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# Improvised drilling and blasting techniques at underground metal mine for faster advance to enhance linear excavation and production – a techno-economic case study

*The paper deals with innovative drilling and blasting techniques that are used for mineral extraction at the deposit located at Kadapa district, whose mineralization belongs to Vempalle formations of Papaghni group of Cuddapah Super Group and occurs in Vempalle carbonate rock, which forms the host rock to the mineralization, is essentially strata-bound type. The host rock is characterized by impure, siliceous, phosphatic, dolomitic limestone with stromatolites, ripple marks and mud cracks thus also named as dolostone. This dolostone is sandwiched between lower massive limestone and upper shale. In relation to drilling and blasting works are described the main technological parameters of existing mining method; room and pillar mining with backfilling. For purpose of higher effectiveness of drilling and blasting, the works are executed by the emulsion explosives and the jack hammer 1.8m -2.4m drilling length), low profile (4.2m drilling length) and extra low profile (3.2m drilling length) electro-hydraulic drill jumbo by modern drilling pattern (modified burn cut) with single deck charging, double deck charging and without deck charging). The mentioned changes are analyzed through techno-economic analysis. The repeated calculation of drilling and blasting parameters for each individual type of faces (declines, advance strike drive, stope drive, ramp) made it possible for the blast design engineers, to provide various corrections in a new blasting plan for faster advance, resulting in higher productivity, reduced production drilling cost and reduced overall blasting cost per blast.*

## 1.0 Introduction

The excavations of ore drives, drifts are common features in any metal mining. The room and pillar mining method, needs a huge quantity of drivages (4.5m × 3.0m) in orebody to make available number of stope blocks for production. Declines, ramps, loading bays, cross cuts, drift for excavation of sumps are the additional requirements for permanent mine construction. The vital component in driving drivage is unlike any other mining method, the absence of initial free faces. Therefore, solid blasting is carried out, for which, blast design is important factor, in order to create free faces for successive rows and column of holes and ultimately a big advances of faces/large pull.

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The important factors on which generally, the progress of drives, ramps, declines, cross cuts etc. depend are as follows:

- Geology of strata and rock mass condition.
- Appropriate blast design including drilling pattern, quality and type of explosive, initiation, its sequence.
- Types of drilling equipment used and length of drilling rod used.
- Dimension of drives.
- Properties and VOD of explosives used.

The cost and time benefit analyses of the excavation are mostly decided by the rate of advance/pull. Therefore, it is utmost important to have proper blast design with optimum quality of explosives used, in order to achieve maximum rate of advance/pull per blast. The “burn cut (parallel holes with reamers) in the blast design is suitable for any large size category of drift/drive excavation and with proper explosives, initiation sequences etc. It can give considerable amount of pull.

## 2.0 Details of the mine

### 2.1 Geology

The mine is located in Kadapa district, Andhra Pradesh, to mine the radioactive ore mineral (pitchblend). Geologically the area is in the SW part of the Cuddapah basin. The mineralization lies within Vempalle formations of Papaghni group of Cuddapah Super Group. The deposit is located in the Survey of India Topo-sheet No. 57 J/3 and 7, between latitudes 14°18’36.6"N and 14°20’20"N, and longitudes 78°15’16.57"E and 78°18”3.33"E. It is situated in the Kadapa district of Andhra Pradesh. The mineralization occurs within Vempalle carbonate rock, which is a strata bound formation. It is basically a dolomitic limestone (dolostone), along with stromatolitic limestone.

The orebody is uniform in its thickness and trend, with an average dip of 15° due N22°E. The extent of the orebody is 5.6 km along the strike and 1 km along dip, with overburden depth ranging from 15 m to 275 m. The orebody consists of two bands: hangwall lode (2.30m in width) and footwall lode (1.70 m in width), separated by 1.50 to 3.0 meter thick lean zone 3.

### 2.2 The status of mining

The country rock in the area is bedded dolomitic limestone. The orebody consists of two bands: 2.30 meter thick hangwall lode and 1.70 meter thick footwall lode, with a 1.50 to 3.0 meter thick lean zone in between. The dolomitic limestone is overlain by a thin clay band and thick red shale, which form the immediate roof strata above the orebody in the mine workings. The weathering zone (weathering grade W2) extends for 40 to 50 m below the surface. Three declines, roughly 5 m in width and 3 m in height, along an apparent dip of 9º due NE, are being driven 15 meter apart to work the two lodes at different levels. The East decline is driven to work the footwall orebody, while the Central and Western declines are originally meant for developing the hangwall orebody. From the East and Central declines, level galleries called Advance Strike Drives (ASDs), are driven at 40 m interval (vertical is 10 m). The strike drives are 4.5 m in width and 3 m in height. The mine development would also include loading points measuring 7 m width, 10 m length and 5.5 m height, and ramps.

