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RESEARCH INTO THE EFFECT OF GALLERY SIZE ON BLAST PULL IN UNDERGROUND COAL MINES

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ABSTRACT

Purpose. To investigate the effect of gallery size on blast pull in underground colliery by drilling and blasting techniques.

Methods. The study conducted in three different underground collieries namely A, B and C located in eastern part of India has been accomplished by solid blasting using milli-second short delay detonators and wedge cut pattern. The trial blasts were conducted in underground mines to investigate the effect of gallery size on blast pull.

Findings. From the study it was found that the pull is related to the cross sectional area of the drive. It increases as the face size increases. This relation is obtained considering width of the gallery in the range of 3.0 – 4.8 m and height of the gallery in the range of 2.2 – 3 m. It is assumed that the angle of the hole and the length of the hole are optimum, charging and connections being appropriate. Advancement per blast round was found to vary from 0.8 to 1.5 m whereas average advancement per blast round was 0.98 m.

Originality. This is a field study and the results are based on the data collected and analyzed on site. Although similar studies have been done by various researchers to improve the productivity of the mine for different conditions, the obtained results are condition, machinery, method and mine specific.

Practical implications. This study is applicable for underground coal mine but can be extended in underground metal and tunneling projects for improving the blast pull.

Keywords: *coal mine, gallery, pull, wedge cut pattern, blasting, gadding*

1. INTRODUCTION

Blasting in underground colliery is controlled by coal mines regulations 1957 and various circulars, issued by Director General of Mines Safety under those regulations. The adopted technique of blasting must conform to the following general directives (Singh, 1997).

Coal must be pre-cut to provide a free face and the length of the short holes must be 15 cm less than the length of the cut (Sarathy, Vidyasagar, Roy, & Singh, 2013).

The maximum charge per hole is limited to 1 kg in case of P3 and P5 explosives and to 0.79 kg in case of P1 explosives. Short holes should be neither overcharged nor undercharged (Kaku, 2009).

In multi short firing the detonators must be connected in series and fired simultaneously.

For blasting off the solid, P5 class of explosives and "carrick" series of short delay detonators are permitted where the number of shots to be fired is in excess of the capacity of the exploder. The first round may be fired with short delay detonators and balance with instantaneous detonators simultaneously (CMR 1957). The maximum delay between the first shot and last shot must be fixed by taking into consideration the make of gas.

The development blast rounds depend on many factors such as: strength of coal, thickness of coal seam, structure of the seam, dip of the seam, nature of roof and floor, strata pressure and ground stresses, method of working etc.

An effort has been made by Murthy & Ray (2002) to improve the pull per round at Tandsi mine while placing emphasis on higher roof stability. A series of trial blasts consisting of conceptual and full face blasts with modified wedge cut were carried out in the dip, rise and level galleries of the mine. From the results it was apparent that the increase in coal availability by 40% was a significant gain along with a marginal improvement in powder factor, 7 and 42% improvement in detonator factor due to change in drilling and firing pattern. The pull obtained with the modified pattern was 39% more, indicating the suitability of the proposed blast pattern. They also suggested that higher pull, normally, results in higher roof vibration due to increased hole confinement. However, this confinement was neutralized by the wider gallery size. The earlier narrow gallery size and improper drilling pattern both were responsible for higher vibration and lower pull.

Adhikari & Venkatesh (1995) suggested that drilling and blasting cost in any project can be as high as 25% of the total production cost. So the design and implementation of a blast must be given some priority. By the blast design parameters optimization the profitability would increase. They observed that to achieve a certain degree of refinement in blast design, scientific and systematic approach is needed.

