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Abstract

In the development of hard rock mines, achieving maximum pull after blasting plays a crucial role. Various machines have been developed for rock cutting, but still, due to flexibility and cost-effectiveness, drilling and blasting are preferred. To enhance the effectiveness of this method, several techniques have been developed, including the use of appropriate stemming material, double-primer placement, selecting optimal initiation locations, improving blast designs, and exploring stress superposition techniques through electronic detonators. This research paper focuses on investigating the effect of the priming and explosive initiation location on pull through an experimental approach. The study specifically examines the influence of different initiation approaches on pull, with a particular focus on inverse initiation without solid decking. The findings indicate that inverse initiation without solid decking reveals the best pull for competent rock. Additionally, the inverse initiation with 1st and 2nd square cut solid decking (double detonators with different delays) and spacers in periphery holes was found to be the best choice to eliminate the post-blast sockets with reasonable pull for weathered competent rock.

Keywords

Mine development; Drill and blast; Rate of advance; Priming; Solid decking; Inverse initiation.

Erratum

The page range corrected from 331-342 to 332-343

RESEARCH ARTICLE

Effect of Priming and Explosive Initiation Location on Pull in Hard Rock Underground Mine

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Abstract

In the development of hard rock mines, achieving maximum pull after blasting plays a crucial role. Various machines have been developed for rock cutting, but still, due to flexibility and cost-effectiveness, drilling and blasting are preferred. To enhance the effectiveness of this method, several techniques have been developed, including the use of appropriate stemming material, double-primer placement, selecting optimal initiation locations, improving blast designs, and exploring stress superposition techniques through electronic detonators. This research paper focuses on investigating the effect of the priming and explosive initiation location on pull through an experimental approach. The study specifically examines the influence of different initiation approaches on pull, with a particular focus on inverse initiation without solid decking. The findings indicate that inverse initiation without solid decking reveals the best pull for competent rock. Additionally, the inverse initiation with 1st and 2nd square cut solid decking (double detonators with different delays) and spacers in periphery holes was found to be the best choice to eliminate the post-blast sockets with reasonable pull for weathered competent rock.

Keywords: mine development, drill and blast, rate of advance, priming, solid decking, inverse initiation

1. Introduction

The design of underground mine blasts depends on a series of factors such as rock properties, geology, explosive characteristics and blast design geometry. Learning about these parameters can speed up drive development through increased pull and produce significant economic and social benefits [1–4]. Many blast design formulae have been developed by researchers but using them directly for a particular site is unsuitable. Therefore, theoretical formulae can only serve as a guide for any blast design or its implementation.

The explosive initiation location determines the detonation wave propagation direction. Consequently, the effect of the initiation location must be addressed in the drill and blast for improved blast results [5–9]. Gao et al. [10] found that the initiation location affects the blast vibration. Fry et al. and Sichel [11,12] also found that the detonation direction affects the dynamic behaviour of the surrounding medium.

"The normal priming method shows the primer positioned at the back of the borehole, while the reverse priming method depicts the primer at the opposite end of the powder column" [13]. Allen et al. [14] reported that with an extended relief hole in burn cut design, the application of reverse priming might be even better suited.

Zhang et al. [15,16], on the basis of collision theory, reported that "if the two primers are initiated at the same time, the collar primer will produce serious back break and even bring about lot of detonation energy loss. If the collar primer is initiated later than the bottom one, the result is not good, either".

Hagan [17] reported that when the stress fields, either tectonic or gravitational (non-hydrostatic) act, the fracture pattern generated around the blast holes is influenced by the non-uniform stress concentrations around the same. In massive homogeneous rock, the cracks which start to propagate radially from the blast holes tend to follow the direction of the maximum principal stresses.

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Decking is a process of leaving a certain vacuum in a blast hole (not to fill explosives); the gap may be filled with any solid substance or air. It is one option to achieve desired blasting results in jointed rock [18]. Decking length and its position play a crucial role in designing and executing the blast [19]. Increasing decking length not only delivers optimal blast effects but also optimizes cost economics by reducing explosive charge [20,21]. The air decking provides the best opportunity to properly disperse the explosive charge in the blast hole, resulting in uniform rock breakage [22–28]. However, air decking is best suited to medium-jointed rock mass rather than highly jointed rock mass; air decking did not produce notable effects in a highly jointed rock mass [29].

2. Objective

The objective of the present study is to investigate the effect of initiation location, priming and decking of blast hole on pull in a hard rock underground mine.

3. Field description

3.1. Mine A: general, geological and mining details

To accomplish the objective, field studies were conducted at two different underground metal mines A and B.

The Mine A deposit is situated in the Cuddapah district of Andhra Pradesh. The ore body is uniform in its thickness and trend, with an average dip of 15° due to N22°E, and its physico-mechanical properties are listed in Table 1.

3.2. Mine-B: general, geological and mining details

The Mine-B deposit is situated approximately 11 km west of Jaduguda in the central region of the Singhbhum Thrust Belt. The mineralization covering a length of over 3 km and its physico-mechanical properties are listed in Table 2.

Drilling and blasting practices: in both the mines, the drilling was performed by Jackhammer (hole dia-32 mm, length of hole 1.8 m/2.4 m) for 3 m × 3 m face size while for large faces' size the drill jumbo (45 mm diameter, hole length of 3.2/3.4/4 m) was used. The explosive used was cartridge emulsion.

Table 2. Physico-mechanical properties of rock in Mine-B.

Rock type/property	Main band	KND ore body
Density (gm/cc)	2.8	average: 2.8
Compressive strength (MPa)	19 to 150, average: 70–80	6–133, average: 6–10
Tensile strength (MPa)	average: 17	0.44 to 17.47, average: 7

The pull achieved was in the range of 67–75%. The details of the drilling, charging and firing patterns for the drill jumbo face are shown in Figs. 1 and 2.

The same pattern as shown in Fig. 2 was used for both Mine-A & Mine-B. The cycle time was recorded to investigate the cost of drivages in the base pattern and the same is given in Table 3.

4. Field problems and research methodology

During trial blasts, it was found that there was a regular failure of burn cut (under blast) in Mine-A and Mine-B that resulted in post blast sockets. The sockets resulted in increased drilling length and, therefore, increased drilling cost, mucking cost, and increased overall cycle time. The achieved pull was in the range of 70–75% in both the mine. Therefore, to improve the pull, a systematic plan was prepared under different cases, as mentioned in Table 4.

