Damage Control Blast Designs for Excavation of Desilting Chambers at Nathpa Jhakri Hydroelectric Project

G. R. Adhikari
R. Balachander
A.I. Theresraj
and
R. N. Gupta

National Institute of Rock Mechanics
Champion Reefs, Kolar Gold Fields – 563 117, India
Tel: (0815) 375008; Fax: (0815) 375002
Email: gradhikari@rediffmail.com

ABSTRACT

This paper deals with a case study of excavation of the hopper, settling trench and flushing conduit of the desilting chambers of Nathpa Jhakri hydroelectric power project, India. The existing blasting patterns were reviewed and the implementation of these designs was carefully observed. The existing blast design for hopper excavation was modified incorporating smooth wall blasting technique and no changes were made in the blast design for settling trench and flushing conduits. A blast vibration study was also conducted to derive a site-specific predictor equation for peak particle velocity. Based on the literature survey, safe limits of blast vibration for rock, steel fiber reinforced shotcrete, concrete lining and fully grouted bolts were recommended for the conditions of the desilting chambers.

Keywords: Controlled blasting, desilting chambers, blast vibration, blasting damage

1. INTRODUCTION

The Sutluj Jal Vidyut Nigam Ltd. (SJVN), formerly Nathpa Jhakri Power Corporation (NJPC), was constructing a 1500 MW (6 x 250 MW) underground hydroelectric power project on the left bank of the Sutluj river in Himachal Pradesh. As a part of this project, four large underground egg-shaped desilting chambers, 525m long, 16m wide at the center, 27m high and 29m apart and parallel to each other were to be excavated. The major rock types in the desilting chambers are augen gneiss and gneiss with bands of pegmatite, amphibolites and biotite schist. Apart from the foliation joints, there are two to three more sets of joints some of which are sheared and filled with crushed rock and clay gauge. The excavation of the desilting chambers was being carried out by drilling and blasting. Pattern rock bolting with steel fiber reinforced shotcrete (SFRS) was the principal support system adopted for the desilting chambers. For the hopper portion and below, the support system consisted of rock bolts, 50mm thick plain shotcrete and 300m thick RCC lining.

Rock blasting by its very nature is a destructive process and the excavation demands greater control over blasting so as to minimise the blasting damage to the surrounding rock mass. At the initial stages of their excavation, the National Institute of Rock Mechanics (NIRM) had carried out field trials to minimise overbreak and to achieve maximum pull and progress (Adhikari et al., 1999). NIRM was approached again by NJPC to review the blasting patterns being followed for the excavation of hopper, trench and conduit portions of the desilting chambers and to monitor and analyse blast vibrations.

2. EXISTING BLAST DESIGNS AND THEIR MODIFICATION

The hopper, trench and conduit of the desilting chambers with their planned dimensions is shown in Figure 1. The width of the hopper was 11.4 m on the top and 4.4 m at the bottom with a height of 5.5m. The settling trench was 4.4 m wide throughout its length but the depth varied from 0.8m at 0 RD to 3.0m at 525 RD. The height of the flushing conduit was 2.1m throughout its length but the width varied from 1.3m to 4.2m (0 RD to 525 RD).

Tamrock drilling machines with a hole diameter of 45 mm were used to drill the holes to a depth of 4 m. Powergel 801, an emulsion explosive from ICI was used. Explosive cartridges of 40mm diameter weighing 0.390kg were used for production holes while cartridges of 20mm diameter weighing 0.125kg, pre-assembled in PVC pipe (Adhikari et al., 1999), were used for perimeter holes. The non-electric initiation system (EXEL) was used to initiate the round.

2.1 Blast Design for the Hopper Portion

The hopper was excavated earlier to a full height of 5.5m. When cracks were observed on the right wall of the chambers, it was excavated in two lifts. The first lift of 1.9 m had already been completed and the second lift of 3.6 m was in progress. The design for the second lift consisted of 17 production holes and 22 presplit holes (Fig. 2). The production holes were drilled with a decreasing burden of 1.60 m to 0.75 m and with a spacing of 1.20 to 1.10 m. The presplit holes were drilled at a spacing of 0.35m. All the holes were drilled to a depth of 4.2m. All production holes were charged with 4.55 kg per hole and the alternate perimeter holes were charged with 1.0kg per hole using spacers.

