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Improvement of Drilling and Blasting in Underground Mine/Tunnel: A Case Study of Cominak Mine Niger Republic

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Abstract: In underground mining and tunnelling, the often-unpredictable variability in ground conditions always leads to application of sub optimal drilling and blasting patterns, which results to poor performance. This research work is aimed at achieving advance of 3 m per round in Akouta Mining Company in Niger Republic. Hence, eleven tests of four types of cuts were carried out. The resultant advance per test is measured. Of the four types of cut only one, cut 9/13 gave advance of 2.9 m, which is close to the desire advance. The poorest of all is cut 12/20 with the entire charge blow out.

Key words: Advance, cut, blow out, round, charge, velocity of detonation, stemming, primer

INTRODUCTION

When blasting a rock tunnel or mine drifts underground, the first hole to be detonated create a cut, an opening toward which the rest of the rock is successively blasted. The rock that are blasted into the cut spread outward from the cut in a pattern of rings of increasing diameter until the required perimeter is reached, which is the outside line of the holes, which is characterized by reduce charge density. A lifter hole that is those holes in the lower part of the round that remove rock down to the floor or inverted level should be properly designed. For best positioning and throw of the muck, the cut should be centrally placed. Cut low in the face tends to produce larger muck and much less throw.

There are two main types of cut, an angle cut in which the blast holes are inclined inward to meet each other and a parallel cut in which the holes are drilled parallel to each other in the direction of advance.

Angle cuts include wedge or v cuts, fan cuts and pyramid cuts. Angle cuts require level holes and less explosive, but more difficult to drill accurately than parallel cuts and gives more erratic throw of the broken muck (Costin *et al*., 1985). Since the advance is limited to about 65% of the heading width, angle cuts are used most often in heading width, such as mine excavations for iron ore and rock salt (Costin *et al*., 1985).

The parallel cut cuts, that includes burn cuts and cylinder cuts have some of the holes left empty to provide release. Burnt cuts have burden greater than diameter of the empty hole and therefore, have to be more heavily loaded than cylindrical cuts with a closer spacing (Costin *et al*., 1985).

An important advantage of parallel cuts is that the advance per round can be adjusted to suit charging ground or support conditions or the degree of experience of the blaster, without modifying the pattern of drilling (Costin *et al*., 1985).

Blast holes drilled in erratic locations; angles and depths are some of the common causes of over break, under break, poor advancement, air blasts, fly rock and high vibration level (Costin *et al*., 1985). To minimize these problems, Costin *et al*. (1985) suggested that the holes location should be carefully marked out on the rock face and during drilling their inclinations are measured and adjusted.

Charges in blast holes must be sealed or stemmed to prevent the gases from escape before its contribution to breaking the rock and to contain and direct explosive energy into the rock where it is needed. Unstemmed or poorly stemmed holes can result in rifling, the shooting of the rock and explosives from the holes with very little or rock damage and the generation of excessive air blast.

Explosive consumption is expressed as powder factor or specific charge (q) defined as the total weight in kilograms of dynamite (or equivalent weight of the explosives) to excavate a cubic meter of rock (Costin *et al*., 1985). Blasting patterns in mines are optimized over months or even years with the help of trial blasting, particularly in the early stages of mine development. Hendron and Oviard recommended that rather than giving maximum powder factors, the specification should state maximum vibration levels together with location and methods of vibration monitoring. This gives the blaster greater flexibility,

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allowing him to come up with low-cost, high production blast designs while demonstrating that these make adequate provision for control of vibrations and damage.

Specifications for controlled perimeter blasting should further give details of stemming materials and holes alignment tolerances and should state the use of either simultaneous firing or millisecond delays to reduce ground vibrations.

These for presplit and cushion blasting should give the diameter, depth and the spacing of the perimeter holes and an approximate range for the charge perimeter of the holes. Unloaded perimeter holes are specified for line drilling. For smooth wall blasting in tunnels, the specifications require that all perimeter holes except the lifter be fired on the last delay period of the round (Costin *et al*., 1985).

Pressure in a blast hole can exceed 10 Gpa (100,00atm), sufficient to shatter the rock near the hole and also to generate a stress wave that travels outward at a velocity of 3-5 km s^{-1} (Bhandari, 1997). The Velocity of Detonation (V.O.D), the speed at which the detonation shock wave travels through a charge of explosives confined in a drill hole, ranges from 1.5-7.6 km s^{-1} (Bhandari, 1997). An explosive reaction that moves through the charge faster than speed of sound is termed detonation while the one that propagate more slowly is termed deflagration. All high explosives and blasting agents detonate whereas low explosive such as black powder deflagrate (Bhandari, 1997).

