Excavations by controlled blasting for the construction of an underground LPG storage cavern at vizag

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Abstract

An underground LPG storage cavern, first of its kind in India, was constructed at Visakhapatnam in the state of Andhra Pradesh having an overall net liquid storage capacity of 1,20,000 m^3. The work involved removal of about 1,50,000 m^3 of rock by blasting with the depth of underground excavations extending to about 190 m below sea level. The work involved excavation of two vertical shafts, tunnels of various sizes made to facilitate the commissioning of the main egg shaped LPG storage cavern with a cross sectional area of 314 m^2 excavated for a length of 357.2 m. Access ramps, water curtain galleries, upper and intermediate connections were also excavated. The main rock type is granite gneiss with predominant alternating of quartzo-feldspathic bands and has been categorized as Fair to Good using Q and RMR classifications. Three sets of major joints were present with two random joint sets. The NATM concept was adopted for excavations with blasting on encountering good rock.

Blasts were designed for jack hammer drill employing 32 mm diameter holes as well as for two boom jumbo drills for larger hole diameters ranging from 40 mm to 105 mm. The designs were made to suit site-specific requirements for various cross sections, shapes, advances, smooth blasting and controlled blasting applications. This paper deals with the excavation of Main LPG Storage Cavern Gallery.

The rock mass quality was invariably considered for sequencing and designing all the blasts. Blast induced damage zones for all types of blast designs were computed and restricted the overbreak within the permissible limits of 300 mm by achieving 70% hole impressions through smooth blasting techniques. The blast vibrations were monitored using five seismographs (one near field with eight channels and four far field seismographs). Predictor equations were derived using 300 data sets with a correlation coefficient of -0.84 giving rise to the equation $V = 190.57(D/ÕQ)^{-1.45}$. Fragmentation was controlled to the desired size of below 300 mm. Overall, the larger size fragments were restricted to about 5 to 7%. Blasts were designed for coupled and decoupled charges and power bulk drive was used for the first time in underground blasts in India. The excavation of LPG Storage cavern was completed successfully in spite of several challenges like varying rock mass conditions, heavy ground water seepages, rock falls, etc.

Introduction

The LPG storage project at Visakhapatnam is owned by South Asia LPG Company Pvt.Ltd (SALPG), India. It is a Joint Venture between Hindustan Petroleum Company Limited (HPCL) and Total Gas and Power India (TGPI), a fully owned subsidiary company of TOTALFINAELF S.A. incorporated in France. SALPG has been formed to develop and operate a LPG underground storage in the Visakhapatnam Port area, Andhra Pradesh, India. The Rs.333 crores liquefied gas (LPG) cavern storage facility is the first of its kind in South East Asia. The underground storage facilities will be used for storing and dispatching LPG received at a nearby jetty, where it will be unloaded from pressurized or refrigerated LPG carriers. The overall net liquid storage capacity is 1,20,000 m^3. The LPG stored will be re-exported via an existing

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Fig. 1: A view of the LPG project site at Vizag

pipeline connected to downstream evacuation facilities/ bottling plant in Visakhapatnam or to the nearby LPG berth for loading into pressurized vessels. The approximate depth of excavation is 190 m below sea level and the volume to be excavated is around 1,50,000 m^3. Figure 1 gives a view of the project site.

Blast Designs

Design Philosophy

In this project, the purpose of the excavation is to create underground storage space for gas with hydraulic confinement. Therefore, smooth blasting was essentially required to minimize damage to the surrounding rock mass. Poor blasting quality could damage the rock and increase the need for support and grouting that would be costly and time consuming. A good blast design is aimed at to maximize the pull, minimize over break, achieve desired fragmentation and ensure safety[1]. A blast primarily induces a compression wave into the rock mass. If there is a free face, this compression wave reflects back as a tension wave. It is this tension wave that is responsible for causing fractures in rock. Once the fractures are created, gaseous energy seeps through these fractures and cause the rock to break. The first task in the design of a blasting pattern is to create a free face. The holes and charging of these holes are designed in order to create a free face. These holes are generally called baby cut/cut holes. Subsequent holes are designed to break the rock mass in a sequence. The holes along the perimeter, called as perimeter holes, are designed such that the final surface is smooth and matches with the desired surface.

A design for blasting of any excavation results in a layout of holes showing charging and initiation patterns[2]. Wedge cut (V cut) method of drilling is used to create the additional free faces for the subsequent rows in the blast. All holes except for those on the two outermost circles are responsible for pull or advance per blast and the volume of breakage of rock per blast. Spacing of perimeter holes < or = 10 times diameter of blast holes whereas the Production holes are designed at a spacing of 12 to 14 times the diameter of blast holes. The perimeter (counter) holes are drilled very closely along the final line of excavation. They are charged lightly which is just sufficient to develop cracks between the holes. The holes next to the perimeter holes are also charged lightly compared to other holes but heavier than the perimeter holes. The purpose of these two groups of holes is to control overbreak so that the drill hole markings are visible after the blasts. In hard rock excavation where pull is a problem, these holes are to be charged heavily so that desirable pull and fragmentation are achieved. Good stemming with suitable material is required to prevent blown out and escape of gaseous energy into the atmosphere. Specific charge for shafts is rather high as compared to tunnels, ranging from 2 to 4 kg/m^3.

The blast holes are to be charged with suitable explosives and initiated in proper sequence using a non-electric initiation system for better performance and safety. Theoretically the perimeter holes should be initiated simultaneously. But due to restriction on ground vibration, holes are to be divided into 2/3 groups of holes. Keeping in view that scattering of delays may affect the smoothness of the walls, the group of holes is connected to a detonating cord, which in turn is connected to the detonator. Sufficient number of delay periods is to be provided to restrict the maximum charge per delay to control ground vibration and to protect the inherent strength of rock mass outside the line of breakage.

