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Extent of Rock Mass Damage Induced by Blasting in Tunnelling

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Abstract

Drilling and blasting is preferred method of rock excavation world-wide due to low initial investment, cheap explosive energy, easy acceptability amongst the blasting engineers, possibility to deal with different shapes and sizes of openings. Although, drill and blast method has witnessed significant technological advancements, it has inherent disadvantage of deteriorating surrounding rock mass due to development of network of fine cracks in it leading to safety and stability problem. The damage in the peripheral rock mass culminates in the form of overbreak and damaged zone. Overbreak increases project cost by more than 15%. The damaged zone extends beyond overbreak. Although significant efforts have been made to assess damage to the surrounding rock mass using different methods, easier solution based on easily available site parameters is still lacking. Authors have carried out field investigations at five different tunnel construction project sites located in Himalaya, India to formulate an empirical correlation for prediction of blast induced damage for wide range of Q values (0.04 – 17.8). The proposed correlation is based on specific charge, perimeter charge factor, maximum charge per delay, advancement and confinement factor and rock mass quality rating Q. All the parameters used in empirical correlations are readily available to the site engineers and does not require laboratory testing. Data sets of 113 experimental blasts are collected from the five tunnel sites. The proposed empirical correlation has been validated using ultrasonic tests on rock core samples obtained from one of the experimental location

1.   Introduction:

Drill and blast method (DBM) is commonly used technique for breaking rocks and minerals in mining, quarrying, tunnelling and other excavation works across the globe. DBM is cheaper than the other available mechanical methods such as Rock Breakers, Tunnel Boring Machines, Road Headers etc. especially with regards to tunnels excavation. DBM offers unmatched degree of flexibility over Tunnel Boring Machines (TBM). Low initial investment, cheap explosive energy, easy acceptability amongst the blasting engineers, possibility to deal with different shapes and sizes of openings and reasonably faster advance rate in a suitable geo-mining condition collectively make DBM preferred method of rock excavation (Innaurato et al., 1998).

Although DBM has witnessed significant technological advancements, it has inherent disadvantage of deteriorating surrounding rock mass due to development of network of fine cracks in it leading to safety and stability problems. Blasting for underground excavation and tunnelling is difficult operation compared to open excavation due to unavailability of the free face. Rrock mass damage is a common problem in tunneling (Gupta et al., 1988, Kadkade, 1991; Adhikari and Babu, 1994). The practicing engineers involved in the rock excavation, would try to achieve faster advancement in tunnel and underground excavation by employing drill jumbo which significantly increases drilling accuracy and reduces drilling time. Although, faster advancement rate may be achieved using greater amount of explosives quantity but it leads to greater extent of

blast induced rock mass damage (Murthy, 2003).

Blast induced rock mass damage has been studied by various researchers such as Langefors and Kihlstrom 1973; Bauer and Calder, 1978; Holmberg and Persson, 1979; Singh, 1993; Scoble et al., 1997; Bäckblom and Martin, 1999; Raina et al., 2000; Singh and Xavier, 2005; Warneke et al., 2007; Ramulu et al., 2009; Fu et al., 2014. Damage around an opening in underground has been described by using terminology such as Blast induced rock mass damage (BIRD), Blast induced damage (BID), Excavation damage zone (EDZ), Rock mass damage zone (RMD) etc. Rock mass damage zone surrounding an underground opening consists of overbreak zone (failed zone), damaged zone and a disturbed zone. Three zones of damage are shown in Fig. 1
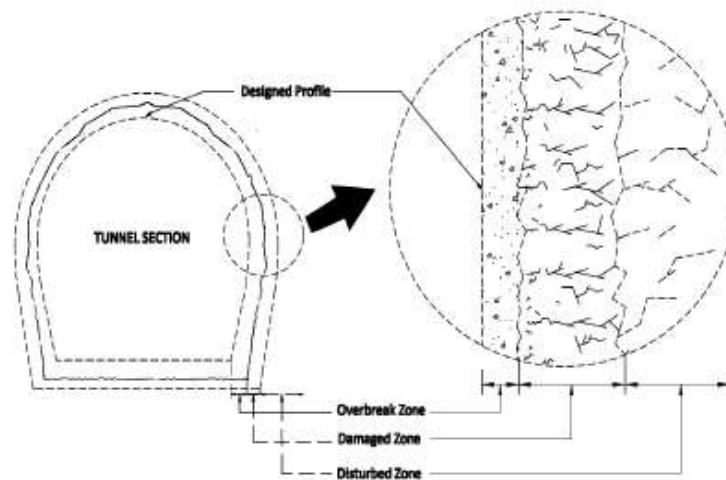