Some researchers such as Bhandari (1997), Costin *et al*. (1985) consider that stress to be the main causes of fracturing with gas pressures acting to widen and extend stress generated cracks or natural rock joint. Geological reasons for deviation of the actual blasted perimeter from the intended perimeter include pronounce rock anisotropy or a substantial anisotropy of rock stresses in the plane perpendicular to the direction of drilling. When the rock developed jointing or schistosity, the drill holes and also blast-generated fractures tends to follow planes of weakness. The joints act as wave-guides, blast energy travels preferentially along the direction of jointing. When the rock is highly and anisotropic ally stressed, the fractures tend to follow a direction perpendicular to the least principal stress. A further problem develops when the ambient stresses are isotropic but high: A very high blast holes pressure may be needed to overcome the stresses sufficient to shatter the surrounding rock.

The blasting pattern must be designed to suit the nature of the rock and the stress field and not borrowed, without modification or checking, from the previous projects where success was achieved, possibly under quite different sets of conditions. Charges to the blast patterns are often needed from place to place, even without the boundaries of a single site or within a single mine.

Harries (1978) stated that there are three mechanisms by which rock fragments. Initially, the strain wave resulting from the expansion creates a series of radial fractures around the drill hole. The gas within the drill hole expands as a result of the explosion and fills these radial fractures. These radial fractures propagate under the gas pressure and circumferential fracturing is introduced. Flexural rupture is the final mechanism of fragmentation. This is where the expansion of the gases causes the rocks to bend until they break under tension (Hemphill, 1981). The expanding gases then heave the rock away from the drill hole, throwing on to the muck pile. Further fragmentation is then caused by the action of the rock particles striking each other (Harries, 1978).

Worsey (2001) noticed that there are number of variable that can be adjusted to increase advancement. Decreasing the burden of the design will increase the powder factor and consequently increasing the energy available to break the rock. Decreasing the spacing the spacing will have the same effect. These two parameters should be adjusted in tandem for if the burden is too small in relation to the spacing, the drill holes will not connect which will result to poor face and inefficient use of the explosives.

Adjusting stemming is another simple way of increasing advancement. The ideal size for rock based stemming is one eight the drill hole diameter. Too fine a stemming material won't hold whist too coarse a stemming introduces flock. According to Hemphill (1981), angular fragment can help prevent the stemming from blowing out. Wet stemming does not hold well though as water acts as lubricant when placed under pressure. As a general rule, increasing stemming length will increase the fragmentation of the rocks, as energy result explosion can be used more efficiently. However, if the stemming is too long, the uncharged length of the drill hole is increased thereby reduce available energy for advancement (Worsey, 2001). Drilling shallower holes and better distribution of explosive charges will prevent this problem (Gregory, 1984). Worsey (2001) stated that the use of stemming plugs is more effective than conventionally stemming and allows for better explosive distribution. Gregory (1984) prefers the use of short delay detonator but gives no reason. Whereas Worsey (2001) noticed that increasing the delay will improve fragmentation, as this will prevent scabbing of rock from adjacent drill holes. He then goes on to suggest that increasing the delay between rows allow the maximum time for relief on the free face and thus improve blasting performance. Henrych

(1979) recommended the use of millisecond delays, as this will allow for constructive interference of shock waves caused by adjacent explosion and in effect improves fragmentation. Langerford and Kihlstorm (1978) suggested that 10 ms is optimum for even fragmentation, which also give best fragmentation.

Reducing the drill holes diameter without changing the burden or spacing has effect on reducing the powder factor, which in turn reduces the fragmentation. However, increasing the drill holes diameter without increasing explosive diameter has effect of reducing the efficiency of the efficiency of the drill holes as the coupling between the explosive and the rock is poor. Whilst water coupling would improve the performance, it is unlikely to compensate for the overall reduction in efficiency; often the drill hole diameter is used as the basis for calculating the burden and spacing for blast pattern. For these designs, smaller drill holes will improve fragmentation. As a general rule, small drill holes with small spacing produce small rock, large drill holes with large spacing produces large rocks (Worsey, 2001).

The most important considerations in determining the drill hole diameter for a blast include the geology; charge diameter, restrictions due to built up areas and production factors (Gregory, 1984).

