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Effective Drilling & Blasting for Tunnel Excavation in Difficult Hilly Terrain of Himalaya

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ABSTRACT

The Government of India, in its first phase, proposed a new broad-gauge railway line project connecting Sivok in the state of West Bengal and Rangpo in the state of Sikkim, which would pass through the steep hilly terrain of Kanchenjunga-mountain ranges foothills, dense reserve forest and Teesta valley of Darjeeling. Out of the total 44.39 km of the rail line, 3.5 km falls in Sikkim state and the rest 40.89 km in West Bengal state. To avoid loss of forests, 14 tunnels with longest tunnel of about 5 km and 17 bridges are planned. The project has its national importance, as once completed, it will make it easier for the movements of civilians and soldiers along the border region.

Drilling and blasting being the preferred method of rock excavation worldwide due to low initial investment, cheap explosive energy and easy acceptability, was considered for excavating all the tunnels. However, it posed enormous problems of underbreak and overbreak especially at the crown that invited innovative design patterns for quick, well-controlled and cost-effective operations. Various drilling patterns were executed for blasting solid rock faces using wedge cut or V cut patterns in contrast to conventional burn cut patterns with parallel-drilled holes. The cut displaces a good wedge of rock out of the face leading to achieve almost 90% pull using 0.9 to 1.1 kg/m^3 charge factor.

Keywords: Rock quality designation; Rock mass ratings; Q-index; Blastability index; Charge factor; Cut-area; Look-out angle

1. INTRODUCTION

Drilling & blasting is the most common technique of rock excavation in tunnelling. It is the cheapest and easiest technique in contrast to TBM (Tunnel Boring Machine) operations, which does not work suitably in Himalayan rocks due to the most challenging ground conditions. This is due to the fact that the rock mass of the Himalayan region is fragile in nature.

An explosive, upon detonation produces heat in the tune of 4500 °C and subsequently followed by large quantities of high-pressure gases in the range of 20-25 GPa that create new cracks to the surrounding rock mass and vents through existing cracks at extremely high force to overcome the confining forces of the surrounding rock formation in producing fragmentation. The energy released due to detonation results in four basic components namely rock fragmentation, rock displacement, ground vibration and air blast.

Characteristically, a tunnel blasting is more constrained than a bench blasting as tunnel blasting is done towards one free face while bench blasting is done towards two or more free faces. Such restrictions compelled a tunnel blasting to limit its round length, and the volume of rock that can be blasted at one time. The rock is thus more constricted in the case of tunnelling and thereby artificial free faces are created towards which the rest of the rocks break and are thrown away from the face. These artificial free faces are produced by a cut on the tunnel face and can be either a parallel hole cut, a V-cut, a fan-cut or other ways of opening up the tunnel face. After the cut opening is made, the stoping towards the cut starts.

Indian Railway Construction Limited (IRCON), under the ownership of Indian Railways, Ministry of Railways, Government of India has been awarded the design and construction of 44.39 km-long new broad gauge railway line project between Sivok (W.B.) and Rangpo (Sikkim), which will provide access to Gangtok in Sikkim. This first-phase project would improve the connections between Sikkim's north-eastern cities – which lie on the Chinese border and the rest of India. The second phase of the project will connect Sikkim to the tail-end of the Chinese border. The project has its national importance, as once completed, it will make it easier for the Indian Army to deploy soldiers along the border in the region. Once operational, it will be the first time to connect Sikkim with the main Indian rail network through Siliguri, the gateway of north-east and the same is expected to boost local tourism and the region's economy. The proposed railway line would pass through steep hilly terrain of Kanchenjunga-mountain ranges foothills, dense reserve forest and Teesta Valley of Darjeeling district of West Bengal and East Sikkim district of Sikkim state (Fig.1).

Fig. 1 - Portal location and alignment of Tunnel-1 of the 44.39 km long tracks connecting Sivok (W.B.) to Rangpo (Sikkim)

2. BRIEF OUTLINE OF THE PROJECT

The 44.39 km long track will have 17 bridges and 14 tunnels measuring 38.65 km of the track. The track is planned to be completed by December 2024 under Northeast Frontier Railway. Bridges over deep gorges and valleys will provide a scenic journey. The track has to be traversed through the foothills of the Kanchenjunga Mountain range and the Teesta River valley. New railway stations will be constructed at Melli, Teesta Bazaar, Geil Khola, Riang, and Rangpo which will provide access to Gangtok in Sikkim.

In 2008, the line was proposed to be 53 km long with 1,676 mm (5 ft 6 in) broad-gauge track but the final alignment became 44.39 km long with 3.5 km in Sikkim state and the rest in West Bengal state. The railways made a proposal in February 2013 to install elephant sensors along the stretch of the proposed railway line in Mahananda elephant sanctuary or run the trains at a speed of only 20 km per hour in the forest area and stop when an elephant is sighted close to the track. The Supreme Court of India approved the project in February 2016 with strict guidelines of the National Wildlife Board that cleared the project in June 2015 but ordered restricted speed, wireless animal tracking sensors and allowed digging of tunnels only during daytime. The railway line is needed for socio-economic and security reasons. The railway line will help troops and armaments move faster towards the Indo-Tibet border. The railway line up to Rangpo is expected to be completed by December 2023. In the second phase, the line will be extended by another 69 km stretch between Rangpo and Bhusuk of the state of Sikkim.

