Explosives utilization at a Witwatersrand gold mine

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Paper written on project work carried out in partial fulfilment of BSc. (Mining Engineering)

Synopsis

Gold bearing deposits of the Witwatersrand basin are generally less than 2m thick and require conventional narrow-reef mining methods for extraction and employ explosives as a means of rock breaking. Optimal utilization of explosives is dependent on the overall design of the blast. The under-utilization of explosives arises when shot-holes are drilled inconsistently, overcharged, and when tamping is absent. This can be rectified by emphasizing the importance of good drilling practices as part of induction programmes and refresher courses. The project was aimed at determining whether or not explosives are being optimally utilized at project site. This was investigated through a study of the properties of explosives, mine standards, and recommendations for usage. Underground observations were made to determine whether or not mine standards were being adhered to. Historic data was obtained to establish the historic relationship existing between the quantity of explosives used (kg) and the production output (m^2). This was then compared to the quantity of explosives the mine expects to use per unit of production. The results obtained were analysed to determine the presence and extent of over- or under-utilization. It was found that explosives are being under-utilized at the mine. More explosives are ordered than expected per unit of production. The explosives' properties are not thoroughly exploited during blasting, thereby requiring the use of more explosives than prescribed.

Keywords

blasting practices, explosives utilization, blast design.

Introduction

The South African gold mining industry is based predominantly in the Witwatersrand Basin. The gold reefs found in this basin are generally less than 2 m thick and extend to depths in excess of 3 km below surface, with approximate dips ranging between 20 and 25° from surface (MRM, 2012, p.20). Mines that extract deposits of this nature are narrow-reef mines. The project site is one of these mines, and conventional drill-and-blast mining methods are employed. Hand-held pneumatic rock drills are used for face drilling, explosives are used to fragment the rock, and electric-powered scraper winch systems clean the working areas by removing broken rock from the face and tipping it to the orepass system through a system of in-stope tipping points.

The nature of the orebody and mining environment necessitates the use of explosives as a rock-breaking mechanism, thus making explosives an integral part of the mining cycle. Without them, production cannot take place.

Explosives utilization is the usage of explosives in a manner that yields the desired results and that exploits every aspect of their ability to break rock. In order to optimize the use of explosives, a thorough understanding of their properties, characteristics, rock-breaking mechanisms, and application is necessary. An understanding of the basic operational functions of explosives will encourage the implementation of techniques that lead to optimal utilization of explosives.

Objectives

The project is aimed at investigating the types of explosives in use at the project site and improving their utilization by at least 10% by determining the following:

- ➤ Factors contributing to explosives utilization
- ➤ Whether explosives are currently being optimally utilized
- ➤ The relationship between explosives and production
- ➤ Mine standard pertaining to explosives utilization
- ➤ Possible causes and consequences of over- or under-utilization of explosives
- ➤ How explosives utilization can be improved by at least 10%.

Explosives consumption

The mine has an expected broken rock output per unit of explosives used (de Sousa, 2013). The ratio of explosives used to production (centares) is obtained empirically using the following parameters:

- ➤ Length of drill steel: 1.2m
- ➤ Length of drill steel chuck: 0.3 m
- ➤ Length of hole: 0.9 m
- ➤ Drilling density: 4 holes per m^2
- ➤ Shock tubes: 4 tubes per m^2
- ➤ Panel length: 30 m
- ➤ Panel width: 1 m

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 ISSN 2225-6253. Paper received Mar. 2015

- Burden spacing: 0.6 m.

One case of explosives (25 kg) breaks rock over a span of 10 m2. A 30 m panel would therefore require 75 kg of explosives and 120 shock tubes.

The quantity of explosives required per panel in kilograms would then be obtained from the sum of the explosives used in shot-holes and those used in preconditioning holes:

Shot-holes:

1 case = 10 m2

3 cases = 3 m2

Therefore 75 kg explosives would be required for 30 m2

= 2.5 kg/m2

Preconditioning holes:

Nine preconditioning holes are expected and there are three cartridges per hole. A 25 kg box of explosives contains 100 cartridges, each with an approximate mass of 0.25 kg.

The mass of explosives in preconditioning holes for the entire panel is

$$\frac{0.25\times 3\times 9}{30^2}$$
$$=0.225 \text{ kg/m}^2$$

The total explosives mass required for a panel is the sum of the mass for the shot-holes and of the mass for the preconditioning holes, which equates to 2.725 kg/m2.

It is important to note that this method of calculating the approximate quantities of explosives required to produce the expected output is based on the following assumptions:

- Face preparation, drilling, charging, and timing are per mine standard
- Panel length is maintained at 30 m and stoping width kept constant
- Secondary blasting is neglected
- Blasting of the gullies is not accounted for.

Blast design

Optimal explosives utilization is dependent on the overall blast design (de Beer, 2013). It is important to ensure that face preparation, drilling, and charging are done correctly.

Face preparation

Blast designs may vary for various reasons, one major reason for this being the stope width. The distance between blast-holes, also known as the burden spacing (G), can be obtained as follows:

$$G=\sqrt{\frac{M_c}{K}}$$
$$=\sqrt{\frac{kg}{m}\div\frac{kg}{m^3}} \qquad [1]$$
$$=\sqrt{m^2}$$
$$=m$$

where

M_c = mass of explosive per metre of blast-hole (kg/m)

K= powder factor (kg/m3).