The type and amount of explosives can affect fragmentation obviously, higher powder factors will increase fragmentation and however the type of explosive is also important. Hemphill (1981) stated that the Velocity of Detonation (V.O.D) is important in the development of radial fractures in the initial stages of fragmentation.

The faster the V.O.D, the more radial fractures that are formed, more fractures mean smaller blocks are created, improving fragmentation explosive with high V.O.D's, however, tend to produce little gas, which reduce fragmentation.

Stagg *et al.* (1990) found that there was an upper limit to the powder factor where it no longer had impact on the fragmentation size. It was found that when the powder factor exceeds 0.84 kg m^{-3}, the amount of finer sized material (<70mm) remain constant at 60%, however, if the powder factor is reduced, the average particle size continue to increase and the optimum burden to spacing ratio was found to be 1.4. As with Langerfords and Kihlstorm (1978) formula, though the data was acquired using bench blasting, it can only be used as a guide for confined blasting. In confined blasting, such as Tunnel and underground mines two key topics are suggested as being the most important parameters namely water and the nature and orientation of discontinuities and zone of weakness.

As mentioned, the presence of water will impact the type of explosive used in the blasting as well as the performance of the selected explosive. The main impact of water on explosive performance is that of coupling. Explosive coupling is analogous to conducting heat through wall of container. When packaged explosive are placed in a drill hole, they leave an annulus between explosives and the rock face. Coupling is the ability of the media filling the annulus to conduct the energy from the explosive, through the annulus into the rock (Worsey, 2001). The better the coupling, the more of the explosive's energy is going to be spent breaking the rock.

Worsey (2001) stated that there are two reasons why water coupling improves blast performance. Firstly, water coupling increase the magnitude of the rock wave that transmitted from the explosive to the rock face. This enhances fragmentation though it does not increase the ground vibration due to the blast. Secondly, water is an incompressible. Consequently, less energy will be required to compress the annulus media than it would in air and less gas pressure would be lost through discontinuities and jointing. This in turn increases the pressure within the hole and increases the amount of radial fracturing.

The presence of joint within a rock mass will have impact on the advancement, for highly jointed rock, small blocks will be present so the explosive need only leave the jointed rock rather that break it. Explosive with low V.O.D's and high gas pressure should be used in this instance. If, however, there is relatively few joints, much larger blocks will be formed, higher V.O.D's explosives produced high shock should be used in these instance as more work will be required to break up the blocks, rather than move them (Worsey, 2001).

Discontinuities act as natural presplits in blasting and can be used to improve the performance of the blast. Horizontal bedding allows pulls to be maximized, as the blast will tend to split horizontally, rather cratering. This removes the need for sub drilling. Horizontal bedding can, however, introduce cap rock. If the stemming length is greater than the bedding thickness, the problem is signified.

Worsey (2001) stated that the rock is broken by tensile stress, hence the uniaxial compressive stress has little to do with the effect of blasting. Brady and Brown (1983) however, disagree with above. Both Worsey (2001) and Brady and Brown (1983) agreed that young's modulus is another mechanical property that will affect the blast performance. Brady and Brown (1983) state that is indicates the rock's capacity to transmit energy.

The explosive energy distribution analysis introduced by Kleine can help accesses distribution of the explosive charge in a plane or volume of the rock mass.

It has generally being applied to optimize design geometries and the charging conditions of ring pattern (Onederra, 2001).

A preliminary 30 explosive energy distribution analysis of the burn cut indicated that the possible contributor to detonation failure was hole deviation and for poor collar positioning. In some cases, holes had been collared too close to one another resulting in localized high-energy zones. This confirmed the importance of accurate hole collaring and drilling. The overall analysis helped define the critical distance between child holes in the burden cut to avoid the incidence of hole dislocation and or explosive desensitization. Immediate improvements in results were achieved.

MATERIALS AND METHODS

Drilling operation: The drilling operation is carried out with the aid of two types of drilling machines: Two booms plutons Hs2F drilling machine (FP22) and 2 booms Jumbo FC3.

Both types are diesel powered. The jumbo FC3 has drilling height of 10 m and maximum drilling width of 9 m. These machines use rotary mechanism for drilling horizontal or inclined holes of about 38-40 mm in diameter with a depth of 3.2 m. The flushing medium used is water. The cross sectional area of the working face is 6×3 m in dimension and it is expected to advance 3 m ahead in the ore deposit after blasting.