Almost 300 blasts in Mine-A and 200 blasts in Mine-B were conducted in five varieties of initiation systems for the six different face sizes and lengths of holes listed above to identify the optimum results. The results were evaluated based on the pull, powder factor, sockets, other blast output parameters etc. Initially, the base blast pattern data for Mine-A and Mine-B were monitored for a period of eight months in various face dimensions. For example, a base pattern in a face dimension of 4.5 m × 3.0 m involves a total of 39 nos of holes with four nos of reamers having diameter of 45 and 89 mm, respectively, and having spacing and burden of 0.80 and 0.90 m respectively. Subsequently, the spacing and burden for other types of face dimensions were adjusted accordingly in line with the face size. The overall performance of these base patterns is mentioned in Tables 5 and 6.

Keeping all the blast parameters the same as being practiced, other priming/initiation location variations were tried in all six cases of the base pattern of Mine A and four cases of Mine-B, which are given below.

Table 1. Physico-mechanical properties of host rock and ore rock in Mine-A.

Rock type/property	Dolostone	Ore body	Red shale
Density (gm/cc)	2.7 to 2.9 average: 2.786	2.4 to 2.89 average: 2.86	2.78
Compressive strength (MPa)	174 to 359 average: 278	306 to 370 average: 345	38 to 183 average: 116
Tensile strength (MPa)	16 to 21 average: 18.9	15.6 to 21.8 average: 18.9	8 to 16.5 average: 12.2

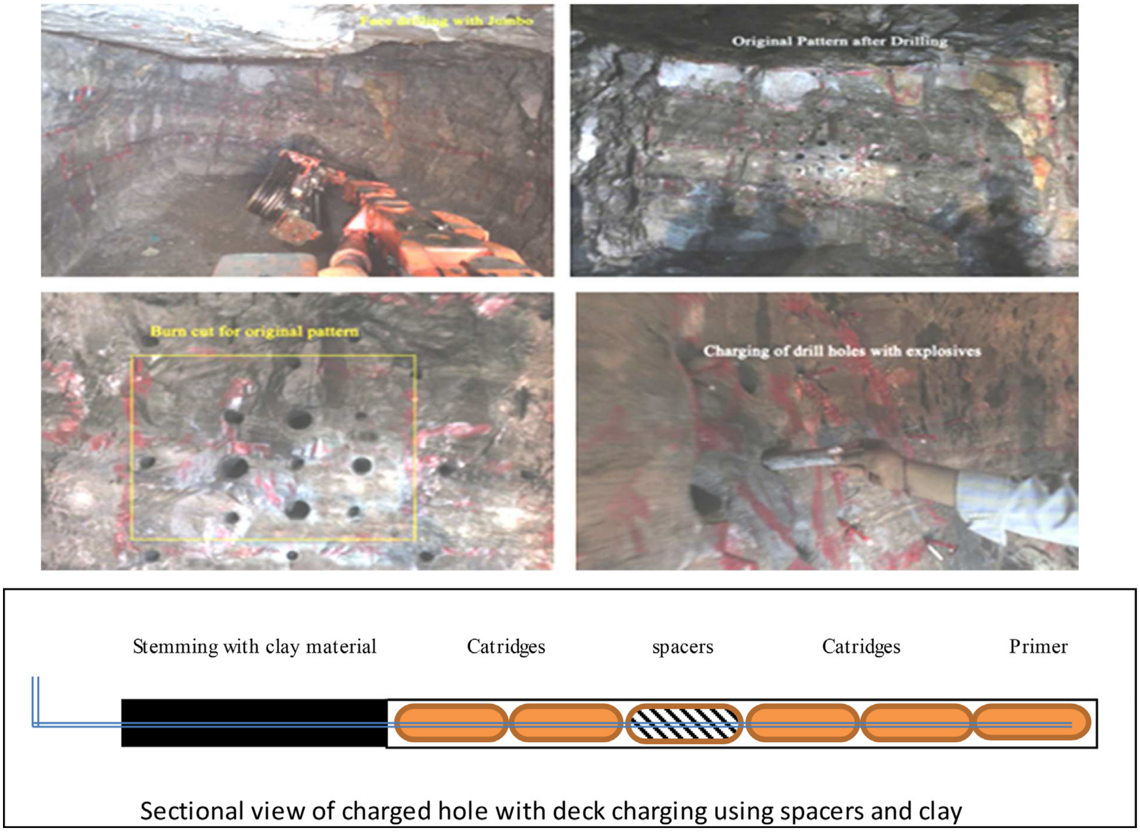

Fig. 1. Sequence of drilling and charging for burn cut pattern before blasting and sectional view of charge hole with deck charging using spacers and clay.

1. V-1: First square cut solid decking with inverse initiation,
2. V-2: First and second square cut solid decking with inverse initiation,
3. V-3: V-1, V-2 and spacers in periphery holes with inverse initiation,
4. V-4: Direct initiation without solid decking,
5. V-5: Inverse initiation without solid decking,

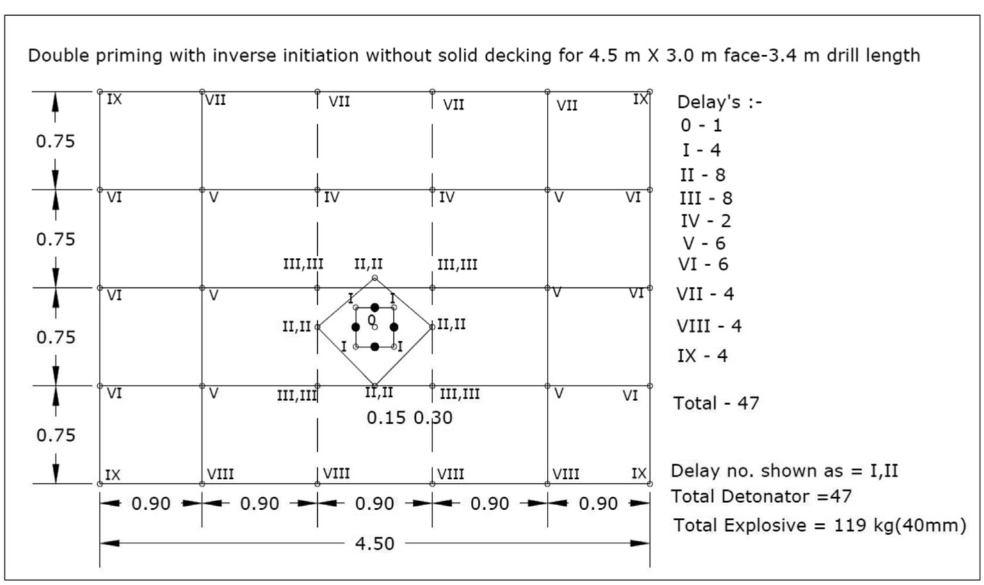

Fig. 2. Base pattern for drive size 4.5 × 3.0 m with 3.4 m drilling length.

