International Journal For Advanced Research In Science & Technology
A peer reviewed international journal www.ijarst.in
ISSN: 2457-0362

A CASE STUDY ON COMPARISON OF THEORETICAL ANDPRACTICAL DESIGN ASPECTS OF BLASTING IN OPENCASTMINES

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Abstract:Blast design is the key factor in mining operationsthat has a profound bearing on the overall economics of the Mining project. Proper selections of blast parameters will ensure the most optimum result with regard to the quality of the blast. Optimization of the blast is dependent on a host of complex factors related to the rock, explosive, initiation, drill-hole parameters and their layout. An optimum blast is also associated with the most efficient utilization of blasting energy in the rock-breaking process, reducing blasting cost through less explosive consumption and less wastage of explosive energy in blasting, less throw of materials, and reduction of blast vibration resulting in greater degrees of safety and stability to the nearby structures. This project deals with the study of theoretical blast design with practical blast design in an Opencast Coal Mine. By comparing of the theoretical blast design and practical blast design, we can know the defects of the blast design and I'm suggesting the optimum blast design to the mine authorities.After knowing the defects of the blast design, I'm suggesting the proper blast design to the experimented opencast coal mine to reduce the cost of the blasting.

INTRODUCTION

Blast design is not a science, but knowledge, experience, studying and analyzing past Practices in relation to rock strata & geology etc., makes blaster to achieve perfection. Thus, for a blaster, valuable tool is the file of blast reports that he builds as he gains experience. Not only do these provide evidence of the quality of his work, but they also provide a wealth of information upon which he can draw as future blasting situations develop. Improvement in production has been achieved with the help of large capacity open cast machineries, continuous mining system with improved design, development of modern generation, explosives and accessories, process Innovations and application of information technologies and increased adoption of computerized mine planning and control. The concept of blast design starts with the estimated production per annum and it undergoes estimated production per month/day, type and number of shovels and dumpers to be deployed type of technology adopted to transport the material, type of drills used, shot hole design, diameter of the shot hole, type of explosives, blast plan and estimated cost of the production. Blast design plays main role in the extraction of minerals with safety and economically. Proper adoption of blast design can contribute significantly towards profitability and therefore optimizations of these parameters are essential.

Optimization means achieving the best i.e. to achieve maximum or minimum value of the operating parameters.

Optimization of blast is dependent on a host of complex factors related to the rock, explosive, initiation, drill-hole parameters and their layout. The present work is a step in the direction of developing a suitable blast design, with simple methodologies which can be accepted by the mining industry to achieve better blasting results is found reasonably accurate. Geological and geo-mechanical properties as well as operational conditions vary from deposit to deposit and from mine to mine. Therefore, blast design parameters estimated from theories and empirical formulae should serve only as a guide. The blast design parameters must be obtained by field testing any estimated values. Expert application of open pit blast design principles improves blast performance, maximizes production and significantly reduces cost.

BLASTING MECHANISM

2.1 Theory of Breakage

Blasting operation is to assist in the conversion of solid rock to several small pieces capable of being excavated or moved by material handling equipment. This entire operation comprises of Fragmentation and Displacement or movement or throw. The extent of fragmentation and displacement is varying from site to site and also nature of job requirement. Both these operations are controlled by the energy liberated from an explosive charge. Breaking of rock involves two

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IJARST ISSN: 2457-0362

basic processes Radial cracking and Flexural rapture. Rock is stronger in compression than in tension. Thus, the easiest way to break rock is to subject it to a tensile stress greater than its ultimate strength in tension as shown in figure14. Since rock formations are not homogeneous and have variable physic mechanical properties, this process gets complicated. This is further complicated by the changes in density.

Sudden and quick release of high pressure takes place when the explosion occurs, with firing of a cylindrical charge. This sends a shock wave through the rock from the blast hole (after crushing /pulverizing a small amount of rock immediately around the blast hole). This compression shock wave shown at different times indicated by A,B and C with a radial cracking proceeding behind, travels from the blast hole throughout the entire rock mass as an elastic wave its speed is a function of the rock density – the denser the rock the faster the wave travels as shown in figure 1. This wave travels towards the nearest available free face and distance travelled is the "burden", and then reflected back to the blasthole. Wherever the wave comes in contact with a strata having change in density, contact with a portion of the wave will return to the blasthole and the remainder will continue through the different material in weakened state.

