in flight, the calculation cycle is repeated. This continues until no unlanded blocks remain. The terrain model then represents the final calculated muckpile shape.

Modelling Rock Structure in Three Dimensions

Rock joints are three dimensional. However, sampled observations are at best two-dimensional, such as when data are gathered from outcrops or rock cuttings.

To solve the problem of rock joint characterization, therefore, it is required to characterize the two-dimensional data, correct for sampling bias, and establish a link with a three-dimensional model of rock jointing.

It is proposed to do this through the three-dimensional rock joint model and calculation methodology of Villaescusa [4]–[8]. This model provides a three-dimensional framework to model rock jointing based on geometrical probability, statistical theory and mathematical stereology. The model allows simulation of the stochastic nature and properties of joint set characteristics such as joint size, location and orientation, based on two-dimensional data. It also allows derivation of aggregate measures of rock jointing such as joint surface density and in-situ block size distribution.

The Villaescusa model includes a rigorous data collection scheme. The method collects data from line sampling of rock joints, but the line transect is placed within a convex "observation window", similar to those of cell mapping. Only those joints intersecting the line are recorded and individual observations are classified according to the number of visible end-points within the observation window. The technique allows for subsequent correction of mapping biases such as edge effects and progressive censoring.

In the Villaescusa model, joints are considered as two-dimensional convex discs geometrically defined in space by their center location, disk diameter and orientation [14]. Joint orientation is simulated using a spherical normal distribution. The ill-posed mathematical problem of joint size estimation by stereology is solved in the Villaescusa model in a credible and elegant way. This makes the model suitable for the proposed calculations, as it provides all the information on joint orientation and location needed for detailed blast gas propagation calculation.

Modelling Blast Gas Pressure in Structured Rock

It is proposed to modify the blast gas pressure calculation described above, to make explicit use of the Villaescusa model. This will make the geometry of the explosive gas calculation more realistic, by introducing explicit information about rock joint structure into the gas calculation. The largest gas forces are built up early in a blast, before bulk overburden movement takes place. It is just these forces which are of interest to slope stability analysis. Thus the model will not need to calculate the full overburden movement and muckpile formation.

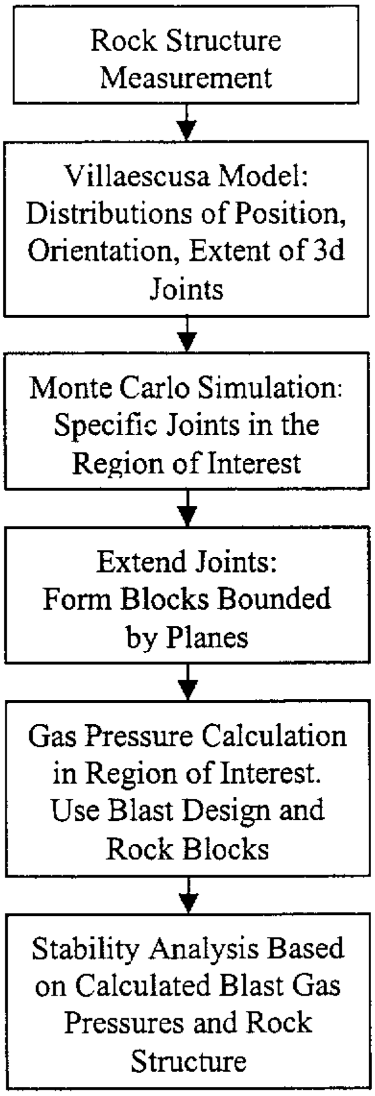


Figure 5 Schematic Illustration of proposed stability analysis due to blast gas pressure.

A schematic illustration of the proposed gas pressure calculation is shown in Figure 5. The proposed calculation will begin with measurements of rock jointing using line sampling of rock joints. It will apply the Villaescusa Model to calculate the statistical properties of the rock mass. These properties are joint surface density, in-situ block size distribution and distributions of joint orientation, joint size and joint location.

These statistical properties will then be combined with the blast design for further calculations. These calculations will use an extended form of the Blast Gas Pressure Model from QED to calculate gas pressure in the block joints.

