Development of Predictive Models for Controlling Blast-Induced Overbreak in Tunnels

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ABSTRACT

Drilling and blasting continues to be the predominant rock excavation technique in driving horizontal openings and tunnels for underground construction and mining purpose. Faster drivages, attempted these days for reducing the long gestation periods of projects, have often resulted in large overbreak. This is due to the unacceptable levels of ground vibration to which the rock is subjected. Factors contributing to this include longer pulls offering more confinement; burn cuts with high explosive concentration per hole and per delay etc. Blast-induced rock damage (BIRD) assessment based on far-field blast vibration (peak particle velocity (PPV)) measurement, when extrapolated near the face, has often resulted in suggesting higher PPV threshold levels thus limiting the maximum explodable charge per delay. This poses great constraint to increase pulls particularly in lined tunnels where concrete setting and preserving its strength is of significant importance. Apart from these disadvantages, the geological and structural features play a dominant role in masking the intensity of blast waves. It is evident from the above reasoning that a rational approach needs to be evolved for suggesting possible “PPV/g” values to control the blast-induced overbreak. Therefore, near-filed monitoring using accelerometers has been attempted in one of the metal mines to study the blast damage in faces excavated by burn cut. Present seismographs available have limited monitoring PPV range upto 2540mm/s and are not suitable for near-field monitoring. This paper reports the investigations carried out for measuring acceleration, PPV and overbreak in a development heading for arriving at a suitable method of predicting blast-induced rock damage. The analysis of both acceleration and PPV measured against overbreak has revealed that near-field acceleration monitoring is more acceptable than the far-field PPV monitoring for predicting blast-induced overbreak.

Keywords: Tunnels, blast-induced overbreak, acceleration, peak particle velocity, threshold levels

1. INTRODUCTION

Underground construction for mining as well as for civil engineering projects requires driving of drifts and tunnels in a large number. In recent years, mechanical excavation with drifting and tunneling rigs (Road headers, TBMs) has advanced considerably, excavating rocks with compressive strengths up to 250MPa. However, excavation with explosives is still widely accepted technique as the mechanical cutting has its inconveniences due to rigid work system (as the sections must be circular), ground to be excavated must not have important variations or geological upsets, curves should have a radius over 300 m, initial excavation is costly and personnel must be highly specialized.

Excavation with drilling and blasting solves most of these problems but is seriously affected by poor drivage rate. Attempts to get more pull, sometimes leads to roof rock damage. In order to control and reduce blast-induced rock damage, assessment of the extent of damage is a pre-requisite. Most of the existing criteria relate damage to ground vibrations resulting from dynamic stresses induced by the blasting process. An attempt has been made to monitor the blast-induced accelerations and peak particle velocity (PPV) simultaneously to arrive at predictive models for controlling overbreak.

2. BLAST DAMAGE ASSESSMENT

Blast-induced rock damage prediction models in tunnels assume greater importance to minimize the same adopting suitable site-specific blast designs. A host of geo-technical, explosive, blast design and operational parameters influence it. However, estimation of overbreak from the ground vibration, in terms of peak particle velocity, is found to have increased application in recent times as discussed in Murthy et al. (2002). This is due to the fact that peak particle velocity (PPV) has been accepted as a parameter to assess the structural/rock damage world over today. The PPV/acceleration based damage estimation models and the damage levels for overbreak are below:

Crandell (1949) proposed that the damage caused by the blast vibrations was proportional to the energy ratio. The energy ratio, ER, was defined as ratio of the squares of the acceleration, a, and the frequency, f.

$$ER = \frac{a^2}{f^2} \tag{1}$$

Langefors and Kihlstrom(1973), Edwards and Northwood (1960) and several others proposed particle velocity as a blast damage criteria.

a) There was a common agreement that a PPV of less than 50 mm/s would have low probability of structural damage to residential buildings.
b) There is scarcity of data relating PPV to rock damage in underground openings.

Langefors and Kihlstrom (1973) have proposed that PPV of 305 mm/s and 610 mm/s results in fall of rock in unlined tunnels and formation of new cracks respectively.

Bauer and Calder (1970) observed that no fracturing of intact rock will occur for a PPV of 254 mm/s, whereas PPV of 254 to 635 mm/s results in minor tensile slabbing and PPV of 635 to 2540 mm/s would cause strong tensile and some radial cracking. Break-up of rock mass will occur at a PPV of 2540 mm/s.

Holmberg and Persson's (1979) stated that damage is a result of induced strain (ε) which is given by,

$$\varepsilon = V/c \qquad (2)$$

where,
V = peak particle velocity and
c = Characteristic propagation velocity of (P/S/Rayleigh wave).