In practice, the required number of presplit holes was not drilled along the final excavation line. Holes were stemmed with the pieces of explosive carton boxes soaked in water to a length of about 0.3m. Improper stemming resulted in low utilization of explosives energy for breaking the rocks and hence the explosive consumption was high. Irrespective of the diminishing burden, the charge per hole remained the same.

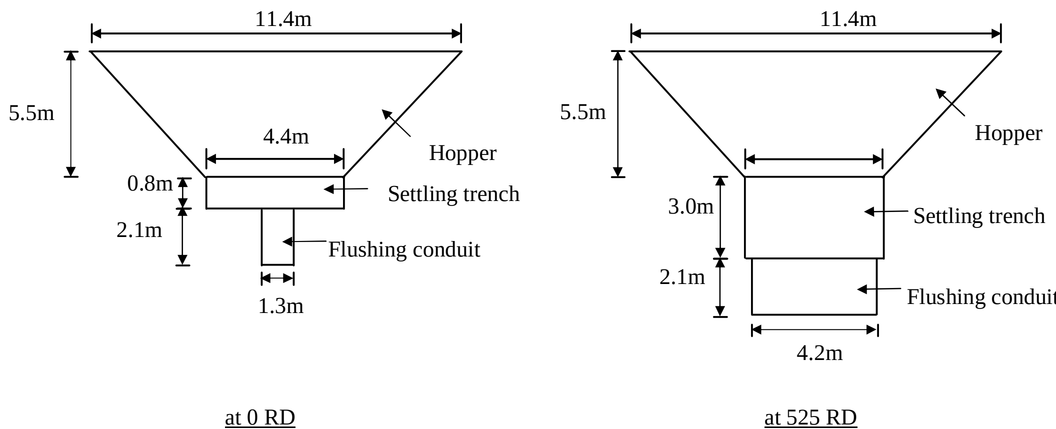


Fig. 1 - Sections showing hopper, settling trench and flushing conduit for desilting chambers

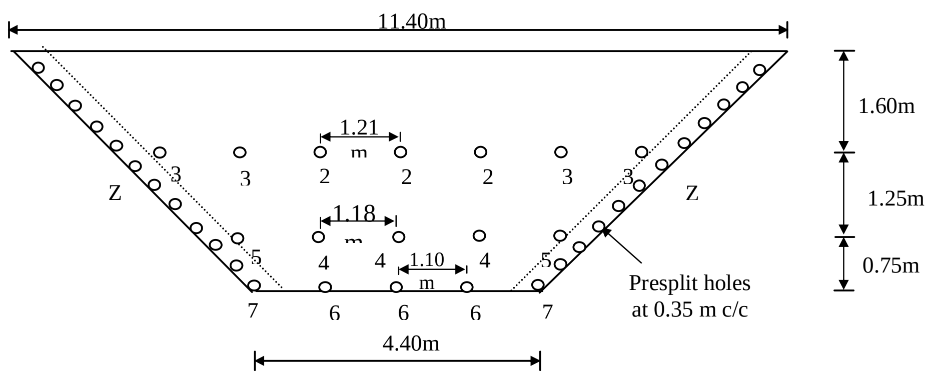


Fig. 2 - Existing blast design for the hopper portion

Considering the above drawbacks, a modified blast design was prepared and executed (Fig. 3). The excavation of the hopper was nothing but horizontal bench blasting with wall control. Smooth wall blasting rather than presplitting was incorporated in the design. This method is widely accepted for controlling overbreak in underground excavations. In this method, holes were drilled along the excavation limits, lightly loaded with well-distributed charges, and fired with last delay in the round. In order to direct the crack along the holes, one dummy hole was left between the charged holes. It was suggested to stem the holes with clay sticks for better utilization of the explosive energy. With the modified design, the overbreak was reduced considerably without affecting the progress of the work.