Table 3. Description and of cycle time for 3.4 m depth drilled with a jumbo drill for 4.5 × 3.0 m drive.

Description of cycle time for 3.4 m depth drilled with drill jumbo-A1 for 4.5 × 3.0 m drive size in the base pattern at Mine-A & Mine-B.

		Mine-A	Mine-B
1	Drilling		
	Time taken for drilling per 3.4 m hole, including position and collaring of hole	1 min 25 s	1 min 45 s
	Time taken for drilling 43 nos of holes	53 min 75 s	62.35 min (1 h 2 min 35 s)
	Time taken for drilling per relief hole (89 mm)	4.0 min	5.0 min
	Time taken for drilling four nos of relief holes	16 min	20 min
	Total time taken for drilling [blast holes and relief holes]	69 min 75 s (1 h 10 min)	82.35 min (1 h 22 min 35 s)
	Operational cost of Jumbo per hour	₹ 10,200	₹ 10,120
	Operational cost of drilling for 69.75 min (1 h 10 min)	₹ 11,858	₹ 13,890
2	Charging & blasting	50 min	50 min
	Manpower cost (1 + 6)	₹ 2400	₹ 2400
	Cost of explosives (120 Kg × ₹ 65)	₹ 7800	₹ 7735
	Cost of detonators (47 nos × ₹ 20)	₹ 940	₹ 940
	Stemming cost	₹ 500	₹ 500
	Total cost for charging and blasting	₹ 11,640	₹ 11,575
3	Re-entry/fume clearance	15 min	15 min
4	Water spraying & loose dressing	30 min	30 min
5	Mucking (1 + 2) set	64 min	114 min
	Total time taken to handle 93 t by LHD	23 min	58 min
	Total time taken to handle 93 t by LPDT (22.5 t capacity)	41 min	56 min
	Operational cost of LHD per engine hour	₹ 9400	₹ 4584
	Operational cost of LHD for 23 min	₹ 3603	₹ 4431
	Operational cost of LPDT per hour	₹ 7450	₹ 5106
	Operational cost of LPDT for 41 min	₹ 5090	₹ 4766
	Total operational cost for mucking	₹ 8693	₹ 9197
6	Rock bolt drilling	45 min	45 min
	Time taken for drilling per rock bolt 2.0 m length	3 min	3 min
	Time taken for drilling 15 no of rock bolt 2.0 m length	45 min	45 min
	Total time for rock bolt drilling	45 min	45 min
	Operational cost of bolter per hour	₹ 10,200	₹ 10,120
	Operational cost of bolter for 45 min	₹ 7650	₹ 7590
	Total cost of rock bolting	₹ 7650	₹ 7590
7	Grouting	90 min	90 min
	Time taken for per rock bolt grouting	6 min	6 min
	Time taken for 15 nos rock bolt grouting	90 min	90 min
	Cost of rock bolt (including men & material)	₹ 900	₹ 900
	Total cost for grouting (15 nos × ₹ 900)	₹ 13,500	₹ 13,500
8	Bottom cleaning and face preparation	60 min	60 min
	Bottom cleaning and face blow	30 min	30 min
	Preparation for drilling	30 min	30 min
	Total cycle time	423.75 min (7 h 4 min)	486 min (8 h 6 min)
	Total cost per round of blast (S. No 1 & 2)	₹ 23,498	₹ 25,465
	Total operational cost per cycle	₹ 53,341	₹ 55,752
	Cost per ton	₹ 573.50	₹ 599.48
	Cost per meter of pull	₹ 21,595.54	₹ 22,755.91

Manpower utilised for water spraying; loose dressing & grouting are the same.

Table 4. Brief details of various face sizes and hole lengths (cases) tried in Mine-A & Mine-B.

Case	Mine-A ($W \times H \times L$ of hole)	Mine-B ($W \times H \times L$ of hole)
Case-I	3 m × 3 m × 1.8 m (JH)	3 m × 3 m × 1.8 m (JH)
Case-II	3 m × 3 m × 2.4 m (JH)	3 m × 3 m × 2.4 m (JH)
Case-III	4.5 m × 3 m × 3.4 m (DJ)	4.5 m × 3 m × 3.4 m (DJ)
Case-IV	4.5 m × 3 m × 4 m (DJ)	4.5 m × 3 m × 4 m (DJ)
Case-V	5 m × 3 m × 3.4 m (DJ)	–
Case-VI	5 m × 3 m × 4 m (DJ)	–

(Legends: JH – Jack Hammer & DJ – Drill Jumbo, W – drive width, H – drive height, L – length of drilled hole).

6. V-6: Double priming with inverse initiation and without solid decking.

A base pattern with solid deck charging and inverse initiation was introduced in both Mine-A and Mine-B. In solid deck charging two nos of detonators were placed in the first square burn cut with inverse initiation in the initial stage, then alternatively tried for the second square cut along with the first square and then lastly investigated for the combination of the first square, second square and

Table 5. Details of the overall performance of base blast pattern practiced in Mine-A for six cases.

Case	Drive size (m)	No of holes	Hole length (m)	E (kg)	D (nos)	RB (t)	Pull (m)	PF (t/kg)	DF	DT (min)
Case-I	3.0 × 3.0	37 + 4	1.8	32	45	32	1.28	0.99	1.40	114.5
Case-II	3.0 × 3.0	37 + 4	2.4	44	45	43	1.70	0.97	1.05	163.5
Case-III	4.5 × 3.0	39 + 4	3.4	120	47	93	2.47	0.78	0.50	69.75
Case-IV	4.5 × 3.0	39 + 4	4	135	47	98	2.60	0.72	0.47	80.2
Case-V	5.0 × 3.0	44 + 4	3.4	135	52	104	2.40	0.77	0.50	80.0
Case-VI	5.0 × 3.0	44 + 4	4	150	52	121	2.87	0.80	0.37	87.2

(Legends: E – explosive, D – detonators, RB – rock broken, t – tons, PF – powder factor, DF – detonator factor, DT – drilling time).

Table 6. Details of blasting parameters of base blast pattern practiced in Mine -B for four types of cases.