It was also observed by them that the proposed generalized PPV equation is valid only for the distance that are long in comparison to charge length, so that charge can be considered as concentrated. For an extended charge of linear charge concentration l (kg/m), they obtained a first approximation of the resulting PPV by integrating the generalized equation for the total charge length.

$$V = K\, l^{\alpha} \left[\int_0^H \frac{dx}{\left\{ D^2 + (D - x)^2 \right\}^{\frac{\alpha}{2\beta}}} \right]^{\alpha} \qquad (3)$$

For arbitrary explosive (not ANFO), weight strength must be made equivalent of ANFO. For competent Swedish bed rock masses the constants used are K = 700, α = 0 .7 and β = 1.5. The computed damage zones is estimated from a plot if V vs. R

Bogdanhoff (1995) monitored near field blast acceleration of an access tunnel in Stockholm. Vibration measurements were done at distances between 0.25 and 1.0 m. outside tunnel perimeter holes with accelerometers. Altogether eight blasts were monitored and the vibrations were filtered and PPV in the assumed damage range was found to be between 2000 and 2500 mm/s.

Blair and Minchinton (1996) proposed that Holmberg model warrants further investigation. The Holmberg model assumes that for blast-hole of length, L the vibrations peaks (such as V1 and V2) may be numerically added at point P to yield the total peak vibration, VT. Blair argued that as this model does not incorporate any time lag for the vibration peaks at point P the model is not capable of providing the correct near field analysis. They developed a Dynamic finite element model to assess the damage zone.

Holmberg and Persson (1997) extended the applicability of their model and showed from comparison of theoretical and experimental values that the effective parts of elemental waves arrive at a point almost simultaneously. They, therefore, neglected the difference in time of the arrival of elemental waves from different parts of charge.

3. DESIGN OF EXPERIMENT AND INSTRUMENTATION SCHEME

Most of the damage threshold levels are arrived at using far-field vibration monitoring and extrapolation to near field. To understand the blast-induced damage it is necessary to monitor close to the blast site to arrive at ground vibration threshold levels for rock damage. One such monitoring by Bogdanhoff (1995) using uniaxial accelerometers has indicated that the PPV range for rock damage could range between 2000 and 2500 mm/s. In Indian conditions no such effort has been made. Near-field ground vibrations levels during blasting are too high to be measured with ordinary/high frequency geophones in the underground and hence accelerometer based seismograph (Fig. 1) with a acceleration measuring range up to 500g has been put to use in the current study. The high frequency geophone based seismograph and triaxial geophone based seismograph were also used for the cross verification of vibration levels and their limitation in predicting rational overbreak threshold levels. The insitu rock strength was determined using Schmidt rebound hammer and laboratory testing was also carried out on the core samples after suitable preparation. To determine the dynamic strength of the rock, P-wave and S-wave velocities were measured using Sonic Viewer of OYO Corporation, Japan. Joint characteristics were also studied in an attempt to determine the RMR (Bieniawski, 1979) and Q-index (Barton et al., 1974). Overbreak for the each blast has been measured using overbreak measuring telescopic rod (Fig. 2), designed and fabricated in Indian School of Mines, Dhanbad, India under the supervision of the authors. The sensors used in the study with their broad specifications are mentioned in Table 1. The overbreak measurement scheme using telescopic offset rod and the fixing arrangement of accelerometers has been shown in Fig. 3 and Fig. 4 respectively.

Table 1 – Major specifications of seismic sensors used in the study

Parameters	Sensor		
	Accelerometer	High frequency geophone	Triaxial geophone
Frequency range	1 Hz to 3 kHz	1Hz to 2 kHz	2 to 300 HZ
Amplitude range	Upto 500 g (4903 m/s^2)	Geophone natural frequency: 28Hz	Upto 254 mm/s
Others	For near-field ground vibration monitoring	Used for near-field high frequency monitoring applications	Measures vertical, transverse and longitudinal ground vibrations.

4.0 FIELD INVESTIGATIONS

4.1 Geology and Geotechnical Investigations

Investigations were carried out in one of the metal mines in eastern India where burn cut is practised on a large scale. Due to higher confinement and inadequate free face the ground vibrations are normally higher irrespective of the best possible delay sequence. A study has been carried out to assess the blast-induced damage to the rockmass. The type of rock is chlorite-sericite-schists of massive metamorphic formation. Some of the geo-technical parameters collected from mine reports are tabulated in Table 2.

Table 2 – Laboratory test results on rock samples

Rock property	Value	Rock Property	Value
R. Q. D.	81.67	Q - Index	5.11
Cohesion strength (MPa)	13.5	Uniaxial compressive strength (UCS) of hangwall (MPa)	77.64
Angle of internal friction (Deg.)	41	UCS of ore (MPa)	64.45
Tensile strength of ore (MPa)	10.45	Tensile strength of hangwall (MPa)	10.27
Young's modulus of ore (GPa)	35.89	Young's modulus of hangwall (GPa)	28.66
P-wave velocity (km/s)	4.5 – 6.1	Poisson's ratio	0.1 – 0.04
S-wave velocity (km/s)	2.5 – 3.5	RMR (Bieniawski)	66

4.2 Mining Subsystems

The mine has both mechanized and manual face workings. The subsystem details of both the faces are described in Table 3. The blasting pattern practiced is burn cut with large dia relief holes. The details of blast and charging pattern for both the workings are shown in Figs.5 & 6 and Tables 4 & 5 respectively.