2.2 Blast Design for the Settling Trench and Flushing Conduit

The settling trench and the flushing conduit were excavated together because, the width of the settling trench was not sufficient to move the drill machine to further drill vertical holes for flushing conduit. A typical blast design followed at 137 RD is given in the Fig. 4. Vertical holes were drilled on a burden of 1.1m and spacing of 0.8 to 1.6 m to a depth of 2.0 m in the trench section and 4.0 m in the conduit section. The 2.0 m deep holes were charged with 5 to 6 cartridges per hole and 4.0m deep holes were charged with 11 to 12 cartridges per hole with a maximum charge per delay of about 17.7kg.

The post blast observations indicated that profile was not conforming to the desired shape of the flushing conduit. The resulting profile was almost of 4.4m wide and 4m deep. The charge per hole for the conduit section may be reduced as it was on the higher side for 4.0m holes (about 11 to 12 cartridges), with a spacing of 0.8m. The charge per hole and the spacing are to be adjusted depending on the width.

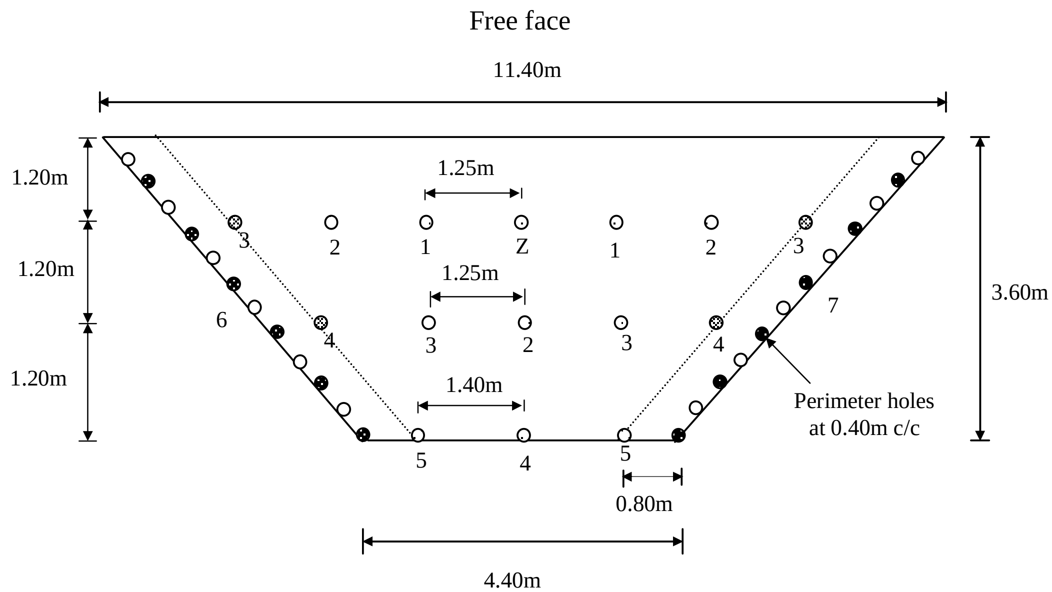
3. GROUND VIBRATION STUDIES

3.1 Monitoring and Analysis of Ground Vibration

Blast vibrations were recorded at different locations with two units of Minimate DS 077 seismographs from Instantel, Canada and one unit of μmx micro monitor from Blastronics, Australia. (Table 1). In total, 34 sets of data were used for regression analysis. Figure 5 shows a plot of PPV against scaled distance on a log-log graph. The scaled distance (SD) is the distance divided by the square root of the maximum charge per delay. The following predictor equation was derived for the underground desilting complex with a correlation co-efficient of -0.91.