3. GEOLOGY

3.1 Project Geology

The project area lies on the Eastern Himalaya, representing the Gorubathan sub-group under the Daling Group of rocks ranging from the geological age of Middle to Late Proterozoic age. The rock type of this area is mainly chlorite quartz schist, chlorite schist, phyllite with or without sericite, quartz-sericite schist, sericite-quartz schist, slaty phyllites, slates and quartzite.

Sikkim-Darjeeling Himalayas are techno-stratigraphically defined by four domains with characteristic stratigraphic and structural characteristics. From south to north, they are (a) Foothill belt; (b) Inner belt; (c) Axial belt and (d) Trans-axial belt.

The state is mostly covered by Precambrian metamorphic rocks of low to medium grade (Daling Group), high-grade gneisses (Darjeeling gneiss and Kanchenjunga gneiss), Chungthang formation (quartzite, calc-silicate rocks, marbles, graphite schist's and occasionally amphibolites) with intrusive granites. The Kanchenjunga gneiss comprises mainly high-grade gneiss. The Chungthang gneiss is quartz-biotite gneiss. A streaky sheared granite gneiss known as "Lingtse gneiss" occurs as a NE-SW to N-S trending strip of rocks and forms a general line of separation between the Daling and high-grade Kanchenjunga gneiss. Gondwana Group of rocks are represented by a basal pebble slate followed by coal-bearing sandstone-shale horizons with occasional plant fossils equivalent to the Damuda group of rocks of the Indian Peninsular shield. The rocks of Buxa formation occur as thrusted wedges along the thrusted contact of the rocks of Gorubathan formation and Gondwanas. The boundary between Gondwanas and Dalings and between Dalings and Darjeeling group of rocks is tectonised and thrusted. During initial phase of blasting, huge overbreak at the crown and underbreak at the sidewalls was encountered, which was subsequently eliminated by the judicial implementation of blast design patterns by studying rock quality indices and rock-geologic parameters. The details of the rock mass type, rock condition, rock mass rating (RMR), expected rock class, joint orientations and other properties are given in Tables 1 and 2.

3.2 Rock Condition at the Working Sites

The paper considers only the design patterns of Tunnel-1 (T1: 4224 m), Tunnel-2 (T2: 896 m), Tunnel-11 (3232 m) and Adit tunnels of T1 & T11. Rock descriptions of these tunnels are given in Tables 1 and 2.

Table 1 - Rock descriptions of T1 and T2

Tunnel No.	Rock conditions	RMR
Tunnel-1 Portal-1 (T1P1)	Sandstone mixed with shale and siltstone. Prominent 3 sets of joints including bedding encountered. Rock strength varies from 23 MPa to 01-02 MPa. Rock class varies from IV to V.	18-35
Tunnel-1 Portal-2 (T1P2)	Mostly un-weathered sandstone, rarely conglomerate layers or carbonaceous shale layers appear intercalated with sandstone. Mainly 3 sets of joints including bedding encountered. Rock strength varies from 53-56 MPa. Rock mass falls in Class III.	54-59
Tunnel-2 Portal-1 (T2P1)	Slightly to moderately weathered sandstone, sometimes thin layers of shale and carbonaceous shale also present. Low grade quarzitic sandstone sometimes appears near bottom of the face. 04 sets of joints including bedding encountered. Rock is hard near bottom and soft near crown. Wedge formation is very common. Rock mass falls in Class V.	34-38

<u>Notation:</u> RMR - Bieniawski's rock mass rating as per 1989 classification

Table 2 - Rock descriptions of T11

Rock Type	Rock Class	Rock description	Overburden	Parameters
Phyllite & Quarzitic-Phyllite	V	Rock is weak in strength. Thinly foliated and highly fractured. Clay infilling between the joints.	60-63 m	F1: 75/270; J1: 50/290 J2: 85/225; J3: 81/290 UCS: 25-50 MPa RQD: 22; Q: 0.37
Phyllite & Quartzose-Phyllite	III	Rock is weathered thinly foliated quartzite. Clay infilling between the joints.	60-63 m	F1: 60/320; J1: 55/195 J2: 75/215; J2: 52/145 UCS: 25-50 MPa RQD: 43; Q: 1.43

4. THE CHALLENGES

The tunnel traverses through vulnerable and challenging geological conditions of the Lesser Himalaya. To counter vulnerability, the latest groundmass and most sophisticated tunnelling technology, i.e., New Austrian Tunnelling Method or NATM had been adopted after the completion of the locational survey, showing the stretch of rail route to about 44.39 km starting from Sivok in West Bengal to Rangpo, the excavation work started with the approval of the Railway Board. As per the survey out of 44.39 km stretch, 38.65 km falls under tunnel numbering 14 of which 3 tunnels would be more than 4 km in length (Table 3).

Table 3 - Tunnel number and its length

Tunnel No.	Length (m)	Tunnel No.	Length (m)	Adit (m)	Length (m)
T-1	4224	T-8	4148	T-1 Adit	865
T-2	896	T-9	538	T-4 Adit	600
T-3	1268	T-10	5300	T-6 Adit	577
T-4	3968	T-11	3232	T-7 Adit	617
T-5	2140.2	T-12	1404	T-8 Adit	1010
T-6	3912	T-13	2560.5	T-10 Adits	No.1: 1144; No. 2: 412.7
T-7	3082	T-14	1977	T-11 Adit	1036

With regard to the second phase of connectivity for a stretch of 69 km connecting Rangpo to Bhusuk, the preliminary engineering traffic (PET) survey has been completed. The survey team has proposed two more stations – one near Saramsa Garden/Chota Singtam and the other at just beyond Bhusuk Junior High School area. It is learnt that the train ferrying state would have stoppages at Rangpo main junction, Singtam, Nimtar, Chota Singtam/Saramsa station and Bhusuk.