Table 3 – Details of mining subsystems in mechanized and manual face

<table>
<tr><th colspan="2">Parameters</th><th>Mechanised face</th><th>Manual face</th></tr>
<tr><td colspan="2">Face size</td><td>5×3.2 m</td><td>4×3 m</td></tr>
<tr><td rowspan="5">Drilling</td><td>Diameter of blasthole (mm)</td><td>38</td><td>32</td></tr>
<tr><td>Diameter of reamer hole (mm)</td><td>64</td><td>32</td></tr>
<tr><td>No. of reamer holes</td><td>4</td><td>1</td></tr>
<tr><td>Drilling length (m)</td><td>3.2</td><td>1.6</td></tr>
<tr><td>Machine used for drilling</td><td>Jumbo Drill, Atlas Copco Make</td><td>Jack hammer with air leg</td></tr>
<tr><td rowspan="2">Blasting</td><td>Explosive and detonator used</td><td colspan="2">Explosive used: Powergel 801, Nobel gel, Belmx, Indorock
Short and long delay detonators manufactured by IEL were used.</td></tr>
<tr><td>Short and long delay used</td><td>Refer Fig. 5 and Table -4</td><td>Refer Fig. 6 and Table -5</td></tr>
<tr><td rowspan="2">Loading and Transportation</td><td>Mucking</td><td>LHD and Scoop Tram</td><td>Rocker shovel</td></tr>
<tr><td>Transportation</td><td>Mine truck of 25t capacity or Low Profile dump truck of 10t capacity dumped in ore pass or directly in stope for filling</td><td>Tub of 0.6m^3 capacity hauled by battery locomotive.</td></tr>
<tr><td>Support</td><td colspan="3">The support system used in the mine was rock bolting. Rock bolts were used as the permanent support for the drifts and declines and as well as for raises and winzes.
For drift/decline: 1.6m × 1.6m grid pattern
Bolt length :1.6 m, Dia : 32 mm dia with twisted surface.
Shotcrete/grouting mixture: 1:1:0.5 (cement: sand : water)
Pull out strength of bolt: 16 ton
Maximum unsupported span from drift face = 2.5m
For Large permanent excavations/junctions: - 1.2m × 1.2m grid</td></tr>
</table>

Table 4 – Explosive charging pattern in the mechanised drift (4.5 m x 3.2)

Hole(s)	Delay No.	No. of Holes	Charge/Hole (Cartridges)	Total Charge (Cartridges)
Center hole	0	1	10 + 1P	11
1st square	1, 2, 5, 8	4	10 + 1P	44
2nd square	II x 4, III×4	8	11+ 1P	96
3rd square	IV x 4, V×4	8	11 + 1P	96
Easers	VI x 6, VII×3	9	12 + 1P	117
Side holes	VIII×6	6	11+1P	72
Top holes	IX x 8	8	10 + 1P	88
Bottom holes	X x 8	8	12 + 1P	104
Total		52		628
Depth of round: 3.2 m Blast hole dia : 38 mm Reamer hole dia : 64 mm Total no. of cartridges: 628			Cartridge dia : 32 mm Wt. of cartridge: 0.220 kg. Total explosive: 138.16 kg. Total Yield: 121 t (expected in 3 m pull)	
Powder factor: 0.87 t/kg				

Table 5 – Explosive charging pattern in the manual face (4 m x 3 m)

Hole(s)	Delay No.	No. of Holes	Charge/Hole (Cartridge)	Total Charge (Cartridge)
Center hole	Reamer(R)	1	0	0
1st square	I×4, II×4	8	4 + 1P	40
2nd square	III×4, IV×4	8	4 + 1P	40
Easers	V×4, VI×4, VII×4	12	5 + 1P	72
Side holes	VII×2, VIII×4	6	5 + 1P	36
Top holes	VII×1, IX×2, X×2	5	4 + 1P	25
Bottom holes	VII×1, IX×2, X×2	5	6 + 1P	35
Total		44+1		248
Depth of round: 1.6 m Blasthole dia : 32 mm Reamer hole dia : 32 mm Total no. of cartridges: 248		Cartridge dia : 25 mm Wt. of cartridge: 0.125 kg. Total explosive: 31 kg. Total Yield: 50.4 t (with 1.5 m pull)		
Powder factor: 1.62 t/kg				

4.3 Ground Vibration Measurement and Analysis

Blast-induced acceleration measurement has been done using accelerometer of 500g range manufactured by Instantel Inc. Canada, for the first time in India. PPV has also been monitored using Minimate 077 of the same manufacturer. The monitored accelerations have been integrated to achieve derived PPV (here in after referred as DPPV). Scaled distance of the each blast has been calculated using the formula (Eq. 4) proposed by Ambraseys and Hendron (1968). Data pertaining to vibration measurement in the mine have been tabulated in Table 6. The acceleration measured in the field are given in unit ‘g’ and it was integrated to achieve PPV given in mm/s. The actual PPV measured are also with the unit mm/s.

$$SD = \frac{R}{\sqrt[3]{W}} \tag{4}$$

where,
SD = Scaled distance,
R = Distance of instrument from blast (m), and
W = Maximum charge per delay (kg).