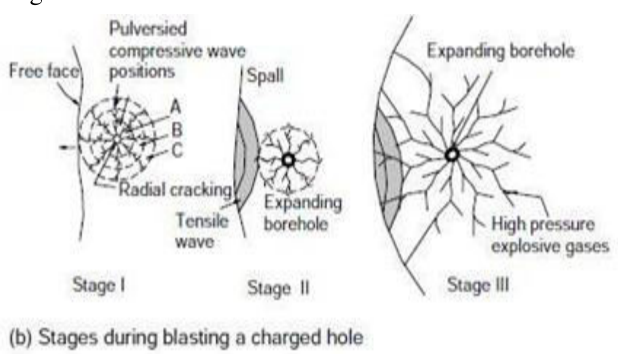

Figure 1: Stages during blasting a charged hole

Proper/desired fragmentation results when there is sufficient force in the compressive wave to travel to the face and back, overcoming the tensile strength of the medium through which it propagates. In any blasting operation only 3% of the explosive energy is used in the compressive wave in the burden area. Along the face the outermost edge is stretched in tension, which causes cracks. Boulders will formed if the energy released is not high enough to travel to the face and return to the source. Boulders are also formed if there is change in density of the strata, or over break from the previous blast.

2.2 Radial Cracks

The compression is not only enlarging short radial cracks that radiate out from the blasthole axis. Rock being not homogeneous is short lived compression waves do not develop much radial cracks. In turn it helps in the expansion of the existing cracks and direct energy that follows as shown in figure 2.

2.3 Flexural Rapture

Rapid expansion of the explosion gases in the blast hole causes flexure or bending. This gas expansion exerts pressure against to the blasthole. The generation of the gas pressuredrives the radial cracks through the burden to the free face and then causes the rock to displace in the direction of this resistance. This gaseous pressure causes a flexural rupture of the rock and is responsible for the fracture of the rock in a direction perpendicular to the blasthole axis as shown in figure 16 below.

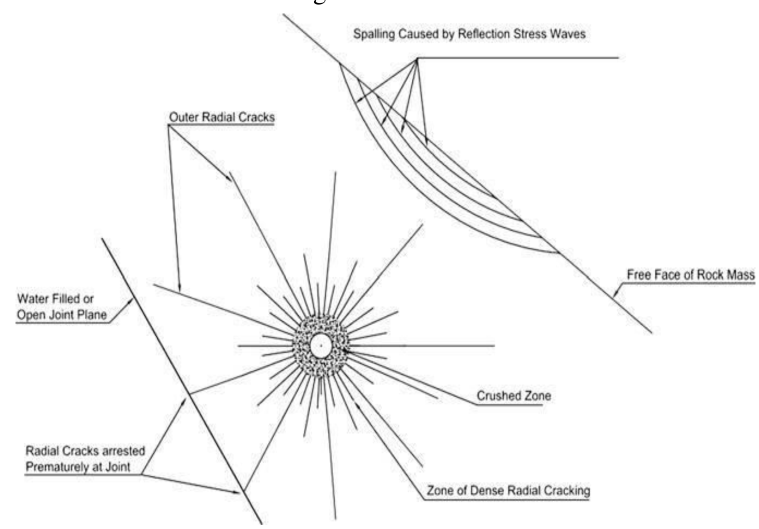

Figure 2: Radial fracturing and spalling caused by reflection of the stress waves(Source: Dongxiang Zo)

In the fragmentation of rocks with explosives at least eight mechanisms are involved, with more or less responsibility, but they all expert influence upon the results of the blasting.

i) Crushing of rock

In the first instants of detonation, the pressure in front of the strain wave, which expands in cylindrical form, reaches values that well exceed the dynamic compressive strength of the rock, provoking the destruction of its inter crystalline and inter granular structure.

ii) Radial fracturing

During propagation of the strain wave, the rock surrounding the blasthole is subjected to an intense radial compression which induces tensile components in the tangential planes of the wave front. When the tangential strains exceed the dynamic tensile strength of the rock, the formation of a dense area of radial cracks around the crushed zone that surrounds the blasthole is initiated as shown in figure 17.

a) Reflection breakage or spalling

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When the strain wave reaches a free surface two waves are generates a tensile wave and a shear wave. This occurs when the radial cracks have not propagated farther than one third the distance between the charge and the free face. Although the relative magnitude of the energies associated with the two waves depends upon the incident angle of the compressive strain wave, the fracturing is usually caused by the reflected tensile wave. If the tensile wave is strong enough to exceed the dynamic strength of the rock, the phenomenon known as spalling will come about, back towards the interior of the rock.