An average pull of 0.8 to 1.0 m and yield of 10 to 16 tonne per blast in Blasting off the Solid (BOS) have never been considered satisfactory for optimum utilization of men and machines at faces. Efforts to mechanize bord and pillar workings by introducing intermediate technology, i.e with SDLs (Side Discharge Loaders) and LHDs (Load Haul Dumpers) as loading machines, could not achieve expected production targets due to poor availability and thither variants of coal at the faces (Roy & Singh, 2011; Mishra, Sugla, & Singha, 2013). Thus, conventional method of solid blasting in underground coal mines in India suffers from low pull and yield per blast leading to underutilization of men and machines at face which is the main reason for low production and productivity of Indian underground coal mines.

2. FIELD STUDY

The study conducted at three different underground collieries namely A, B and C located in eastern part of India has been accomplished by solid blasting using milli-second short delay detonators and wedge cut pattern as shown in Figure 1.

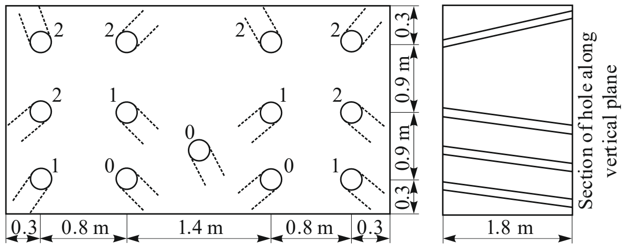

Figure 1. General wedge cut pattern with blasting sequence

These mines deploy a fleet of Side Discharge Loaders (SDLs) and Load Haul Dumpers (LHDs) for mucking the blasted coal. It was found in practice that the utilization of these costly machines is very low adding to the cost of mining. This urged an immediate need to make more coal available by resorting to an optimum blast design.

The gallery width practiced in the collieries was from 3 to 4.8 m with height varying from 2.5 to 3 m. The permitted explosive being used is Senatel 5000, Pentadyne Solar Coal 5. The seam thickness was 6.5 to 11 m and the depth of excavation was at 180 to 320 m. The optimum operating width required by the LHDs/SDLs was 4 m. Therefore, an optimum blast pattern which could yield higher pull was needed to meet the mine production target. This study assesses the reasons, factors and parameters behind the considerable difference between the theoretical advancement rate and the pull obtained practically.

3. RESULTS AND DISCUSSION

The study was conducted for the three collieries. The field observations are presented in Tables 1, 2 and 3.

Table 1. Data obtained and calculated for colliery-A

Face	Face height, m	Face width, m	Pull, m	Normalized pull, m*	Normalization factor for gallery dimension, N2**	Volume of broken rock, m^3	Amount of broken coal, te	Normalized tonnage, te	Powder factor, te/kg
10R/7SL	2.90	4.23	1.52	1.26	1.06	18.65	26.10	27.60	3.82
10R/7SL	2.74	4.57	1.42	1.18	1.04	17.78	24.89	25.79	3.57
10R/7SL	2.90	4.80	1.5	1.25	0.92	21.23	29.72	27.24	3.77
10R/7SL	3.00	4.26	1.57	1.30	1.00	20.40	28.56	28.51	3.94
10R/7SL	2.98	4.52	1.50	1.25	0.96	20.20	28.29	27.24	3.77
10R/7SL	2.90	4.57	1.48	1.23	0.98	19.61	27.46	26.88	3.72
6D/13SL	2.59	4.80	1.52	1.26	1.03	19.09	26.73	27.60	3.82
6D/13SL	2.74	4.80	1.52	1.26	0.98	20.07	28.10	27.60	3.82
6D/13SL	2.74	4.78	1.10	0.91	0.99	14.41	20.17	19.98	2.76
7D/12SL	2.59	4.72	1.58	1.31	1.06	19.32	27.04	28.69	3.97
7D/12SL	2.59	4.80	1.22	1.01	1.04	15.23	21.32	22.16	3.06
10R/8SL	2.82	4.38	1.30	1.08	1.05	16.06	22.48	23.61	3.27
10R/8SL	3.00	4.80	1.37	1.14	0.87	20.39	28.55	24.88	3.44
10D/8SL	2.90	4.49	1.41	1.17	1.00	18.36	25.70	25.61	3.54
10D/8SL	2.82	4.20	1.32	1.10	1.10	15.63	21.89	23.97	3.32
10D/7SL	2.90	3.95	1.23	1.02	1.13	14.09	19.73	22.34	3.09

*Normalized pull is calculated considering the drill rod length. For colliery-A drill rod length = 1.8 m as against 1.5 m used in colliery B and C. Therefore, Normalization factor (N1) for pull is 1.5/1.8 = 0.833.