Table 6 – Blast vibration monitoring details in the horizontal drivages

Max. charge/ delay (kg)	Distance (m)	Acceleration (g)	PPV (mm/s)	PPV derived from acceleration (mm/s)	Acceleration derived from PPV (g)	Scaled distance (D/Q^1/3)
21.2	56		14.9			
26	58	2.06		57.1		19.57
22	30	3.41		10.5		10.70
16.2	61	1.35		13.4		24.10
18.2	63	1.73		29.3		23.95
9	34		1.37		0.0732	16.34
8	33		1.96		0.0832	16.5
8	27	12.7		53.6		
18	20	2.95		32.1		7.63
15.4	13	9.19		156		5.22
15.4	45		180		4.71	18.08
15.4	22.7	31.5		576		
15.4	52		9.01		0.447	20.90
34	59	2.31	8.48	74	0.298	18.21
15	24	1.97		13.9		9.73
7.2	78		7.32		0.419	40.39
6.9	60	2.16	33.8	17.5	1.58	31.51
6	61	2.97	53.4	13.9	1.45	33.56
6	68	1.94		9.15		37.42
4.05	70.5	2.09		4.56		44.22
4.8	71	2.03		8.62		42.08

Regression analysis has been carried out between the scaled distance and measured acceleration (Fig. 7). The predictor equation found has a correlation coefficient of 0.84 and is given below:

$$a = 20.84 \times \left(\frac{R}{\sqrt[3]{W}} \right)^{-0.72} \tag{5}$$

where,
a = Acceleration (g),
R = Distance of instrument from blast (m), and
W = Maximum charge per delay (kg).

Similarly the best-fit curves for derived PPV (Fig. 8 and Eq. 6 with correlation coefficient of 0.83) and for actual PPV monitored in the field (Fig 9 and Eq. 7 with correlation coefficient of 0.88) are established and are presented below:

$$DPPV = 289.95 \times \left(\frac{R}{\sqrt[3]{W}} \right)^{-0.90} \tag{6}$$

where,
DPPV = PPV derived from acceleration by integration (mm/s),
R = Distance of instrument from blast (m), and
W = Maximum charge per delay (kg).

$$V = 1006.4 \times \left(\frac{R}{\sqrt[3]{W}} \right)^{-1.59} \tag{7}$$

where,
V = Actual PPV measured in the field (mm/s),
R = Distance of instrument from blast (m), and
W = Maximum charge per delay (kg).

4.4 Statistical Significance

The ‘t’ test has been conducted for determining the statistical significance of the hypothesis for the Acceleration, DPPV and PPV predictors shown in Eqs. 5, 6 and 7. The t values calculated for Acceleration, DPPV and PPV are 4.79, 4.78 and 4.6 respectively. The t value obtained from the student t table at a confidence level of 95% are 2.228, 2.228 and 2.447 respectively which are less than the calculated values. Thus, the hypothesis is accepted and and the above predictors are statistically acceptable.

4.5 Overbreak Prediction

The acceleration and PPV threshold levels, for the actual overbreak measured in the tunnel, which is 0.4m, have been derived from each vibration predictor and are shown in Fig 10 (for Acceleration and Derived PPV) and Fig. 11 (for PPV). The damage threshold level for acceleration is found to be around 76.38g and for derived PPV (DPPV) arrived by integration of acceleration is 1467.36 mm/s. On the contrary, the damage threshold level using the extrapolated PPV, indicates a value in the order of 18005.15 mm/s, which is significantly higher than the earlier reported values of Bogdanoff (1995) and cannot be a reliable tool for overbreak prediction. However, the derived PPV value is observed to be well within the suggested range as described in the article 2.0. It is also seen that predicting overbreak using acceleration measurements would result in arriving at accurate maximum charge per delay values due to higher correlation coefficient obtained in comparison to far-field PPV measurements and derived PPV (DPPV) values apart from higher significance values.

The percentage overbreak at different maximum charge per delay has been shown in Fig 12. It may be used for determining the maximum charge per delay to control overbreak to a desired degree. In case of general production blasting, if the roof holes could be drilled 0.20 m below the desired excavation line, theoretically, the maximum charge per delay, to be used in roof holes for zero overbreak, works out to be around 1.8kg (≈ 2kg).

5.0 CONCLUSIONS

The existing criteria for rock damage assessment based on ground vibration have been reviewed. Data has been generated from laboratory and field testing of rock and trial blasts in the mine to study the blast- induced damage in burn cuts. The ground acceleration and PPV have been monitored for each blast. The measured accelerations have also been carefully integrated to arrive at the corresponding derived PPVs. The vibration predictors for the acceleration, derived PPV and measured PPV have been established. Vibration predictor derived from the near-field acceleration monitoring has the maximum correlation coefficient indicating the clear dependency of overbreak. The threshold levels for damage/overbreak have been established and are found to be around 76.38g for acceleration, 1467.36 mm/s for derived PPV, 18005.15mm/s for measured PPV. It is very much interesting that the predictor equation derived from PPV measurement is having a very high correlation coefficient, but still fails to predict overbreak. This could happen only because the measured PPV is of far-field in nature and hence, may be unsuitable for extrapolation to near-field. Thus, acceleration measurement in the near-field is a better choice for damage prediction. It is necessary to include more observations before suggesting a definite relationship.