Case	Drive size (m)	No of holes	Hole length (m)	E (kg)	D (nos)	RB (t)	Pull (m)	PF (t/kg)	DF (nos/t)	DT (min)
Case-I	3.0 × 3.0	37 + 4	1.8	32	45	30	1.18	0.93	1.50	115
Case-II	3.0 × 3.0	37 + 4	2.4	45	45	39	1.54	0.86	1.15	164
Case-II	4.5 × 3.0	39 + 4	3.4	120	47	93	2.45	0.86	1.15	81
Case-III	4.5 × 3.0	39 + 4	4.0	135	47	93	2.45	0.77	0.50	84
Case-IV	5.0 × 3.0	44 + 4	3.4	135	52	102	2.44	0.75	0.51	89

Table 7. Pull percentage for a base pattern with double priming for Mine-A and Mine-B.

Double priming with inverse initiation	Avg. hole depth (m)	Face dimension	No of holes	Pull percentage Mine-A	Pull percentage Mine-B
Case-I	1.8	3.0 m × 3.0 m	37 + 4	71.28	65
Case-II	2.4	3.0 m × 3.0 m	37 + 4	70.93	64
Case-III	3.4	4.5 m × 3.0 m	39 + 4	72.70	72
Case-IV	3.4	5.0 m × 3.0 m	44 + 4	72.50	72
Case-V	4	4.5 m × 3.0 m	39 + 4	64.88	–
Case-VI	4	5.0 m × 3.0 m	44 + 4	71.55	–

Table 8. Initiation Investigation blast pattern in Mine-A was tried for six types of cases.

Case	Drive size (m)	No. of holes	Drilling depth (m)	E kg	D (nos)	RB (t)	Pull (m)	PF (kg/t)	DF
Case-I	3.0 × 3.0	37 + 4	1.8	32	37	35	1.38	1.09	1.06
Case-II	3.0 × 3.0	37 + 4	2.4	45	37	48	1.89	1.06	0.78
Case-III	4.5 × 3.0	39 + 4	3.4	120	39	106	2.81	0.88	0.37
Case-IV	4.5 × 3.0	39 + 4	4	135	39	120	3.16	0.88	0.33
Case-V	5.0 × 3.0	44 + 4	3.4	135	44	115	2.73	0.85	0.38
Case-VI	5.0 × 3.0	44 + 4	4.0	150	44	129	3.08	0.86	0.34

spacers at periphery holes to provide better free space with spacing and burden as 0.90 m and 0.75 m respectively. Subsequently, base patterns with direct priming and inverse priming were studied in Mine-A and Mine-B, respectively, as shown in Table 7. The final results of the investigation for the best initiation/priming are shown in Tables 8 and 9.

5. Results and discussions

In both mines all the faces were drilled in burn-cut drilling pattern and fired on double priming with one of the primers kept at the bottom of the blast hole and another kept at the middle of the blast hole. Double detonators were placed in the first square and second square of the burn cut and initiated with the same delay.

Table 9. Initiation investigation blast pattern in Mine-B was tried for four types of cases.

Case	Drive size (m)	No of holes	Drilling depth (m)	E (kg)	D (nos)	RB (t)	Pull (m)	PF (kg/t)	DF
Case-I	3.0 × 3.0	37 + 4	1.8	32	37	33	1.31	1.03	1.12
Case-II	3.0 × 3.0	37 + 4	2.4	44	37	46	1.7	1.04	0.81
Case-III	4.5 × 3.0	39 + 4	3.4	119	39	102	2.70	0.86	0.38
Case-IV	5.0 × 3.0	44 + 4	3.4	135	44	110	2.61	0.81	0.40

(Legends: E – explosive, D – detonators, RB – rock broken, PF – powder factor, DF – detonator factor).

Table 10. Effect of initiation on pull results in all variants, different face dimensions with several other lengths of the hole in Mine-A.

Face size	Variants	No of holes & reamers	Hole depth (m)	Pull (m)	Pull percentage (%)	T (t)	E (kg)	D (nos)	PF (kg/t)	DF (no/t)	Drilling length (m)
3.0 m ×	V-1	37 + 4	1.8	1.00	55.28	25	31.8	41	0.79	1.64	81
3.0 m	V-2	37 + 4	1.8	1.13	62.50	28	31.8	41	0.89	1.59	81
Drive size	V-3	37 + 4	1.8	1.33	74	34	31.8	45	1.06	1.35	81
	V-4	37 + 4	1.8	1.21	67.00	30	31.8	37	0.96	1.22	81
	V-5	37 + 4	1.8	1.38	76.78	34	31.8	37	1.10	1.06	81
	V-6	37 + 4	1.8	1.28	71.28	32	31.8	45	1.02	1.40	81
3.0 m ×	V-1	37 + 4	2.4	1.56	65.00	39	44.5	46	0.88	1.04	108
3.0 m	V-2	37 + 4	2.4	1.64	68.38	41.	44.8	45	0.92	1.09	108
Drive size	V-3	37 + 4	2.4	1.78	74	45	44.7	45	1.01	1.00	108
	V-4	37 + 4	2.4	1.55	64.70	39	44.7	37	0.88	0.95	108
	V-5	37 + 4	2.4	1.89	79	43	44.7	45	0.96	0.86	108
	V-6	37 + 4	2.4	1.70	71	86	118	43	0.73	0.50	108
4.5 m ×	V-1	39 + 4	3.4	2.30	67.62	93	120	47	0.78	0.50	160
3.0 m	V-2	39 + 4	3.4	2.47	72.53	95	119	47	0.80	0.49	160
Drive size	V-3	39 + 4	3.4	2.52	74.03	87	119	39	0.73	0.45	160
	V-4	39 + 4	3.4	2.32	68.10	106	119	39	0.89	0.37	160
	V-5	39 + 4	3.4	2.81	82.6	93	119	47	0.78	0.42	160
	V-6	39 + 4	3.4	2.47	72.7	94	134	48	0.71	0.51	176
5.0 m ×	V-1	44 + 4	3.4	2.26	66.50	103	134	52	0.76	0.51	176
3.0 m	V-2	44 + 4	3.4	2.45	72.06	105	134	52	0.78	0.50	176
Drive size	V-3	44 + 4	3.4	2.50	73.53	101	134	44	0.75	0.44	176
	V-4	44 + 4	3.4	2.40	70.44	115	134	44	0.85	0.38	176
	V-5	44 + 4	3.4	2.74	80.53	100	134	52	0.75	0.44	176
	V-6	44 + 4	3.4	2.39	70.29	86	134	43	0.65	0.50	170
4.5 m ×	V-1	39 + 4	4	2.30	57.48	98.28	135	47	0.73	0.48	170
3.0 m	V-2	39 + 4	4	2.60	65.00	102.4	135	47	0.76	0.46	170
Drive size	V-3	39 + 4	4	2.71	67.78	101	135	39	0.75	0.39	170
	V-4	39 + 4	4	2.66	66.50	119.7	135	39	0.89	0.33	170
	V-5	39 + 4	4	3.17	79.23	98.09	135	47	0.73	0.41	170
	V-6	39 + 4	4	2.60	64.88	112.5	150	47	0.75	0.43	170
5.0 m ×	V-1	44 + 4	4	2.68	67.00	115.0	150	52	0.77	0.45	208
3.0 m	V-2	44 + 4	4	2.75	68.85	117.7	150	52	0.79	0.44	208
Drive size	V-3	44 + 4	4	2.80	70.08	113	149	44	0.75	0.39	208
	V-4	44 + 4	4	2.68	67	129	150	44	0.86	0.34	208
	V-5	44 + 4	4	3.08	77.03	121	149	52	0.81	0.37	208
	V-6	44 + 4	4	2.87	72	121	150	52	0.81	0.37	208