Figure 1 Blast induced Rock Mass Damage Zone around an Underground Opening

Overbreak zone represents the zone beyond minimum excavation line of the designed periphery from where rock blocks/slabs detach completely from the rock mass. It is a measure of difference in excavation between 'as designed profile' and 'as excavated profile'. Overbreak zone is undesirable and leads to cost overrun due to extra excavation and backfilling, shortcrete, concrete or other material as per designed support system. Overbreak varies from 5 % to 30% which incurs significant cost and increases cycle time of tunneling operation.

Damaged zone is a zone around tunnel beyond overbreak zone. The irreversible changes in the rock mass properties take place in this zone due to presence of fine networks of micro-cracks and fractures induced by the blasting and excavation process. This zone is characterized by deterioration in mechanical and physical properties and increase in transmissivity properties. Disturbed zone is a zone in the rock mass immediate beyond the damaged zone where changes in the rock mass properties are insignificant and reversible. This zone is dominated by changes in stresses and hydraulic heads. In the case of civic tunnels, the disturbed zone may not play significant role and hence the productivity and safety of tunnel construction is relatively unaffected by the disturbed zone.

Overbreak as well as damaged zone has significant impact on the project cost, construction period, safety and performance of the underground structures. In case of the

civic tunnels, damaged zone adversely affects the stability of underground opening and hence need to be accounted while designing support system of opening. Large extent of damage zone endangers safety of the front line workers as it may considerably reduce stand-up time of the rock mass. Functionality and post-construction performance of the structure will also be affected due to large extent of damage zone.

In light of above observation, efforts have been made in this study to develop empirical correlations for estimation extent of damaged zone based on data obtained from the field investigations.

2. Field Investigations:

Field experiments have been carried out to gain insight of these influencing parameters at five tunnel construction sites. The sites are integral parts of three major hydro power projects located in Hiamalya. These sites are Access Tunnel AA10R and AA7 from Pump Storage Plant (PSP) Project of THDC India Limited, at Tehri, Head Race Tunnel (HRT) of Singoli-Bhatwari Hydroelectric Power Project (SBHEP) at Rudraprayag, HRT and Bypass Tunnel (BPT) of Tapovan Visnhnugaad Hydroelectric Power Project (TVHEP) at Tapovan. The data was obtained from 113 blasts undertaken at five tunnel construction sites. Rock mass characterisation, blast vibration monitoring, overbreak assessment and estimation of damaged zone was carried out during each blast operation.

All the experimental blasts under observation were closely monitored. The data on drilling pattern including spacing and burden, with emphasis on holes in perimeter and penultimate row and hole depth were collected during drilling operation. Parameters related to explosive such explosive consumption in a hole as well as total round, initiation system and firing sequence, maximum charge per delay were recorded during charging of holes in a blast round. Record of pull in each round was obtained after surveying of tunnel profile and advancement. Parameters such as total charge used in blast round (T), maximum charge per delay (W), Pull (l), hole depth (d) were directly available from the records. Other parameters such as such as advancement factor, confinement factor and perimeter charge factor were calculated from the recorded observation for each round of blasts. In the present study, Specific Charge (q), Maximum Charge per Delay (W), Perimeter Charge Factor (q_p), Advancement Factor (A_f) and Confinement Factor (C_f) have been used to represent the blasting operation in underground excavation. In the present study, following parameters are used:

- Specific Charge (q) (kg/m3): Specific charge is defined as ratio of total quantity of explosive used and rock volume broken. It is expressed in kg/m3.
- Maximum Charge per Delay (W) (kg): It is maximum quantity of explosive fired in a delay series. This is obtained from record of delay distribution in a blast round and charging pattern in each hole.
- Perimeter Charge Factor (q_p) (kg/m3): Similar to specific charge, perimeter powder factor is the quantity of explosive used in perimeter holes and the volume of rock corresponding to burden of the contour holes.
- Advancement Factor (A_f): It is ratio of pull (l) and hole depth (d) in a blast round.

- Confinement Factor (C_f): It is ratio of hole depth (d) and cross-sectional area of tunnel (a).