The said underground metal mines working for the excavation of hard dolomite based rock of compressive strength of range 300-350 MPa and only development work has been done. In underground, the development headings provide mine access for men and materials, ore and waste transportation and ventilation paths. The development needs to be done at faster pace to prepare the mine, start stoping operation/production, for which various types of burn cut had been tried. Following are the major excavation for development work being done at said underground mines.

- Declines with dimension -5m × 3m
- Ore drives/drift for all the levels (dimension – 4.5m × 3m)
- For connecting the ore drives ramps are driven (4.5m × 3m) at an interval of 100-150 m.

### 2.3 Drilling equipment

As far as drilling equipment for drive is concerned, jack hammer and single boomed electro-hydraulic jumbo drills (low profile and extra low profile) are used for linear excavation. The parallel and reamers holes can be drilled very easily and rate of penetration is also quite faster in case of low profile and extra low profile and single-boomed electro-hydraulic jumbo drills. The best pull or rate of advance is found in extra low profile equipment. The single boom electro hydraulic drilling machine designed to work in excavations with headroom as low as 1.70 meter. The robust universal boom has large optimum shaped coverage, 360º rotation and full automatic parallelism for fast and easy face drilling. The exceptional ‘V’ shaped layout is designed for good visibility and balance, equipped with powerful four wheel drive articulated carrier ensure fast and safe maneuvering even in low head room conditions. The technical specifications of one of the models i.e. Sandvik make DD210L drill jumbo are as follows.

- Dimensions of the machine (L × W × H) : 12260mm × 2250mm × 1950mm
- Rock drill (HL×5) power: 20kW
- Power pack: Hydraulic pump with electrical motor of capacity 75 hp.
- Feed : TF500 with feed force 25 kN (cylinder – wire rope type)
- Boom: B 26 XLF with parallel holding and 360º feed roll over.
- Engine: 74kW, deutz BF4M2012
- Stabilizers: 04 nos. (02 hydraulic front jacks and 02 hydraulic rear jacks)
- Total installed power: 70kW
- Operating voltage: 360 volts to 660 volts (±10%)

## 3.0 Result and discussions (blasting variation in phases and its technical results)

### 3.1 The original blast design pattern

The original blast design and drilling and blasting for excavation of various development activities at the said underground mines is as below:

For faces blasting, 40mm dia. cartridges emulsion explosives (each cartridges length 300mm, contain 390gm explosives) with VOD of about 4000 m/s are used for 45 mm dia. holes. Drilling length kept mostly 3.2 m or 4.2 m (or sometimes 4.0m). Reamers of dia. 89 mm/110m are also used. Long delay detonators are used for initiation. The pull ranges from 2.5m-2.75m. The details of the original pattern is shown in Fig.1 and Table 1.

### 3.2 The decked burn technique blast design: phase-I

Face blasting for declines, ramps, ore drives and cross cuts “decked-burn” technique was used, using LDD (long delay detonator). The salient features of the decked burn system are as follows:

- The collar portion of hole is blasted prior to bottom.
- Mid-column decking between the two charges in a hole is kept about 0.5m

TABLE 1: BLAST DETAILS OF THE FACE IN ORIGINAL PATTERN

| Face (with dimension) | No. of holes drilled | No. of holes charge | Explosive used in a round (kg) | Detonators used (no.) | Stemming length kept (m) |
|---|---|---|---|---|---|
| 4.5m × 3m face (with 3.4m drilling length) | (1) 46 holes with 45m dia.<br>(2) 4 holes with reamer 89mm dia. (46+4) | 42 | 120.0 | 50 (in burn holes two detonators were used) | 1.3 to 1.6 m |

TABLE 2: BLAST DETAILS OF THE FACE IN PHASE-I

| Face size | No. of holes drilled | No. of holes charge | Explosive used in a round (kg) | Detonators used (no.) | Stemming length kept (m) | Reduction in drilling (no. of holes) w.r.t. original pattern | Reduction in explosives and detonators quantity w.r.t. original pattern | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | Exp. | Det. |
| 4.5m × 3m face (with 3.4m drilling length) | (1) 41 holes with 45m dia.<br>(2) 4 holes with reamer 89mm dia. (41+4) | 37 | 118.5 | 45 (in burn holes two detonators were used) | 0.8 to 1m | Decked-burn, reduced no. of holes by 5 (45mm) | 1.5 kg | 5 no. |