ACKNOWLEDGEMENT

This paper forms a part of the research and development project on “Development of Predictive Models for Blast-Induced Rock Damage Assessment (BIRD) in Tunnels” sponsored by Ministry of Human Resource Development, Government of India. The authors are grateful to MHRD for sanctioning the project. The authors are also thankful to Director, Indian School of Mines, Dhanbad and Head, Department of Mining Engineering for providing the all the facilities and encouragement for writing this paper. The authors gratefully acknowledge the wholehearted help provided by the mine officials during the field investigations.

References

Ambraseys, N.R. and Hendron, A.J. (1968). Dynamic Behavior of Rock Masses, Rock Mechanics in Engineering Practices, (Eds) Stagg K.G. and Zienkiewiez O.C., John Willey and Sons, London, pp 203-207.

Barton, N., Lien, R. and Lunde, J. (1974). Engineering Classification of Rock for the Design of Tunnel Support, Norwegian Geotechnical Institute, Publication No. 106, Oslo, pp 1-115.

Bauer, A. & Calder, P.N. (1978). Open pit and Blasting Seminar, Mining Engg. Dept. Publication", Queen's University, Kingston, Ontario, p.3.

Bieniawski, Z.T. (1979). The Geomechanics Classification in Rock Engineering Applications. Reprinted from: Proc. 4th Congress of the International Society for Rock Mechanics/Comptes-rendus/Berichte-Montreux, Suisse, 2-8 Sept., 2208 pp., 3 Vols., A.A.Balkema, Rotterdam, Netherlands

Blair Dane and Minchinton Alan (1996). On the Damage Zone Surrounding a Single Blasthole, Proc. Rock Fragmentation by Blasting, FRAGBLAST-5, (Ed) Mohanty, Montreal, Quebec, Canada, 23-24 August, pp121-130.

Bogdanoff Ingvar (1995). Vibration measurements in damage zone in tunnel blasting", Proceedings of Rock fragmentation by blasting, FRAGBLAST-5, (Ed) Mohanty, Montreal, Quebec, Canada, 23-24 August, pp177-185.

Crandell, F.J. (1949). round Vibration due to Blasting and its Effect upon Structures, J. Boston Society of Civil Engineers, 36(2).

Edwards, A.T. and Northwood, T.D. (1960).Experimental Studies of Effects of Blasting on Structures, The Engineer, 211.

Holmberg, R. and Persson, P.A. (1979). Swedish Approach to Contour Blasting, Proc. Fourth Conf. Explosive and Blasting Techniques, pp.113-127.

Holmberg, R. and Persson, P.A. (1997). The Relation between Strain Energy, Rock Damage, Fragmentation and Throw in Rock Blasting, Proc. Rock Fragmentation by Blasting, FRAGBLAST-5, (Ed) Mohanty, Montreal, Quebec, Canada.

Gustafsson, R. (1973). Swedish Blasting Technique, SPI Gothenburg, Sweden, pp. 77 - 143.

Jimeno, C. L., Jimeno, E. L. and Carcedo, F.J.A. (1995). Drilling and Blasting of Rocks, A. A. Balkema, Rotterdam, Brookfield Publication, pp. 200-287.

Langefors, U. and Kihlstrom, B. (1973). The Modern Techniques of Rock Blasting, John Wiley and sons, pp. 34 - 73.

Murthy, V.M.S.R. and Dey, K. (2002). Development of Predictive Models for Blast-Induced Rock Damage Assessment (BIRD) in Tunnels, Interim Report, MHRD Project No: MHRD (27)/99-00/111/ME, pp. 12-43.

Murthy, V.M.S.R., Dey, Kaushik and Rajesh Raitani (2002). Prediction of Overbreak in Underground Tunnel Blasting – A Case Study, Proc. of 5th North American Sym. and the 17th Tunnelling Association of Canada Conference, NARMS-TAC 2002, Toronto, Ontario, Canada, 7-10 July, pp. 1499-1505.

Fig. 1- Accelerometer based seismograph with accessories, Instantel Make

Fig. 2- Telescopic overbreak measuring rod (fabricated)