The explosives in use have a density of 1.15 g/cm3. Using the expression $M=\rho V$, the mass of explosives contained in a hole and subsequently, a panel can be obtained. Underground observations carried out on the western panel of the 16th level, 31st crosscut are used below to derive the burden spacing of 60 cm as per mine standard.

The panel has on average 66 shot-holes and 7 preconditioned blast-holes. The total mass of explosives contained in the panel is obtained as follows.

For the shot-holes:

$$M_p=\rho V$$
$$=\rho\pi\times l\times r^2 \qquad [2]$$
$$=1.15\times\pi\times 27\times 1.35^2$$
$$=177.778 \text{ g}$$

where

l = length of priming cartridge (cm)

ρ = density of cartridge (g/cm3)

R = effective radius of shot-hole (cm).

$$M_s=\rho V$$
$$=\rho\pi\times l\times r^2$$
$$=1.15\times\pi\times 31\times 1.35^2$$
$$=204.116 \text{ g}$$

where

l = length of column charge cartridge (cm)

$$M_{T1}=66\left[M_s+M_p\right]$$
$$=25\ 205.004 \text{ g}$$

For the preconditioned holes:

$$M_p=\rho V$$
$$=\rho\times\pi\times l\times r^2$$
$$=1.15\times\pi\times 27\times 1.35^2$$
$$=177.778 \text{ g}$$

$$M_s=\rho V$$
$$=\rho\times\pi\times h\times r^2$$
$$=1.15\times\pi\times 31\times 1.35^2$$
$$=204.116 \text{ g}$$

$$M_{T2}=14M_s+7M_p$$
$$=(14\times 204.116)+(7\times 177.778)$$
$$=4102.07 \text{ g}$$

Therefore, the total mass of explosives in the panel is

$$M_T=M_{T1}+M_{T2}$$
$$=29\ 307.206 \text{ g}$$

In order to determine the burden spacing, one needs to take into account the powder factor. This is the mass of explosive required to break one cubic metre of rock, and is calculated using the expression:

$$K=\frac{M_T}{Volume\ of\ rockblasted}$$
$$=\frac{29.307}{0.8\times 1.0\times 20}\text{ kg/m}^3 \qquad [3]$$
$$=1.22 \text{ kg/m}^3$$

The burden spacing is given by

$$G=\sqrt{\frac{M_c}{K}}$$

M_C is the mass of explosives contained per blast-hole. The value has been derived by dividing the total mass contained in the shot-holes by the number of shot-holes,

$$\frac{M_{T1}}{66} = 381.894 \text{ g}$$

The length of the blast-hole that actually contains the explosive is then found. Since the holes are 1.2 m long and the total length of the combined cartridges is 580 mm, only 1.2 × 0.580 m is fitted with explosives. As a result, the mass of explosive contained per blast-hole becomes

$$M_c = \frac{381.894}{0.696}$$
$$= 0.5487 \text{ kg/m}$$

Substituting this and the K value into the burden spacing equation yields:

$$G = \sqrt{\frac{M_c}{K}}$$
$$= \sqrt{\frac{0.5487}{1.22}}$$
$$= 67.1 \text{ cm}$$

This is the maximum burden that the explosives can effectively handle.

Drilling

The mine has regulatory policies (mine standards) for all activities carried out during the ore extraction process, which should be adhered to at all times. The mine standards for drilling are as follows:

- ➤ All drill-holes must be drilled on the position marked on the face and aligned underneath the direction line
- ➤ Holes are to be drilled to the full length of the drill steel
- ➤ All the holes marked on the face should be drilled ensuring that each hole has the same burden to break
- ➤ Holes must be drilled at an angle no less than 75° to the face
- ➤ Temporary support is to be installed prior to commencement of drilling.

Charging

The mine standards prescribe the following when charging up and blasting (de Beer and Ross, 2012):

- ➤ The primer is prepared by inserting the metal end halfway into the cartridge. This should be done in a safe, approved priming bay away from the blast site to minimize the risk of accidental firing, which could be caused by stray currents or electromagnetic radiation
- ➤ Blast-holes are to be de-sludged using an aluminium 3-way blowpipe and an approved scraper wire. Safety goggles are to be worn at all times when de-sludging blast-holes
- ➤ Explosives should then be transported to the working face in elephant bags. The cartridges and accessories should be transported separately in approved containers (elephant bags)
- ➤ The primer should be inserted into the hole first and pushed to the bottom of the hole using a square-ended charging stick
- ➤ The column charge is then inserted into the blast-hole.