Site Specific Requirements

The following site specific requirements were mandatory as per the basic engineering design.

- The damage zone should be restricted within 300 mm from perimeter holes and penultimate holes to avoid damage to the surrounding rock mass and the blast induced block instability.
- After blasting, at least seventy-five per cent (75%) of perimeter row blast holes (half barrels) must be measurable. If not so, the round shall not be considered as excavated according to "smooth blasting".
- Blast induced vibration levels shall be controlled and recorded in order to avoid any damage to third parties or to any part of construction and surrounding rock mass.
- All blast holes shall be drilled horizontally, unless otherwise specified. Vertical holes for bench blasting are forbidden.
- The built line shall remain between O-line (over break) and U-line (under break). For shafts the O-line is set 30 cm outside the T-line (Theoretical) and U-line is 0 cm inside the T-line, for access tunnel and galleries the O-line is set 40 cm outside the T-line and U-line is set 20 cm inside the T-line.
- The maximum length of any blast hole is: Four (4) metre in galleries, including storage gallery top headings, Six (6) metre in storage gallery benches and Two and half metres (2.5 m) in shafts.

Field Investigations

Description of the Excavation Site

Field investigations at the project site were started in February 2004. A resident project scientist has been posted at LPG project site to supervise and monitor all the blasts. All necessary blast records and vibration records were regularly updated. To approach the LPG storage cavern gallery (16m max. x 25m x 357.2m), there are two vertical shafts running parallel to each other namely Access shaft and Operation shaft with 6.5m and 4.0m diameter respectively. The storage cavern gallery is located at El -162.0 (top) and El -186.0 (bottom). Storage cavern gallery is connecting two horizontal galleries called upper connection (10.0 m x 8.0 m) at El -162.0 (top) El -170.0 (Bottom) and intermediate connection (8.0 m x 8.0 m) at El -170.0 (top) and at El -178.0 (Bottom). One access ramp (8.0 m x 6.0 m) is running downward (15%) connecting the upper connection, intermediate connection and bottom of the storage cavern gallery. There is one horizontal gallery at EL -144.0 called water curtain gallery (4.0m x 3.5m) for maintaining the water level to fulfill hydrodynamic containment principle.

The excavations are made mainly in granite gneiss with predominant alternating quartzo-feldspathic bands. Near the surface, the rock was fractured and amenable for excavation without blasting up to EL -15.65 m and up to El -8.85 m in the access shaft. The rock formation was categorized as Fair to Good

Table 1: Dimensions of various excavations

Name of the excavation	Height x Width (m)	Shape	Cross sectional area (m^2)	Volume (m^3)
Access shaft	6.5 m dia, Depth=170m	circular	34.2	5816.13
Operating shaft	4 m dia, Depth=182m	circular	13.2	2344.39
Water curtain gallery	4 x 4 Length=210m	'D'	14.28	2940
Water curtain/Access shaft connection	4 x 4 Length=7.5m	'D'	14.28	105
Water curtain/Operation shaft connection	4 x 4 Length=14.5m	'D'	14.28	203
Upper shaft connection	8 x 5.5 Length=12m	'U'	44	528
Lower shaft connection	5 x 4.5 Length=50m	'U'	22.3	1115
Access Ramp	8 x 7 Length=114m	'U'	57	6498
Upper connection	8 x 5.5 Length=72m	'U'	44	3168
Intermediate connection	8 x 5.5 Length=64m	'U'	44	2816
Main gallery (Egg shape, 314 m^2 and 355 m length)				
Pilot heading (1)	8 x 8	'D'	57	20235
Crown sides (1a)	5 x 2.7	arc	8.25	2929
Intermediate benches (2 & 3)	5 x 8	rectangular	40	14200
Intermediate Sides (2a & 3a)	5 x 3.9	arc	18	6390
Bottom bench (4)	6 x 8	rectangular	48	17040
Bottom sides (4a)	5 x 3.9	arc	21	7455

using Q and RMR classifications. Three sets of major joints were present with two random joint sets. The NATM concept was adopted for excavations with blasting on encountering good rock. The first blast took place on 15th June, 2004 in operation shaft and on 27th July, 2004 in access shaft. The dimensions of the excavation are given in Table 1.

Blast Vibrations Monitoring

The blast vibrations were monitored using five seismographs (Fig. 2), one near field with eight channels and four far field seismographs.

Fig. 2: MiniMate Eight channel recording system

The monitoring stations were identified with respect to the critical structures. They are: Buried LPG pipeline, Buried crude oil pipeline, Water tank, Booster pumps and HPCL control room. For every blast one instrument is kept at the respective station and blast vibrations are recorded. The controlled blasts were executed from the first blast and all the blasts monitored were found to be safe with respect to the structures. Two pipe lines were buried at a depth of 1.5m from surface, one is passing between two shafts

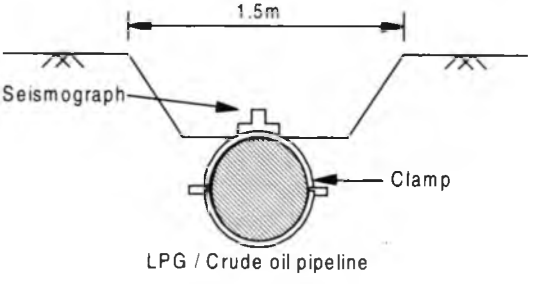


Fig. 3: Details of the monitoring pit at pipeline locations and placement of seismograph

and is carrying LPG the other is carrying crude oil and is at 68.8 m from access shaft. A special pit was made to keep the MiniMate Plus directly over the pipe lines for measuring blast induced vibrations (Figs. 3 and 4).