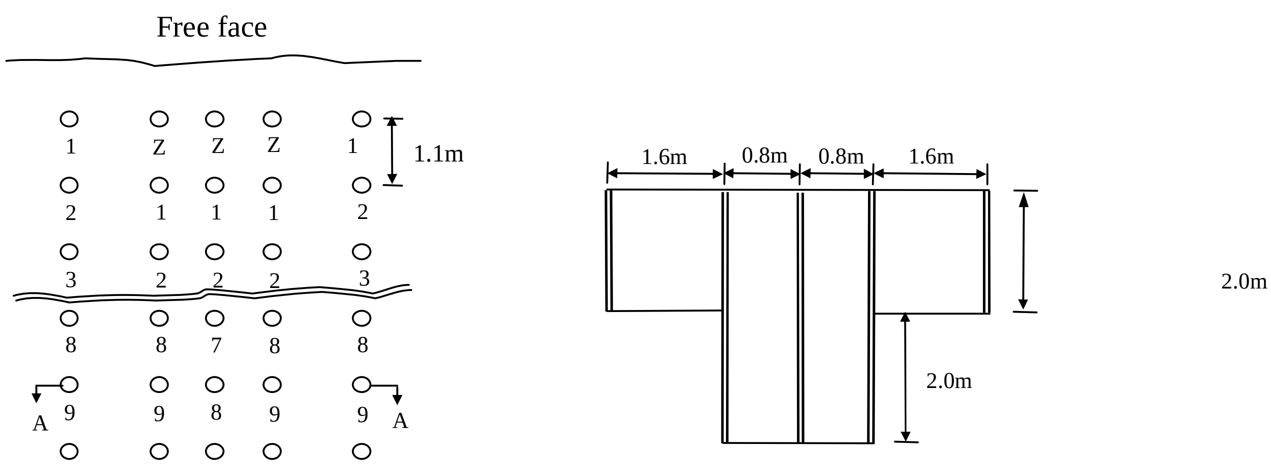
$$V = 334.40(D/\sqrt{Q})^{-1.32} \tag{1}$$

where V is the peak particle velocity (mm/s), D is the radial distance from blast to monitoring station (m), and Q is the maximum charge per delay (kg).



- 11 production holes @ 9 cartridges of 40mm dia (3.51 kg) per hole
- 4 adjacent to production holes @ 7 cartridges of 40mm dia (2.73 kg) per hole
- 12 alternate perimeter holes @ 8 cartridges of 25mm dia pre-assembled in PVC pipe (1.0 kg) per hole

Fig. 3 - Modified blast design for the hopper portion



Section along A-A

2.0m depth holes @ 5-6 cartridges of 40mm dia per hole
4.0m depth holes @ 11-12 cartridges of 40mm dia per hole