Achieved Pull = 1.52 m so Normalized Pull = 1.52·0.833 = 1.26 m.

**Normalization Factor N2 is calculated considering an arbitrary gallery dimension = (4.6·2.8) as standard dimension. Hence, the normalized tonnage = actual tonnage* N2.

N2 = 4.6·2.8 actual gallery cross section.

Table 2. Data obtained and calculated for colliery-B

Face	Gallery height, m	Gallery width, m	Pull, m	Normalization factor for gallery dimension, N2**	Volume of broken rock, m^3	Amount of broken coal, te	Normalized tonnage, te	Powder factor, te/kg
3D/12EL	2.50	3.75	1.08	1.38	10.13	14.18	19.61	3.27
3D/12EL	2.75	3.90	1.05	1.21	11.26	15.77	19.07	3.18
3D/12EL	2.50	3.65	0.90	1.42	8.21	11.50	16.34	2.72
4D/13EL	2.80	3.80	0.85	1.22	9.04	12.66	15.44	2.57
4D/13EL	2.90	3.90	1.02	1.15	11.54	16.15	18.52	3.09
4D/13EL	2.60	3.72	1.04	1.34	10.06	14.08	18.89	3.15
3R/13EL	2.72	3.82	0.92	1.25	9.56	13.38	16.71	2.78
3R/13EL	2.54	3.74	1.05	1.37	9.97	13.96	19.07	3.18
3R/13EL	2.62	3.65	1.02	1.36	9.75	13.66	18.52	3.09
3D/13WL	2.80	3.80	1.10	1.22	11.70	16.39	19.98	3.33
3D/13WL	2.92	3.90	1.08	1.14	12.30	17.22	19.61	3.27
3D/13WL	2.88	3.85	0.95	1.17	10.53	14.75	17.25	2.88
4R/13WL	2.91	3.85	0.92	1.16	10.31	14.43	16.71	2.78
4R/13WL	2.72	3.68	1.02	1.30	10.21	14.29	18.52	3.09
4R/13WL	2.62	3.72	0.92	1.33	8.97	12.55	16.71	2.78

Table 3. Data obtained and calculated for colliery-C

Face	Face height, m	Face width, m	Pull, m	Normalization factor for gallery dimension, N2**	Volume of broken rock, m^3	Amount of broken coal, te	Normalized tonnage, te	Powder factor, te/kg
4D/137EL	2.5	3.8	0.97	1.37	9.22	12.90	17.62	2.94
4D/17EL	2.8	3.9	1.05	1.19	11.47	16.05	19.07	3.18
4D/17EL	2.5	3.6	1.10	1.44	9.90	13.86	19.98	3.33
5R/17EL	2.7	3.8	1.08	1.26	11.08	15.51	19.61	3.27
5R/17EL	2.5	4.1	1.15	1.27	11.79	16.50	20.88	3.48
4R/18EL	2.6	3.8	1.00	1.31	9.88	13.83	18.16	3.03
4R/18EL	2.8	3.7	0.90	1.25	9.32	13.05	16.34	2.72
4R/18EL	2.6	3.0	1.15	1.66	8.97	12.56	20.88	3.48
5D/17WL	2.5	3.5	1.00	1.48	8.75	12.25	18.16	3.03
5D/17WL	2.4	3.6	1.02	1.50	8.81	12.34	18.52	3.09
5D/17WL	2.6	3.8	1.05	1.31	10.37	14.52	19.07	3.18
5D/17EL	2.2	3.7	1.10	1.59	8.95	12.54	19.98	3.33
5D/17EL	2.5	3.5	0.95	1.48	8.31	11.64	17.25	2.88
5D/17EL	2.4	3.8	1.00	1.42	9.12	12.77	18.16	3.03
5R/17EL	2.5	3.6	0.99	1.44	8.91	12.47	17.98	3.00