At present, all the activities related to construction of tunnels, bridges and stations on this project are in progress and targeted to complete by December 2024.

5. IMPORTANCE OF CUT IN TUNNEL BLAST DESIGN AND GUIDELINES FOR PARALLEL CUT

The principle behind tunnel blasting is to create an initial opening or artificial free-face by means of a cut. If a pull obtained from the cut area is poor, it would not be possible to achieve good yield. Optimization of pull from the cut area is the most important part in any tunnel blast design in order to achieve maximum pull (Fig. 2).

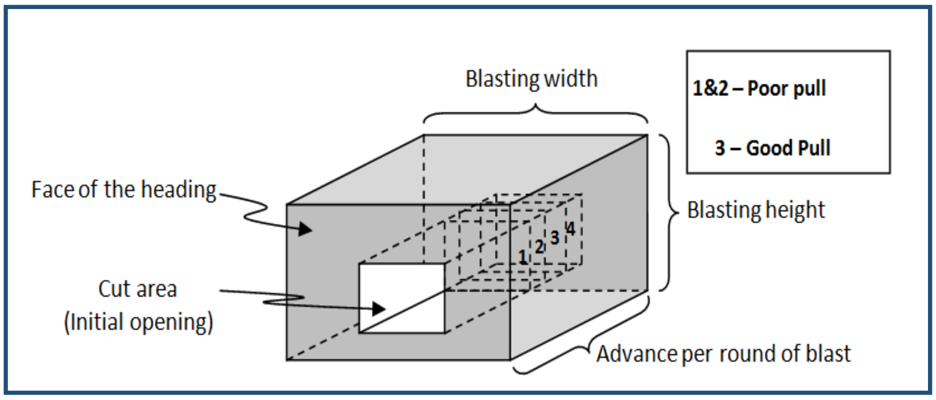

Fig. 2 - Pictorial view of cut area and advance per round of blast

Jimeno et al. (1995) has pointed out that due to the mechanized drilling based upon the hydraulic jumbos, the inclination has been towards parallel hole cuts as they are easier to drill, do not require a change in the feed angle and therefore, the advances are not as conditioned by the width of the tunnel, as it happens in wedge cut and angled cuts. After thorough experiments, Holmberg (1982) and Jimeno et al. (1995) had given a number of constitutive equations to determine the design parameters for effective tunnel blasting operations (Table 4).

The general geometric pattern of a four-section cut with parallel blastholes is shown in Figure 3. The distance between the central blasthole and those of the first section (B_1) should not be more than 1.7 times D_2 to obtain fragmentation and satisfactory movement of the rock. Generally, the cut area consumes higher explosive charge and the charges are highly confined.

Table 4 - Guidelines for parallel-cut design parameters (After Jimeno et al., 1995)

Abbreviation	Parameter	Empirical formula
NB	Number of blasthole	Integer of $[\{(AT + 2L \times \sin \gamma)/B\} + 2]$
L	Depth of blasthole (m) (four-section cuts; Fig. 3)	$0.15 + 34.1\ D_2 - 39.4\ D_2^2$ For D_2 = 76 /100 mm; L = 2.5 / 3.00 m
For Cut and cut-spreader holes		
B_1	1st section burden (m) When drill deviation is more than 1%	$1.5 \times D_2$ (Fig. 3) $1.7\ D_2 - E_p$
B_2	2nd section burden (m)	$\sqrt{2} \times B_1$
B_3	3rd section burden (m)	$1.5 \times \sqrt{2} \times B_2$
B_4	4th section burden (m)	$1.5 \times \sqrt{2} \times B_3$
B_z	Practical Burden	$B_z = B - L \sin \gamma - E_p$
S_z	Practical Spacing	$S - L \sin \gamma$
I_f	Length of bottom charges	$1.25\ B_z$
I_c	Column charges	$L - I_f - 10\ D_1$
E_p	drilling error (m)	$(\alpha \times L + e')$
Q_l	Lineal charge concentration (kg/m)	$55\ D_1\ [B/D_2]^{1.5} \times [B - D_2/2] \times [C/0.4] \times [1/PRP_{ANFO}]$
X	Average advance per round (m)	$0.95 \times L$
S/B	Spacing/Burden	Lifters: 1; Stoping: 1.2; Row nearest to contour: 1.1; Contour: ~ 0.9
	Blastability	
	Good	Poor
Lifters	1 – 1.25 m	0.8 m
Stoping	1.1 – 1.15 m	1.0 m
Row nearest	1.0 m	0.9 m
Contour (average value)	0.8-0.9 (0.85) m	0.7-0.9 (0.8) m

Notations: AT = tunnel width (m); L = blasthole depth (m); B = design burden (m); γ = lookout angle, D_1 = Diameter of charged hole (m); D_2 = Diameter of the relief blasthole (m); E_p = drilling error (m); α = angular displacement (m/m); e' = collaring error (m); C = rock constant (usually 0.4) and PPR_{ANFO} = relative weight strength of explosive w.r.t. ANFO (typically for cartridge explosives); Q_l = 1.17 – 1.3 kg/m