Rock mass characterisation has been carried out using tunnel rock mass quality index Q. Q-system has been recommended specifically for tunnels and caverns with an arched roof. It is observed that Q-system is preferred method of rock mass classification for the construction of civic tunnel and cavern etc. A large number of field and design engineers/ geologists are using Q-system for various purposes such as support design, engineering analysis of rock mass etc. Prevailing stress environment influences damage to the surrounding rock mass. In Q-system, Stress Reduction factor (SRF) is one of the parameters which account for actives stresses during construction of an underground opening. Therefore, Q-system has been selected for rock mass characterisation in the present study.

Data sets of 113 experimental blasts are collected from the five tunnel sites. Rock mass of the experimental tunnel sites varies from poor class to good rock class (Q values range from 0.03 to 17.8). This range of rock classes represents commonly encountered geo-mining conditions for civil construction industry. Therefore, the suggested method could be applicable to a wide range of rock mass conditions encountered by tunnel construction industry.

3. Structures Estimation of damage distance (D_d):

During field investigation in each tunnel, vibration monitoring have been carried out in all the sites for determination of attenuation characteristics of blast induced ground vibration. Blast induced ground vibration were measured using three different engineering seismographs namely MinimatePlus (MMP), Minimate Blaster (MMB) and Minimate (MM), (manufactured by Instantel Canada). Monitoring of the blast induced ground vibration was carried out as per the guidelines given in IS: 14881 (2001), ISRM suggested method (ISRM, 1992) and Standard Operating Procedures recommended in Instantel Minimate User Manual (Instantel Manual, 2009). Ambraseys and Hendron (1968), developed a model for of blast induced ground vibration attenuation for spherical charge geometry. The suggested blast vibration prediction model is given in Eq. 1.

$$V_{ppv} = K \left(\frac{R}{\sqrt[3]{W}} \right)^{-\beta} \tag{1}$$

where

V_{ppv}	=	Peak particle velocity, mm/s
K, β	=	Site constant (function of characteristics of propagating media)
R	=	Distance of measurement, m
W	=	Maximum charge per delay, kg.

The constant K and β have been determined using least square regression analysis for all the sites. In the present study, the attenuation characteristic is derived at 95% confidence interval and same is used for prediction of the blast vibration in all the five sites. The attenuation characteristics of the vibration obtained at five sites has been presented in Table 1.

Table 1
Geotechnical Properties of Rock in Experimental Tunnels
(Source: Detailed Project Report of respective Projects)

Tunnel Site	Predominant Rock Type	Vibration Attenuation Equations	σ_t MPa	V_p m/s	E MPa	V_{cr} mm/s
HRT SBHEP	Quartz Bio-tite Schist	$V_{ppv} = 1825.1 \left(\dfrac{R}{W^{0.33}} \right)^{-1.20}$	6.71	3267	12600	1739.8
HRT TVHEP	Augen Gneiss	$V_{ppv} = 1390.5 \left(\dfrac{R}{W^{0.33}} \right)^{-1.598}$	8.7	5400	27900	1683.8
BPT TVHEP	Quartzite	$V_{ppv} = 2079.1 \left(\dfrac{R}{W^{0.33}} \right)^{-1.334}$	12.4	6200	55500	1754.5
AA7 PSP	Phyllitic Quartzite Thinly Bedded (PQT)	$V_{ppv} = 441.9 \left(\dfrac{R}{W^{0.33}} \right)^{-1.16}$	4.3	5400	10500	221.65
AA10R PSP	Phyllitic Quartzite Massive (PQM)	$V_{ppv} = 576.2 \left(\dfrac{R}{W^{0.33}} \right)^{-1.03}$	7.2	6000	12700	340.15

Notations: σ_t : Tensile strength, V_p: P-wave velocity, E: Young Modulus; V_{cr}: Critical peak particle velocity; SBHEP: Singoli-Bhatwari Hydroelectric Project, TVHEP: Tapovan Vishnugaad Hydroelectric Project, Tapovan, PSP: Pump Storage Plant Project, Tehri.

Rock mass damage is a result of the induced dynamic stress during detonation. For an elastic medium, induced dynamic stress can be calculated as a function of peak particle velocity (Vppv) and longitudinal wave velocity (V_p). Therefore, blast induced damage arising in the rock mass is widely correlated with peak particle velocity of blast induced vibration. In this study, extent of damage to the surrounding rock mass is calculated using Critical Peak Particle Velocity (Vcr) (Singh, 1993; Forsyth, 1993; Homberg and Persson, 1994; Known et al., 2009)). Critical peak particle velocity for each of experimental predominant rock mass is obtained using Eq. 2.

$$v_{cr} = \frac{V_p \, \sigma_t}{E} \quad (2)$$

where

v_{cr}	=	Critical peak particle velocity, mm/s
V_p	=	P-wave velocity of rock, m/s
σ_t	=	Tensile strength of rock, MPa, and
E	=	Young's modulus of rock, MPa.