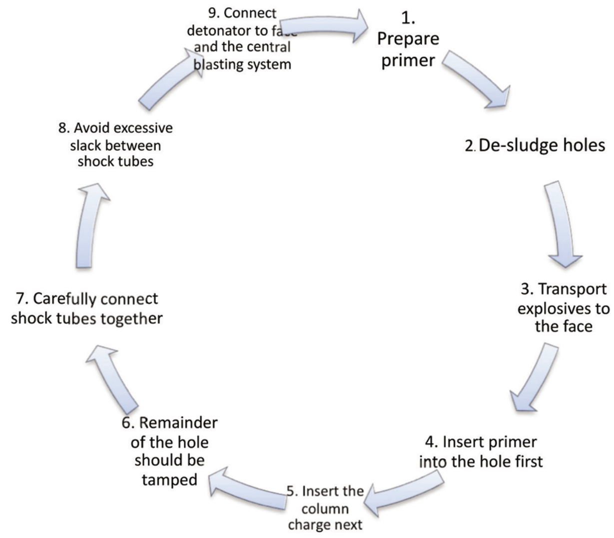

Figure1 – Mine standards for charging and blasting

Proper coupling should be ensured by pushing the column charge as far into the hole as is possible without damaging it

- ➤ The remainder of the hole should be tamped to contain gases inside the hole using clay tamping provided by the mine
- ➤ Shock tubes should then be carefully connected to each other. The connector blocks should be more than 10 cm apart
- ➤ Excessive slack between the shock tubes should be avoided in order to prevent whiplash and damage
- ➤ Lastly, the shock tube starter is connected to the charged face, and this is connected to the central blasting system, which is controlled from the control room on surface.

Results

The results presented include historic results obtained from the explosives supervisor and observations recorded underground during the project. The expected explosives utilization is calculated based on the ratio used by the mine – 2.725 kg per m^2. Ordered explosives are calculated based on order and delivery forms obtained from the mine, and the ratio obtained by dividing the mass of explosives used by the production throughput for the month.

Historic data

The data here enables a direct comparison to be made between the planned and actual explosives consumption, based on the planned production output (obtained from the mineral resource management (MRM) department) and the actual production output (obtained from the production personnel at the shaft) for the period from September to December 2013. The graphs were constructed by comparison of the total planned and actual production in relation to the explosives quantities used.

Underground observations

Observations were made in two panels, on levels 18 and 16, to gain an understanding of the quantity of explosives used per blast and to determine whether blasting was conducted as per

Table I

Stoping production results for September 2013

Panel	Total production (m²)	Miner	Explosives expected (kg)	Explosives ordered (kg)	Shock tubes expected	Shock tubes ordered
V1	127	A	346	525	508	300
V2	190	B	518	300	760	300
V3	102	C	278	500	408	900
V4	145	C	395	200	580	0
V5	203	D	553	1150	812	1010
V6	99	E	270	750	396	400
V7	37	E	101	0	148	0
V7	0	F	0	275	0	300
V8	89	G	243	500	356	0
V9	86	G	234	250	344	700

Table II

Stoping production results for October 2013

Panel	Miner	Total production (m²)	Explosives expected (kg)	Explosives ordered (kg)	Shock tubes expected	Shock tubes ordered
V1	A	190	517.8	150	760	0
V10	B	128	348.8	300	512	100
V11	C	95	258.9	300	380	0
V11	C	153	416.9	600	612	300
V13	E	0	0	600	0	0
V5	D	185	504.1	0	740	0
V14	D	0	0	600	0	600
V6	E	93	253.4	0	372	0
V7	E	156	425.1	0	624	0
V7	F	130	354.3	275	520	200
V8	G	202	550.5	0	808	0
V15	G	51	139.0	450	204	300
V16	H	0	0.0	125	0	100

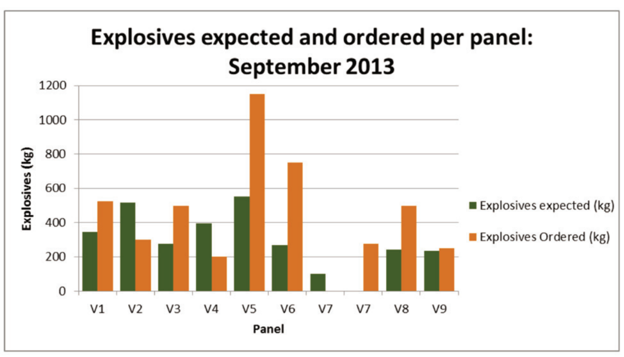

Figure 2 – Comparison of expected and actual quantity of explosives ordered for September 2013

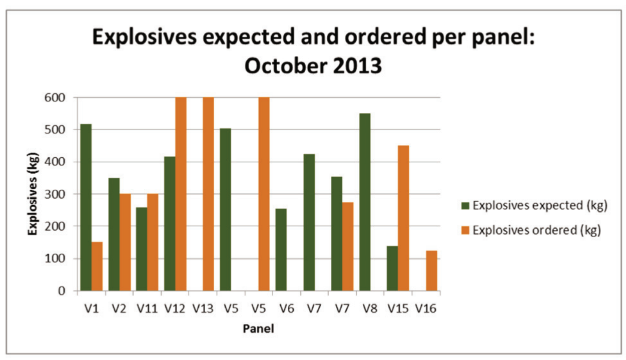

Figure 4 – Comparison of expected and actual quantity of explosives ordered for October 2014