Fig. 4 - Blast design for excavation of the settling trench and flushing conduit

Table 1 - Details of vibration monitored at desilting chambers

Date	Bl. #	Blast location	Location of vibration instrument	D (m)	Q (kg)	SD	PPV, mm/s
21/10/02	1	C3, Hopper 30 RD	C2, Lower beam, left side, 125 RD	95.0	12.48	26.89	2.41
			C2, Middle beam, right side, 260 RD	230.0	12.48	65.11	3.35
22/10/02	2	C3, Hopper 27 RD	C3 behind the blast, 2 RD	25.0	12.48	7.08	37.50
			C4, Lower beam, right side, 15 RD	30.0	12.48	8.49	39.50
25/10/02	3	C3, Hopper 20 RD	C3, Middle beam, right side, 35 RD	15.0	12.48	4.25	39.80
			C3, Top beam, right side, 20 RD	18.0	12.48	5.10	27.90
25/10/02	4	C3, Hopper 16 RD	C3, Middle beam, right side, 100 RD	100.5	14.01	26.85	4.98
			C4, Lower beam, right side, 10 RD	27.7	14.01	7.40	23.40
27/10/02	5	C2, Hopper 5 RD	C3, Lower beam, right side, 150 RD	145.0	15.60	36.71	1.87
			Transition 3, 0 RD	65.5	15.60	16.58	6.00
27/10/02	6	C3, Hopper 12 RD	C3 Lower beam, right side 150 RD	138.0	12.48	39.06	5.97
			Transition 3, 0 RD	62.0	12.48	17.55	13.60
23/11/02	7	C3, Hopper 105 RD	Behind the blast	15.0	12.48	4.25	22.30
			Behind the blast	61.0	12.48	17.27	12.60
			C4, Lower beam, left side, 105 RD	32.0	12.48	9.06	30.55
24/11/02	8	C4, Trench and Conduit 127-137 RD	Behind the blast	30.0	17.75	7.12	33.80
			Behind the blast	90.0	17.75	21.36	23.40*
			Behind the blast	126.0	17.75	29.91	16.83*
27/11/02	9	C3, Hopper 100 RD	C3, Transition 3, 0 RD	150.0	15.65	37.92	2.27
			C3, Transition 3, 40 RD	124.0	15.65	31.34	4.25
			C2, Lower beam, left side, 100 RD	32.0	15.65	8.09	38.80
30/11/02	10	C3, Hopper 90 RD	C2, Lower beam, left side, 60 RD	43.9	13.00	12.16	7.57
			C2, Lower beam, left side, 30 RD	68.0	13.00	18.86	5.94
			C2, Lower beam, left side, 90 RD	32.0	13.00	8.88	21.41
01/12/02	11	C4, Trench and Conduit 147-157 RD	C3, Transition 3, 0 RD	204.5	18.00	48.20	1.30
			C3, Transition 3, 40 RD	155.3	18.00	36.60	2.48
			C2, Lower beam, left side, 86 RD	78.0	18.00	18.38	4.80
01/12/02	12	C3, Hopper 88 RD	C3, Transition 3, 0 RD	138.0	12.50	39.03	1.30
			C3, Transition 3, 40 RD	101.0	12.50	28.57	2.27
			C2, Lower beam, left side, 86 RD	32.0	12.50	9.05	13.70
03/12/02	13	C3, Hopper 84 RD	C3, Transition 3, 0 RD	152.0	11.70	44.44	1.05
			Adit 1	212.0	11.70	61.98	2.81
			C2, Lower beam, left side, 82 RD	32.0	11.70	9.36	19.12
05/12/02	14	C3, Hopper 80 RD	C2, Lower beam, left side, 50 RD	43.0	13.65	11.64	19.70
			C2, Lower beam, left side, 20 RD	68.0	13.65	18.41	7.48
			C2, Lower beam, left side, 78 RD	32.0	13.65	8.66	19.12

***** *Not considered for regression analysis; C2, C3 and C4 denote Chambers 2, 3 and 4 respectively*

Using this predictor equation, peak particle velocities were calculated for distances up to 30m for maximum charge per delay (Q) of 10kg, 15kg and 20kg (Fig. 6). This distance covers the partition between the chambers and the crown and this charge meets the requirement of the blast designs. In order to restrict the extrapolation of peak particle velocity beyond the measured range of data, near field vibration monitoring is suggested in such cases.

3.2 Effect of Vibration on Rock Masses and Support Systems

Damage is the deterioration of rock mass strength due to the presence of newly generated or extended fractures or opening and shearing along cracks and joints. Damage may occur as a result of adjustment of stresses around the excavation and effect of blasting. Various codes and standards have been prescribed for ground vibration limits in different countries for surface structures. There are no such standards of blast vibration for underground structures. The research work carried out by NIRM (Adhikari et al., 1994) on vibration levels vis-à-vis damage to underground structures is reproduced in Table 2. A PPV up to 250 mm/s may be considered as tolerable limit. The suggested PPV is also consistent with the results of other investigators (Yu and Croxall, 1985; Anon, 1987)

Table 2 - Nature of damage with respect to peak particle velocity (Adhikari et al., 1994)

Nature of damage	Peak particle velocity, mm/s	
	For fair rock at Kalyadi Copper Mines, HGML	For poor rock at Ramagiri Gold Mine, BGML
No damage	Less than 153	Less than 52
Opening & widening of joints	153 – 217	52 – 195
Dislodging of loose pieces	217 – 367	195 – 297
Induced cracking	367 – 604	297 – 557
Excessive damage	Greater than 604	Greater than 557

All blasts monitored by NIRM were safe as no cracks were observed in SFRS and no falls were noted from the roofs and walls of the chambers. The overbreak was on the higher side, though at some locations it was due to unfavorable orientation of discontinuities

The mechanical performance of fully grouted bolts subjected to close proximity blasting is not influenced by the blast vibrations. Therefore, fully grouted bolts can be used close to blast (Stjern and Myrvang, 1998).