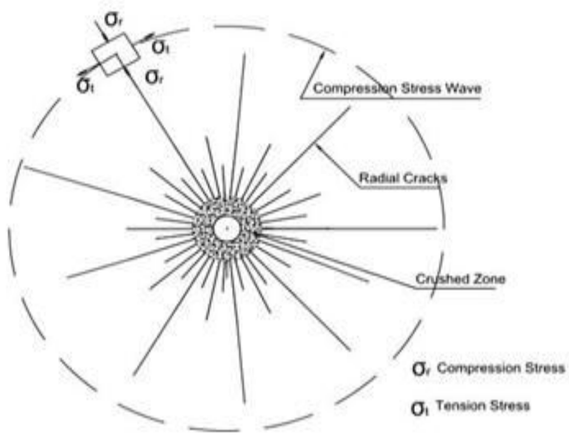

Figure 17: Tensile components in the tangential planes of the compression stress wave front forms radial cracks in the rock

(Source: Dongxiang Zo)

b) Gas extension fractures

After the strain wave passes, the pressure of the gases cause a quasi-static stress field around the blasthole. During or after the formation of radial cracks by the tangential tensile component of the wave, the gases start to expand and penetrate into the fractures. The radial cracks are prolonged under the influence of the stress concentrations at their tips.

c) Fracturing by release-of-load

Before the strain wave reaches the free face, the total energy transferred to the rock by initial compression varies between 60 and 70% of the blast energy. After the compressive wave has passed, a state of quasi-static equilibrium is produced, followed by a subsequent fall of pressure in the blasthole as the gases escape through the stemming, through the radial cracks and with rock displacement. The stored Stress Energy is rapidly released, generating an initiation of tensile and shear fractures in the rock mass.

f) Fracturing along boundaries of modulus contrast of shear fracturing

In sedimentary rock formations when the bedding planes, joints etc., have different elasticity modulus or geo mechanic parameters, breakage is produced in the separation planes when the strain wave passes through because of the strain differential in these points.

g) Breakage by flexion

During and after the mechanisms of radial fracturing and spalling, the pressure applied by the explosion gases upon the material in front of the explosive column makes the rock act like a beam embedded in the bottom of the blasthole and in the stemming area producing the deformation and fracturing of the same by the phenomena of flexion.

h) Fracture by in-flight collisions

The rock fragments created by the previous mechanisms and accelerated by the gases are projected towards the free face, colliding with each other and thereby producing additional fragmentation.

2.4 Explosive energy utilization

Detonation of explosive follows release of chemical energy in form of heat, light, seismic wave, sound, shock and pressure. A part of the total energy used for rock breaking, fragmentation and its displacement and rest escapes into the environment. A good blast should use maximum share of its energy in rock fragmentation and create less noise, air over pressure, ground vibrations and fly rocks. The major share of the energy of the explosive is used up in generating vibration and air over pressure. The energy balanced of the blasting operation as been estimated as follows:

- Fracturing in situ <1%
- Breakage 15%
- Displacement 4%
- Crushing in the vicinity of the hole 1.5-2%
- Fly rock <1%
- Deformation of the solid rock beyond the shot <1%
- Ground vibration 40%
- Air over pressure 38-39%

The useful share of explosive in mining is only the shock energy and gas energy the combined effect of these two forms of energy is responsible for final breakage of rock. It is concluded that nearly 20% - 25% of the energy is used for breakage of the rocks and 75% - 80% of the energy is wasted in the form of unwanted ground vibrations and produces air over pressure.