Geotechnical properties of the predominant rock mass encountered at experimental tunnel site are presented in Table 1. The critical peak particle velocity (V_{cr}) value was obtained using Eq. 2 for each experimental sites. The damage distance from a blast round can be back calculated using Eq. 1. Following this method, damage distance for all the observed blasts have been obtained.

4. Analysis of the Data:

The data obtained were analysed to gain insight in to the contribution of the individual parameters in the damage induced by the blasting operation. Figure 1 shows the variation of average damage distance with rock mass quality. In general, the damage distance decreases with the increase in Q-values. The average damage distance for rock mass having Q-value less than 1 is greater than 5.0 m. It sharply reduces to approximately 3.0 m for rock mass with Q-value greater than 4. The impact of rock mass quality is significant in lower classes of the rock mass having Q-value less than 4. In higher classes of rock mass, impact of the rock mass quality remains fairly uniform and possibly other parameter of blast design play pivotal role in defining damaged zone around an opening.
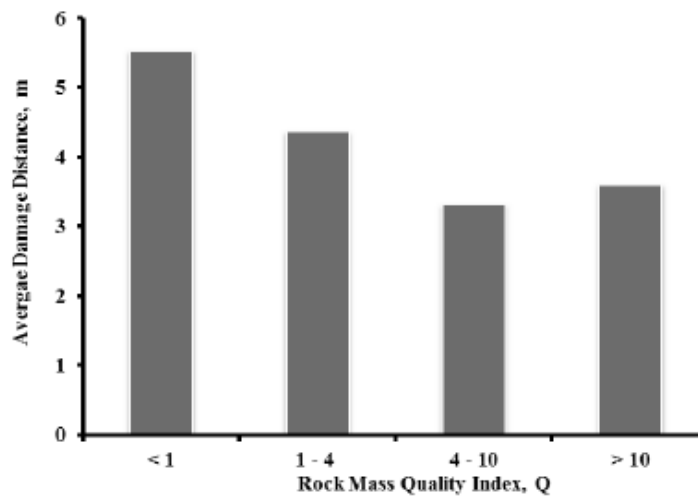

Figure 2 Variation of Damage Distance (D_d) with Rock Mass Quality Index, Q

In any underground blasting operation, progressive enlargement of the free face shall be achieved by designing the firing sequence of holes using different delay series. Proper distribution of the delay series ensures free face to each hole. The holes fire in the direction of free face thus utilising the explosive energy in breaking and displacing the rock. The maximum charge per delay, W depends on the number of delay series used in a blast round. Improper delay distribution gives excessive burden and spacing to the holes which leads to generation of the blast induced ground vibration resulting into greater extent of damaged zone. Thus maximum charge per delay, W influences significantly the damage distance amongst other parameters.

The variation of observed average damage distance with maximum charge per delay is shown in Fig 3. In experimental blast rounds, range of W varied between 15 kg to 45 kg. Therefore, range of 15 to 45 kg of maximum charge per delay, W has been considered in the analysis. It may be noted from Fig 3 that the damage distance increases with increase in maximum charge per delay.
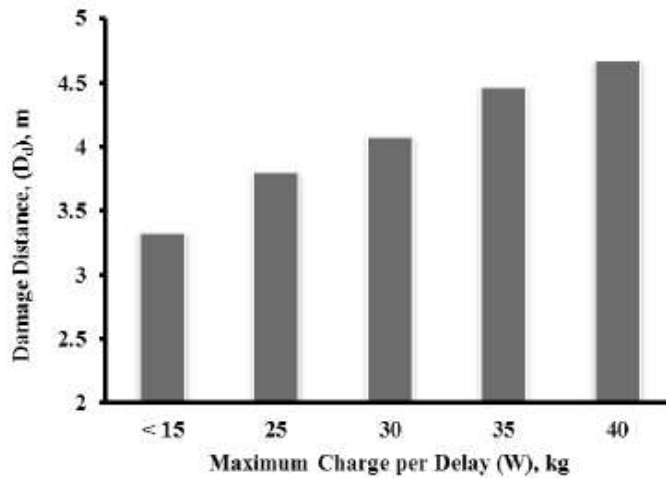
Figure 3 Variation damage distance (D_d) with Maximum Charge per Delay (W)