Blasting operation

Explosive: The type of explosives in used in Akouta deposit is Emulstar 8000 U.G. It is an emulsion explosive cartridge of 30 mm diameter and 32 cm long. It has the following characteristics as shown in Table 1-3.

Two sequences of the high intensity electric detonators are used during the charging process for the initiation of the Emulstar cartridges. The Short-Period (SP) delay detonators driving 25 millisecond delay interval between one another and long-period or ordinary delay detonators with 0.5 sec delay time. This is illustrated in Table 4.

Explosive charging: The 40 mm diameter and 3.2 m deep horizontal or inclined drilled holes are firstly cleaned with the aid of a tamping rod to ensure that the holes are not obstructed. This is followed by loading of explosive cartridges. The primer is charged first, then followed by 5 or 6 other cartridges in section C, B and A, respectively. These sections are as follows:

Section A: Represents the cut and cut spreader or easier holes. This section is charged with 7 cartridges including the primer.

Table 1: The measure characteristic

Explosives	Emulstar 8000UG
Average density	About 1.28 for diameter>40 mm About 1.25 for small diameter
Coefficient of self excitation (cm)	> 3
Choc sensibility (Joules)	> 1200
Velocity of Detonation (VOD) (m s^{-1})	
Diameter of 30 mm	About 5300
Diameter of 80 mm	About 5700
Average massic energy: cal g^{-1}	1150
Mg kg^{-1}	481
Critical diameter of detonation (mm)	<20

Table 2: Calculated characteristics

Explosives	Emulstar 8000UG
Volume of gas (1 kg^{-1})	710
Oxygen balance (g 100 g^{-1})	-3.4
Energy cal g^{-1}	1342
Mg kg^{-1}	5.61
Strength	1.10
Detonation pressure	Gpa
Diameter 30 mm	About 8.8
Diameter 80 mm	About 10.4

Table 3: Compression resistance characteristics

Resistance to static pressure	For diameter less than 40mm:> 30 bar For diameter greater than 40mm: 130 bar
Resistance to dynamic pressure	For diameter less than 40mm: about 200 bar For diameter greater than 40mm: about 300bar

Table 4: Detonator used

Short period	Long period
NO: Z=0	NO: I
„ 4	„ II
„ 8	„ III
„ 12	„ IV
„ 15	„ V
	„ VI

Section B: Represents the stopping holes with breakage upward and downward, wall or perimeter holes, floor or lifter holes and these limits the mineralized Part. This study is charged with 6 cartridges including the primer.

Section C: Includes roof holes and represents the non-mineralized part or the Gauge and is charged as section C.

Blasting: When all the connection is carried out (in series), the resistance of all the circuit is measured with the aid of an ohmmeter. The connecting line is divided in two distinct lines called primary and secondary line. The primary line, which is a soft white wire, has a resistance of 12Ω/100 m. The later is connected to the blasting machine called schapler exploder for firing.

Blast design in used at Akouta Mine: The principle behind this design is that an opening is blasted by means of a cut and then stopping is carried out towards the opening. With the designs in used in the company, the prime requirement of the company, which is to achieve the advance of 3m, could not be met. For this research work, a series of tests were carried out by changing specially the old cut or sometimes by modifying the charging procedure.

RESULTS AND DISCUSSION

The results obtained from the test conducted on the four different types of cuts are as shown in Table 5. The cut 13/17 gave minimum advance of 2.3 m and maximum of 2.6 m (Fig. 1-5).

Table 5: Summary of the results of the tests

Cut types	Test No.	Quantity of explosive	Number of holes	Advance
13/17	1	92.75	53	2.3
	2	71.75	41	2.3
	3	71.75	41	2.3
	4	71.75	41	2.5
	5	71.75	41	2.6
16/24	1	77.0	44	2.85
	2	77.0	44	2.80
	3	80.5	46	2.50
	4	105.0	60	2.20
9/13	1	64.75	37	2.90
12/20	1	70.0	40	0