(Legend: T — tons, E — explosive, D — detonators, PF — powder factor, DF — detonator factor).

5.1. Effect of priming on the pull

5.1.1. Results of Mine-A

It is observed from Table 10 and Fig. 3, that the pull percentage of the base pattern is in the order of 65–72% in Mine-A. There was a need to improve the blast results in terms of fragmentation. Therefore, two primers were placed at 1/3 and 2/3 charged parts of each blast hole. The position of a primer in the double-primer placement may be changed according to practical situations. Hence, the first primer was put in near to middle and

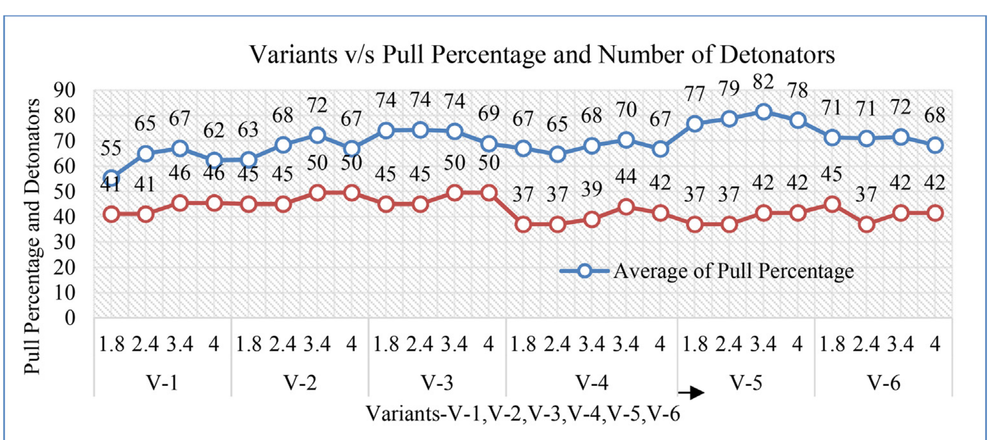

Fig. 3. Relation between all variations and pull and nos detonator for Mine-A.

another at just above the blast hole. After the experiment with double primer, the following results were obtained:

1. Improved fragmentation, which may be due to concentrated total detonation energy in a shorter time and with higher amplitude.
2. The rock mass was highly shattered in the double-primer.

In this case of the blast pattern, good fragmentation was achieved, but the pull achieved was 73%, which needs to increase. Therefore, another five varieties of initiation/priming methods were tried.

Principally, Variations V-1 and V-2 were used in view of generating free face in two stages, which may result in the creation of a sufficient volume of void and is consequently critical to achieving a proper initial cut up to a full depth of the hole and advance per blast in case of incompetent and weathered rock with joints and fractures. Variation, V-3 was introduced as an addition, where spacers (made of wooden sawdust) were used to reduce the explosive concentration in the periphery holes in order to minimize the blast damage.

Results of variations V-1 to V-6 for all cases in Mine-A

5.1.2. Results of Mine-B

Table 11 presents the field observations and blast performance results, and Fig. 4 shows that the pull percentage of the base pattern is in the range of 64–72% in Mine-B. Several blasts were conducted in Mine-B to accomplish the stated research objectives (optimization of initiation/priming). The blast patterns practiced in Mine-B were the same as in Mine-A, except for hole length.

During the field observation with the aforesaid variation (variation in placement of detonators in first square cut, second square cut and spacers in periphery holes etc.), it was revealed that the pull was of the order of 73–79% in case of V-5 and 61–74% in case of V-3. Hence, it was anticipated to better utilize the explosive energy by optimization

Variation	Brief about Variation in Mine-A	Results, including all cases
V-1	Use of two detonators of different delay numbers in each hole of the first square of the burn cut. The primer at the bottom of the hole was followed by four cartridges, then stemming by clay pills up to 50 cm, which again was fitted with mid-collar primer followed by charging four cartridges and stemming at the mouth of the hole. The sequence of initiation in the holes of the first square cut was to initiate the mid-collar primer first, followed by the bottom primer. A total number of detonators used is 41, 43 and 48 for face dimensions of 3.0 m × 3.0 m, 4.5 m × 3.0 and 5.0 m × 3.0 m, respectively.	Pull achieved was 8–22% lesser than the base pattern in 1.8 and 2.4 m length of blast holes and 5–12% lesser in 3.4 and 4.0 m, respectively.
V-2	It is the same as V-1; the only change was two primers were used in the second square cut. A total number of detonators used is 45, 47 and 52 for face dimensions of 3.0 m × 3.0 m, 4.5 m × 3.0 and 5.0 m × 3.0 m, respectively.	Pull achieved was 3–12% lesser than the base pattern with 1.8 and 2.4 m length of blast holes and similar in the case of 3.4 and 4.0 m length of blast holes.
V-3	This variation is the combination of spacers in periphery holes with inverse initiation and V-1 and V-2, in which spacers (made of wooden sawdust) were used in the periphery holes, and the cartridge saved due to the application of spacers is placed in the first and second square cut charge holes, thereby increasing the explosive concentration per hole in burn cut. A total number of detonators used is 45, 47 and 52 for face dimensions of 3.0 m × 3.0 m, 4.5 m × 3.0 m and 5.0 m × 3.0 m, respectively.	The pull percentage obtained was 3–5% more than the base pattern with 1.8 m and 2.4 m length of blast holes and 2–5% more in case of 3.4 and 4.0 m length of blast holes. The post-blast socket was minimized at periphery holes in case of development in weathered rocks; the same was also found in hard rock.
V-4	Direct initiation, keeping primer between the explosive column and stemming column at the collar of the charge hole. A total number of detonators used was 37, 39 and 44 for face dimensions of 3.0 m × 3.0 m, 4.5 m × 3.0 m and 5.0 m × 3.0 m, respectively.	Pull achieved was 5–9% less than the base pattern with 1.8 m and 2.4 m length of blast holes and 0–5% less in case of 3.4 and 4.0 m length of blast holes, respectively. The back break was also observed in these blasts.
V-5	Inverse initiation followed by explosive cartridges and stemming. A total number of detonators used was 37, 39 and 44 for face dimensions of 3.0 m × 3.0 m, 4.5 m × 3.0 m and 5.0 m × 3.0 m, respectively.	Pull obtained was 7–11% more with 1.8 and 2.4 m length of blast holes and 6–22% more in the case of 3.4 m only.
V-6	Double priming with inverse initiation and without solid decking.	This was the base pattern