Figure 4 is a plot of damage distance with normalized advancement factor. Advancement factor is normalized with rock mass quality index to make data comparable for different rock mass conditions. In general, damage distance increases with increase in normalized advancement factor. This can be attributed to the fact that in experimental blasts, higher advancement has been achieved in most of the case due to increase in either hole depth or total charge used in blast round. Higher value of total charge in a blast round will increase advancement but will also cause more damage to the rock mass.
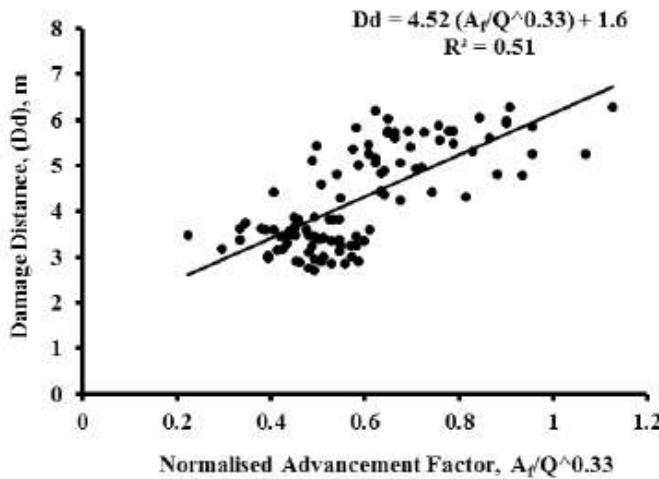

Figure 4 Variation of Damage Distance with Normalized Advancement Factor

Contrary to this observation, increase in advancement factor by optimizing blast design parameter for a given size of tunnel should ideally decrease the damage distance. The reason behind decrease in damage distance may be due to fact that the increase in advancement factor also leads to optimum utilization of the explosive energy which otherwise would have converted in blast induced ground vibration. In blasting operation, explosive energy appears in three forms, maximum charge per delay (W), specific charge (q) and perimeter charge factor (qp). Whereas maximum charge per delay is in-

fluenced by the initiation and firing sequence of the blast round, perimeter charge factor and specific charge are dependent on advancement of a blast round. For a fixed amount of explosive energy, greater advancement rate reduces perimeter charge factor as well as specific charge. All the three explosive energy parameter W, qp and q are grouped as factor Z as given in Eq. 3

$$Z = q_p^{0.15} \sqrt{W + q} \qquad (3)$$

where,

q_p	=	perimeter charge factor, kg/m³,
W	=	maximum charge per delay, kg,
q	=	specific charge, kg/m³,

Figure(s) 5 and 6 are plots of damage distance with factor Z for experimental tunnel with Q values less than 1.67 and greater than 1.67 respectively.
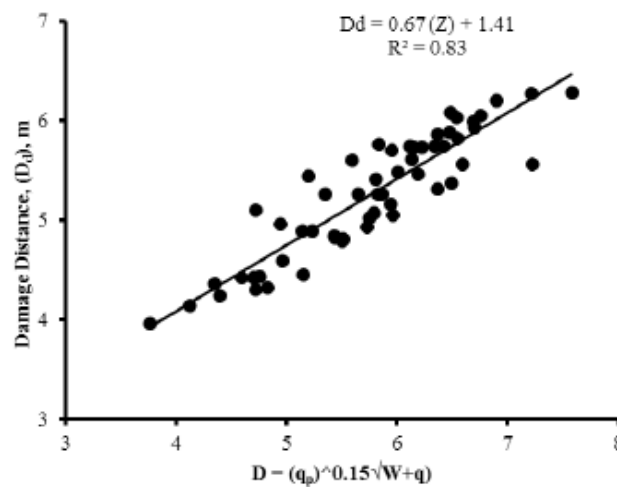