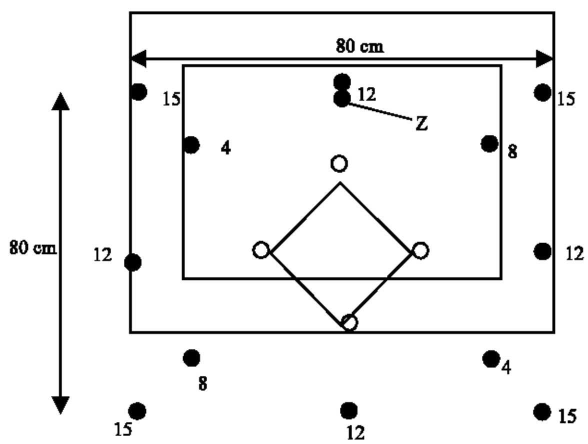

Fig. 1: Diagram of Test 1 for 13/17 cut designs

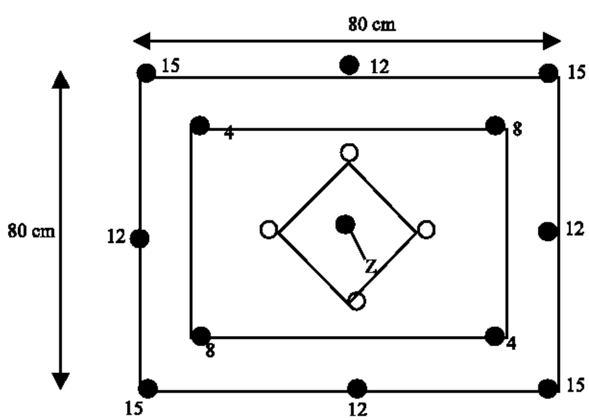

Fig. 2: Diagram of Test 2 for 13/17 cut designs

The cut 16/24 gave minimum advance of 2.2 m and maximum advance of 2.85 m (Fig. 6-9). The cut 9/13 gave advance of 2.90 (Fig. 10). While cut 12/20 gave advance of zero that is all the charges blew out (Fig. 11). From the result obtained for far, it can be seen that cut 9/13 will best suit the required advance for Akouta Mine.