Table 11. Effect of initiation on pull results in all variants, different face dimensions with several other lengths of the hole in Mine-B.

Face size	Variants	No of holes & reamers	Hole depth (m)	Pull (m)	Pull (%)	T (t)	E (kg)	D (nos)	PF (kg/t)	DF (no/t)	Drilling length (m)
3.0 m × 3.0 m Drive size	V-1	37 + 4	1.8	0.94	52.39	23	32	41	0.75	1.73	81
	V-2	37 + 4	1.8	1.06	58.61	27	32	45	0.83	1.69	81
	V-3	37 + 4	1.8	1.34	74.44	34	32	45	0.87	1.34	81
	V-4	37 + 4	1.8	1.07	59.33	26.9	32	37	0.85	1.40	81
	V-5	37 + 4	1.8	1.31	72.89	33.0	31	37	1.05	1.12	81
	V-6	37 + 4	1.8	1.18	65.44	29	31.7	45	0.94	1.30	81
3.0 m × 3.0 m Drive size	V-1	37 + 4	2.4	1.4	61.75	37.3	44.4	41	0.84	1.10	108
	V-2	37 + 4	2.4	1.5	64.92	39.2	44.5	45	0.88	1.15	108
	V-3	37 + 4	2.4	1.6	69.13	41.8	44.5	45	0.86	1.08	108
	V-4	37 + 4	2.4	1.4	58.75	35.5	44.5	37	0.80	1.05	108
	V-5	37 + 4	2.4	1.8	75.25	45.5	44.30	37	1.03	0.81	108
	V-6	37 + 4	2.4	1.5	63.96	38.6	44.50	45	0.87	0.98	108
4.5 m × 3.0 m	V-1	39 + 4	3.4	2.1	63.18	81.1	120	43	0.67	0.53	160
	V-2	39 + 4	3.4	2.4	72.12	92.6	119	47	0.78	0.51	160
	V-3	39 + 4	3.4	2.5	73.68	94.6	120.8	47	0.78	0.50	160
	V-4	39 + 4	3.4	2.3	68.97	88.6	118.7	39	0.75	0.44	160
	V-5	39 + 4	3.4	2.7	79.38	102.	119.3	39	0.86	0.38	160
	V-6	39 + 4	3.4	2.4	72.06	92.6	118.7	47	0.78	0.42	160
5.0 m × 3.0 m Drive size	V-1	44 + 4	3.4	2.1	62.29	88.9	133	48	1.50	0.67	177
	V-2	44 + 4	3.4	2.3	70.18	100	135	52	1.35	0.74	177
	V-3	44 + 4	3.4	2.3	69.38	99.0	134	52	1.36	0.74	177
	V-4	44 + 4	3.4	2.2	65.65	93.7	135	44	1.46	0.69	177
	V-5	44 + 4	3.4	2.6	76.97	109	135	44	1.23	0.81	177
	V-6	44 + 4	3.4	2.4	71.62	102	135	52	1.33	0.75	177

of initiation from V-3 to V-5. The concept adopted in this case is similar to the one adopted earlier for Mine-A.

5.2. Results of effect of initiation on pull

5.2.1. Results of Mine-A

On perusing the variation of initiation/priming results (Table 10) and results of variation of Mine-A, it was found that the results of all the variations in terms of pull are not very encouraging except the one with Inverse initiation followed by explosive cartridges and stemming in comparison to the base pattern. Since the maximum pull of 83% was observed in inverse initiation in the face size of 4.5 m × 3.0 m, which exhibits an increasing trend line in the composite graph as shown in Fig. 3 from 1.8 m to 3.4 m length of the hole, but decreases in case 4.0 m length, hence (V-5, case-III) was found the most suitable in case of highly competent rock. This means that a face dimension smaller or larger than the optimum face (4.5 m × 3.0 m) implies a wastage of explosive energy. For smaller dimension, as is amply evident from the present study, the excess explosive energy leads to the unwanted shattering of rock and even the generation of back breaks, due to which the

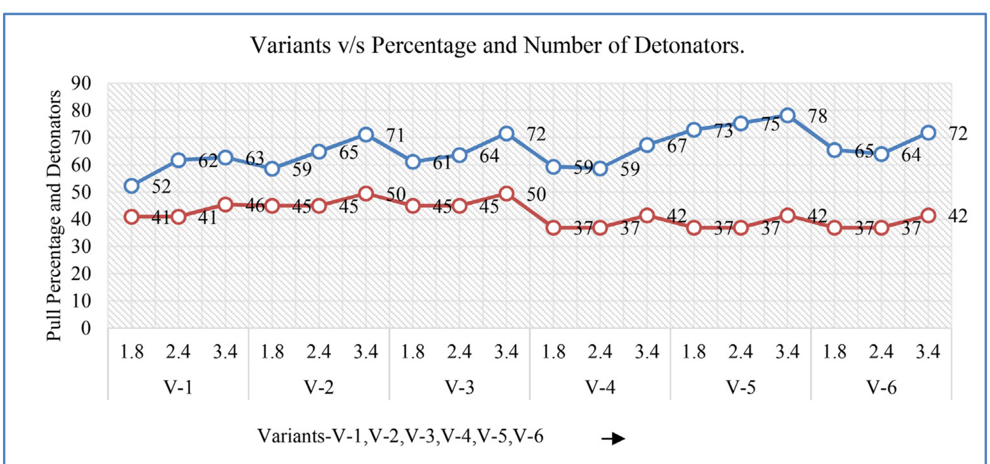

Fig. 4. Relation between all variations and pull and nos detonator for Mine-B.