Fig. 4: Blast induced vibration monitoring at LPG pipeline

Blast vibrations were also monitored and controlled blasts were designed for those structures, which came up during the time of excavation. They are diaphragm wall, first aid station, canteen, store room and site office buildings. All these structures were identified as critical during the construction phase.

Safe Limits of Vibration for structures and pipelines

Authorities throughout the world have experienced difficulty in defining acceptable damage standards with reference to pipes. One of the main reasons is the variability in standardization of structure, its materials of construction and place where it is located (in liquid or solid). Attempts to establish safe levels of vibration, therefore, tend to be on the conservative side. Since it is unlikely that the physics of cracking change at national borders, these national variations are certainly influenced by several factors in addition to the crack susceptibility of structures. Various codes and standards have been prescribed for ground vibration limits in different countries for residential and industrial structures. The Office of the Surface Mining, USA has adopted a modification of the Bureau of Mines

Alternative Safe Blasting Criteria[2]. The recent trend is to refer to the frequency of the ground motion. Low frequency waves cause more damage to structures particularly in case of multi-storied buildings. It may be noted that all of these standards recognize particle velocity as suitable damage criteria but the threshold values vary over a wide range. Previous experience on monitoring, analysis and control of ground vibration in different conditions is essential. The Permissible and expected peak particle velocity (PPV) for surface structures[3] are given in Table 2.

Table 2: Permissible and expected peak particle velocity (PPV) for surface structures

Structure	Permissible PPV mm/s	
	8-25 Hz	> 25 Hz
LPG pipe line	127	
Crude pipe line	127	
Booster pumps	20	25
Water tank	20	25
HPCL control room	20	25

The permissible PPV values for nearby structures (RCC structures), are based on curing time of concrete and they are given in Table 3.

Table 3: Permissible peak particle velocity for RCC structures

Curing time (hrs)	Permissible PPV, (mm/s)
< 24	No Blasting
24 to 72	25
72 to 148	50
148 to 240	100
> 240	200

Blast vibration Analysis

For all the blasts, blast vibrations were monitored at required locations. In total 300 sets of readings were used for regression analysis. Figure 5 shows a plot of vector sum against square-root scaled distance on a log-log graph. The following predictor equation was derived based on the data generated.

$V = 190.57(D/ÖQ)^{-1.4358}$ Eq (1)

N = 300 r = -0.84

N = Number of data sets r = Correlation co-efficient

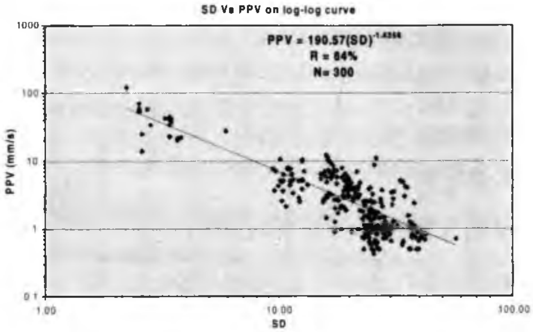


Fig. 5: Peak particle velocity Vs scaled distance

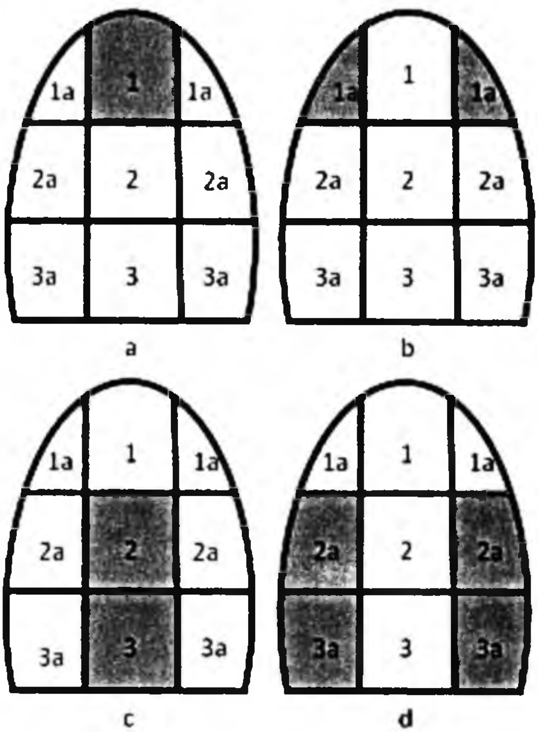
The maximum peak particle velocity (PPV) recorded at LPG pipeline is 10.7 mm/s (recorded when a blast was taken in the Access Shaft) and the minimum recorded is < 0.5 mm/s. From the overall data, the maximum peak particle velocity monitored was 122.0 mm/s at ring beam of Access Shaft when the blast was taken in the Access Shaft.

Excavation of the Main Storage Cavern

Sequence of excavation

Considering the large size of the egg shaped main gallery having dimensions of 24m height, 16m width (max.) and 355 m length with a total cross sectional area of 314m^2, The sequence of excavation is shown in fig 6. It involved crown portion, intermediate and bottom benches. The crown portion was made with a central pilot heading (a) side slashing (b), followed by intermediate and bottom benches (c). Finally the side slashing of intermediate and bottom benches was done (d).

In this case, a 8 x 8 m pilot heading with a cross sectional area of 57.468m^2 was driven to full length at the crown. The pilot heading was named as sequence 1. In the next stage, side slashing of the crown portion was taken up. The side portions are named as 1a. After the excavation of the crown portion was completed, the bottom sections were excavated by benching method. The benching cycle comprised of two benches, i.e. one intermediate and bottom. The benches were of 8 m high having a width of 8 m and length of 6 m. The benches were named as 2 and 3 respectively. The advance per blast was 6 m.