The calculation will proceed similarly to the above description. Explicit rock blocks, calculated from the Villaescusa model to reflect the in-situ rock structure, will be used in the calculation. These blocks will be oriented according to the joint set orientation calculated by the model.

The joints calculated from the Villaescusa model give a statistical description of the rock mass. This statistical description can be used to calculate a specific realization of rock joints in the region of interest (i.e. location, extent and orientation of each joint). This is done using a "Monte Carlo" simulation to generate specific joints according to the necessary statistical distributions. The generated joints can then be extended to link up and form "blocks". Such blocks are solid elements bounded by planes. They will be assumed to be "free" i.e. no cementing of rock joints.

The extension and freeing of the rock joints described above is justified by rock breakage due to blast shock. Additional breakage within blocks (i.e. formation of new joints) introduced by blast shock will not be introduced into the calculation. This will give a conservative estimate of gas pressure, as additional breakage will create more space between blocks during initial block movement and hence lower gas pressure.

The calculation of block motion will require extension of the present distinct element calculation, to allow for arbitrarily-shaped blocks bounded by planes. Theory for this is available in published literature [15].

It is proposed that both these calculations (the rock structure model and gas pressure propagation model) will be presented in a computer program allowing users to go step-by-step through the data input and calculations. The three-dimensional gas pressure calculation is quite computer-intensive, and may take some hours to produce a result.

The calculated gas pressure and rock structure can then be used by engineers in a stability analysis, to assess the risk of failure for the given slope, rock structure and blast design. This analysis may be similar in style to the Wong and Pang stability analysis for blast vibrations currently used by GEO [9]. Development of this risk assessment technique forms part of the proposed work. Some initial ideas for an approach to this based on the Wong and Pang [9] method are presented below.

Model calibration and validation is proposed through measurement of gas pressure in boreholes drilled near test blasts. Such measurements do not measure the gas pressure in rock joints directly, but rather the pressure changes in the boreholes due to various processes which occur during the blast. Pressure in such boreholes can even decrease during the blast gas propagation [10], [11]. Nevertheless these techniques provide the most direct method available to tie the predicted pressures to measurements.

The proposed development of gas calculation will consist of three stages:

- Initial development of model calculation
- Measurements of gas pressure in boreholes near blasts
- Model calibration to reproduce measured pressures

Once the calibrated model has been developed, it can be used in the risk-assessment method. This method can be developed in parallel with the gas pressure and rock structure calculations.

Development of a Risk Assessment Technique

Once the gas pressure for a given blast design, rock structure and rock slope has been calculated, this pressure can be used in a stability analysis to assess risk of failure. The blast gas pressure calculated by the method proposed above will vary with time and with position of the blast relative to the rock slope of interest.

The simplest way to use the modelled pressure, is to take its average maximum value in a region of interest, and apply this value in a stability analysis. This will give a conservative assessment of stability. It is possible to develop other assessment techniques which will use more of the information on rock structure and gas pressure variation generated by the model. For example, the joint orientation will give likely orientation of blocks relative to the rock slope.

This section will describe a possible approach to stability assessment due to gas pressure, using a single static pressure value and assuming that blocks are oriented along the rock slope.

Stability of a rock block resting on a rock slope is defined by the so-called static factor of safety FOS. This is given by the following expression:

$$FOS = \frac{\tan \phi}{\tan \alpha} \tag{15}$$

where α is the slope inclination and ϕ is the rock friction angle.

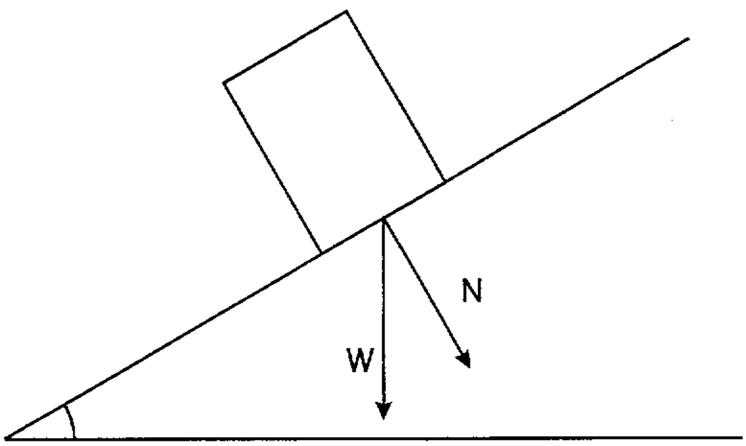


Figure 6

When FOS>1, the rock block remains in its initial position, otherwise it slides down along the slope.