3.1 Trends of powder factor in studied mines

It is evident from the Figure 2 that the powder factor (te/kg) achieved is higher in the case of colliery-A than the collieries B and C.

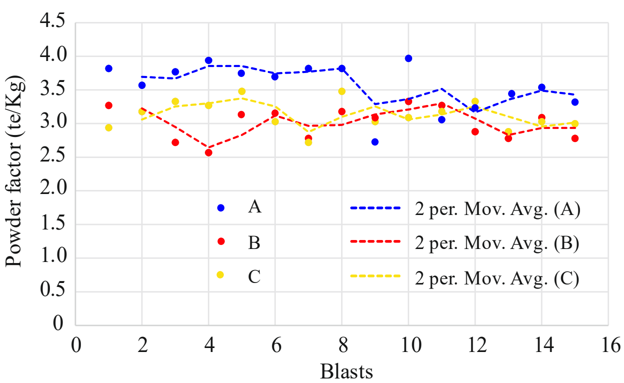

Figure 2. Trends of powder factor in different collieries

This is due to higher pull achieved in wider gallery, which, in turn, increased the overall production from the face.

3.2 Relation between gallery size and pull achieved

Figure 3 shows the graph of relationship between the developing face cross section and the pull achieved in all the three collieries.

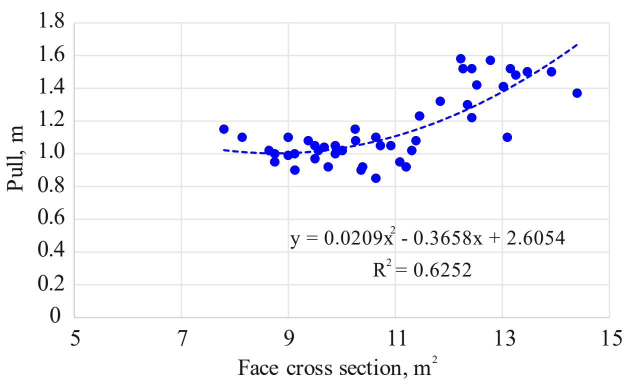

Figure 3. Developing gallery cross section vs achieved pull

It shows that the advancement (pull) increases as the face cross section increases. The increase in face cross section helps in forming the correct angle of wedge during drilling at face and drilling extra holes to create free face. The correct angle of holes forms a correct wedge and creates proper free face, which helps increase pull. It was also observed that in case of colliery-A, the pull achieved is higher than in the other two collieries, because of correct drilling of holes and keeping uncharged holes in the face for creating free face.

4. CONCLUSIONS

The following conclusions may be drawn from this study:

– pull is related to the cross-sectional area. The general trend shows that it increases as the face size increases. This relation is obtained considering width of gallery in the range of 3 – 4.8 m and height of gallery in the range of 2.2 – 3 m;

– maximum pull is obtained when the drive size is 13 m^2;

– higher pull can be achieved by improved drilling technique as was used in colliery-A, with modified wedge cut pattern;

– generally, use of a longer drill rod, 1.8 m (colliery-A) as against commonly used 1.5 m (colliery-B and colliery-C) provides an improved normalized pull;

– the angle of drilling must be optimum to achieve the requisite pull during each blast;

– drilling up to the full length of the drill rod must be ensured for pull maximization;

– proper stemming should be carried out to prevent escape of blast energy.