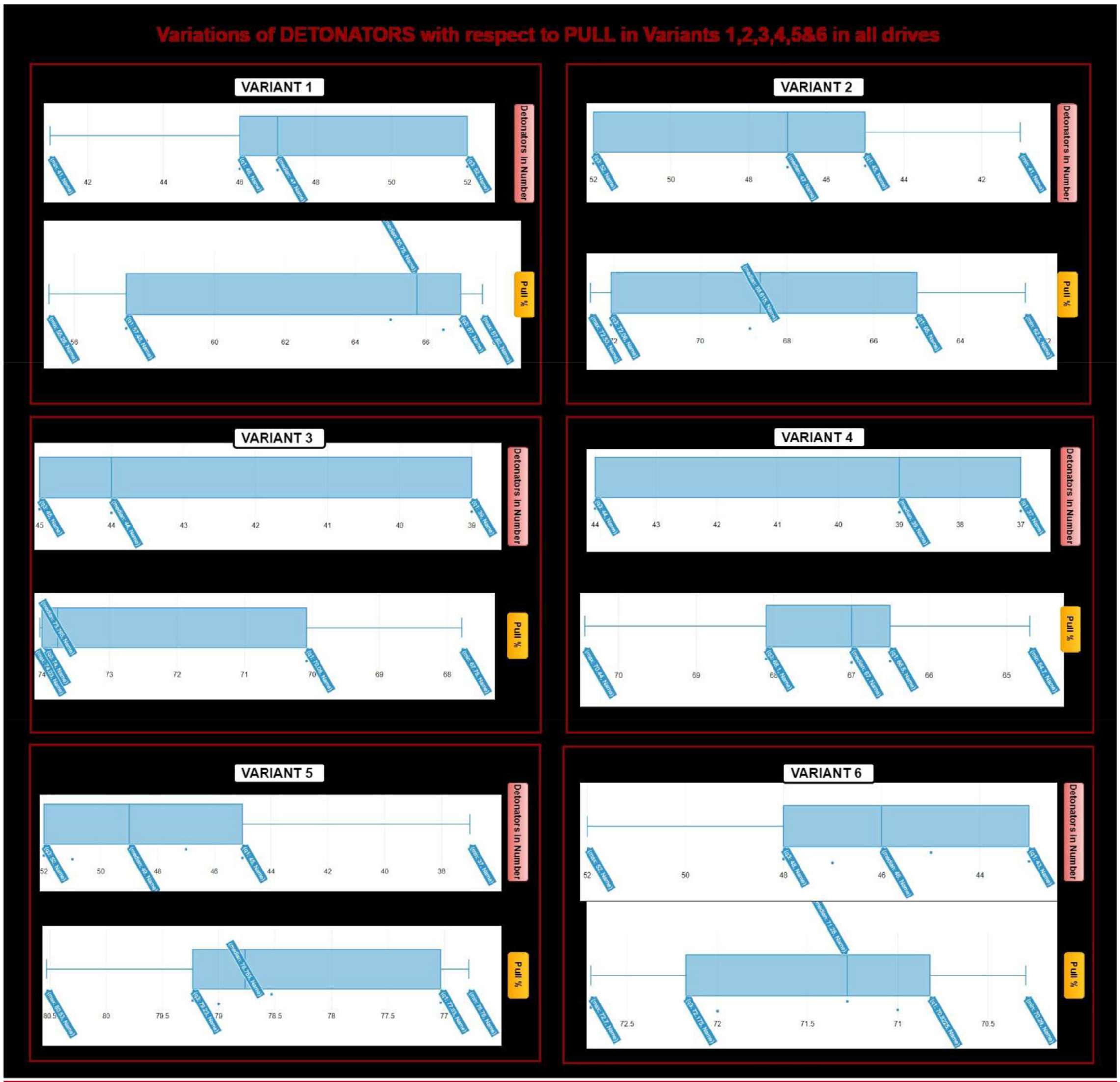

Fig. 5. Variation of detonator with respect to pull in V-1 to V-6 for Mine-A.

use of spacers were encouraged. The larger face dimension, on the other hand, resulted in generation of a large number of boulders in the blasted muck, which, in turn, more than offsets the savings on the part of explosives by increasing the loader cycle time and operating hours of rock breakers; besides generating the back breaks as well.

Subsequently, in variation V-3 (face size of 4.5 m × 3.0 m), a similar trend line of pull was observed for hole length ranging from 1.8 m to 3.4 m, i.e. maximum up to 74%, but has shown a slight decline in pull percentage in case of 4.0 m hole length in competent and weathered rock drivages, but the same was not with the case of competent rock, as there was the frequent occurrence of under blast. Hence, it may be inferred from field results that V-3 has a limitation as it is best suitable for competent and weathered rock only, and V-5, inverse variation, is the most effective option for competent rock.

Since this study is more inclined towards highly competent and hard rock, the case V-5 was considered to be the most suitable.

The composite graphical representation in Fig. 3 exhibits the zone V-3 and V-5 variations as the most favourable options for delivering the progressive pull. Conclusively, the zone of V-5 in the graph clearly indicates the achievement of optimum pull with reduced detonators in use.

5.2.2. *Results of Mine-B*

On perusing the variation of initiation/priming results (Table 11) and results of variation of Mine-B, it was found that the results of all the variations in

