Optimization of blasting in narrow reef mines in South Africa

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ABSTRACT: A sequence of interventions was conducted at Kopanang Mine with the aim of optimizing the blasting process and thereby achieving enhanced workplace safety, improved explosive efficiency, reduced blasting cost and reduced emission of noxious gasses. The programme began with conversion from capped fuse-igniter cord initiated blasts of ANFO. It moved through the use of cartridged emulsions to shock tube initiation of pumpable explosive products in both the stoping and development activities on the mine. The new processes have been successfully migrated to other mines in the Anglogold Ashanti stable. Significant rationalization of the explosives and accessories commodity basket has been achieved with concomitant savings in costs of stock holding and simplification of packaging, transport and storage requirements. The carbon footprint has also been reduced. These encouraging results have come about through cooperation between explosives suppliers, mining groups, research organizations and government departments over many years.

1 INTRODUCTION

1.1 *Overview*

For most gold and platinum mines in South Africa, the operating depth, complex infrastructure, narrow seam width and unskilled and semi-literate labour force creates a challenge to the implementation of new technologies not found in opencast or massive underground mines. Despite the difficulties, transition from old capped fuse – igniter cord (CFIC) initiation of ANFO-type explosives systems to pumpable products and shocktube initiation has been achieved at Anglogold Ashanti mines in South Africa.

1.2 *Historical perspective*

A vision of an “explosives-free” mining environment has been in the minds of safety conscious mine operators and legislators for many years. Its realization has been delayed by a number of factors including restricted access to modern explosives and blasting practices and limited competition in the explosives supply industry in South Africa. The scattered nature of mining activities together with their lack of mechanization was not conducive to the implementation of newer explosives and initiation systems. Attempts at improved performance in blasting were initially addressed through cost-cutting initiatives; the aim being to find the cheapest and most widely suitable product. In this way, ANFO-type explosives known commercially as Anfex, Panfex, Expanfo or Danfo became widely used. Initiation was by fuse and igniter cord. Stocks were stored near all workings throughout the mine. Rock Mechanics Engineers had repeatedly voiced their concerns regarding the destructive and uncontrollable effects of ANFO type explosives when used in conjunction with CFIC on the sidewalls and hangingwalls of excavations. Questions from ore-accounting departments were periodically raised about the possibility for gold loss into cracks created during the blasting activity. Wastage arising from both the packaging weakness (spillage and discard of partially used bags) and from the charging method (spillage and overcharging of holes) were ongoing problems.

The continuing need to reduce costs, together with increased emphasis on health and safety of mineworkers and the increasing influence of technically skilled graduates in service departments like Rock

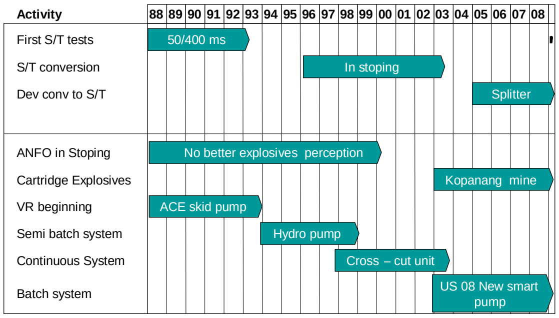

Figure1. Blasting projects undertaken on South African mines from the 1980s

Engineering resulted in an appreciation of the possibility for greater sophistication in the blasting process. Although the technologies of shock tubes, emulsion explosives and watergel explosives were developed nearly fifty years ago, they had not found wide acceptance on South African mines. Among the reasons for this were a perception of high cost, the higher level of sophistication required for their proper utilization, pricing structures and business approach of suppliers, and lack of incentives for the labour force. The last years of the eighth decade and the beginning of the ninth decade of the previous century saw increased attention being paid to explosives technologies. Work began with shock tube initiation and pumpable explosives in development. Some small-scale trials were undertaken in stoping. Electronic delay detonators were also introduced around this time but results achieved in laboratories were not repeated underground and the potential for productivity and safety improvements was underestimated by production officials. Despite sporadic good results, the project was shelved.

1.3 *Anglogold Ashanti's activities*

Blasting technology was one of a number of topics being actively addressed by Anglogold Ashanti's research and development department. It devoted 5% of the $6 million annual research budget to this area. By this means, test sites were made available to explosives suppliers; labour and other support together with project management personnel were also provided. Their role was pivotal in the basting process optimization from the replacement of ANFO, through the introduction of cartridged emulsions, to the implementation of bulk pumpable explosives. The paper will trace the salient steps of the work which can be summarized in Figure 1.

1.4 *Structure of the paper*

Following a general background description, each section has a similar structure. Beginning with a brief description of the theoretical details of the topic being discussed together with a summary of the desired outcomes, it then progresses into the experiences gained during the conversion process

1.5 *Background*

Anglogold Ashanti has interests in gold mining activities across the world as shown in Figure 2. Major gold mines in South Africa lie along an arc from Johannesburg in the north to Welkom roughly southwest of Johannesburg in a geological location known as the Witwatersrand Super Group. The South African region of Anglogold Ashanti operates gold mines in the Klerksdorp and Carletonville areas.