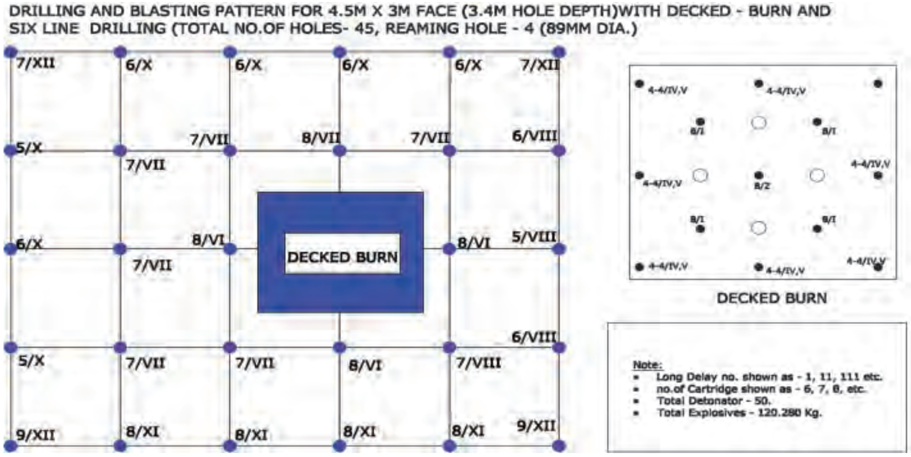


Fig.1 Original pattern design with explosives and detonator details

Original pattern (4.5m × 3m) face size drilling hole length – 3.4m.

*3.2.1 Discussion and review on decked burn pattern*

(1) As number of holes are more, smaller quantity of explosives used per hole (6 to 8 cartridges per hole), resulting in large stemming length (1.2 to 1.6m), causing a peculiar kind of under blast failure, in which only inside is blasted and fragmented, whereas outside (collar) rock appears solid and intact (Fig.2).

(2) Difficult to handle the blasted rock with LHD, especially when there is under blast.

(3) Difficult to deal with post blast sockets; generated sockets are hollow at the end.

Under blast reasons and its handling:

- Stemming length is more, due to which, thick rock colar of hole is not easy to break.
- Rock breaks in hole till where the explosives are filled in hole, forming the cavity.
- Mucking by LHD in such case is difficult.
- Handling post blast socket is very difficult, as the sockets holes are very less in diameter, thereby causing difficulty in charging and re-blasting.
- The above reasons under blast can be solved, if explosive length is increased and stemming length is reduced to 0.6 to 0.8 meter.

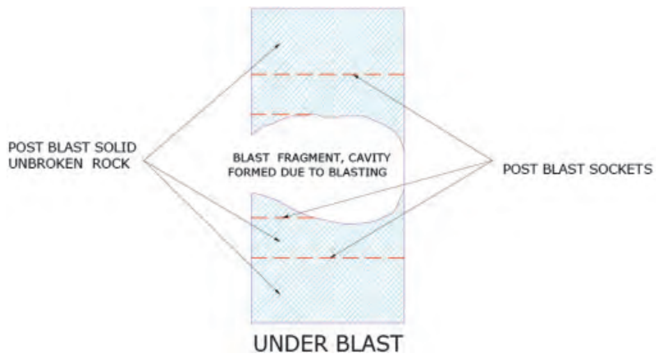


Fig.2 The section of under blast failure

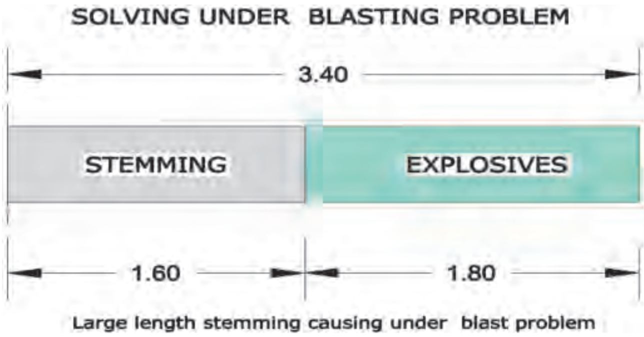


Fig.3.Section of large length stemming causing under blast problem

*3.2.2 Modification in decked burn pattern*

(1) It is thought to modify the decked burn pattern by reducing number of holes, in order to put more quantity of explosives (7 to 10 cartridges) in individual holes to reduce stemming length (0.6 to 0.8m), to overcome above type of under blast failures.

(2) Also simplification of explosive charging procedure is thought of by doing away with the decked charging procedure. In the first phase, one vertical line is reduced in order to reduce the number of holes in the pattern and the same "decked burn" kept intact.