Reinforced shotcrete maintains its functionality and sustains only minor damage when exposed to PPV of 1500 to 2000 m/s due to nearby blasting (McCreath et al., 1994). It is thus able to survive well beyond the threshold values of PPV at which rock fracturing would be anticipated.

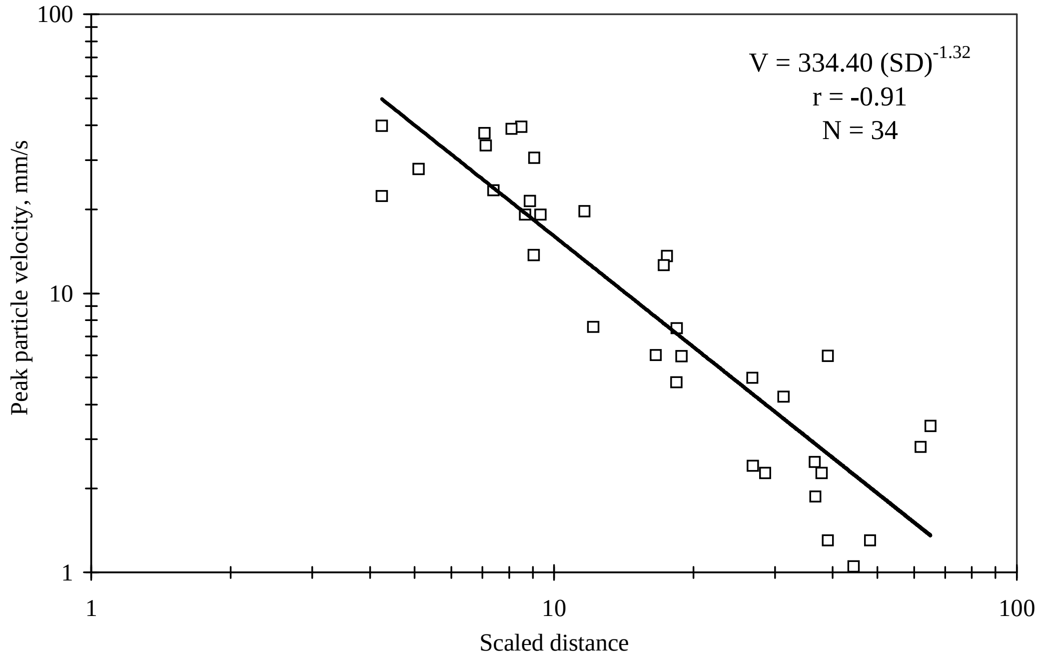


Fig. 5 - Peak particle velocity versus scaled distance for desilting complex

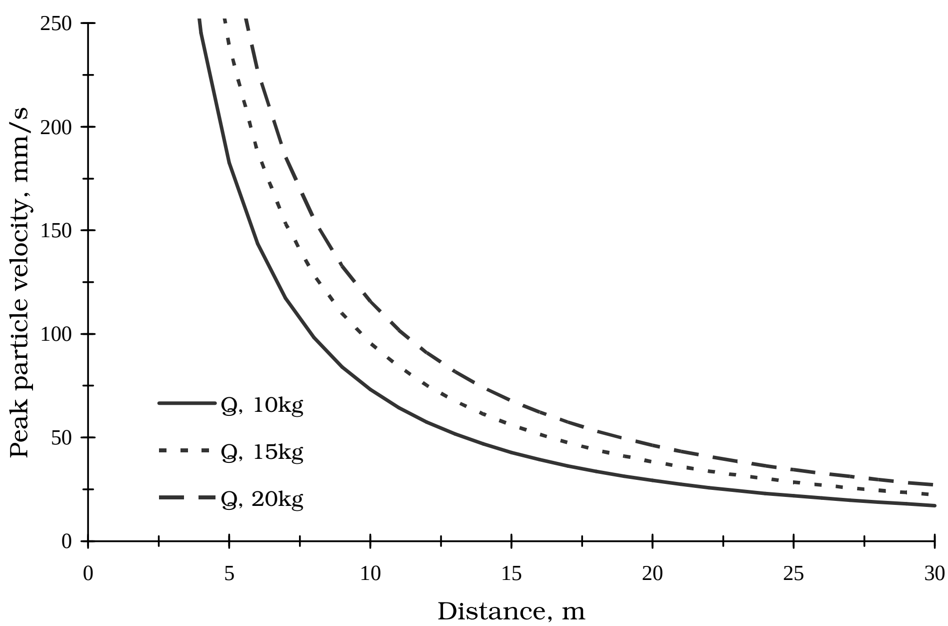


Fig. 6 - Peak particle velocity versus distance for different maximum charges per delay

Permissible PPV for concrete depends on its curing or hardening time (Jimeno et al., 1995). During the hardening period of 0 to 4 hours, the concrete is still not hard and permissible levels are relatively high. From 4 to 24 hours, it begins to harden slowly, and after 7 days it reaches a strength that is approximately 2/3rd of the final product (28 days), allowing a progressive intensification of the vibrations.

4. CONCLUSIONS

- The blast design for excavation of the hopper portion of the desilting chambers has been modified incorporating smooth blasting but no changes were made in the blast design for settling trench and flushing conduits.
- All the blasts reported in this study did not cause any noticeable damage to the walls and roofs of the chambers. As some damage to SFRS particularly on the right side wall of the chambers in geologically weak zones were reported, controlled blasting has to be carefully conducted for further excavation of the desilting chambers.
- There are no standards of blast vibration for underground structures. The safe limit of blast vibration for rock and SFRS can be approximately taken as 250 mm/s. In case of concrete, it depends on the curing time of concrete. Research is to be conducted in this area.