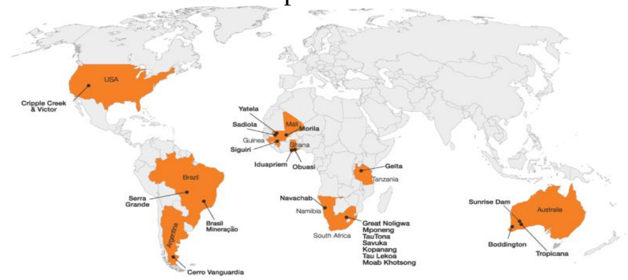

Figure 2. Anglogold Ashanti worldwide gold mining activities

Platinum mines occur in the Bushveld Igneous Complex which covers an area to the north of Pretoria and north-west of Johannesburg. Figure 3 provides an indication of the general locations
The Witwatersrand Super Group is a large sedimentary basin. Within it, the Central Rand Group is the world's largest repository of gold. Economic concentrations occur in a number of different reefs. Surface outcrops were the first to be mined in Johannesburg more than 100 years ago. Sub-outcrops have given rise to some of deepest mines in the world.

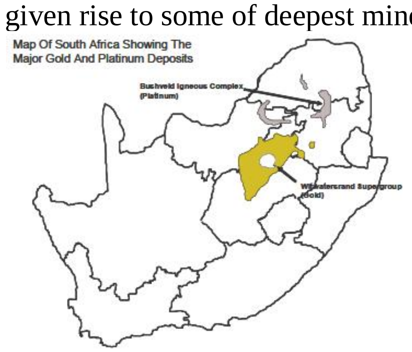

Figure 3. Gold and platinum mining areas in South Africa

They are exploited through deep shafts and tunnels cut at the minimum practicable dimensions.
The Bushveld Igneous Complex is a large igneous intrusion which has been tilted and eroded and outcrops around what appears to be a geological basin. It contains some of the richest ore deposits in the world, including the largest reserves of platinum group metals (PGM). It is divided into an eastern limb and a western limb, with a further northern extension.

Gold mines and the older platinum mines are similar insofar as they utilize relatively unskilled labour to mine narrow reefs, having a seam height of the order of 1 m at dips between 5 and 30 degrees. Traditionally, CFIC initiation has been used for blast initiation for reasons of low cost and ease of use. Narrow reef mining requires that the minimum practicable stoping width is maintained to minimize dilution. Noxious blast fumes created by blasting must be completely removed from the workings before any work may be permitted to continue. This involves the complete evacuation of workings while the atmosphere is cleared, and constitutes a loss of operational time. Storage, handling and transport of explosives have associated hazards and are controlled by legislative and certification requirements. Safety and flexibility of operations would be significantly enhanced in an "explosives-free" environment. The steps along this path of change and innovation will be described in the following sections of the paper. Firstly the conversion to shock tubes in stoping will be described, followed by the work to achieve the same change in development ends. The next section deals with the change from ANFO to cartridged emulsion explosives, and from there to bulk pumpable systems. Whereas stoping was converted before development in the case of shock tubes, development was addressed before stoping for bulk explosives conversion. The different approaches were a function of the state of existing technologies and logistics.

2 FROM FUSE AND IGNITER CORD TO SHOCKTUBES IN STOPING

2.1 *Shock tube assembly*

A shock tube consists of a small diameter plastic tube which has been internally coated with layers of explosive material, as depicted in Figure 4. In this case, the material consists of a mixture of HMX and fine Aluminium powder applied at a rate of 20 g/m. It has a velocity of detonation of approximately 2000 m/s. This small amount of explosives is insufficient to cause the tube to rupture as the detonation shock passes through it, and has no effect on surrounding explosives.

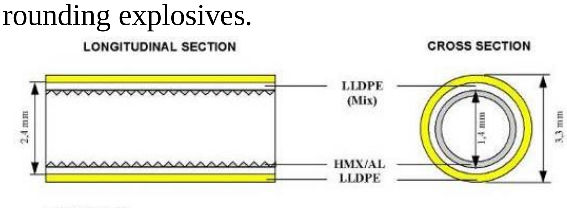

Figure 4. Cross-sectional schematic of a shock tube

There are two types of the shock tube assembly in use on narrow reef mines namely:

- Long period delay (LPD) shock tube assemblies; in this type the tube is heat sealed and has a clip on one end and detonator with the delay element on other.
- Uni – delay shock tube assemblies where one end has a connecting block containing a mini detonator representing the inter-hole delay i.e. 200 ms, and the other end has a detonator with down-the-hole delay i.e. 4000 ms. All assemblies have the same set of inter-hole and down-the-hole delays.

They are shown diagrammatically in Figure 5.