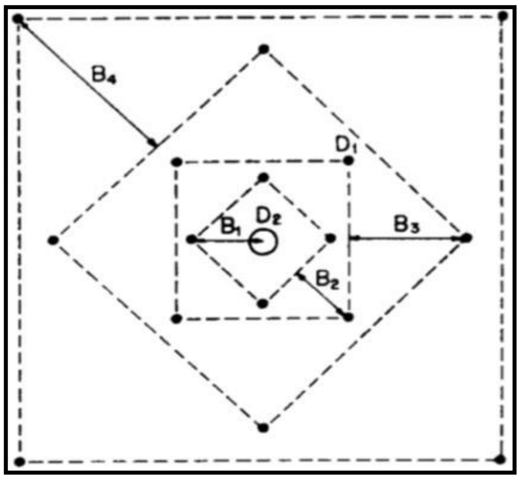

Fig. 3 - A typical four-section cut usually applies to tunnel blasting (Persson et al., 2001)

6. ROCK MASS BLASTABILITY

Rock mass blastability classification provides a theoretical basis for blast design of rock mass, which is used to estimate the unit explosive consumption, and to determine the effective design parameters for smooth excavation.

Lilly (1986) developed a blasting index based on rock mass description, Joint density & orientation, specific gravity and hardness. This index can closely be related to the powder factor (Ghose, 1988). To use Lilly's blastability Index, it is required to establish a site-specific relationship between this Blastability Index and the Powder Factor (Table 5) as well as guiding values for burden (B), spacing (S) and stoping area (Table 6). Bieniawski (1989) provided the rock mass class vs. suggested excavation patterns with the help of historical blast records and from the trial blast results (Table 7). Extending it further, based on several test blasts in various rock classes to determine the optimum design patterns, the authors established correlations between RQD, RMR, Q and Rock Class with varying charge factors for determining suitable blast patterns applicable for a particular class of rock with minimum overbreak and underbreak (Table 8).

Lilly (1986) proposed the following formula for blastabilty index (BI):

$$BI = 0.5 \times (RMD + JPS + JPO + SGI + H) \qquad (1)$$

where

RMD (Rock mass description)	= 10 – for powdery/friable rock mass,
	= 20 – for blocky rock mass,
	= 50 – for totally massive rock mass,
JPS (Joint plane spacing)	= 10 – for closely spacing (< 0.1 m),
	= 20 – for intermediate (0.1 – 1.0 m),
	= 50 – for widely spacing (> 1.0 m),
JPO (Joint plane orientation)	= 10 – for horizontal,
	= 20 – for dip out of the face,
	= 30 – for strike normal to face,

= 40 – for dip into face,

SGI (Specific gravity influence) = 25 x Specific Gravity of rock (t/m^3) – 50

H = Hardness in Mhos Scale (1 – 10)

Table 5 - Relationship between blastability index and powder factor (Ghose, 1988)

Parameters	Value				
Blastability Index	30-40	40-50	50-60	60-70	70-85
Powder Factor (kg/m^3)	0.7-0.8	0.6-0.7	0.5-0.6	0.3-0.5	0.2-0.3

Table 6 - Guiding values for burden (B), spacing (S) and stoping area (F_s) (NTNU, 1995)

Type of hole	45 mm drillhole diameter		
	Blastability quality	Burden (m)	Spacing (m)
Contour	Good Poor	0.8-1.0 0.7-0.9	0.7-1.0 0.6-0.9
Row nearest to contour	Good Poor	1.0 0.9	1.1 1.0
Lifters	Good Poor	1.0 0.8	1.0 0.8
Stoping (easers) F_s = S x B S/B = 1.2	Good Poor	F_s = 1.6 m^2 F_s = 1.2 m^2	

Table 7 - Rock mass class vs. suggested excavation patterns (After Bieniawski, 1989)

Rock mass class	Excavation	
I - Very good rock *RMR*: 81-100	Full face, 3 m advance.	Guidelines for excavation and support of 10 m span rock tunnels in accordance with the RMR system.
II - Good rock *RMR*: 61-80	Full face, 1-1.5 m advance	
III - Fair rock *RMR*: 41-60	Top heading and benching. 1.5-3 m advance in top heading	
IV - Poor rock *RMR*: 21-40	Top heading and bench 1.0-1.5 m advance in top heading.	
V – Very poor rock *RMR*: < 20	Multiple drifts 0.5-1.5 m advance in top heading.	

Table 8 - Estimated charge factor at different RQD, RMR, Q and rock class

RQD (%)	RMR	Rock type	Q	Rock class	Charge factor (kg/m^3)
-	< 10	Extremely poor	0.001-0.01	Class VI	0.5 - 0.6
< 25	10 - 20	Very poor	0.01 - 0.1	Class V	0.7 - 0.8
25 - 50	21 - 40	Poor	0.1 - 1.0	Class IV	0.9 - 1.0
50 - 75	41 - 60	Fair	1 - 3	Class III	1.1 - 1.2
75 - 90	61 - 80	Good	3 - 10	Class II	1.2 - 1.5
90 - 100	81 - 100	Very good	10 - 40	Class I	1.5 - 2.0

Using data parameters given in Tables (5, 6, 7 and 8), the overall design guidelines were compiled and shown in Table 9. These guidelines were used to determine the design parameters for tunnels T1, T2, T5, T11 etc. which were widely tested and analysed for various rock classes and found to be optimum for successful progress of the tunnels (Table 10). The pictorial views of such designs are shown in Figures 4-8.