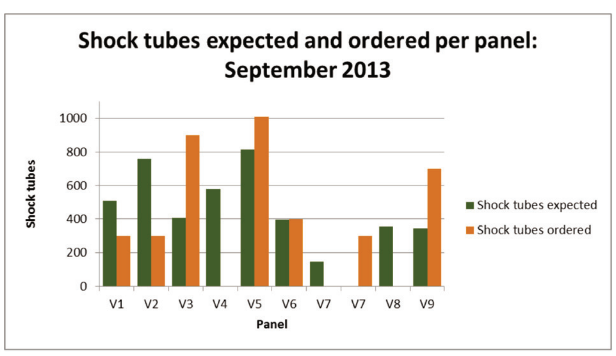

Figure 3 – Comparison of expected and actual number of shock tubes ordered for September 2013

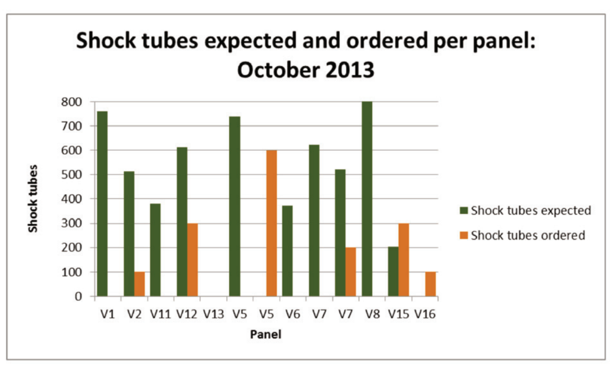

Figure 5 –Comparison of expected and actual number of shock tubes ordered for October 2013

Table III

Stoping production results for November 2013

Panel	Miner	Total production (m^2)	Explosives expected (kg)	Explosives ordered (kg)	Shock tubes expected	Shock tubes ordered
V1	A	0	0	550	0	300
V17	A	0	0	0	0	0
V7	B	97	264.33	275	388	0
V11	C	148	403.3	200	592	200
V11	C	228	621.3	750	912	300
V5	D	243	662.18	250	972	200
V18	F	0	0	0	0	0
V7	F	107	291.58	250	428	200
V19		6	16.35	0	24	0
V8	G	164	446.9	0	656	0
V15	G	77	209.83	300	308	100
V16	H	0	0	125	0	0
V20	H	40	109	125	160	100
V10	B	102	277.95	50	408	100
V21	I	0	0	125	0	0
V14	D	0	0	800	0	300
V22	J	0	0	600	0	0
V23	B	0	0	50	0	0

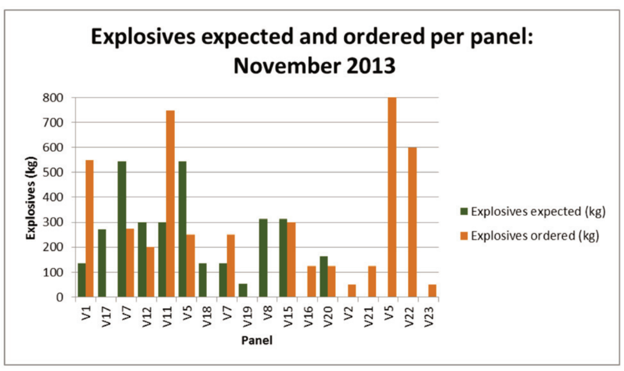

Figure 6 – Comparison of expected and actual quantity of explosives ordered for November 2013

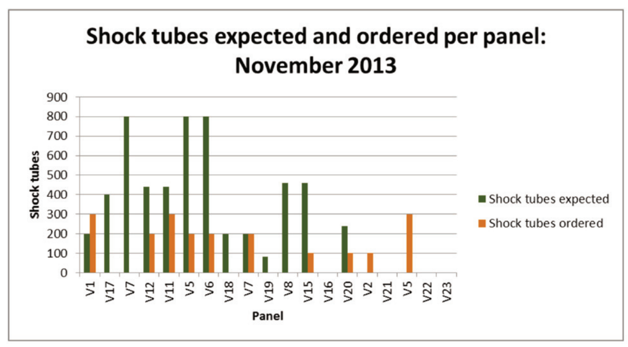

Figure 7 – Comparison of expected and actual quantity of shock tubes ordered for November 2013

Table IV

Stoping production results for December 2013

Panel	Miner	Total production (m^2)	Explosives expected (kg)	Explosives ordered (kg)	Shock tubes expected	Shock tubes ordered
V1	A	86	235	0	344	200
V17	A	39	106	0	156	0
V7	B	105	286	225	420	0
V3	C	68	185	0	272	0
V4	C	152	414	0	608	0
V5	D		0	100	0	0
V6	B	180	491	0	720	0
V18	F	32	87	0	128	0
V7	F	98	267	125	392	100
V24	D	83	226		332	
V8	G	0	0	50	0	100
V25	G	0	0	0	0	0
V15	G	0	0	200	0	100
V16	H	0	0	100	0	0
V20	H	75	205	0	300	0
V22	J	274	747	0	1096	0
V11	C	0	0	200	0	100
V21	I	0	0	200	0	200
V11	C	0	0	250	0	100
V14	D	0	0	400	0	600
V22	D	0	0	275	0	100