FIELD INVESTIGATION

3.1 Genaral Information

A Case studyof kalyanikhani opencast project, mandamarri area, the singareni collieries company limited (sccl). Location/Addres s of the mine Longitude North : 18°

59' 44" to 19° 03' 42" Latitude East : 79° 26' 32" to 79° 28' 47" Village Mandamarri, Dist. from Hyd. – by road / rail Road/Rail : 260 KM / 280 KM. Geological information of the mine are Pranahita-Godavari basin, a NNW-SSE trending basin deposit, covering an area of 17000sq.km is on Pre-Cambrian platform following the course of Pranhita and Godavari rivers over a strike length of 470 km. The south eastern sector over 350 km length lying in the districts of Adlibbed, Karimnagar, Warangal and Khammam of Andhra Pradesh state is referred as Godavari Valley Coalfield. The continuity of coal seams is broken and missing at places due to faulting and therefore different coal bearing areas occurring are generally treated as different coal belts.

This Project is a conversion of the part of the Abandoned KK-2, KK-2A, SMG-1, SMG-1A & SMG-3 underground mines into open cast project. At Kalyanikhani Open Cast Project there are eleven (11) Seams existing in the mine property viz., I, IA, IB, II T, II B, LB, III B, III A, III. IV T and IV B seams. Out of which I, I A, III, III A, & IV seams are partly developed/depillared and other seams are virgin. The detailed investigation within the block has established the presence of 11 co- relatable coal seams/horizons viz. IB, IA, I, IIT, IIB, LB, IIIB, IIIA, III, IVT and IVB in descending order.

3.2 InvestigationandAnalysis

I have observed the three overburden blasts at Kalyanikhani Opencast Coal Mine. The resultsoftheobservations arementioned in thetable5, table 6 and table7.

3.2.1 DetailsoftheBlastno.1:

Table1: Detailsof the blast no.1

S.NO	PARAMETERS	BLASTNO 1	Remarks
1	Dateofblast	29-09-2020	
2	Timeofblast (Hours)	3:30 pm	
3	Locationofblast	Subsoil/860RL	
4	Diameter(mm)	150	
5	Burden(m)	5	ClosertoBH,Notadvisable,33.5D
6	Spacing(m)	7	GreaterthanBH,Notadvisable
7	Drillingpattern	Diagonalpattern	
8	Depthofthehole(m)	5.6	
9	Stemming(m)	3.5	Charge length – 2.1/5.6=37.5%
10	Numberofrows	5	
11	Numberofblastholes	179	Avg Charge- 39.485
12	Maximumchargeperdelay(Kg)	100	
13	Water	Nil	
14	Fragmentation	Satisfactory	
15	Deckingcolumn(m)	-	
16	Totalcharge(kg)	7067.9	

17	Volume(m^3)	18114.8	AvgCum/hole-101
18	Boosters(kg)	17.9	
19	SME(kg)	7050	
20	Powderfactor(m^3/kg)	2.56	

3.2.2 Details of the Blast no.2

Table2: Detailsof the blast no.2

S.NO	PARAMETERS	BLASTNO 2	Remarks
1	Dateofblast	25-09-2020	
2	Timeofblast (Hours)	3:30 pm	
3	Locationofblast	Subsoil/860RL	
4	Diameter(mm)	150	
5	Burden(m)	4	76%holedepth,too high 27D-tooless
6	Spacing(m)	5	
7	Drillingpattern	Diagonalpattern	
8	Depthofthehole(m)	5.25	
9	Stemming(m)	3.5	ChargeLength–1.75/5.25=33.33%
10	Numberofrows	5	
11	Numberofblastholes	169	
12	Maximumchargeperdelay(Kg)	100	
13	Water	Nil	
14	Fragmentation	Satisfactory	
15	Deckingcolumn(m)	-	
16	Totalcharge(kg)	7265.9	AvgCharge-42.99kg/hole
17	Volume(m^3)	18900	Avg cum/hole-111.83

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A peer reviewed international journal
ISSN: 2457-0362
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18	Boosters(kg)	16.9	
19	SME(kg)	7240	
20	Powderfactor(m^3/kg)	2.61	