Table 9 - Design guidelines for smooth and controlled tunnel excavation

Parameters	Observations
Empty hole diameter	Most suitable diameters are 89 to 102 mm (Smaller diameter may work well but their numbers have to be more)
Empty hole spacing	Class I & II, 30-35 cm c/c. Class III & IV, 20-25 cm c/c.
No. of empty holes	For 76 mm: 3 to 4; For 89 mm: 2 to 3; For 100 mm: 1 to 2; For 150 mm: 1
Spacing between main cut holes and empty holes	Class I & II – 3 to 4 times the diameter of dummy holes Class II & III – 2 to 3 times the diameter of dummy holes
Spacing between main cut holes and cut spreader holes	More than 30– 35 cm to avoid sympathetic detonation
Charge factor (kg/m^3)	In cut holes area varying between 2 and 2.5; cut spreader 0.7 and 0.9; side wall 0.5 and 0.6; roof 0.4 and 0.5 and bottom 0.9 and 1.2
Delay timing	For better progress 25-50 ms in the cut area holes and > 50 ms for remaining holes to have better overbreak control. For LDD, minimum delay interval is 100 ms
Lookout angle	Within 10 cm+3 cm/m of holes depth
Trajectory deviation	Feed pressure between 30 and 40 bars is optimum for cut holes to stop deviation
Age of explosive	Use within 6 months of the manufacturing date
Stemming length (m)	Not less than 40% of the hole depth using moist clay sticks

Table 10 - Blast design parameters established for tunnels for effective performance

Parameters	Rock class-II/III	Rock class-IV	Rock class-V	Rock class-III	Rock class-II/III
	Explosive Diameter (40 mm)	Explosive diameter (32 mm)	Explosive diameter (32 mm)	Explosive diameters (32 mm & 40 mm mixed)	Explosive diameter (32 mm)
Rock type	Sandstone/ shale/siltstone/ carbonaceous shale	sandstone/ shale/siltstone/ carbonaceous shale	Sandstone/ shale/siltstone/ carbonaceous shale	Sandstone/ shale/siltstone/ carbonaceous shale	Phyllite & quarzitic-phyllite
Drilling pattern	Wedge-cut	Wedge-cut	Wedge-cut	Wedge-cut	Wedge-cut
Blasthole diameter (mm)	45	45	45	45	45
Blasthole length (m)	3.0	2.2	2.0	3.0	3.2
Length of cartridge (mm)	300	300	300	200/300	200
Weight per cartridge (g)	390	390	390	200/390	200
Area of tunnel (m^2)	60.60	61.90	60.08	60.60	61.90
Muck generated (m^3)	151.52	105.23	78.10	151.52	173.32
No. of charged holes	80	79	64	80	108
No. of dummy holes	25	29	42	25	27
Total no. of holes	105	108	106	105	135
Total explosive used (kg)	163.02	117.39	82.29	134.52 (40 mm/ 104.52 kg; 32 mm/30 kg)	185
Power factor (kg/m^3)	1.07	1.11	1.05	0.88	1.1

Table 10 cont…

Pull achieved (m)	2.5	1.7	1.3	2.5	3.0
Initiation system	Nonel / IDD	Nonel / IDD	Nonel / IDD	Nonel / IDD	Nonel / IDD
Type of explosive	Cartridged emulsion	Cartridged emulsion	Cartridged emulsion	Cartridged emulsion	Cartridged emulsion