Figure 5 Damage distance versus Explosive Energy Parameters for $Q < 1.67$

In case of tunnel AA10R and AA7, rock mass was of lower quality whereas in other three experimental tunnel Q values were higher than 1.67. Data for these two cases are plotted separately due to difference in rock mass classes and also in blasting practices. In experimental tunnel AA7 and AA10R rock mass are of very poor category, cross-sectional area are higher and hole depth is less than 2.5 m. In other tunnels, rock masses are of better quality, tunnel cross-sectional area is less than 40 m2 and hole depth were greater than 3.0 m. As shown in Fig(s) 5 and 6, factor Z is directly proportional to the damage distance. In both the cases correlation coefficient are 0.83. Maximum charge per delay and specific charge are measures of explosive energy in the blast round whereas the perimeter charge factor is parameter introduced to measure the damage created by the explosive energy in the contour hole.
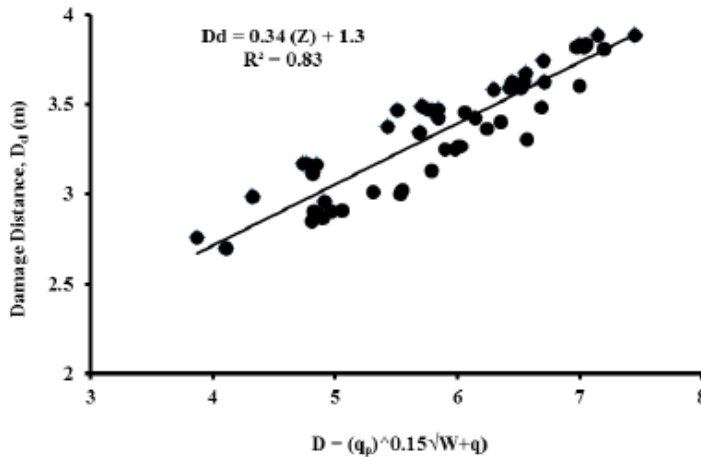
Figure 6 Damage distance versus Explosive Energy Parameters for $Q > 1.67$

In a blasting operation in underground excavation, contour holes are fired in the last delay series. All the holes are assigned same delay and spacing in these holes are lesser than the burden of the contour holes. Such firing arrangement creates a fracture line along the final excavation line. Such arrangement of delay for periphery holes, although these holes are lightly charged, contribute as maximum charge per delay due to large number of holes fired in same delay time on several instances. These conditions compound the effect of maximum charge per delay with perimeter powder factor. Although the periphery holes provide a line of fracture along final line of excavation, due to it's initiation in the last delay series, the effect of W is not restricted by the lightly charged contour holes. Therefore, the damage distance is enhanced by the perimeter charge factor.

5. Empirical Correlation for Prediction of Damage Distance (D_d):

As discussed in preceding section, damage distance is influenced by the blast design parameters and rock mass quality. A regression analysis of various parameters considering the general trend of the field data and also the damage mechanics was performed to develop empirical correlation for prediction of damage distance, Dd. Figure 7 shows plot of a factor D and observed damage distance (Dd). It is a plot for all the observed Dd values obtained during field investigation at five tunnel construction sites.
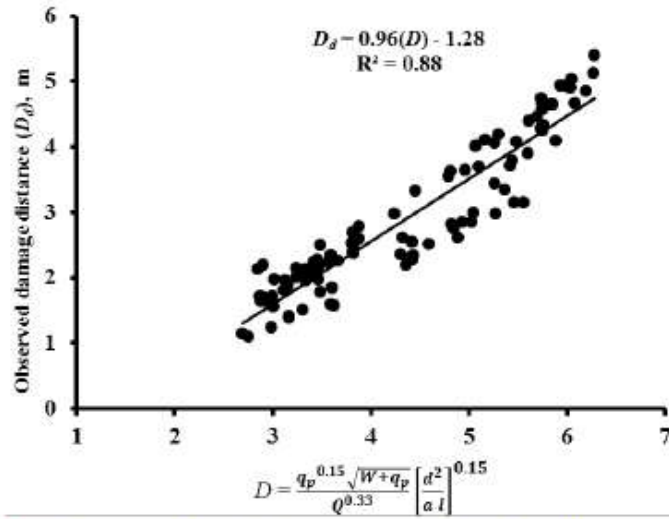
Figure 7 Plot of factor D and observed Damage Distance, D_d

$$D = \frac{q_p^{0.15}\sqrt{W+q}}{Q^{0.33}}\left(\frac{d^2}{a\,l}\right)^{0.15} \qquad (4)$$

$$D_d = 0.96\left[\frac{q_p^{0.15}\sqrt{W+q}}{Q^{0.33}}\left(\frac{d^2}{a\,l}\right)^{0.15}\right] - 1.28 \qquad R^2 = 0.88 \qquad (5)$$

where,

D_d	=	damage distance, m,
q_p	=	perimeter charge factor, kg/m³,
W	=	maximum charge per delay, kg,
q	=	specific charge, kg/m³,
Q	=	rock mass quality index (Barton' Q-system)
d	=	hole depth, m,
1	=	pull, m, and
a	=	tunnel cross-sectional area, m².