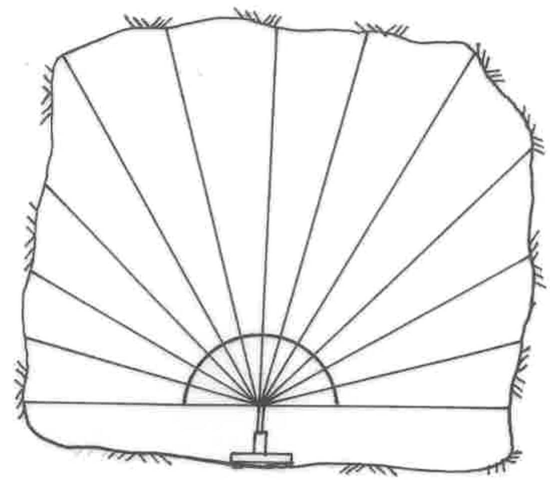

Fig.3– Tunnel overbreak measuring setup

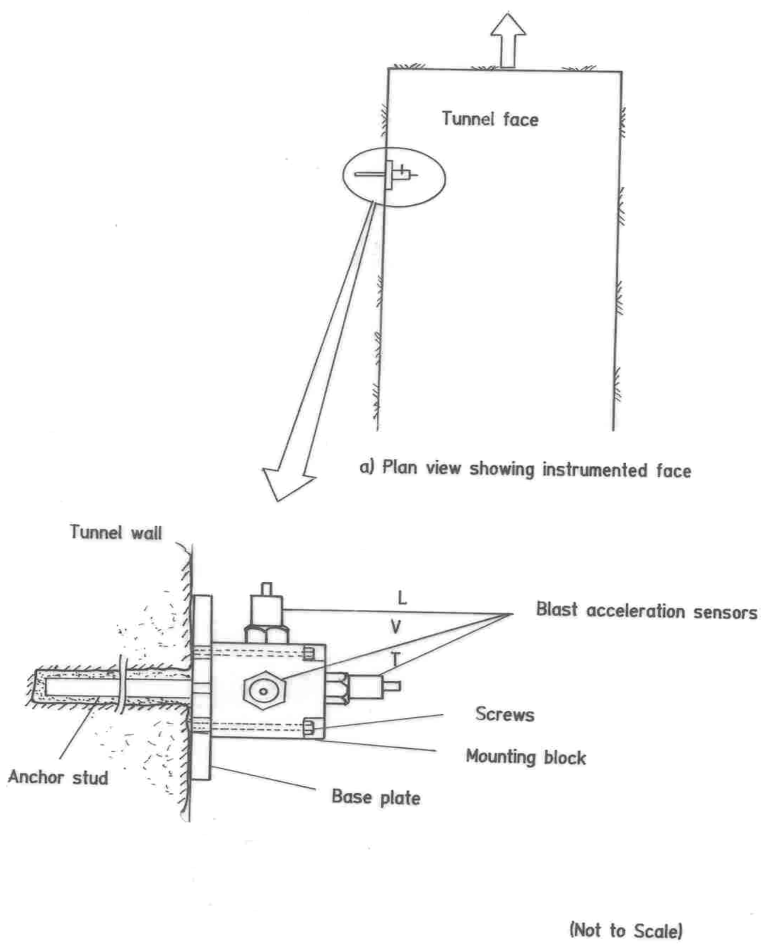

Fig. 4 – Fixing arrangement for acceleration measurement in tunnel wall

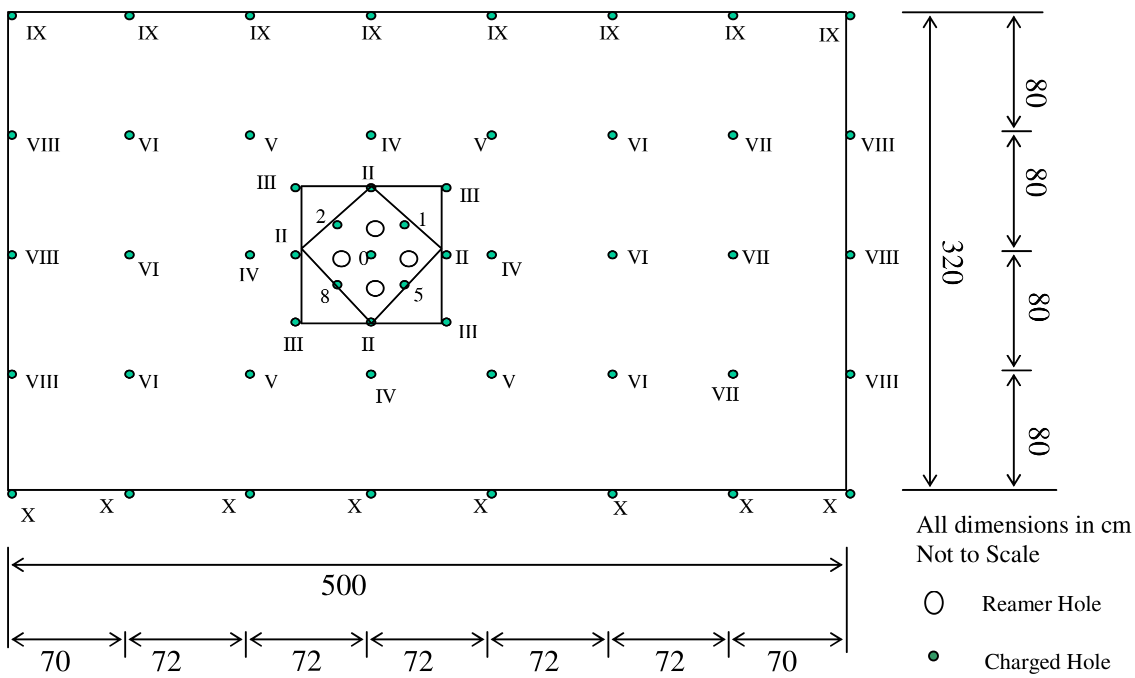

Fig. 5 - Blast pattern for drift with mechanised drilling

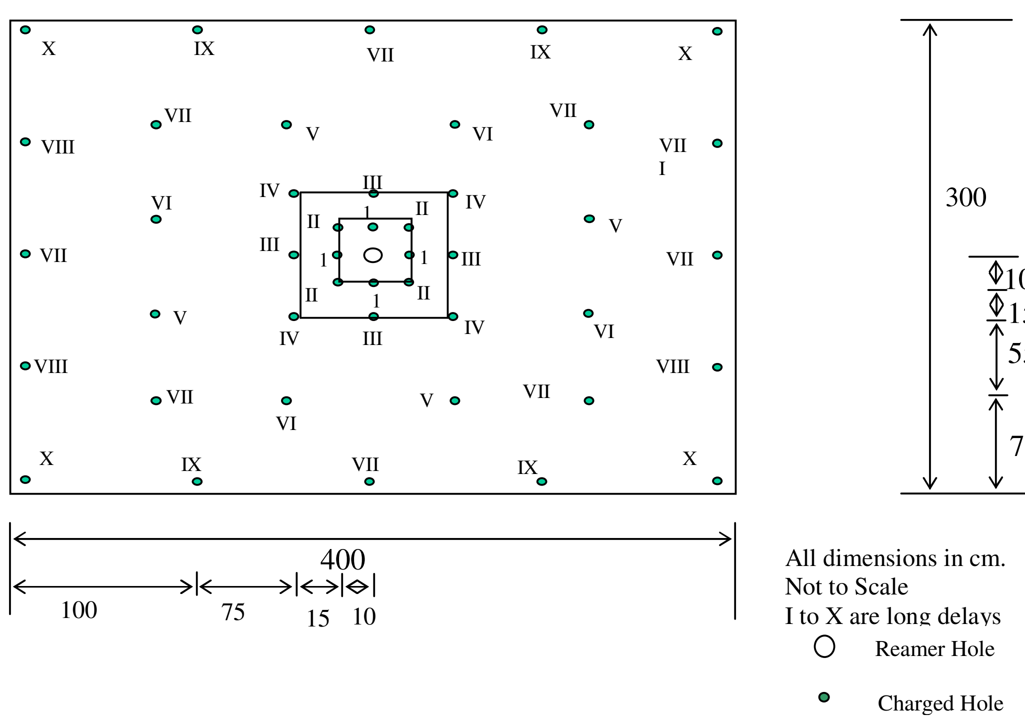