EXCAVATION SEQUENCE

Fig. 6: Excavation sequence for the main gallery.

Blast Volume: The blast volume for various cross sectional area and shapes as per the sequence of excavation is given in Table 4.

Table 4: Blast volumes for different stages of excavation

Name of the excavation	Height x width (m)	Shape	Cross sectional area (m^2)	Volume (m^3)
Pilot heading (1)	8 x 8	'D'	57.468	227.82
Crown sides (1a&1b)	5 x 2.7	arc	7.629	30.52

Blast design for the Pilot heading (1)

The basic blast design for the pilot heading (1) is given in Figure 7a & 7b. The hole depth was kept at 3.2 m, which was calculated using Persson & Holmberg[4] equation. Considering an advance of 95% of the drilled hole depth, the estimated pull was 3.0 m.

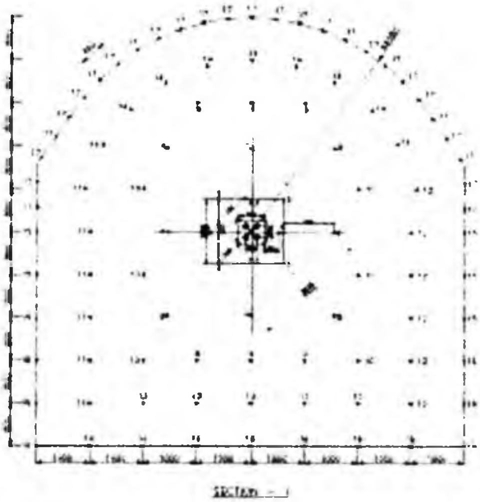


Fig. 7a: Blast pattern for Pilot heading

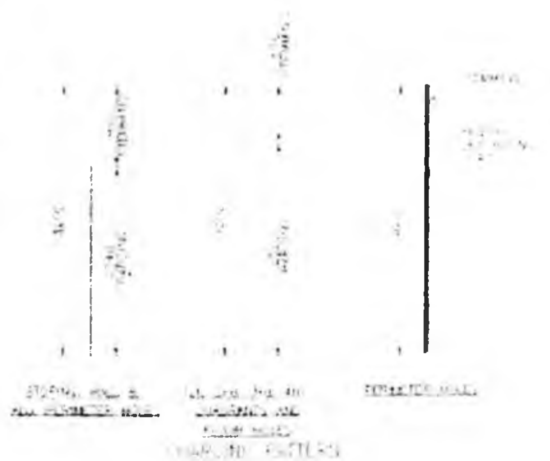


Fig. 7b: Charging pattern for Pilot heading

An alternative blast design using power bulk drive explosive system was suggested and used for expediting the progress and achieving the smooth profiles. The use of power bulk system, first of its kind in the country in underground blasting has improved the overall blast performance in terms of good and uniform fragmentation, desired pull and easy handling of explosives thus reducing the charging time. Moreover, it was easy to achieve the desired charge in the perimeter

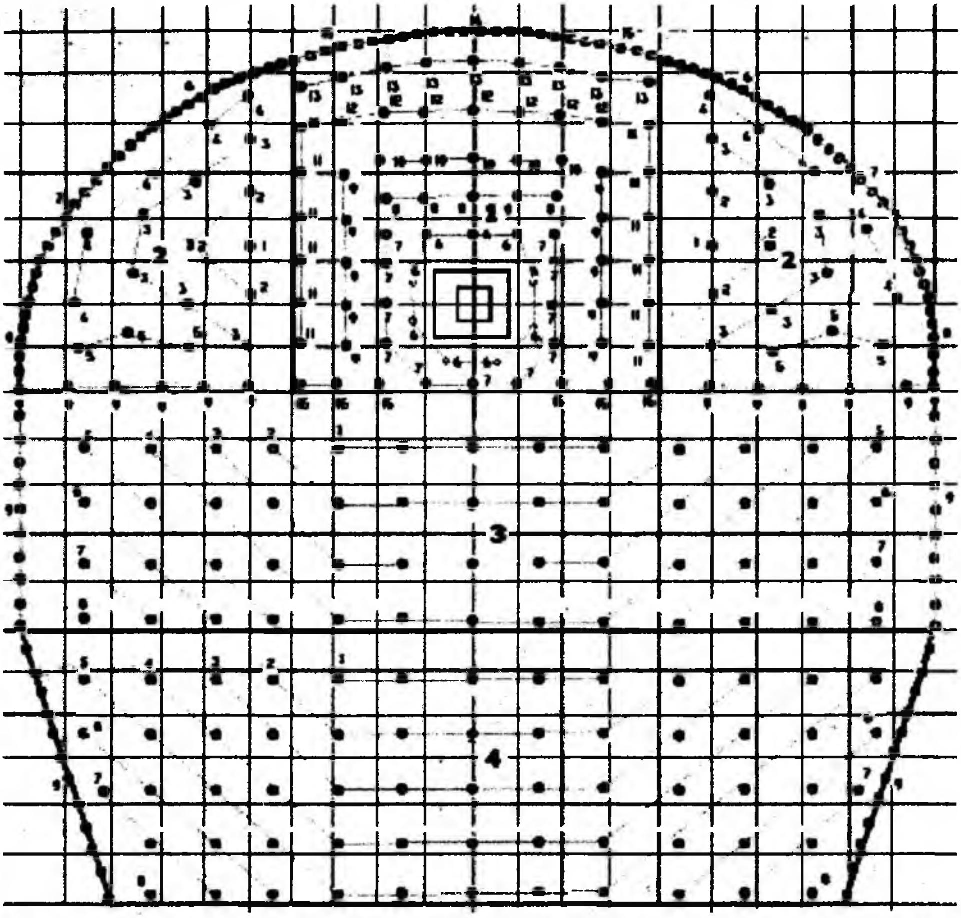


Fig. 8: Blast pattern for the main cavern excavation using power bulk drive system

holes by controlling the density of charges for achieving the smooth profiles and reducing damage to the surrounding rock mass. The blast pattern using power bulk drive is given in Figure 8.