Eq. 5 can be further simplified as Eq. 6 and 7

$$D_d = 0.96\left[\frac{q_p^{0.15}\sqrt{W+q}}{Q^{0.33}}\left(\frac{\frac{d}{a}}{\frac{l}{d}}\right)^{0.15}\right] - 1.28 \quad (6)$$

$$D_d = 0.96\left[\frac{q_p^{0.15}\sqrt{W+q}}{Q^{0.33}}\left(\frac{C_f}{A_f}\right)^{0.15}\right] - 1.28 \quad (7)$$

Confinement factor, $\qquad\qquad C_f = \frac{d}{a}$

Advancement factor, $\qquad\qquad A_f = \frac{l}{d}$

where,
d = depth of drill hole, m,
a = tunnel cross-sectional area, m², and
1 = pull of blast round, m.

Equation 7 can be used for prediction of the damage distance induced by blasting in underground excavation.

In Eq 7, explosive energy parameters, blast design parameters and rock mass quality are included. Equation 7 gives impression that energy parameters are used repetitively in the recommend correlation. It may be noted that all these three parameters are mutually exclusive. In a same blast design, values of W, q_p and q can be altered without changing other parameters. In blast round, having same drill hole depth and total charge, arrangement of firing sequence will change the values of W. Pull of the blast from such changed configuration will alter values of q_p as well as q. Inclusion of these three parameters will therefore be able to assess their effect on blast induced damage distance. The correlation for assessment of damaged zone has been validated from ultrasonic test data of rock core samples obtained from ten locations from HRT of SBHEP project. Percentage reduction of P-wave velocity with depth computed from the ultrasonic test data. As per Liu et al., (2009) and Fu et al. (2014), the threshold of damage is defined as a 10% reduction in the P-wave velocity as compared to the P-wave velocity of the undisturbed rock mass. The damage distance obtained from the ultrasonic test data were in close agreement with the predicted damage distance obtained using Eq. 7.

6.   Conclusions:

A comprehensive field investigation have been carried out at five tunnel construction sites to evolve empirical correlation for estimation of damage distance using readily available site parameter. Observations of 113 blasting experiment have been taken in different rock mass from extremely poor to good rock mass class. An empirical correlation has been suggested using Specific Charge (q), Maximum Charge per Delay (W), Perimeter Charge Factor (q_p), Advancement Factor (A_f) and Confinement Factor (C_f) and rock mass quality index Q.

It is suggested that the proposed correlation for estimation of damage distance may be used to fix a range of allowable blasting parameters based on the anticipated rock mass characteristics. Using proposed correlation, the extent of damaged zone may be computed for known values of Q and blast design parameters. The rock mass support shall be designed by considering damaged zone also

Acknowledgements:

References:

1. Adhikari, G. R. and Babu, R. (1994). Controlled blasting in tunnel - some issues, Tras. Institution of Engineers (I) - MN, Vol. 75, pp. 56-60.

2. Ambraseys, N. R. and Hendron A. J., 1968. Dynamic behaviour of rock masses, rock mechanics in engineering practices, (Eds) Stagg K.G. and Zienkiewiez A.C., John Willey and Sons, London, pp. 203-207.

3. Backblom, G., and Martin C. D. (1999). Recent experiments in hard rock to study the excavation response: Implication of the performance of a nuclear waste geological repository, Tunnelling and Underground Space Technology, Vol. 14, pp., 377-394.

4. Bauer, A. & Calder P.N. (1978). Open Pit and Blasting Seminar Mining Engg. Dept. publication, Queen's University, Kingston, Ontario.

5. Forsyth, W. W. (1993). A discussion on the blast induced overbreak around underground excavations, Proceedings of Fourth International seminar on Rock Fragmentation by Blasting, FRAGBLAST - 4, (Ed) Rossmanith, Vienna, Austria 5-8 July, pp. 161-166.

6. Fu, H., Wong, L. N. Y., Zhao, Y., Shen, Z., Zhang, C. and Li, Y. (2014). Comparison of excavation damage zones resulting from blasting with NONEL detonators and blasting with electronic detonators, Rock Mech Rock Eng, Vol. 47, pp. 809–816.