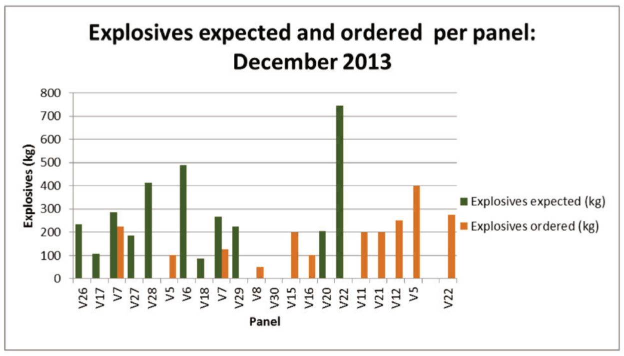

Figure 8 – Comparison of expected and actual quantity of explosives ordered for December 2013

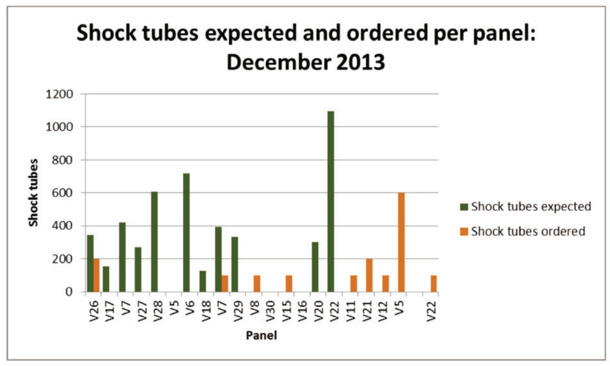

Figure 9 – Comparison of expected and actual number of shock tubes ordered for December 2013

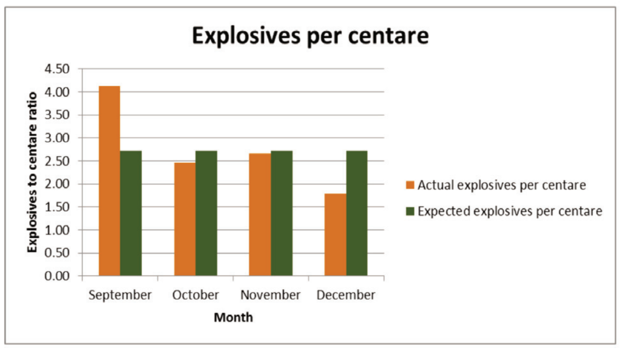

Figure 10 – Comparison of expected and actual explosives used per centare

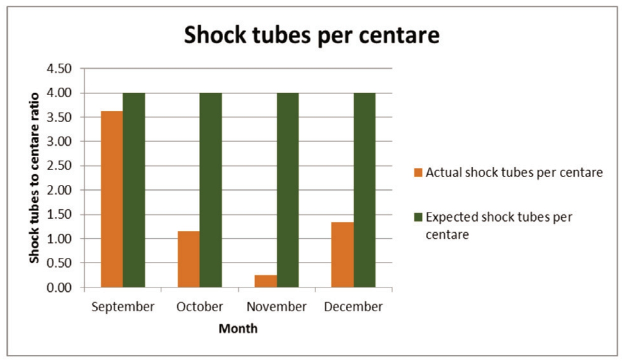

Figure 11 – Comparison of expected and actual number of shock tubes used per centare

Table V

V2 breast panel

Panel characteristics	Week 1	Week 2	Week 3	Average
Panel length (m)	20	20	19	19.7
Stoping width (m)	1.2	1.1	1.1	1.1
Number of marked holes	77	76	70	74
Number of preconditioned holes	7	7	6	7
Average burden spacing (cm)	58	60	60	57
Number of cartridges used	200	200	180	193
Number of shock tubes used	100	100	100	100
Advance (m)	0.8	0.8	0.8	0.8

Table VI

V20 wide raise

Panel characteristics	Week 4	Week 5	Week 6	Average
Panel length (m)	11	11	15	13
Stoping width (m)	1.2	1.2	1.2	1.2
Number of marked holes	42	45	40	43
Number of preconditioned holes	3	3	3	3
Average burden spacing (cm)	58	60	55	58
Number of cartridges used	100	100	100	100
Number of shock tubes used	50	50	50	50
Advance (m)	0.8	0.76	0.8	0.79

mine standard, as well as to investigate the effects of not following the mine standard. Underground observations were limited to two panels because monitoring of the input and output parameters and subsequent analysis was to be done over a series of blasts to increase the accuracy of the results.

Observations were recorded for each shift spent in the respective working places. The weekly averages were then calculated and from these Tables V and VI were compiled. The number of marked holes is inclusive of the holes marked to blast the gullies in both cases, but excludes the contribution of secondary blasting.

Analysis of results

In September 2013, the expected explosives utilization was exceeded by 51.3%. This was calculated by direct comparison of the explosives ordered and the production throughput (Table I). The general trend for the month was that more explosives were ordered than expected. Eleven cases of explosives and 300 shock tubes were ordered for panel V7, yet there was no production from that panel. The mine records this as explosives unaccounted for (wasted). Upon investigation, it was found that the 16th level is seismically active with bad ground conditions. This particular panel had been badly affected by a seismic event and had been closed, and the miner was assisting in panels V2 and V7, since all three panels are on the same working level. Panel V2 ordered only 58% of expected explosives, and V7 ordered no explosives. Possibly, the first miner was placing explosives orders for the two panels he was assisting in. An explosives order may only be placed by a miner for a workplace officially assigned to him (de Sousa, 2013). Therefore, 11 cases and 300 shock tubes can be accounted for. The remainder of the panels ordered more explosives than expected, and the possible reasons for this are discussed in detail later.