Fig. 6 - Blast pattern for drift with manual drilling

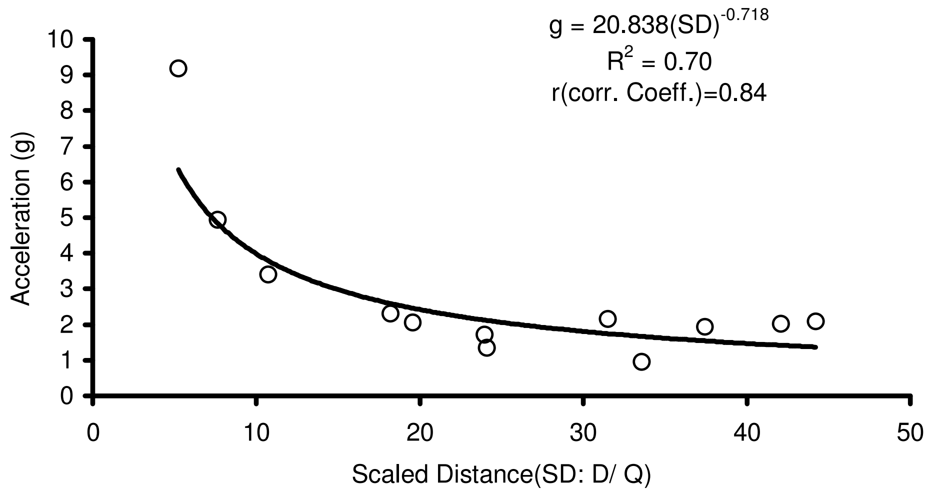

Fig. 7 – Acceleration predictor in horizontal drifts

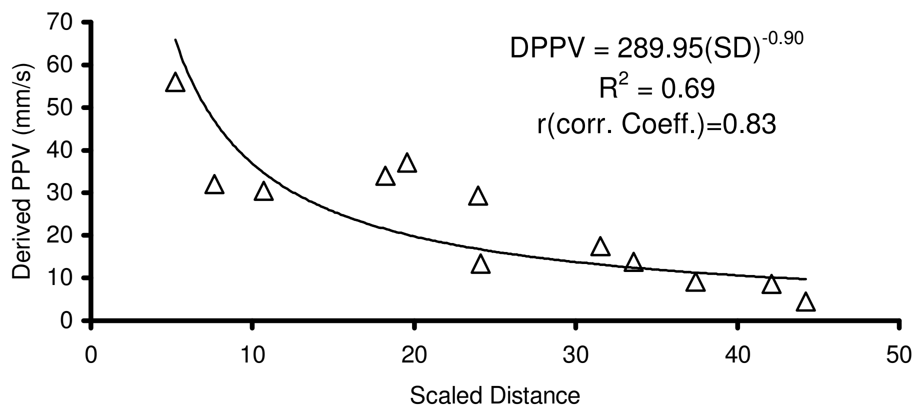

Fig. 8 – Derived PPV predictor from acceleration measurements

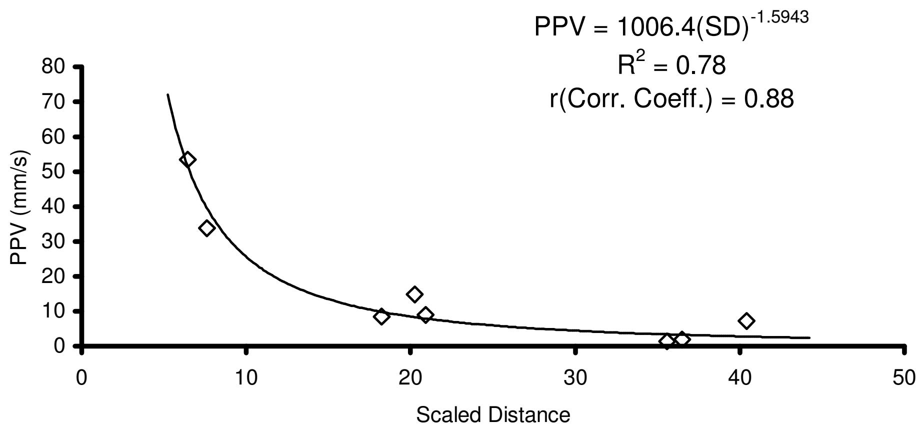

Fig. 9 – PPV predictor in the horizontal drifts