3.2.3 Details of the Blast no.3

Table3: Detailsof the blast no.3

S.NO	PARAMETERS	BLASTNO 3	Remarks
1	Dateofblast	28-09-2020	
2	Timeofblast (Hours)	3:30 pm	
3	Locationofblast	Subsoil/850RL	
4	Diameter(mm)	150	
5	Burden(m)	4	27 D,tooless69%BH, large
6	Spacing(m)	5	
7	Drillingpattern	Diagonalpattern	
8	Depthofthehole(m)	5.8	
9	Stemming(m)	3.8	Charge length - 2/5.8 =34.4%
10	Numberofrows	5	
11	Numberofblastholes	191	
12	Maximumchargeperdelay(Kg)	100	
13	Water	Nil	
14	Fragmentation	Satisfactory	
15	Deckingcolumn(m)	-	
16	Totalcharge(kg)	8379.1	Avg charge/hole- 43.87kgs
17	Volume(m^3)	22098	Avg Volume/ Hole– 115.69
18	Boosters(kg)	19.1	
19	SME(kg)	7240	
20	Powderfactor(m^3/kg)	2.64	

RESULTSANDDISCUSSIONS

Based on observations, the charge distribution in a blast hole is hardly not more than 40 % and it is only 33% -39% of the blast hole from the data. As per blasting theory, charge column should be equal to burden or minimum 50% of the bench height (BH). In this mine, the bench height is only 6m. As per the blast design principles, the diameter of the holes is equal to BenchHeight/60 to BenchHeight/100. As per the blast design formula, the required diameter is 100mm to mm but using larger diameter for unknown reason. If the correct diameter is used, the charge distribution shall be good. If we use 110mm diameter drill, the charge density is 10 kg/m. In this rock, the burden is goes up to 35 – 40 times of the diameter, but it is less due to the limited bench height and hard strata. The delay used in the same row for a spacing of 5m is 25ms, then 25/5 =5ms/m and the delay between the rows for burden of 4m is 67ms, then 67/4 = 17ms/m. These delay intervals are satisfied as per the theory.

I. CONCLUSION

The diameter of the hole, Bench height is very important for achieving blast design, blastefficiency,blast economics and blast safety. Ifthediameteroftheholeischangesto110mm,thechargedistributioninaholeincreases,sofragmentation may improve. TheKKOCCOALMINEusing150mmdiameterholefor5-6mbenchheight,whichislarger. As per the blast design, 110mm dia is adequate, the charge distribution increases, itmayincreases efficiencyand safety.

5.1 LIMITATIONS

These studies were carried out in a mine using the dia of the blast hole is 150mm.Theanalysisis carried out notconsideringthe geologicaldisturbancesofthestrata.

REFERENCES

[1] Dongxiang Zo, A textbook of Theory and Technology of Rock Excavation for Civil Engineering, chapter 8 Bench Blasting pp258-312
[2] U. Langefors and B. Kihlstrom, A textbook of The Modern Techniques of Rock Blasting, third edition.
[3] Partha Das Sharma, Empirical equations supporting blast design, Discussion on blast design parameters and related matter
[4] T.N Hagan (1982), the design of blasting rounds- the key parameters, Journal of mines, metals & fuels pp.12-22.
[5] Chiappetta P N ,R F; Srihara, H N; Worsey, Design of overburden casting by blasting - recent developments, J Min Metal Fuels V38. N9, Sept 1990. P194-208
[6] Bhandari, Innovative developments in blasting in surface mining, S J Min Metal Fuels V38, N9, Sept 1990, P233-237, 232
[7] JimenoL.C&Jimeno L.E., Drilling & Blasting of Rocks
[8] D.J. Deshmukh, A textbook of Elements of Mining Technology vol 1, Chapter 8 Explosives and accessories
[9] Samir Kumar Das, A textbook of Surface mining technology, Second edition 2008, chapter 2 various aspects of surface layout and design pp 21-50.
[10] Gyanindra Kumar Pradhan, A textbook of Explosives and blasting techniques, second edition 2001, Chapter 1 Explosives, Blasting accessories pp 1-34; chapter 3 BlastingAccessories pp 53-70.
[11] Samir Kumar Das, A textbook of Explosives and Blasting Practices in Mines, second edition 2001, Chapter 2 Explosives and blasting practices in mines pp 117-253.