The Parallel Hole cut pattern of drilling was used in the blast design. All the holes were drilled parallel to each other. The relief-hole diameter of 102mm considered and five number of relief holes were drilled at the centre with a hole spacing of 200mm centre to centre. The blast-hole diameter was 45mm. The burden for the first quadrant was 1.5 times diameter of the relief-holes which is equal to 0.22m. The burden for the second quadrant was 0.7 times the rectangular area to be blasted in first quadrant, which is equal to 0.3m. Similarly the burden for 3rd quadrant was taken as 0.7 times the rectangular area to be blasted in the 2nd quadrant which is equal to 0.5 m and for the final quadrant it was again taken as 0.7 times the rectangular area to be blasted in the 3rd quadrant. The burden for the stopping holes were kept at 800mm with a spacing of 1000mm with the floor holes drilled at 1000mm spacing. The burden from perimeter holes was taken as 1000mm for the sides and 800mm in the roof. Smooth blasting was considered for the roof section. The spacing between these holes in the round was 800mm. The spacing of perimeter holes was restricted to 10 times the hole diameter (= 10 x 45 mm = 450 mm). A charge concentration of 0.72 kg/m was considered for calculations. The consumption of explosives in various hole types is given in

Table 5. The stemming length was considered to be 10d, i.e. 450 mm. However the stemming length was varied slightly based on experience in similar type of blasting.

Table 5: Details of charge

Hole Name	Number of holes	Charge per hole (kg)	Total Charge (kg)
1st Quadrant	4	3.51	14.04
2nd Quadrant	4	3.51	14.04
3rd Quadrant	4	3.51	14.04
4th Quadrant	4	3.51	14.04
Stoping holes	21	3.12	65.52
Wall holes	10	3.12	31.20
Floor Holes	9	3.51	31.68
Adjacent to perimeter holes	21	3.12	65.52
Roof holes for smooth blasting	27	0.68	18.36
Total	**103**		**268.44**

For perimeter roof holes, 420 grams of bottom charge was used along with a linear charge of 80g/m of detonating cord. This was required to obtain a charge concentration of around 676g in the entire hole for achieving smooth walls.

Total number of perimeter roof holes = 27,
Total number of production holes = 76

Charge factor = 1.54 kg/m^3 Hole depth = 3.2 m, Expected pull = 3.0 m

Maximum charge per delay = 31.6 Kg

Estimation of "Blast Induced Damage Zone" from the present blast design

The following paragraphs enumerate the effectiveness of the proposed drilling & blasting design in terms of damage control to surrounding rock mass and compliance with the method statement for smooth blasting. Persson et al. has provided the following values (Table 6) of peak particle velocity and their associated damage for granite and gneissic type of rocks.

It has been proved that for a rock mass the incipient fracture will take place at vibrations with particle velocity in the range of 700-1000 mm/s. Charging of penultimate (or adjacent) rows should not cause the crack spreading further into the remaining rock than from the smooth blasted row or perimeter row. Considering these points, the blast design

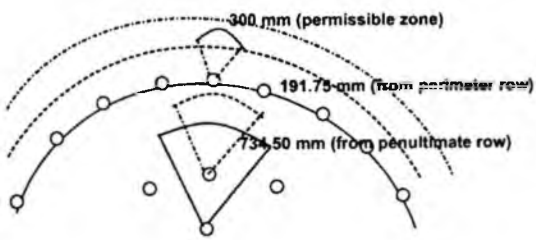


Fig. 9: Damage zone and the permissible zone for section-1

for controlled blasting of this particular case was prepared. The following paragraphs are intended to substantiate the above. For calculation of crack spreading zone, Persson et al. has provided a graph for 'estimated velocity as a function of distance for different

Table 6: Damage and fragmentation effects in hard Scandinavian bedrock resulting from vibrations with different values of the peak vibration particle velocity.

Peak particle velocity (PPV), (m/s)	Tensile stress, (MPa)	Typical effect in hard Scandinavian bedrock
0.7	7.0	Incipient swelling
1.0	10	Incipient damage
2.5	25	Fragmentation
5.0	50	Good Fragmentation
15	150	Crushing

linear charge densities'. Based on this, an estimation of blast induced zone was made for the present blast design (Fig. 9).

Accordingly, the linear charge for adjacent to perimeter holes is 0.975 kg/m, for this value the cracked zone is 734.5 mm, this value is less than the effective burden (800 mm) for final row. So, the damage zone of penultimate row is restricted within perimeter row. For perimeter row the linear charge is 0.21 kg/m, and the damage zone is 191.75 mm only which is less than the allowable zone of 300 mm (shown Fig. 9). The details of blast induced damage zone are given in Table 7.

Table 7: Blast induced damage zone

Location of holes	Linear charge kg/m	Damage zone (up to 1000 mm/s) mm
Perimeter holes	0.210	191.75
Adjacent to perimeter holes	0.975	734.50

Estimation of Blast Induced Pressure[5] from the present blast design: The blasting pressure released from the perimeter row can be estimated by using equation (2)

Pressure developed (s) = v/Cp x E Pa (2)

Where v = peak particle velocity (m/s)

Cp= primary wave velocity (m/s)

E = Elastic Modulus (GPa)

The values considered for the rock type (Garnet gneiss, massive, fresh, with dominant Fe Mg minerals) for depth between 160 m and 190 m from borehole data are: average elastic modulus 86.76 GPa, and the average propagation wave velocity 6670 m/s. Substituting these values along with critical peak particle velocity (1000 mm/s) in Eq. (2), the developed pressure is estimated at 13.01 MPa. The unconfined compressive strength of the rock mass at cavern depth is 109.4 MPa to 147.72 MPa. Hence, the pressure developed due to blasting is well within limiting values of damage.