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ДОСЛІДЖЕННЯ ВПЛИВУ РОЗМІРІВ ВИРОБКИ НА ПОСУВАННЯ ВИБОЮ В ПІДЗЕМНИХ ВУГІЛЬНИХ ШАХТАХ

Б.С. Чоудхарі

Мета. Дослідження впливу розмірів виробки на посування вибою у вугільній шахті при веденні буропідривних робіт.

Методика. Дослідження проводилися на трьох шахтах A, B і C, розташованих у східній частині Індії. Підривання цілика здійснювалось методом клинового врубу із використанням детонаторів з мілісекундним уповільненням. Вибухова речовина, що використовувалась – Senatel 5000, Pentadyne Solar Coal 5. Випробувальні вибухи були проведені в шахтах для вивчення впливу розмірів виробки на посування вибою.

Результати. Експериментальними шахтними дослідженнями встановлено, що посування вибою залежить від поперечного перерізу штреку та зростає зі збільшенням площі поверхні забою за поліноміальною залежністю, отриманою для виробок шириною 3.0 – 4.8 м і висотою 2.2 – 3.0 м. Визначено, що кути встановлення шпурів та їх довжина були оптимальні, а забійка заряду і з'єднання відповідають чинним нормативам. Встановлено, що посування вибою за один вибух склало 0.8 – 1.5 м при середній величині посування забою 0.98 м.

Наукова новизна. Отримані результати впливу розмірів виробки на посування її вибою при проведенні є новими для певних умов шахт східної Індії, механізмів і методів видобутку.

Практична значимість. Збільшення поперечного перерізу поверхні вибою дозволяє визначити оптимальний кут клину при бурінні шпурів. Отримані результати мають практичне значення не лише для вугільних шахт, а й для поліпшення посування вибою при будівництві тунелів і видобутку металевої руди.

Ключові слова: *вугільна шахта, гірнича виробка, посування вибою, клиновий вруб, підривання, буріння шпурів*

ИССЛЕДОВАНИЕ ВЛИЯНИЯ РАЗМЕРОВ ВЫРАБОТКИ НА ПОДВИГАНИЕ ЗАБОЯ В ПОДЗЕМНЫХ УГОЛЬНЫХ ШАХТАХ

Б.С. Чоудхари

Цель. Исследование влияния размеров выработки на подвигание забоя в угольной шахте при ведении буровзрывных работ.

Методика. Исследования проводились на трех шахтах A, B и C, расположенных в восточной части Индии. Взрывание целика производилось методом клинового вруба с использованием детонаторов с миллисекундным замедлением. Используемое взрывчатое вещество – Senatel 5000, Pentadyne Solar Coal 5. Испытательные взрывы были проведены в шахтах для изучения влияния размеров выработки на подвигание забоя.

Результаты. Экспериментальными шахтными исследованиями установлено, что подвигание забоя зависит от поперечного сечения штрека и растет с увеличением площади поверхности забоя по полиномиальной зависимости, полученной для выработок шириной 3.0 – 4.8 м и высотой 2.2 – 3.0 м. Определено, что что углы установки шпуров и их длина были оптимальны, а забойка заряда и соединения соответствуют действующим нормативам. Установлено, что подвигание забоя за один взрыв составило 0.8 – 1.5 м, при средней величине подвигания забоя 0.98 м.

Научная новизна. Полученные результаты влияния размеров выработки на подвигание ее забоя при проведении являются новыми для определенных условий шахт восточной Индии, механизмов и методов добычи.

Практическая значимость. Увеличение поперечного сечения поверхности забоя позволяет определить оптимальный угол клина при бурении шпуров. Полученные результаты имеют значение не только для подземных угольных шахт, но и для улучшения подвигания забоя при строительстве туннелей и добычи металлической руды.

Ключевые слова: *угольная шахта, горная выработка, подвигание забоя, клиновый вруб, взрывание, бурение шпуров*

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