Blasting may cause displacement of the block down the slope even when the static factor of safety is greater than unity. There are two possible mechanisms of this effect.

- The first is connected to blasting vibration. High frequency (30 … 100 Hz) pulses transmit energy which reduces the actual stability of the rock.
- The second concerns gas pressure. Its action is equivalent to increasing the actual factor of safety.

In [9] the stability of rock slopes subjected to blasting vibration is assessed using equations based on the principle of conservation of energy. In this approach the peak particle velocity PPV of the rock block is a key parameter. To estimate the rock stability a critical value PPV_c is calculated using parameters of the rock joint. In some cases if PPV exceeds PPV_c infinite downslope displacement, i.e. complete failure, will occur.

The influence of the gas pressure P_w (normalized to the base area of the rock block), can be taken into account by correcting the static factor of safety FOS according to the following relations

$$FOS' = (1-\frac{P_w}{N})FOS\ , \tag{16}$$

where N=Wcosα, W=mg.

This gives us the critical value of the gas pressure corresponding to FOS'=1 as follows:

$$P_w = (1-\frac{1}{FOS})W\cos\alpha\ , \tag{17}$$

or

$$\frac{P_w}{W} = (1-\frac{\tan\alpha}{\tan\phi})\cos\alpha\ . \tag{18}$$

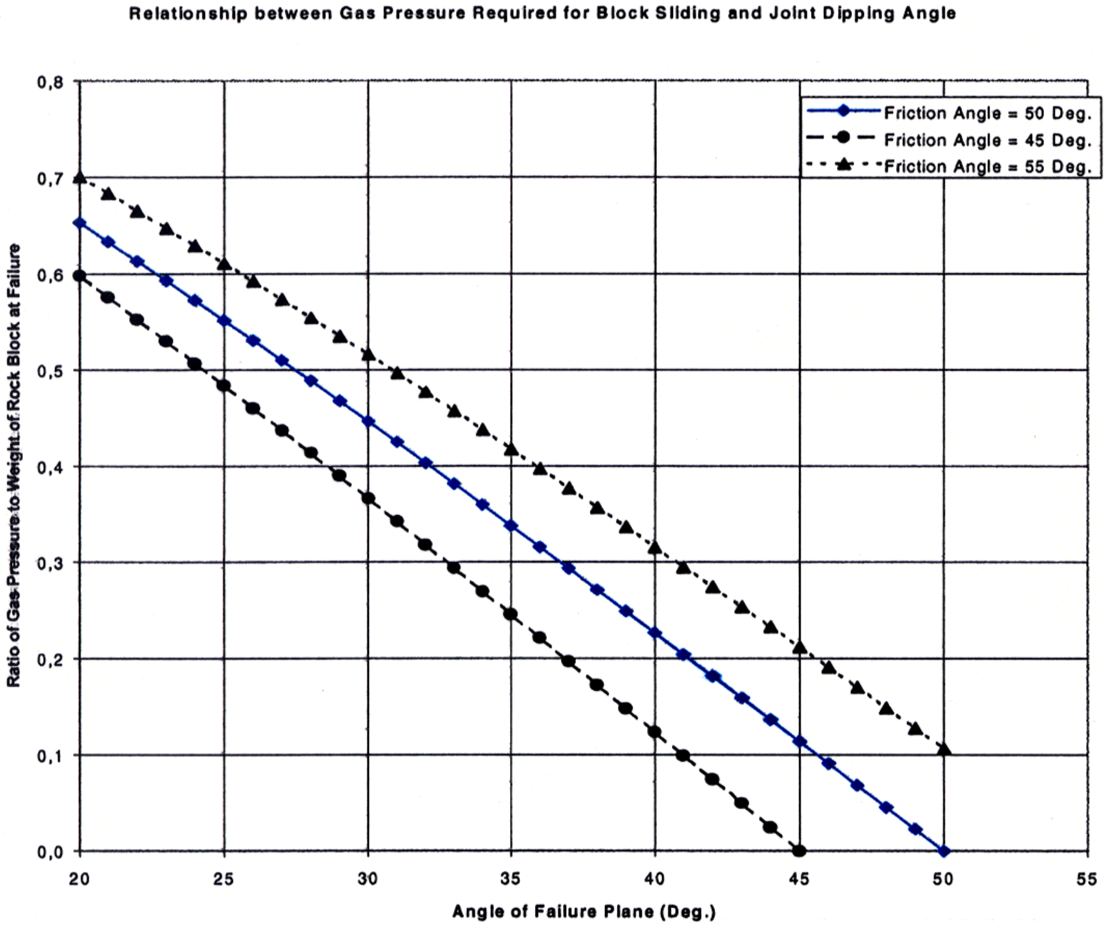


Figure 7

Figure 7 shows the relationship between critical gas pressure and joint dipping angle given by expression (18) for three fixed values of joint friction angle.

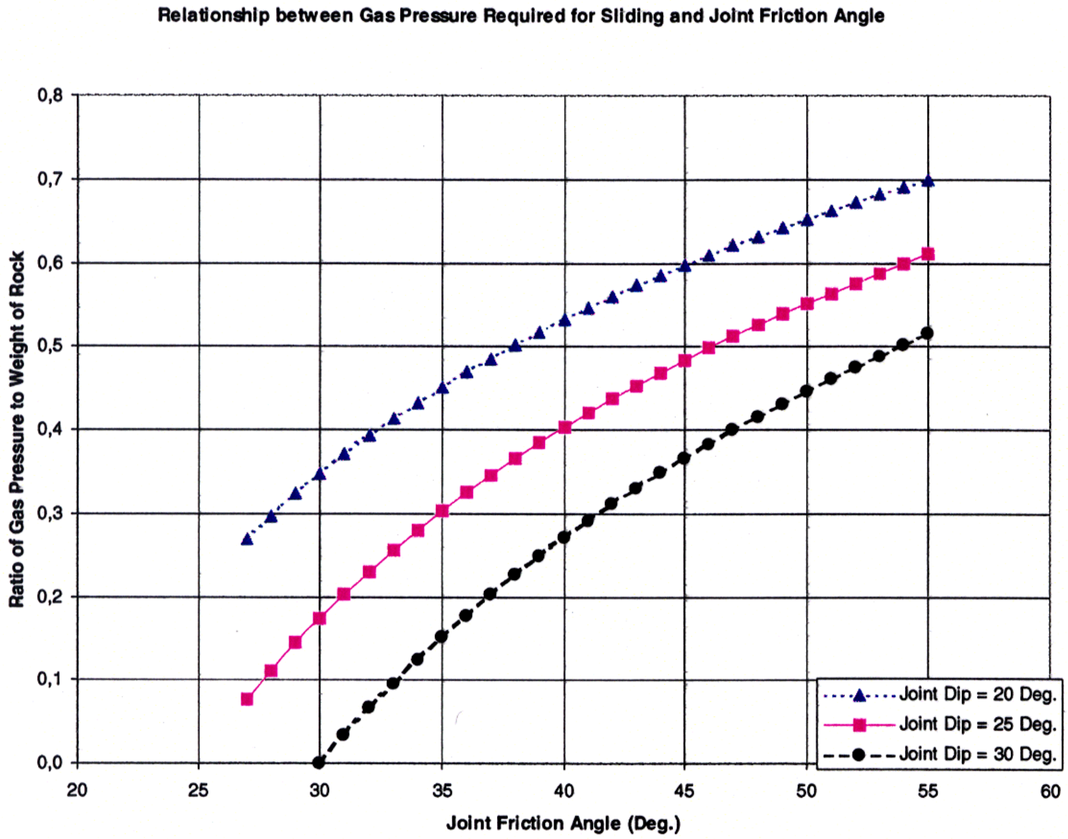


Figure 8

Figure 8 illustrates how the critical gas pressure depends on joint friction angle while joint dipping angle is constant.

One can see that even for reasonably high values of static safety factor FOS (for example when friction angle ϕ is equal to 50° and joint dipping angle α is equal to 20° FOS>3) if gas pressure exceeds 0.7 of rock block weight, the block loses stability.

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