(3) By implementation of one reduced line, 5 holes are reduced in the pattern, accommodating more number of cartridges in a hole in order to reduce stemming within 1m.

*3.2.3 Reasons to phase out decked burn technique*

(1) For UG drives charging of explosives is difficult and time consuming.

(2) Interchanging delays between inside and outside column of explosives takes place and because of that blast failure occurs.

(3) Idea of re-designing and to replace with " non-decked" b `urn came up in order to simplify the procedure of charging of explosives and to prevent blast failure due to inadvertent interchange of delay timings between inside and outside explosives column.

3.3 THE NON-DECKED BURN TECHNIQUE BLAST DESIGN: PHASE-II

A new design of rectangular burn (with four reamers) has been put forth, wherein deck charging has been discarded. In second phase another 4 number of holes are reduced and also "deck charging" is removed providing simplification of explosives charging process and further reducing stemming length to about 0.6 to 0.8 m.

*3.3.1 Discussion on trails and performance of blasts*

(1) Extensive trail blast conducted at all the dimensions of drives, i.e. 4.5m × 3m, 5m × 3m and 5m × 4m with 3.4m and 4.0m length of drilling.

(2) No failure as of "under blast" observed.

(3) No re-blasting is carried out.

(4) The performance of blasts (average pull obtained) is either same or better than the earlier pattern.

(5) It is observed that, with 4m length of drilling, at ramp up pull is better than ramp down.

(6) Side and top corner sockets observed when there is deviation of hole. Chances of hole deviation with 4m length are more than with 3.4m length (Fig.4).

3.4 INTRODUCTION OF THIRD PHASE: PHASE-III

*3.4.1 Salient features*

(1) There is reduction of one reamer, keeping quantity of explosives and detonators timing same without affecting quality of blasts and pull. The three reamer burn also called shielded burn cut rotated left side of the face.

(2) Number of trails are taken up with this three reamer system (89mm dia.) at both 4.5m × 3m and 5 × 3m faces.

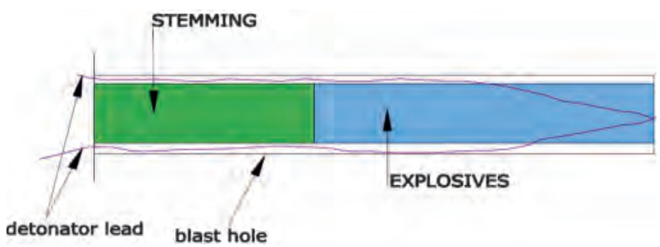


Fig.4. The details of charging pattern of non decked holes

TABLE 3: BLAST DETAILS OF THE FACE IN PHASE-II

| Face size | No. of holes drilled | No. of holes charge | Explosive used in a round (kg) | Detonators used (no.) | Stemming length kept (m) | Reduction in drilling (no. of holes) w.r.t. phase-I | Reduction in explosives and detonators quantity w.r.t. pahse-I | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | Expl. | Det. |
| 4.5m × 3m face (with 3.4m drilling length) | (1) 37 holes with 45m dia. (2) 4 holes with reamer 89mm dia. (37+4) | 33 | 117.5 | 33 (in burn holes two detonators were used) | 0.8 to 1m | Non decked -burn, further reduced in no. of holes by 5 (45mm) | 1.0 kg | 12 no. |

TABLE 4: BLAST DETAILS OF THE FACE IN PHASE-III

| Face size | No. of holes drilled | No. of holes charge | Explosive used in a round (kg) | Detonators used (no.) | Stemming length kept (m) | Reduction in drilling (no. of holes) w.r.t. phase-II | Reduction in explosives and detonators quantity w.r.t. pahse-II | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | Expl. | Det. |
| 4.5m × 3m face (with 3.4m drilling length) | (1) 36 holes with 45m dia. (2) 3 holes with reamer 89mm dia. (36+3) | 33 | 117.5 | 33 (in burn holes two detonators were used) | 0.6 to 0.8m | Non decked -burn, further reduced in no. of holes by 2 (one 45mm and one 89mm dia. reaming hole | - | - |