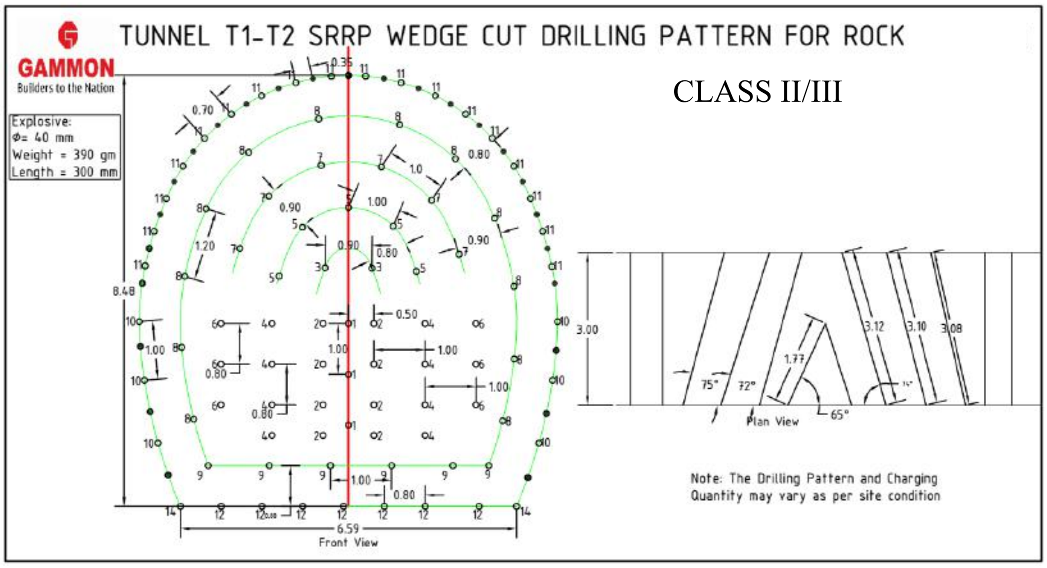

Fig. 4 - Blast design for rock class-II/III using 40mm/300mm/390 g explosive

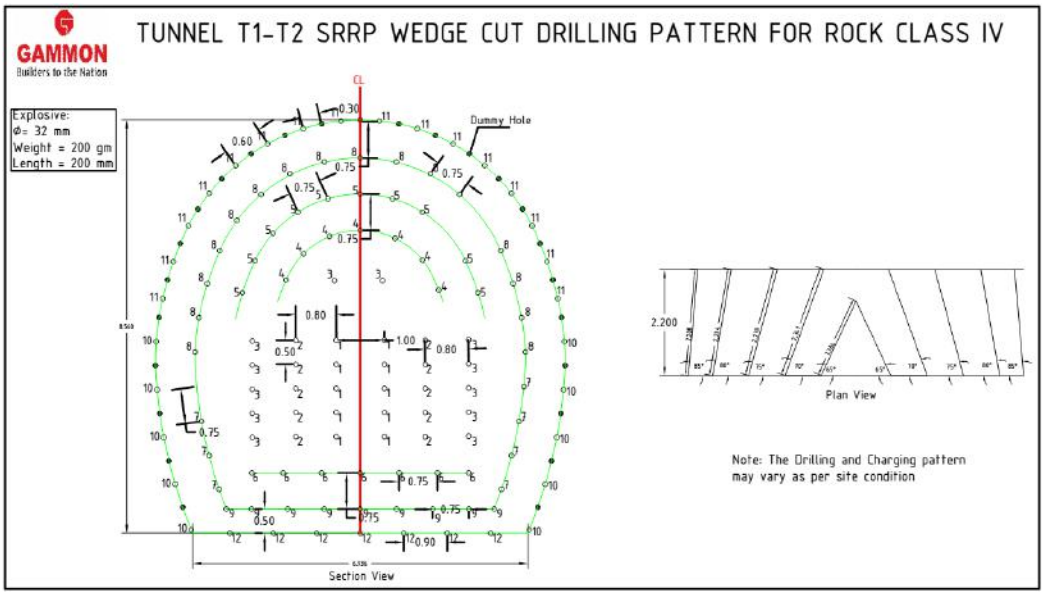

Fig. 5 - Blast design for Rock Class-IV using 32 mm/200 mm/200 g explosive

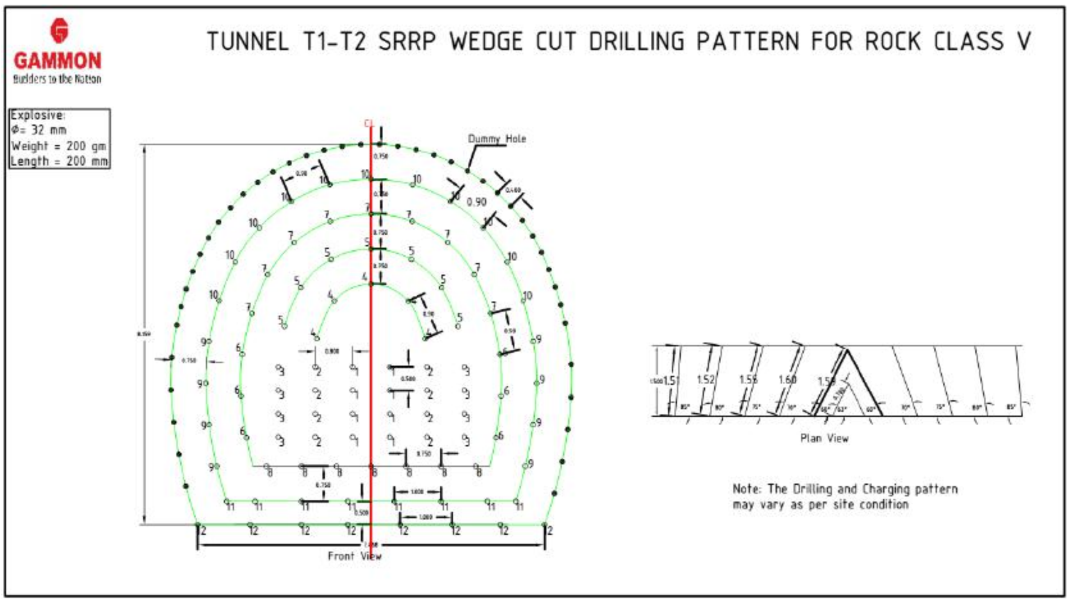

Fig. 6 - Blast design for Rock Class-V using 32 mm/200 mm/200 g explosive

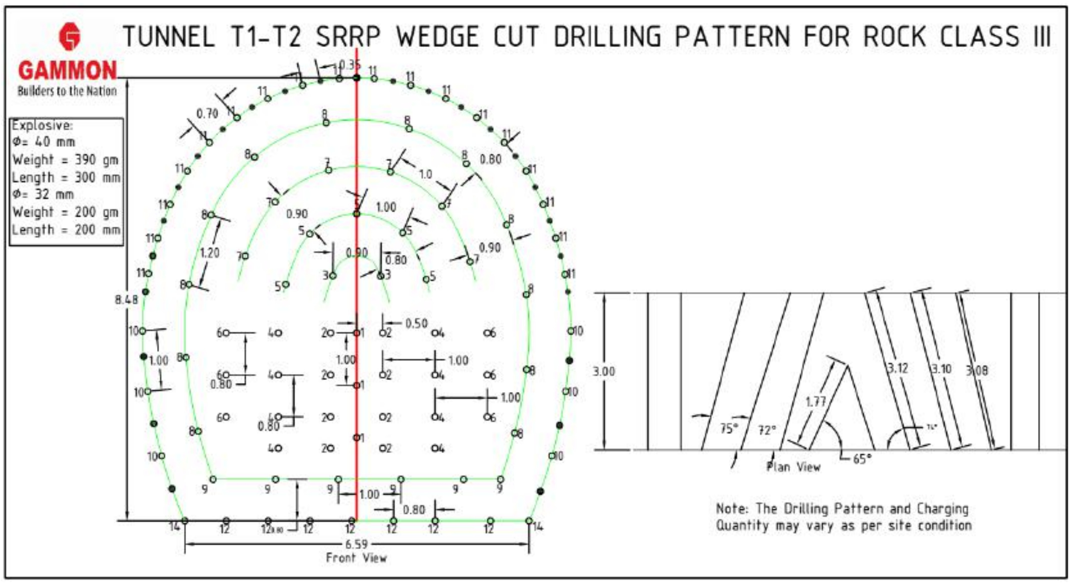

Fig. 7 - Blast design for Rock Class-III using explosive cartridges of both 40 mm and 32 mm diameter