The results obtained for October indicate that more explosives were ordered than expected. There were again panels that received explosives yet showed no production. In this case, V5 and V14 were under the administration of the same miner who received 24 cases of explosives for one panel that were actually intended for another panel. The same applies to panels V13 and V6.

During the month of November, six panels ordered explosives with no production throughput confirming where they have been used. No relationship can be established between panels that ordered explosives without producing and those that produced without ordering explosives. A total of 90 cases of explosives and 600 shock tubes were ordered and these remain unaccounted for. Nothing can be said about their utilization and these explosives can be concluded to have been wasted. December shows the same trend- explosives were ordered yet nothing produced.

Occurrences of November and December are, for the purposes of this report, extreme cases that have required extensive research and enquiries about exactly what happened during that period. The remainder of the cases are those where more explosives were ordered than were expected by the mine.

The ratio of explosives (kg) to production output (m^2) expected by the mine is 2.725:1, and 4:1 for shock tubes. Figures 10 and 11 indicate the performance of the mine in relation to the expected figures. Variations in the ratios are evident, indicating cases of both over- and under-utilization of explosives. Over-utilization occurred when fewer explosives were used than expected, and under-utilization when more explosives were used than planned. The contributing factors to both over- and under-utilization of explosives, based on mine standards and underground observations, are discussed in detail below.

Inconsistent blast-hole length and drilling angle

Underground observations made revealed that at times, the blast-holes are drilled to a shorter length than specified in the mine standard. The impact of shorter blast-holes is demonstrated using the following simple example.

The ideal case (according to mine standard), assuming a 30 m long panel with a 1m stoping width, is as follows:

- Blast-hole length: 0.9 m
- Advance per blast: approx. 0.8 m
- Explosives used per blast: 29 307.21 g
- Advance over 20 blasts: 16 m.

The effect of short blast-holes can be seen from the following calculation:

- Blast-hole length: 0.85 m
- Advance per blast: 0.75 m
- Explosives used per blast: 29 307.21g
- Advance over 20 blasts: 15 m

When blast-holes are drilled shorter than prescribed by the mine standards due to incorrect drilling angles, the advance is reduced although the same quantity of explosives is used as for the full-length blast-holes. This results in under-utilization of explosives because the full potential of the explosives is not used. The calculation above (case 2) is exaggerated slightly because it assumes that all holes in the panel are drilled at 0.85 m length. However, this calculation demonstrates the effect of shorter blast-holes on the utilization of explosives. In addition, if blast-holes are drilled to insufficient lengths, 4.8 cm of face advance is lost per blast (de Beer, 2013).

Incorrect burden spacing

For every 10 cm increase in burden spacing, 10% face advance is lost per blast (de Beer 2013). A burden spacing of 60 cm ensures optimal fragmentation, due to the interaction between adjacent charges.

When the burden spacing is increased, the explosive energy needs to travel further than 0.3 m to effectively break rock from the adjacent blast-hole. Thus the explosive energy is depleted before optimum fragmentation is achieved. This results in poor hangingwall and footwall conditions and an uneven face shape, as well as over-utilization of explosives.

If the burden spacing is reduced, the explosive energy released is more concentrated, leading to finer fragmentation of the rock mass, but also to overbreak of the hangingwall. Any deviation from the prescribed burden spacing results in approximately 10% overutilization of explosives, an uneven face shape, and poor fragmentation.

Overcharging

Much of the explosives energy concentrated in the blast-hole is not evenly distributed but is concentrated within the confines of the surrounding rock mass. As has been observed underground, there is a misconception that overcharging is beneficial to the advance achieved. However, when more cartridges are placed in a blast-hole than the quantity required, more energy is released into the blast-hole. This energy, if tamping is sufficient, causes both overbreak and fine fragmentation, as were as over-utilization of explosives.

Drill bit deterioration

The drill bits used in the working places are 34 mm in diameter. According to Jijingubo (2013) deterioration due to wear and tear results in the gradual reduction of the drill bit diameter, thus causing a reduction in the diameter of the blast-hole. Jijingubo suggested that this reduces free movement of the cartridge inside the blast-hole, thus rendering explosives less effective than they would be when using fairly new drill bits.

Poor or no tamping

The importance of tamping should not be underestimated. Underground observations showed that adequate tamping of blast-holes is often neglected when charging up, especially close to the end of the shift. Figure 12 illustrates the importance of tamping.

Explosive energy released into the blast-hole uses two primary mechanisms for rock fragmentation: shock and heave. For effective fragmentation, the explosive energy should be contained in the blast-hole long enough to cause expansion of the cracks induced by the shock mechanism. Tamping aids in this regard by enabling the explosive itself and the energy it releases to remain in the blast-hole and cause expansion as the gaseous products from detonation penetrate the induced cracks. The absence of tamping or even poor quality installation of tamping allows the gas to escape and hence the energy is released into the surrounding environment. This sometimes causes damage to permanent support elements and overbreak, because the energy is not fully released into the blast-hole but is allowed to escape to other areas where it is not desired.