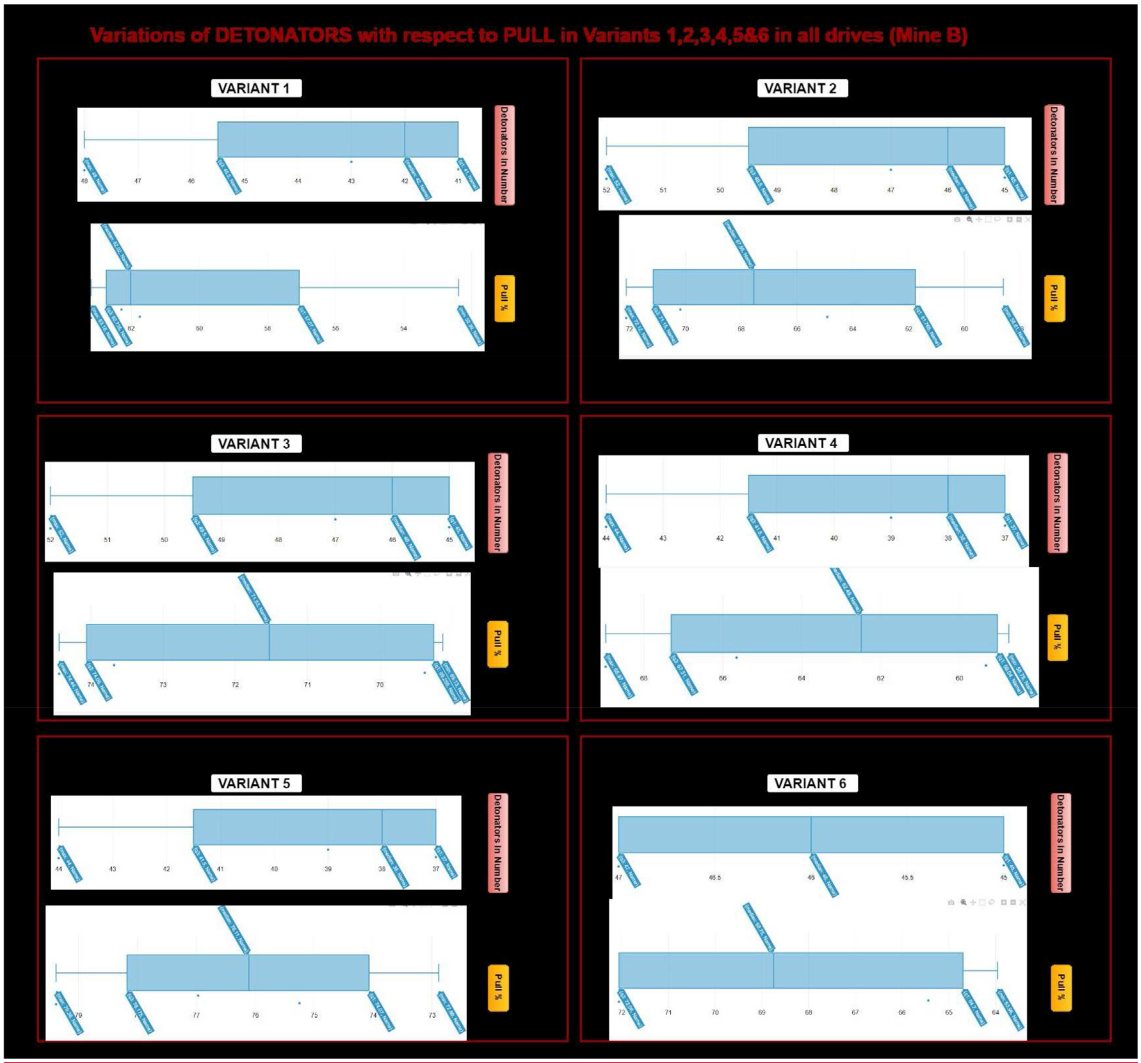

Fig. 6. Variation of detonator with respect to pull in V-1 to V-6 for Mine-B.

terms of pull except the one with Inverse initiation were not very encouraging. Since the maximum pull of 79% was observed in inverse initiation in the face size of 4.5 m × 3.0 m, which exhibits an increasing trend line in the composite graph as shown in Fig. 4 from 1.8 m to 3.4 m length of the hole, but decreases in case 4.0 m length, hence (V-5, case-III) was found best suitable in case of highly competent rock.

Subsequently, in variation V-3 (face size of 4.5 m × 3.0 m), a similar trend line of pull was observed for hole length ranging from 1.8 m to 3.4 m, i.e. maximum up to 74%, but has shown a slight decline in pull percentage in case of 4.0 m hole length in competent and weathered rock drivages, but the same was not with the case of competent rock. Hence, the same conclusion on the effective option of initiation variation may be derived as found in the case of Mine-A. Since this study is more inclined towards highly competent and hard rock, the case of V-5 is established to be most suitable in Mine-B.

The composite graphical representation of Mine-B in Fig. 4 exhibits the zone V-3 and V-5 variation as the most favourable options for delivering the progressive pull. Conclusively, the zone of V-5 in the graph clearly indicates the achievement of optimum pull with reduced detonators in use.

6. Statistical analysis

The results are analysed through the boxploting and are shown in Figs. 5 and 6 for Mine-A and Mine-B, respectively.

Figure 5 shows that among all variations from 1 to 6, the variant with the highest pull percentage −with a median of 76.7, a lower value (q1) of 77, and an upper value (q3) of 79.2 − is variant 5. Due to the fact that holes were drilled in accordance with the rock's compressive strength, the deployment of detonators was slightly higher in the same variant. This was done to maximise the shattering impact and provide a maximised pull. The median, lower and higher values of detonators in this variant are 49, 45 and 52, respectively.

Figure 6 reveals that, out of all variants from 1 to 6, variant 5 has the highest pull percentage, with a median of 78.1, a lower value (q1) of 74.07, and an upper value (q3) of 78.17, despite having a very low number of detonators in that variation. The obtained values are 39, 37, and 41.5 as the median, lower, and upper values.

7. Conclusions

The following conclusions are drawn from the study:

- From the literature and experimental results, it was found that the inverse initiation and direct initiation affect the pull.
- The pull achieved in the case of direct initiation (V-4) is 5–9% less than the base pattern for 1.8 m and 2.4 m length of blast holes and average 0–5% less for 3.4 m and 4.0 m length of blast holes in both the mines.
- Pull obtained in case of inverse initiation (V-3) is in the range of 4–13% more for 1.8 m and 2.4 m length of blast holes and 2–4% more for 3.4 m and 4.0 m length of blast holes.
- Variation V-3 helped to eliminate the post-blast sockets and under-blast failure with a reasonable increase in the pull, but not in all types of rock.
- The most suitable initiation system in any type of rock condition is inverse initiation without solid decking for optimum pull (V-5).
- The effect of the initiation location on pull is sensitive to the charge length.
- The variation in the 1st and 2nd square cut having solid decking (double detonators with different delay) and spacers in periphery holes with inverse initiation is the best option for weathered competent rock for the face size of 4.5 m × 3.0 m with 4.0 m length of blast hole to get maximum pull.
- The inverse initiation without solid decking (V-5) is ideal for getting maximum pull in competent and hard rock for the face size of 4.5 m × 3.0 m with a hole length of 3.4 m.

Conflicts of interest

The authors declare no conflict of interest.

Ethical statement

The authors state that the research was conducted according to ethical standards.

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