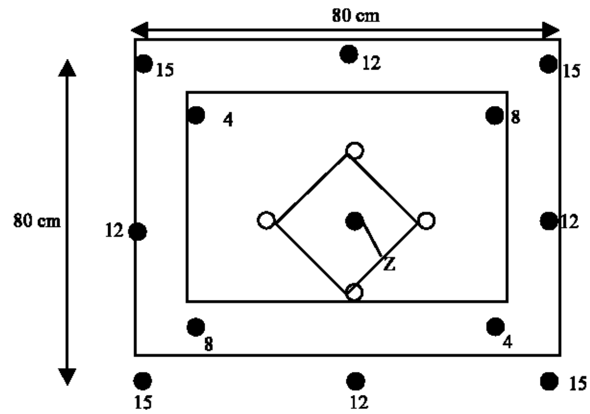

Fig. 3: Diagram of Test 3 for 13/17 cut designs

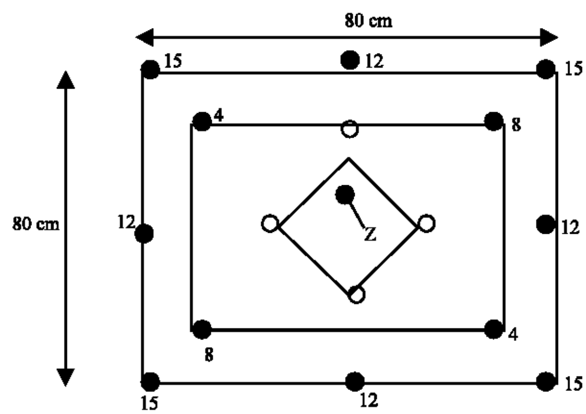

Fig. 4: Diagram of Test 4 for 13/17 cut designs

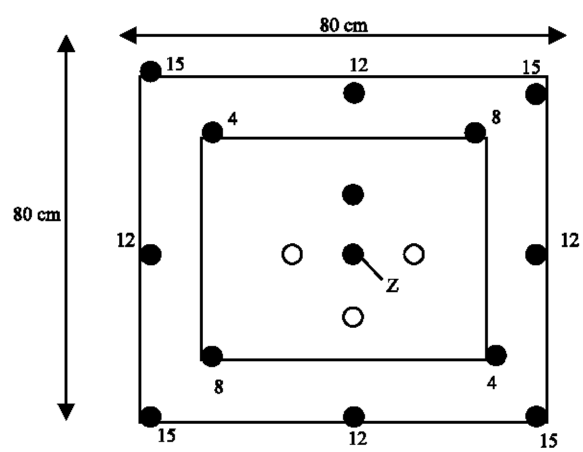

Fig. 5: Diagram of Test 5 for 13/17 cut designs

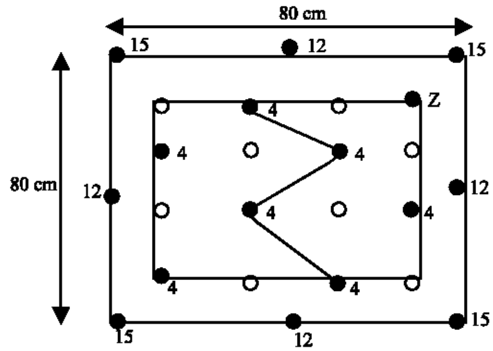

Fig. 6: Diagram of Test 1 for 16/24 cut designs

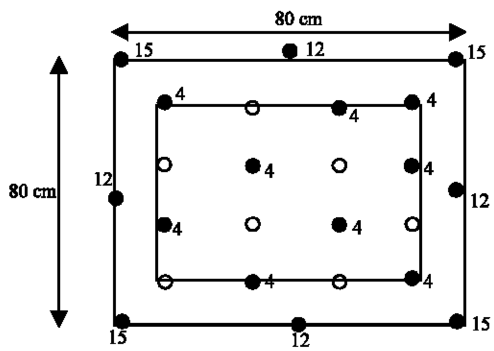

Fig. 7: Diagram of Test 2 for 16/24 cut designs

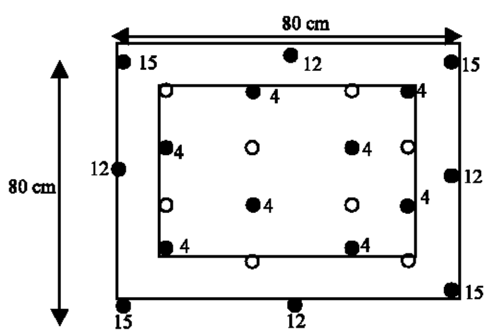

Fig. 8: Diagram of Test 3 for 16/24 cut designs

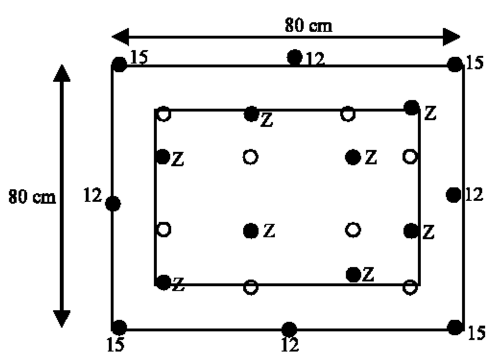

Fig. 9: Diagram of Test 4 for 16/24 cut designs

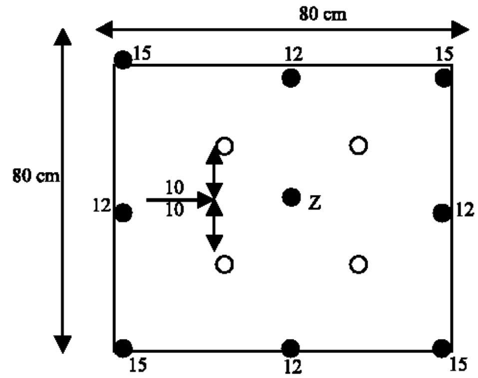

Fig. 10: Diagram of Test 1 for 9/13 cut designs

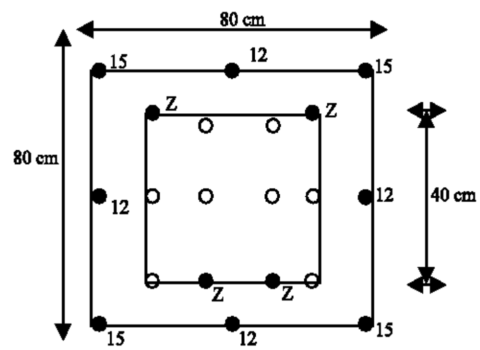

Fig. 11: Diagram of Test 1 for 12/20 cut designs

CONCLUSION

From the analysis of the results obtained, the following conclusions are made:

- Cut 13/17 upon all the modification can not give advance that is more than 2.6 m.
- Cut 16/24, though gave advance of 2.85 for test 1 can also result to advance of 2.2 if design of test 4 is selected.
- Cut 9/13 gave the best advance so far and this can still be improved on by further studies on the design for this particular cut.
- Cut 12/20 should be totally avoided, as it does not give any advance.

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