Figure 13 illustrates the effect of tamping on face advance. Because the absence of tamping allows gases to escape, the end of the blast-hole is often not blasted, leaving sockets behind and subsequently reducing the advance achieved per blast. For 0.9 m holes, no tamping results in 12 cm loss per blast (de Beer, 2013).

Unused cartridges and shock tubes remaining at the face

The mine standards require that unused explosives and accessories be returned to the explosives box and locked away. Strict explosives control policies are employed at the mine – all explosives should be accounted for. The miners keep a record of the quantity of explosives and accessories in storage, and upon receipt of a new batch the quantities are adjusted accordingly. A record of explosives used is to be kept as well. Underground observations proved non-compliance to this requirement, since in both panels observed, no unused explosives were returned to the explosives boxes, and it was assumed that all explosives and accessories taken into the face were used.

Blasting of gullies and secondary blasting unaccounted for

The mine standards require that gullies be blasted such that they lead the face. This is to ensure that the ore blasted has a free face to break into. The centre gully should always lead the face, while following the survey line pegs (Figure 14). In practice, the quantity of explosives used to blast an entire panel includes the explosives used to blast the gully, as well as the face. However, the means of determining the quantity of explosives required per square metre does not distinguish explosives used for blasting gullies. Thus the results over-estimate the utilization of explosives to blast the face, whereas some of these were used to keep the gully ahead of the face. Blasting of the gullies is such an important aspect of production that this usage should be allocated an explosives consumption factor.

The blasting of gullies in a 20 m panel entails five blast-holes and would consume approximately 12–15 cartridges, 5–7 shock tubes, and detonating cords. This may appear insignificant, but it increases the amount of explosives used while not contributing to production. The resulting higher-than-expected explosives utilization factor can be corrected by including the explosives used to blast gullies (centre and strike) in the planning of the quantity of explosives expected to be used in a panel.

Secondary blasting is usually unaccounted for when allocating explosives to working places. This is done when removing obstructions such as large rocks from grizzlies and when blasting bad hangingwall conditions, including brows. Unlike the case of blasting gullies, secondary blasting is used only irregularly and is considered a result of poor primary blasting. However, it is a contributing factor to the apparent over-utilization of explosives.

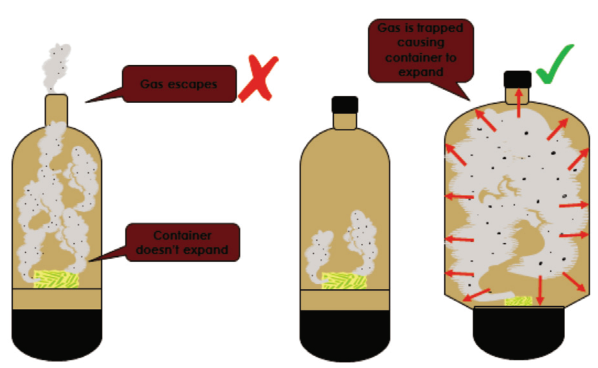

Figure 12 – The effect of tamping on explosives effectiveness (de Beer, 2013)

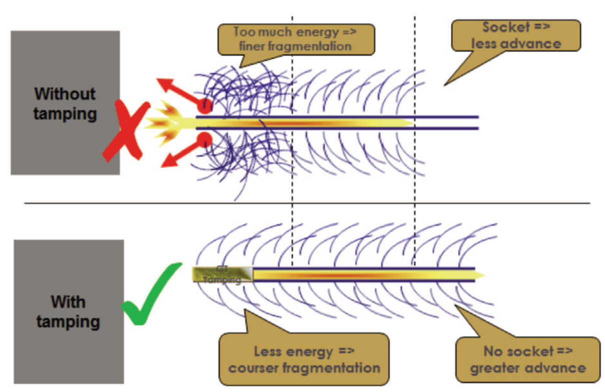

Figure 13 – The effect of tamping on face advance (de Beer, 2013)

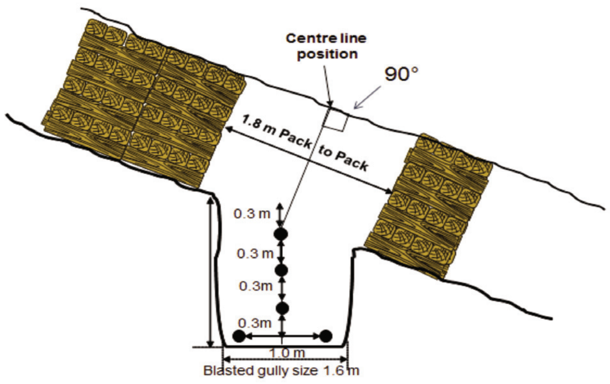

Figure 14 – Marking of the centre gully

Limitations of the record-keeping/monitoring system

The mine has a record-keeping system in place in which all miners order their explosives and accessories for specific workplaces. As can be seen from the results obtained, some workplaces have placed orders for explosives while there is no production to account for the usage. However, it is common for a miner in charge of multiple panels that are relatively close to each other to order explosives for panel A but use them to blast panel B. Comparisons of the expected and actual quantity of explosives used per panel exaggerate the extent of the problem, since the trade of explosives between panels is not taken into account.