7. OVERBREAK AND LOOK-OUT ANGLE

Determination of blast-induced overbreak is a very critical factor in underground excavations where drilling and blasting methods are used. Overbreak, as an inevitable side-effect, significantly affects the cost and safety of underground constructions.

The look-out angle is the angle between the practical (drilled) and the theoretical tunnel profile. If the contour holes are drilled parallel to the theoretical line of the tunnel, the tunnel face gets smaller and smaller after each round. The amount of this look-out overbreak on the contour perimeter, which is defined as the void creator during the excavation in excess of an established perimeter or pay line, is usually correlated with the damage extension zone. It measures the quality of the blast overbreak and underbreak and mainly influenced by the geotechnical condition of the rock mass (i.e., rock disturbances and rock strength) and blast design parameters such as the explosive type, the charge concentration, the blast timing, the drill pattern and the drilling deviations (Oggeri and Ova, 2004; Singh and Xavier, 2005).

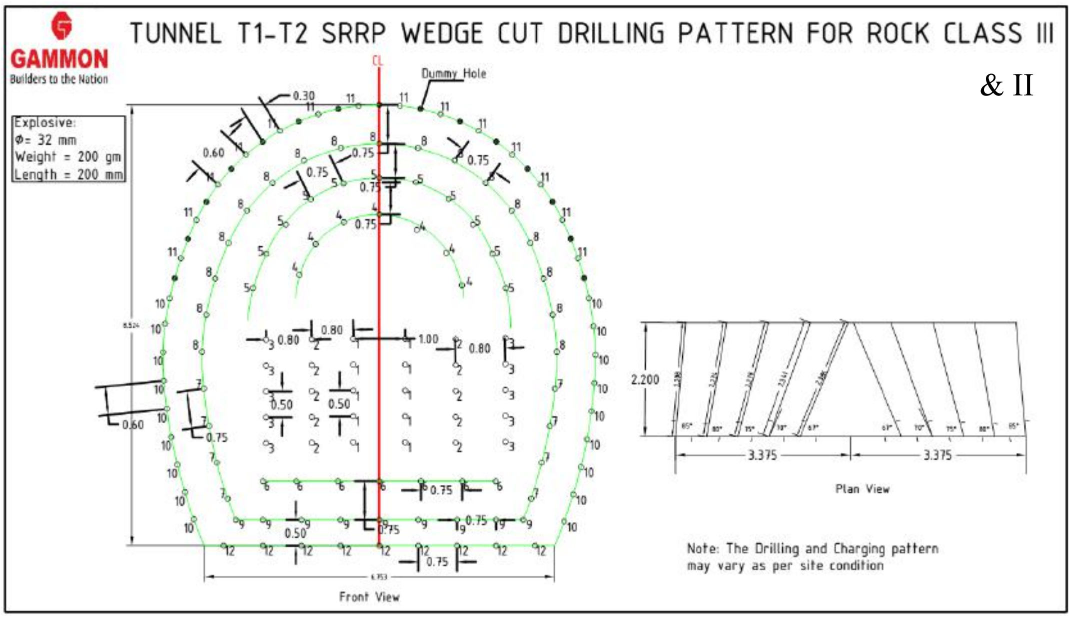

Fig. 8 - Blast design for Rock Class-II/III using 32 mm/200 mm/200 g

Blasting affects the rock mass structure because of shock wave propagation that it creates in the form of vibration, gas pressure, and stress redistribution. The look-out angle is usually taken as 3^0 with the tunnel profile (Jimeno et al., 1995). In case of drill hole length of 3 m, the burden at the toe of the blasthole becomes 3.0 x sin (3^o) = 3 x 0.0523 = 0.15 m = 15 cm. It shows a clear-cut indication of natural overbreak that is uncontrollable.

Similarly, for the purpose of forepole a minimum look-out angle is also required for lower class of rocks to stabilize the crown of the tunnel in advance. The burden for forepole becomes slightly greater as the length of the forepoles are 4.0 m. After considering the overlap between two consecutive forepoles, the effective length of a single forepole will be 3.6 m. therefore the burden at the toe of the Forepole becomes 3.6 x sin (3^o) = 3.6 x 0.0523 = 0.18 m = 18 cm. This 18 cm overbreak is uncontrollable and unavoidable.

Rock-geologic parameters being uncontrollable, it can be seen that amongst the controllable factors, lookout angle contributes the maximum role in overbreak (Fig. 9). Apart from blast design patterns, the skill of drilling operators and strict supervision of the site-engineers can significantly reduce the overbreak.