Blast design details for side slashing (1a)

The drill hole diameter is 45mm drilled to a depth of 4 m. All the holes are drilled horizontal. The burden and spacing for the production holes is 0.9m x 1.2m. Accordingly, there are 2 Nos. of production holes. However, the number of holes varied

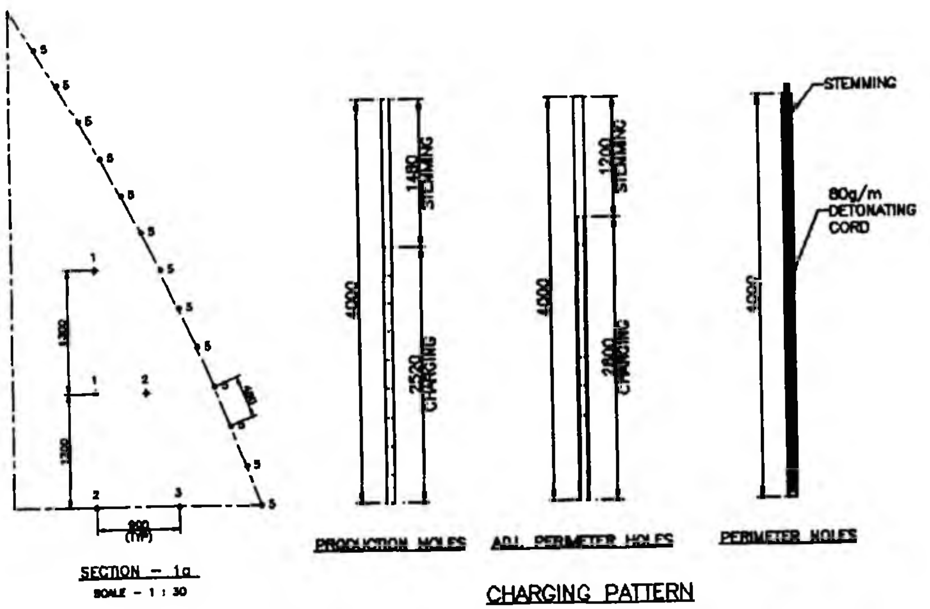


Fig. 10: Blast design for side slashing

depending on the face profile that was modified based on trials. The burden from perimeter holes was taken as 0.9m. The smooth blasting was considered for this section. The spacing between these holes in the round was 1.3 m. This spacing was modified after trials. The spacing of perimeter holes is restricted to 10 times the hole diameter (= 10 x 45 mm = 450 mm). A charge concentration of 1.0 kg/m was considered for calculations. The consumption of explosives in various hole types is given in Table 6. The stemming length[6] is considered to be minimum 10d, i.e. 450 mm. However the stemming length was varied slightly based on experience in similar type of blasting. In the perimeter holes 80g/m Detonating Cord was used as a linear charge along with 2 cartridges of 25mm diameter Powergel-801 emulsion explosives as bottom charge. The blast design for side slashing is given in Fig. 10.

Table 8: Details of charge for side slashing

Hole Name	Number of holes	Charge per hole (Kg)	Total Charge (Kg)
Production Holes	2	3.520	7.04
Adjacent to perimeter holes	3	3.900	11.70
Perimeter holes	14	0.615	8.61
Total	26		27.35

Charge factor = 0.9 kg/m^3 Hole depth = 4.0m, Expected pull = 4.0m, Maximum charge per delay = 11.70 kg, Theoretical volume = 30.52m^3

As per the design, the linear charge for adjacent to perimeter holes is 0.975 kg/m, for this value the cracked zone is 663.20 mm, this value is less than the effective burden (900mm) for final row. So, the damage zone of penultimate row is restricted within perimeter row. For perimeter row the linear charge is 0.15 kg/m, and the damage zone is 141.80mm only which is less than the allowable zone of 300mm. The details of blast induced damage zone are given in Table 9.

Table 9: Blast induced damage zone for side slashing (1a)

Location of holes	linear charge kg/m	Damage zone (up to 1000mm/s) mm
Perimeter holes	0.15	141.80
Adjacent to perimeter holes	0.975	663.20

After excavating the crown portion, the bottom sections were excavated by benching method. The benching cycle comprised two benches i.e., intermediate and bottom. The benches were of 8m high having a width of 8m and length of 6m. The benches were named as 2 and 3 respectively. The advance per blast was 6m. In the benching method, centre portion of the main gallery having a width of 8m was advanced to a minimum of 20 m length. The side portions were excavated by maintaining the same lag or more depending on the operations convenience. For benching, the sequence of operation is 2-2a and 3-3a.

Blast Volume: The blast volume for various cross sectional area and shapes as per the sequence of excavation is given in Table 10.

Blast design details for Benching (2 & 3)

The blast design for both the benching operations was same (Fig. 11). But they are named as sequence 2 and 3 for eliminating confusion during excavation. The drill hole diameter is 45 mm vertically drilled to 8.4 m depth. The drilling of vertical holes was recommended for the following reasons:

Table 10: Blast volumes for different stages of excavation

Name of the excavation	Height x width (m)	Shape	Cross sectional area (m^2)	Volume (m^3)
Benches (2 & 3)	8 x 8	rectangular	64.0	384.00
Intermediate and bottom Sides (2a & 3a)	5 x 3.9	arc	28.383	113.20

- Predictably good fragmentation, More advance per round, Reasonably good side profiles, which otherwise would pose problems while dealing with the side portions, Requires relatively less charge factor as compared to horizontal holes, Possible to provide a good muckpile and desired throw thus making mucking operation easier and faster and Greater control over the blast vibration.