7. Gupta, R. N., Singh, R. B., Adhikari, G. R. and Singh, B. (1988). Controlled blasting for underground excavations, International Symposium on Underground Engineering, (Ed.) B. Singh, 14 - 17 April, New Delhi, India, pp. 449 - 458.

8. Holmberg R. and Persson P. A. 1979. Swedish approach to contour blasting, Proceedings of Fourth Conference on explosive and blasting techniques, pp. 113-127.

9. Holmberg, R. and Persson, P. A. (1978). The Swedish approach to contour blasting, Proceedings of 4th Annual Conference on Explosives and Blasting Research, Explosives Reference Database on CD-ROM, International Society of Explosives Engineers, Ohio, USA, 1997, pp. 113-127.

10. Innaurato, N., Mancini, R., and Cardu, M. (1998). On the influence of rock mass quality on the quality of blasting work in tunnel driving, Tunnelling and Underground Space Technology Vol. 13, No. 1 pp. 81 – 89.

11. Instantel, 2003. Canada. MiniMate Plus Operator Manual. Software version 7.1.

12. IS: 13030 (1996). Method for laboratory determination of water content, porosity, density and related properties of rock mass, Bureau of Indian Standards, New Delhi.

13. IS: 14881 (2001). Method for blast vibration monitoring – Guidelines, Bureau of Indian Standards, New Delhi.

14. IS: 15404 (2003). Recommended practice for measuring ultrasonic velocity in materials, Bureau of Indian Standards, New Delhi.

15. Kadkade D., G. (1991). An overview of tunnelling project in India, Proc workshop on Tunnel, Mine Roadways and Caverns, Ooty, Part-II, pp. 1- 24

16. Kwon S., Lee C. S., Cho S. J., Jeon, S. W. and Cho W. J. (2009). An investigation of the excavation damaged zone at the KAERI underground research tunnel, Tunnelling and Underground Space Technology, Vol. 24, pp. 1-13.

17. Langefors, U. and Kihlstrom, B. (1973). The Modern techniques of rock blasting, John Wiley and Sons Inc. New York, 473 p.

18. Liu, M.G., Zhang G.H., and Liu, S.B. (2009). Research on accumulative damage effect of interlaid rock in DAMAOSHAN tunnel group with small clear distance. Chin J Rock Mech Eng Vol. 28, No. 7, pp.1363–1370

19. Murthy, V. M. S. R. and Dey, K. (2003). Predicting Overbreak from Blast Vibration Monitoring ina Lake Tap Tunnel - A Success Story', Fragblast, 7: 3, pp.149 - 166.

20. Pal Roy, P. (2005). Rock blasting effects and operations, Taylor and Francis, ISSN.0415372305.

21. Raina, A.K., Chakraborty, A.K., Ramulu, M. and Jethwa, J.L. (2000). Rock mass damage from underground blasting, a literature review, and lab- and full scale tests to estimate crack depth by ultrasonic method. FRAGBLAST-International Journal for Blasting and Fragmentation, Vol. 4, pp. 103-125.

22. Ramulu, M., Chakraborty, A.K., and Sitharam, T.G. (2009). Damage assessment of basaltic rock mass due to repeated blasting, Tunnelling and Underground Space Technology, Vol. 24, pp. 208–221

23. Scoble M., Lizotte Y., Paventi M. and Mohanty B.B. (1997). Measurement of Blast, Mining Engineering Journal, June, 1997, pp 103-108.

24. Singh, S. P. (1993). Prediction and determination of explosive induced damage, 4th IntI. Symp. on Rock Fragmentation by Blasting, FRAGBLAST, pp.183-192.

25. Singh, S. P., and Xavier, P. (2005). Cause, impact and control of overbreak in underground excavation, Tunnelling Underground Space Technology, Vol. 20, pp. 63 - 71.

26. Singh, S.P. (1993). Damage causing potential of different explosives, Proc, 9th annual symposium on Explosives and Blasting Research, San Diego, California, pp. 325-337.

27. Warneke, J., Dwyer, J.G. and Orr, T. (2007). Use of a 3-D scanning laser to quantify drift geometry and overbreak due to blast damage in underground manned entries. In: E. Eberhardt, D. Stead and T. Morrison (Editors), Rock Mechanics: Meeting Society's Challenges and Demands. Taylor & Francis Group, London, Vancouver, Canada, pp. 93-100.