TABLE 5: COMPARATIVE DETAILS OF VARIOUS PRODUCTION PARAMETERS FOR DIFFERENT TYPE OF PATTERNS

| | Name of the pattern | Drill hole length (meter) | Pull (meter) | Avg. face OMS | Avg. P.F ($kg/m^3$) | Avg. D.F. | Avg. monthly production (tonnes) | Avg. linear excavation /month (meters) | Avg. total drilling length/ month (meters) |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Original pattern | 3.4 | 2.5 | 3.7 | 1.22 | 2.6 | 38669 | 1023 | 60560 |
| 2 | Phase-I | 3.4 | 2.75 | 3.9 | 1.15 | 2.48 | 41730 | 1104 | 58130 |
| 3 | Phase-II | 3.4 | 2.9 | 4.2 | 0.96 | 3.32 | 49215 | 1302 | 53520 |
| 4 | Phase-III | 3.4 | 3 | 4.4 | 0.95 | 3.43 | 50072 | 1325 | 51184 |
| 5 | Phase-III jack hammer 2.4 m | 2.4 | 2.1 | 0.95 | 1.1 4 | 1.57 | 35078 | 928 | 36722 |
| 6 | Phase-III jack hammer 1.8 m | 1.8 | 1.45 | 0.9 | 1.23 | 1.56 | 24230 | 641 | 27917 |

(3) All the trail blasts are successful with similar performance as of four reamers.

(4) Drilling cost and time are reduced further.

*3.4.2 Discussion on techno-economic advantages of new pattern*

(1) With new blast design, blasts failure of under blast and re-blasting are eliminated completely.

(2) As this new blast design does not use decked charge at the burn, the number of detonators used are reduced considerably and also gives ease in charging (charging time is reduced).

(3) As the number of drilled holes is considerably less, the drilling (percussion) time is reduced.

(4) Thus, cycle time of drilling and blasting is reduced considerably, working efficiency enhanced.

(5) This is evident from the fact that, now two full faces are drilled and blasted in one shift by single drill jumbo, thereby enhancing efficiency in faster linear excavation.

(6) Number of detonators are reduced considerably per blast and also quantity of explosives (thus, powder factor, detonator have been improved). Thereafter, the drilling cost and overall blast has reduced drastically and also improved OMS and production achieved.

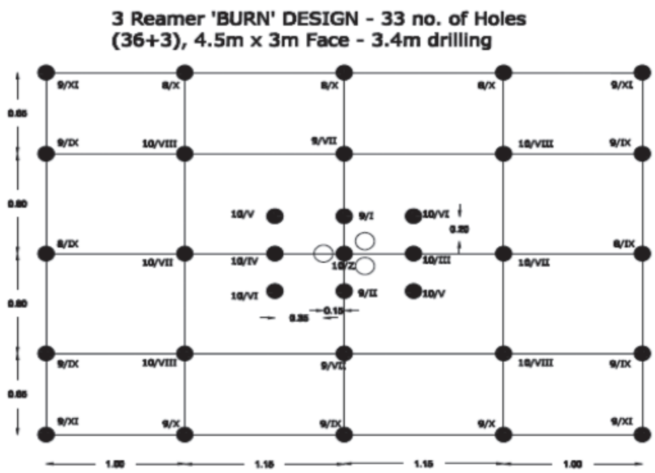


Fig.5.The details of 3 reamer burn design with stemming, explosive and detonators

(7) Now, three reamer holes system is followed instead of four reamer system in both 4.5m × 3m and 5m × 3m faces,

TABLE 6: DETAILS OF DRILLING AND OVERALL COST PER BLAST AND ITS COMPARISON WITH PULL AND DRILL HOLE LENGTH FOR ALL SIX TYPE OF PATTERN

| | Name of the pattern | Avg. drilling cost/blast (Rs. per blast) | Avg. overall cost per blast (Rs. per blast) | Avg. overall cost/m (w.r.t pull) (Rs./mts) | Avg. drilling cost/m (w.r.t drill hole length) (Rs./mts) |
|---|---|---|---|---|---|
| 1 | Original pattern | 7651.6 | 17559.6 | 7023.84 | 2250.47 |
| 2 | Phase-I | 7486 | 17317 | 6297.09 | 2201.76 |
| 3 | Phase-II | 6723 | 16173 | 5576.9 | 1977.35 |
| 4 | Phase-III | 6531 | 16147 | 5382.33 | 1920.88 |
| 5 | Phase-III jack hammer 2.4m | 8841.84 | 16279 | 7751.9 | 3684.1 |
| 6 | Phase-III jack hammer 1.8m | 6720 | 13304 | 9175.17 | 3733.33 |

which further reduced cost and enhanced drilling and blasting efficiency.

## 4.0 Results and discussions of the studies conducted in the mine for various pattern and machineries

Total six varieties of blasts of various pattern are tried in a span of 3 years (2014 to 2016). Each phase is tried and tested for 445 blasts. It is evident from the Table 5 that in comparison with original pattern of drill and blast design, each phase is shown a vital amount of improvement in all productive parameters (pull, face OMS, PF, DF, monthly production, linear excavation) and cost of drilling per blast, overall cost per blast etc. Phase III pattern with electro-hydraulic drill jumbo (3.2m) is found to be ideal in contrast to jack hammer (conventional) of any drill length.