An advance of 6m was considered. The burden and spacing for the production holes is 1.5m x 1.6m. Accordingly, there were 4 rows of production holes. A charge concentration of 1.0 kg/m was considered for calculations. The consumption of explosives in 1st, 2nd, 3rd and 4th rows varied and is given in Table 11. The stemming length is considered to be minimum 10 d, i.e. 450 mm.

Table 11: Details of charge for vertical benching (2 & 3)

Hole Name	Number of holes	Charge per hole (kg)	Total Charge (kg)
1st Row	6	5.8	34.8
2nd and 3rd Row	6 in each row	6.2	74.4
4th Row	6	6.5	39.0
Total	**24**		**148.20**

Charge factor = 0.4 kg/m^3 Hole depth = 8.40m, Expected pull = 8.0m, Maximum charge per delay = 23.8 kg, Theoretical volume = 384 m^3

Blast design details for intermediate side slashing (2a)

The drill hole diameter is 45 mm to a drilled depth of 4m. All the holes were drilled horizontal. The burden and spacing for the production holes is 1.5m x 1.6m. Accordingly, there were 5 Nos. of production holes. However the burden for each hole decreased upwards depending on the face profile obtained from benching. This was modified based on the trials. The burden from perimeter holes is taken as 1.0 m. Smooth blasting is considered for this section. The spacing between these holes in the round is 1.6m. The position of the holes was slightly adjusted to suit the geometry of the section to be blasted. The burden and spacing was modified after trials. The spacing of perimeter holes is restricted to 10 times the hole diameter (= 10 x 45mm=450mm). A charge concentration of 0.4 kg/m is considered for calculations. The consumption of explosives in various hole types is given in Table 12.

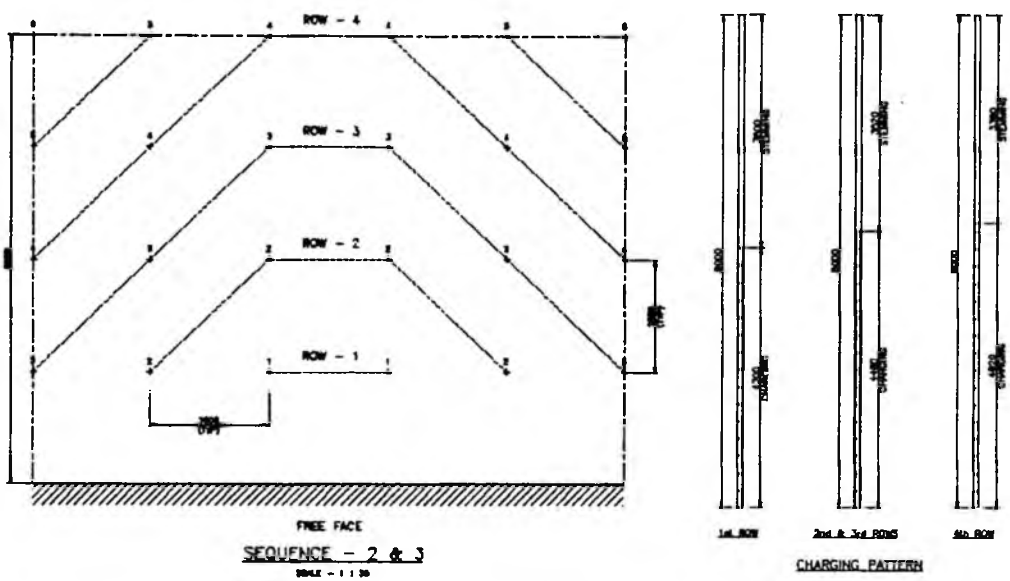


Fig. 11: Blast design for the benches, sequence 2 and 3

Table 12: Details of charge for intermediate side slashing (2a)

Hole Name	Number of holes	Charge per hole (kg)	Total Charge (kg)
Production Holes	5	3.52	17.600
Adjacent to perimeter holes	5	3.90	19.500
Perimeter holes	19	0.615	11.685
Total	**29**		**48.785**

The stemming length is considered to be minimum 10d, i.e. 450 mm. In the perimeter holes 80g/m detonating cord is used as a linear charge along with 2 cartridges of 25mm diameter Powergel-801 emulsion explosives as bottom charge.

Charge factor = 0.40 kg/m^3 Hole depth = 4.0m, Expected pull = 4.0m, Maximum charge per delay = 11.7 kg, Theoretical volume = 113.2m^3

Blast design details for side slashing (3a)

The blast design for side slashing is given in Fig. 12. The drill hole diameter is 45 mm drilled horizontally to a depth of 4 m. The burden and spacing for the production holes is 1.5m x 1.6m. Accordingly, there were 5 Nos. of production holes. However the burden for each hole decreased downward depending on the face profile. The burden from perimeter holes is taken as 1.5m. This burden also decreased downward according to face profile. Smooth blasting was considered. The spacing between these holes in the round is 1.6m. The position of holes was slightly adjusted to suit the geometry of section to be blasted. The spacing of perimeter holes is restricted to 10 times the hole diameter (=450mm). A charge concentration of 0.4 kg/m is considered for calculations. The consumption of explosives in various hole types is given in Table 4. The stemming length is considered to be minimum 10d, i.e. 450 mm. In the perimeter holes 80g/m detonating cord is used as a linear charge along with 2 cartridges of 25mm diameter Powergel-801 emulsion explosives as bottom charge.