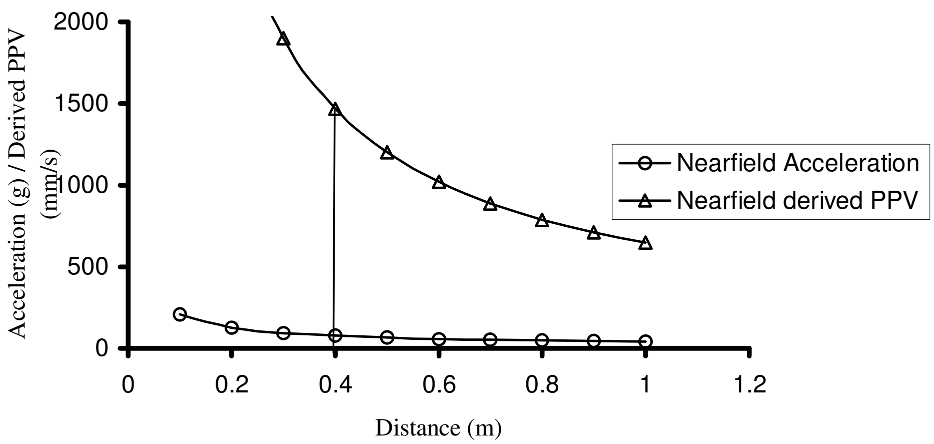

Fig. 10 –Threshold level for overbreak in terms of ground acceleration and derived PPV

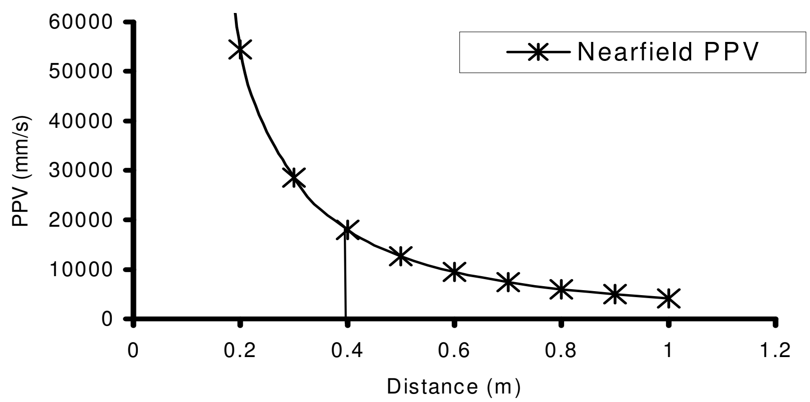

Fig. 11 – Threshold level for overbreak in terms of PPV

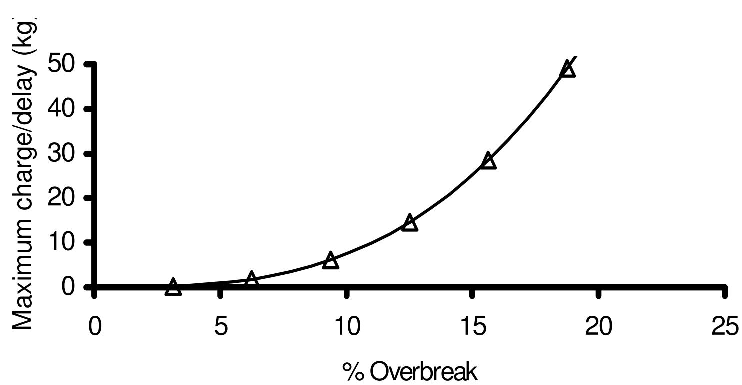

Fig.12 – Determination of maximum charge per delay
for controlling overbreak