### 4.1 OBSERVATION OF THE STUDIES

The original method and its pattern are compared to third phase pattern as it is considered to be the ideal and optimum design for achievement of optimum productivity. The effect of phase III pattern on various parameters is described herewith: there is a total increase of 16.7% in pull practically, which contributes to 16% growth in avg. face OMS, an increase of 22.8% in avg. monthly production, wherein reduction in drilling length per month achieved is 18.30%, with optimum powder factor of 0.95kg/m$^3$, which is best/optimum in case of development. The reduction in drilling length per month has contributed to 17.20% reduction in avg. drilling cost per blast and 8.7% reduction in avg. overall cost per blast.

The phase III benefits the organization to the range of 30.5% in case of avg. overall cost per meter in contrast to original pattern and 17.20% reduction in drilling cost per meter for same drill hole length of 3.40 m. The jack hammers are much costlier if the avg. drilling cost and overall cost per

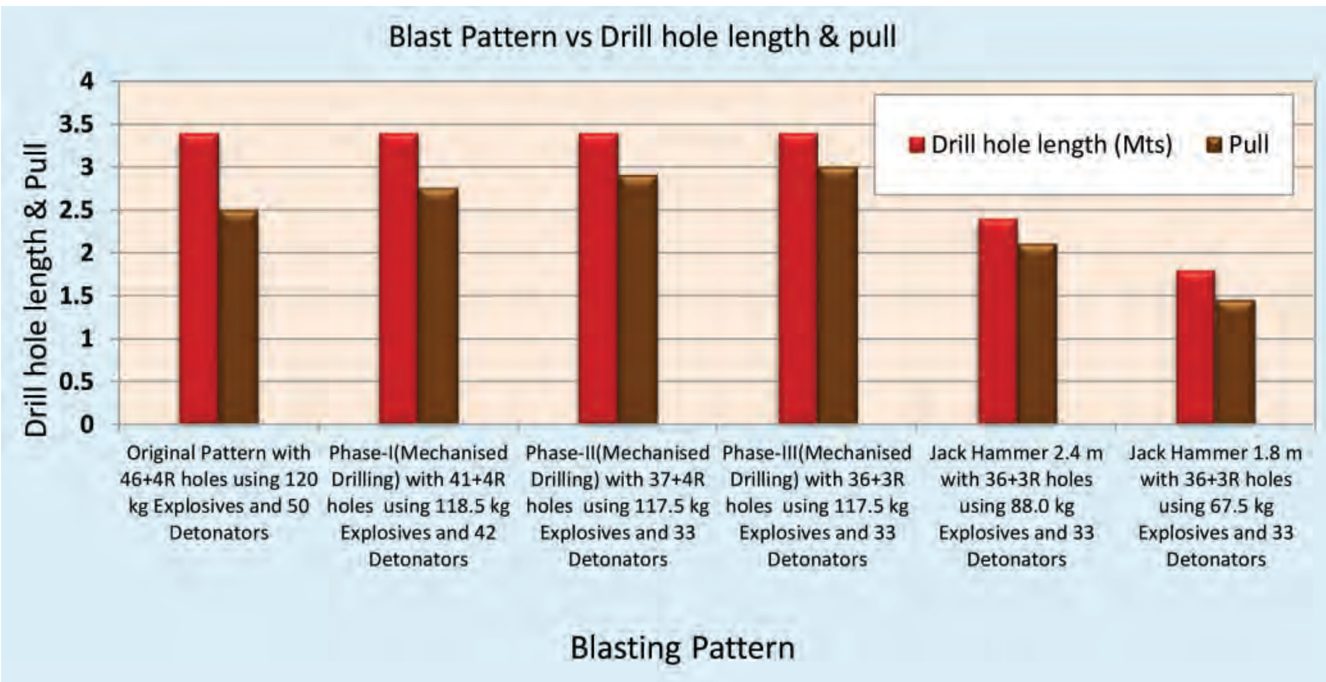


Fig.6 Drill hole length vs pull for all six varieties

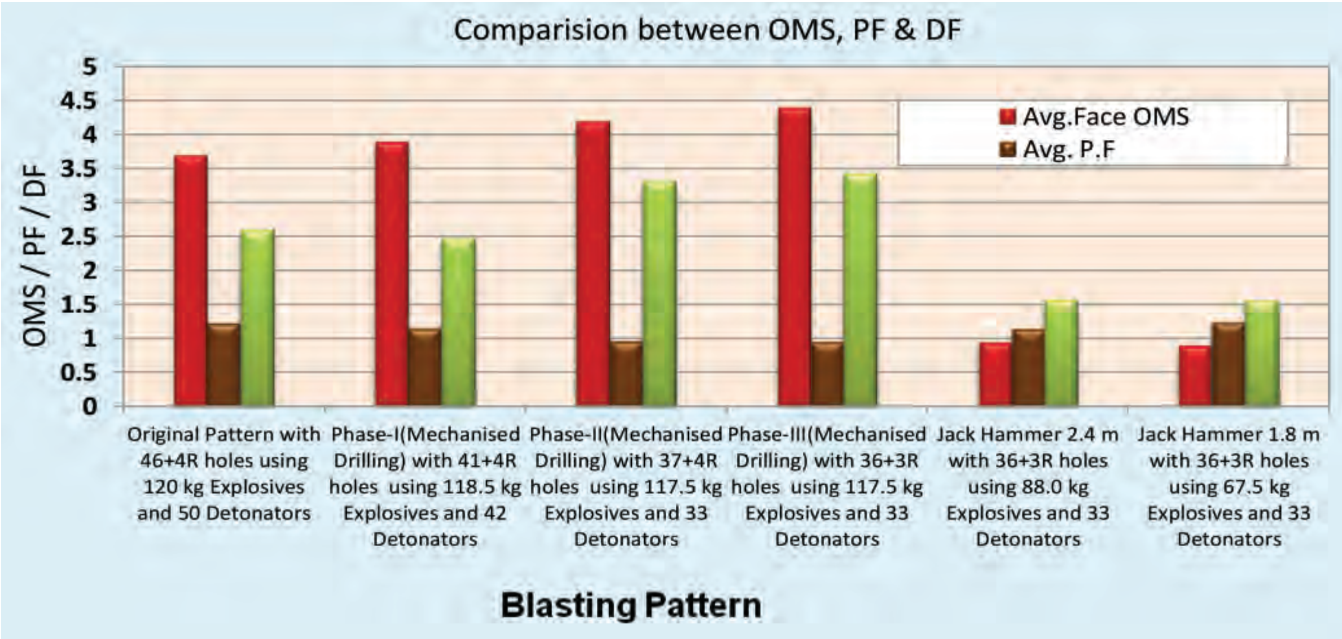


Fig.7 Comparison between OMS, PF and DF

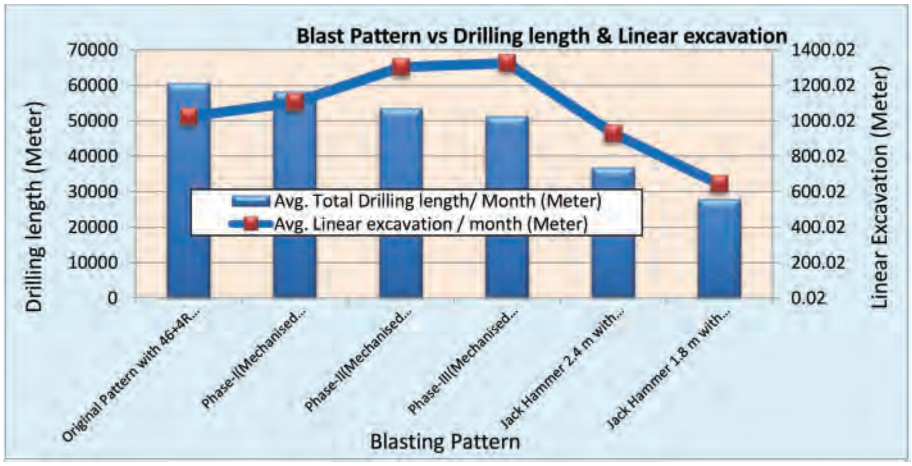


Fig.8.Comparision of total drilling length vs avg. linear excavation per month for all six varieties of pattern