Conclusions

Explosives are a vital component of hard-rock mining operations using conventional mining methods for ore extraction. Mine standards are in place to ensure that all activities involved in the production process are carried out in a way that ensures employee safety and maximizes production output. This study indicates that explosives are not being utilized to their full capacity at the mine. The biggest contributor to the apparent under-utilization of explosives is the limitations of the system that tracks the usage of explosives underground.

The system does not allow a miner to order explosives unless they are for a specified panel officially assigned to him. There are currently no means of determining how much of the ordered explosives is actually used underground and how much is returned to the explosives boxes. Other factors contributing to under-utilization of explosives are directly related to the overall blast design. These include, but are not limited to; overcharging, incorrect drilling lengths and drilling angles, secondary blasting not being accounted for, as well as the somewhat impractical expectation of explosives consumption that the mine currently has. Under-utilization of explosives also leads to poor ground conditions and increased costs because the mine has to purchase more explosives than required yet the production output remains unchanged.

The utilization of explosives can be improved by implementing changes in the explosive ordering process and providing a means of tracking whereby ordered explosives are used. By so doing, no explosives will be unaccounted for and the utilization problems encountered in the stopes can be addressed with a realistic picture of the extent of under-utiliation.

Recommendations

The results of this project indicate that the current system used by the mine to calculate the amount of explosives that should be used per square metre has the following limitations:

- It is based on a panel length of 30 m, which is not the average panel length for the shaft
- Blasting of gullies and secondary blasting is not accounted for when calculating the expected explosives consumption
- Once explosives are delivered to the miner, no further records are kept of their distribution among the various working places
- Miners are permitted to order explosives only for the panels officially assigned to them. The system assumes no trading of explosives takes place between miners.

These limitations exaggerate the extent of explosives unaccounted for and the extent of under-utilization. In order to improve the utilization of explosives, it is important that explosives are used to obtain the best results and not under-estimated. The mine can apply the following measures to improve the utilization of explosives.

The explosives usage calculator

The explosives usage calculator can be introduced into the system to aid in monitoring of explosives usage underground. This form (Figure 15) would be made available together with the explosives order form. After blasting, the form should be inserted into the communication book at the end of the shift. The availability of this information is aimed at encouraging the miner to directly monitor explosives usage and compare it to that which is expected.

Adjustment of consumption parameters

Planning for explosives consumption at the mine is somewhat unrealistic. The benchmark of 2.725 kg/m^2 is based on a panel length of 30 m and constant stoping width. This is not a true reflection of the mining conditions, since pillar extraction is the predominant mining method and the panel lengths are constantly adjusted owing to ground conditions and intersection of geological structures (Tsibuli, 2013). Instead of a fixed benchmark, the mine can employ a consumption calculation method that allows for flexibility due to changing localized conditions and accounts for the blasting of gullies as well as secondary blasting. This will present a practical model from which consumption parameters can be calculated and reduce the apparent extent of under-utilization, thereby improving utilization in future.

Training

Formal training

Scientific details of rock-breaking should be included in induction programmes and refresher courses to broaden the knowledge of explosives handling personnel and help them understand the importance of a 60 cm burden spacing. Miners should constantly be reminded that overcharging is in no way beneficial to mining operations. In addition, employees should be informed and constantly reminded about the financial implications of face advance loss per blast and how this affects them.

Explosives Usage Calculator(Stope)

Shaft/ Section:		
Working Place/Shift Overseer:		
Date:		

General Information

Shock tubes per case		
Explosives per case (kg)		
Detonators per Case		

Input	Planned	Current
Length of Panel		
Number of Fuses per m of face		
Stoping width		
Advance per blast		
Number of Blasts per Month		

Output

Monthly Call		
Total number of shock tubes per month		
Total explosives mass per month		
Shock tube cases to be ordered per month		
Explosives cases to be ordered per month		
Shock tubes ordered per week		
Explosives cases ordered per week		
Detonators ordered per month		
Number of fuses used per blast		
Explosives used per blast (kg)		

Figure 15 – The explosives usage calculator

Informal training

Diagrammatic representations in the form of clearly visible laminated posters at waiting places and in the change houses informing employees about the impact of poor drilling practices on the centares they produce monthly and their inability to reach set targets.

Models made of rubber, clay, or any recyclable material displayed at various places in the shaft. These should be designed such they show the goal (reaching the mine call factor) and all the factors that prevent the set targets beting reached, such as incorrect burden, shorter shot-holes, poor tamping, overcharging *etc.* These factors could be represented by *e.g.* parasites feasting on the target – something everyone can relate to and work together against.

Introduction of light, flexible 60 cm long strings made of recyclable materials that can be folded into 10 cm or 5 cm portions. These would be made available to all stoping crews. The aim here is to involve the crew in adhering to a consistent 60 cm burden spacing, and holding the miner accountable for any inconsistencies, which can then be raised by the crew instead of production supervisors. This is an example of the bottom-up management approach.

Acknowledgements

EurIng C.R Beaumont: Project Supervisor
Mr C.K Tsibuli: Production Supervisor, Project Site
Mr D. Setshoantsho: Production Supervisor, Project Site
Mr R. de Beer: Blasting Design Engineer, Project Site
Mr B. Prout: Lecturer.

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