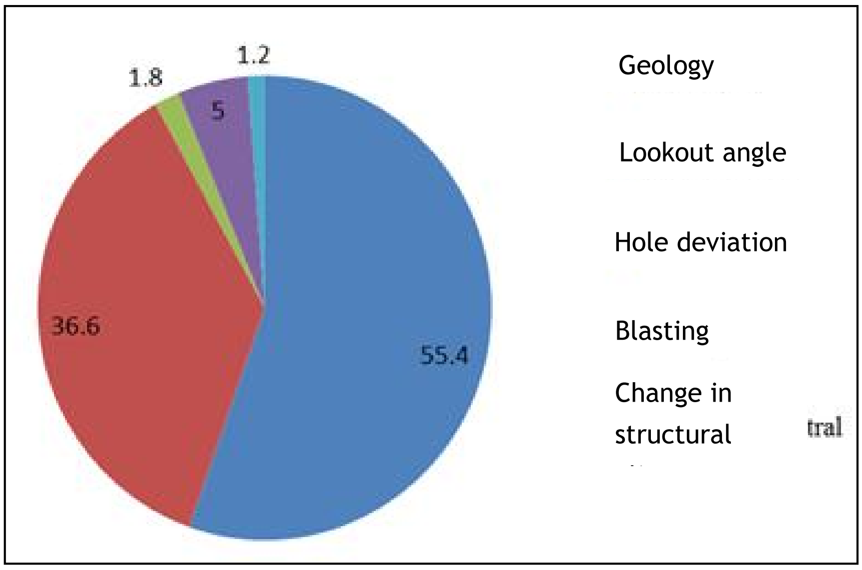

Fig. 9 - Influence of various parameters on overbreak in tunnel blasting

Guidelines on construction have established an overbreak magnitude of 15–20 cm and 10–15 cm in crown and sidewalls, respectively (Cunningham and Goetzsche, 1990). The maximum overbreak distance allowed depends on each national legislation and special terms which can be arranged between the concerned parties in the contract.

To analyze the extension of the overbreak, studies have been carried out by comparing the laser profile of the excavated perimeter with the designed tunnel profile. By comparing scanner profiles of the excavated sections with the blasthole positions, a methodology has been developed to obtain an Excavated Mean Distance (EMD) between the blastholes and the excavated profile, which may be considered a damage measure.

8. CONCLUSIONS

Tunnelling through the medium-hard, weak, jointed and fragile rock masses of Himalaya has always been a difficult and challenging task due to unpredictable geological features leading to huge amount of overbreak at the crown and underbreak at the side walls. Under such prevailing ground conditions, the traditional V cut patterns work well in symmetrically drilled parallel and angled cut-holes with restricted progress and pull. The authors through their varied experience observed that the typical tunnel blasting in Himalayan rocks needs the charge factor (kg/m^3) between 0.8 and 2 kg/m^3 depending upon the size of the tunnel. The charge factor may vary from 1 kg/m^3 in a tunnel with an opening size greater than 30 m^2 to more than 3 kg/m^3 for a size less than 10 m^2, in the same type of ground. Whereas, the typical drill factors vary between 0.8 and 4 m/m^3. However, in the present case studies at the experimental tunnels, charge factor varied between 1.0 to 1.2 kg/m^3 for rock classes II/III/IV and 0.85 to 0.9 kg/m^3 for rock class V, which were encouraging and cost-effective. The drill factor varied between 2.0 to 2.58 for all classes of rocks.

Excavation with a V-cut pattern is similar to that of an inverted surface excavation application. The V-cut is also based on surface blasting principles in which the angle for rock expansion equals or exceeds 90 degrees. Maintaining the proper angle in the cut area is the main difficulty in V-cut drilling; and, the correct drilling angle limits round length in narrow tunnels. The cut displaces a wedge of rock out of the face in the initial blast and this wedge is widened to the full width in subsequent delayed blasts. Blasthole placements should be carefully pre-planned and the alignment of each hole should be accurately drilled for better results.

Due to ease of applications, for tunnels of 60-70 m^2 cross-sections, full-face excavation gives maximum economy and efficiency. Full-face excavation is easily applicable for rock classes II, III and with little cautions to IV and V too. Blastholes are drilled into tunnel faces either at right angles to the face (Burn-cut) or at an angle to the face (Wedge-cut). Nowadays, for good roof conditions and with availability of better support systems, tunnels having larger cross-section greater than 60-70 m^2 can also be effectively excavated by full-face method.

It is revealed during experiments that an angle of approximately 3^o (normally called look-out angle) is required for the contour holes to drill with the theoretical line of the tunnel otherwise if they are drilled parallel to the axis of the tunnel, the tunnel face will get smaller and smaller after each round. It showed that for a look-out angle of 3^o, a minimum of 15 cm overbreak along the crown level and 18 cm overbreak in case of forepole are inevitable, which can't be controlled through blast design parameters.

To judge the performance of the excavation, two parameters are often calculated from a blast design; the powder factor or specific charge (kilograms of explosives per cubic meter of blasted rock) and the drill factor (total length of drill holes per volume of blasted rock (meter/cubic meter). These are the indicators of the overall economy of blasting and permit easy comparison among different blast patterns.

The paper shows how rock parameters and its quality indexes like RQD, RMR, Q and BI become the determining factors of various blast design and charge distribution parameters. It also indicates that the use of 32 mm diameter explosive at the perimeter with alternate dummy holes at the crown and for the rest of the holes 40 mm diameter explosive in 45 mm diameter blastholes yielded the best results with an approximate powder factor of 0.95 kg/m^3.

Conflict of Interest Statement

On behalf of all the authors, the corresponding author states that there is no conflict of interest.

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on their long-experience and experimental data generated at the sites, which are obviously site-specific and may change at different rock-geologic conditions.

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