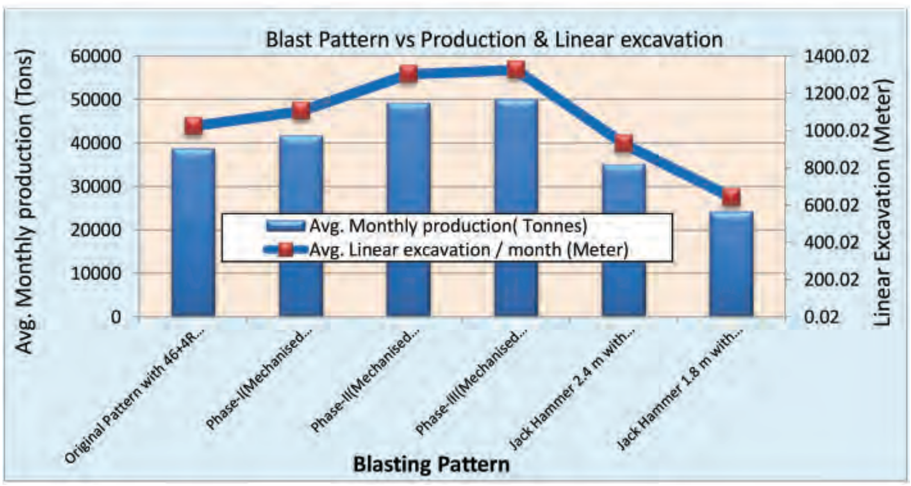


Fig. 9 Comparision of avg. monthly production and avg. linear excavation for all six varieties of pattern

blast is considered. All the above patterns are studied for 450 blasts each. The improvement in all aspects of faster development especially pull, took about three years of study (2014, 2015 and 2016).

### 5.0 Conclusions

As the number of holes has been reduced considerably, the percussion time and drilling meterage are saved. Cost of drill bits and rock tools is also saved and so the reduction in the drilling costs and overall cost per blast. Similar, advantages obtained for 5m × 3m faces with 3.4 m drilling and also with 4m drilling at 4.5m × 3m faces. Upon obtaining satisfactory blast performance, enhancement in productivity parameters and efficiency and eliminating blast failure such as under blast, the new blast pattern are fully incorporated for all the development section of the said underground mine. Similar pattern is tried for jack hammer of 2.4m and 1.8 m drilling length, but the drilling cost is so high, it will not be much productive and feasible performer for the mechanized mine.

The other option apart from drill jumbo and jack hammer, roadheader remains as a third option, whose

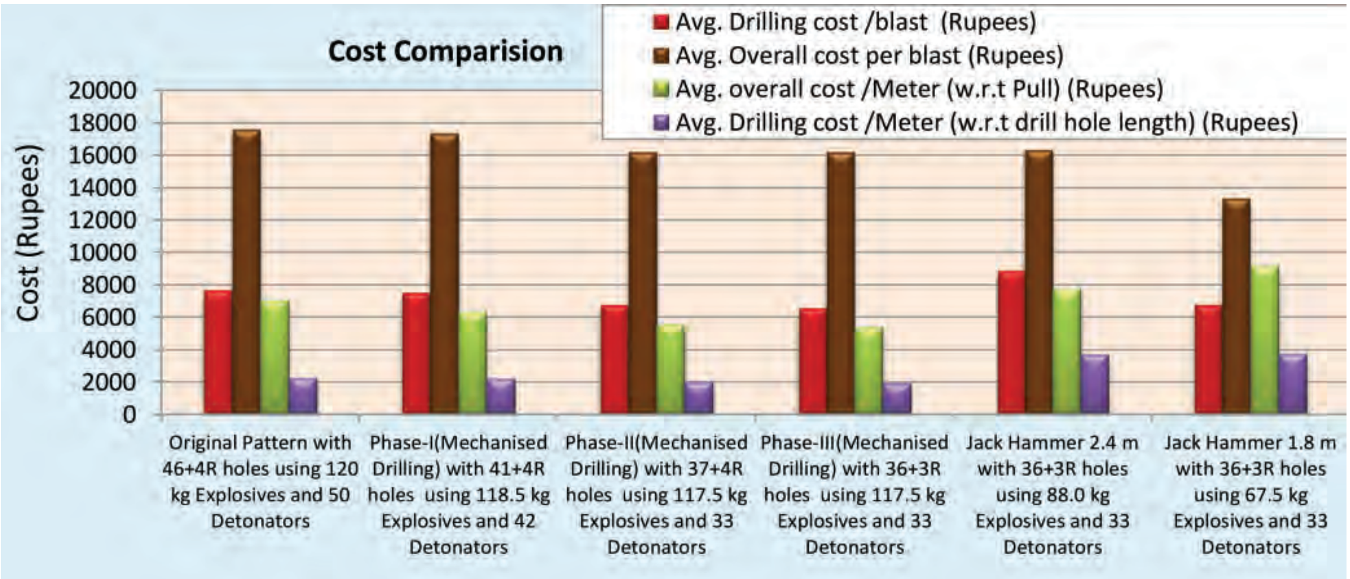


Fig.10 Drilling cost and overall cost per blast overall cost w.r.t pull and avg. drilling cost w.r.t. drill hole length for six patterns

prospects for use in hard rock underground metal mines (compressive strength range of 200Mpa-350 Mpa) is to be explored. The mining Industry and R&D organization, should expedite to develop a roadheader for excavation of the above compressive strength.

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