- The peak particle velocity can be estimated from the derived equation or from Fig. 5. It can be controlled by restricting the maximum charge per delay as low as possible. Vibration should be monitored on regular basis to check the actual vibration levels and to ensure field control and safety.

Acknowledgments

The work reported in this paper was conducted under a project sponsored by the Satluj Jal Vidyut Nigam Ltd. The authors thank the engineers of SJVN and the contractors for their support during the field investigation.

References

Adhikari, G. R., Rajan Babu, A., Balachander, R. and Gupta, R. N. (1999). On the application of Rock Mass Quality for blasting in large underground chambers, Tunnelling and Underground Space Technology, Vol. 14, No.3, pp 367 – 375.

Adhikari, G. R., Rajan Babu, A., Venkatesh, H. S. and Theresraj, A. I. (1994). Blast damage assessment for underground structures, Proc. National Symp. on Emerging Mining & Ground Control Technologies, Feb. 28-19, BHU, Varanasi, pp. 247-255.

Anon (1987). Explosives and Rock Blasting. Atlas Powder Company, Dallas, Texas, USA.

Jimeno, C.L., Jimeno, E.L. and Carcedo, F.J.A. (1995). Drilling and Blasting of Rocks. A.A. Balkema.

McCreath, D.R., Tennant, D.D. and Langille C.C. (1994). Survivability of shotcrete near blasts. Rock Mechanics, Nelson & Laubach (eds.), Balkema, Rotterdam, pp. 277-284.

Stjern, G. and Myrvang, A. (1998). The influence of blasting on grouted rock bolts, Tunnelling and Underground Space Technology, Vol. 13, No. 1, pp. 65-70.

Yu, T. and Croxall, J. (1985). Kidd Creek blasthole stoping, Proc. International Symposium on Large Scale Underground Mining, Lulea, pp. 97-106.