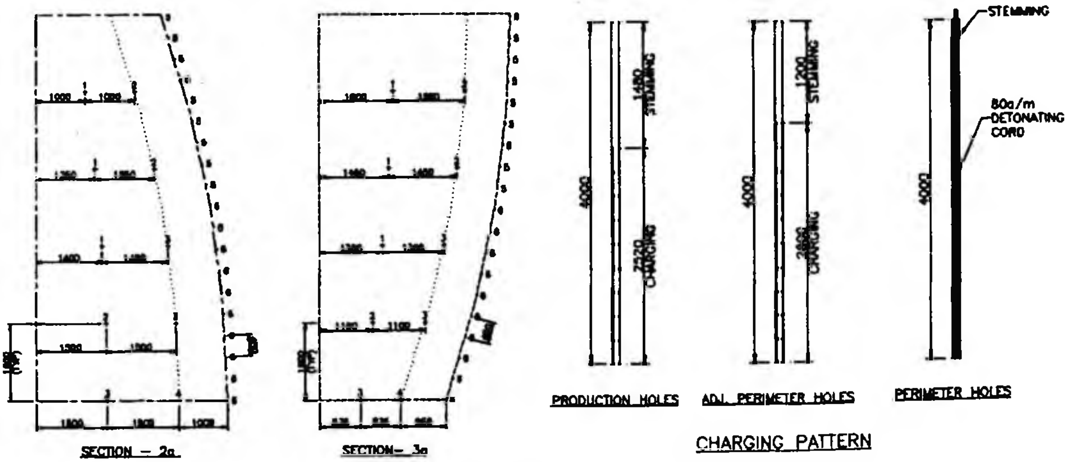


Fig. 12: Blast design for side slashing, sequence (3a).

Table 13: Details of charge for side slashing (3a)

Hole Name	Number of holes	Charge per hole (Kg)	Total Charge (Kg)
Production Holes	5	3.52	17.600
Adjacent to perimeter holes	5	3.90	19.500
Perimeter holes	19	0.615	11.685
Total	**29**		**48.785**

Charge factor = 0.40 kg/m^3 Hole depth = 4.0m, Expected pull = 4.0m, Maximum charge per delay = 11.7 kg. Theoretical volume = 113.2m^3

Conclusions

1. The underground excavation storage facility for LPG was first of its nature in the country and the excavations were completed successfully. The excavations were made mainly in granite gneiss and the rock formation was categorized as Fair to Good using Q and RMR classifications. The joints were used favorably during blasting whenever found feasible.

2. Blasts were designed for jack hammer drill employing 32mm diameter holes as well as for two boom jumbo drills for larger hole diameters ranging from 40mm to 105mm. The designs were made to suit site specific requirements for various cross sections, shapes, advances, smooth blasting and controlled blasting applications. Various types of cuts were designed. They include Wedge cut pattern, Burn cut patterns, Parallel hole cut pattern, vertical benching, horizontal benching, slashing, niche making, etc.

3. The power bulk drive system was used in underground blasts for the first time in the country. The system has provided excellent results in terms of smooth profile, uniform and desired fragmentation and more than 90% advance with less fumes. The application has reduced the cycle times and made explosive loading very easy. Moreover, it was possible to provide various charge densities in the holes that helped to produce smooth profile and good blasting control.

4. The advances per blast were designed for pulls ranging from 1 m to 4.3 m in the headings. For shafts, the advance per round varied from 0.75 m to 2.5 m. More than 2.0 m pull was achieved in both Access and Operation Shafts with manual excavation in difficult conditions. This is again a record achievement.

5. Smooth blasting was carried out restricting the overbreak within permissible damage zone and hole impressions were achieved on an average to 70% and more. Detonating cord of 80 g/m was suggested for perimeter holes to restrict the blast induced damage within 300mm and to get about 75% half barrels. The same was used for a few blasts and observed above 90% half barrels.

6. Blast vibrations were restricted well within permissible limits through controlled blast designs and strict implementation. Predictor equations were derived using 300 data sets generated at the site. During first phase of the study, 127 mm/s resultant peak particle velocity for any frequency was suggested as the safe limit of ground vibration for the safety of LPG Pipeline. These blast induced vibrations were restricted well within limits with proper blasting designs in execution stage. The maximum peak particle velocity recorded at LPG pipe line was 10.7 mm/s. The blast induced vibrations were less than 0.5 mm/s (instrument not triggered) at remaining four locations, viz. Crude Oil Pipeline, Water Tank, Booster Pump and HPCL control room.

7. Fragmentation was controlled to the desired size of below 300 mm. Overall the larger size fragments were restricted to about 5 to 7%. Blast induced damage zones for all types of blast designs were computed and restricted the damage zone through smooth blasting techniques.

8. Blasts were designed for coupled and decoupled charges. In both the cases the desired results were achieved through prudent blast designs.

Acknowledgements

We are very much thankful to Mr. D Laxminarayan, Chief Executive, L&T-Ramboll Consulting Engineers Limited for awarding this work to us. We express our gratitude and special thanks to Mr A. Madan Kumar and Mr. Dominic Xavier for their keen interest and co-operation during field investigations. We are also thankful to Mr. Manoj Kumar Garnayak, Project Manager, and Mr. K. Vijaya Bhaskar Reddy for their kind cooperation and providing all necessary arrangements at the site. We are also thankful to Mr. Pierro Roux UGM Geostock, Mr. Anoop Kumar Sinha and Mr. S. Anand SALPG for their co-operation and kind attention during technical discussions.

We are thankful to Mr. H. R. Ramesh, Excavation Manager, Mr. Elongovan, Excavation Manager and their team members for their active co-operation in this study. We extend our thanks to Mr. Balakumar, Draftman for his cooperation in preparation of drawings and arrangements at site.

We are extremely thankful to Dr. P. C. Nawani, Director, NIRM for permission to publish this paper. We also thank those who directly or indirectly helped during the studies and in preparation of this paper.

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