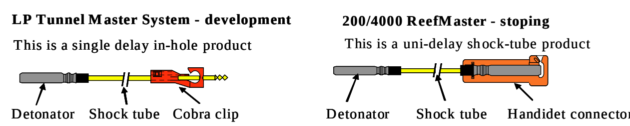

Figure 5. Types of shocktube assemblies

The principle of use is shown in Figure 6.
Following initiation, a low energy shock wave is transmitted in both directions to the detonator and next connecting block.

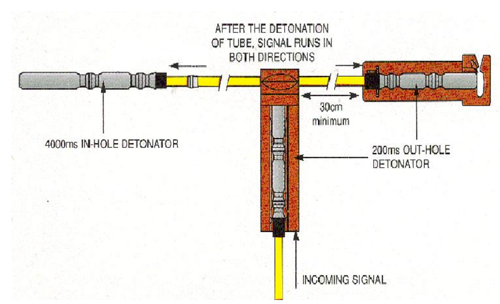

Figure 6. Principle of use for a uni-delay shocktube system

2.2 *Designation of shock tubes*

The out-of-hole detonator delay is the first number in a shock tube designation, and the down-the-hole delay is the second number. Thus a shock tube designated 200/4000 has an out-of-hole delay of 200 ms which represents igniter cord and a down-the-hole delay of 4000 ms which represents fuse itself. Blast design involves the selection and combination of these parameters in shock tube components to obtain an optimal burning front. The combination of delays can be varied depending on the timing and rock breaking requirements.

2.3 *Rationale for conversion to shock tubes*

Anglogold Ashanti's Kopanang Mine had been successfully utilizing uni-delay shock tubes for stoping blast initiation; LPD shock tubes with detonating cord were being employed in development, but CFICs were still in use for off-reef development. Management at the mine was keen to standardize explosives accessories, and was looking to eliminate CFIC, LPD and detonating cord from the explosive stock inventory. The reasons for this included enhanced safety, simplification of training, reduction in stock holdings, and cost savings. Innovations perceived to affect the security of employment of the labour force were sometimes not met with enthusiasm, and although difficult to prove, instances of sabotage of experiments and trials were encountered. This response was not limited to innovations in blasting processes, and has become an element to be addressed in all change management activities.

2.4 *Results from early trials.*

Over a period of more than twenty years, Anglogold Ashanti mines had conducted numerous blast surveys on various shock tube projects. Natural conservatism resulted in any alternative to the entrenched CFIC explosives initiation system giving rise to questions as to potential benefits and justifiability of the associated costs. In the early nineties, only one type was available for use in South African mines, it was 50/400 ms delay. Despite this, in addition to the enhanced workplace safety, the following list of benefits arising from its use could be identified, noting that not all of them are quantifiable.

- Millisecond blasting is achievable
- Sequential firing can be guaranteed
- Longer burning front is created, therefore fewer cut-offs
- No fire hazard exists
- Reduced plastic contamination
- Higher reliability, fewer misfires
- System is more user-friendly system having only a single component
- Less fumes generated
- Improved advance per blast

Unfortunately, the results from the actual trials were unconvincing. Notwithstanding the benefits arising from greater control over timing and the advantages of millisecond delay, a number of problems presented themselves. The main ones were

- Low reliability of the shock tubes.
- Dynamic interaction between blasted holes resulting in nearly half of the muck pile being thrown into the gully,
- Damage to the stoping excavation and equipment as a result of the above
- Lack of local manufacturing capacity
- Cost 300% of that for CFIC until 2000
- Competition from EDD

Experience gained by various researchers during these trials led to the identification of the 200/4000 ms delay system as being the most suitable for narrow reef mining. (MU Ozbay and J Brinkman).The trials also showed that blasting efficiency, as measured by the ratio between advance per blast and average drilled hole length is 6,5% higher than that achieved with CFIC. For future estimates and comparison purposes, this was reduced to a more conservative 5%. The results obtained with shock tubes (Handidet or EZ-Stoper) compared to CFIC are shown in Table 1 below.

When shock tube technology became available within South Africa, and competition was created in the market, costs reduced to a level which made it justifiable to replace CFIC with shock tubes. The graph in figure 7 shows the relative change in costs between 2002 and 2009.

In the middle of 2002, a feasibility study carried out on all Anglogold Ashanti mines showed that the cost ratio between CFIC and shock tubes was 2,2 and

Table 1. Results of trials within the Anglogold Ashanti group between March 1993 and May 2004 on various initiation systems.

Mine name code	CFIC		Shock tube	
	Advance per blast (m)	Blast efficiency	Advance per blast (m)	Blast efficiency
K	0.87	75%	0.94	80.6%
S			0.81	76.4%
S			0.90	94.3%
E			0.87	73.4%
T	0.54	61.5%		
G	0.70	64.1%		
G	0.86	78.5%		
M			0.89	84.0%
G	0.86	73.8%		
SP	0.73	70.0%		
PS	0.96	85.8%		
PS	0.83	82.7%		
PS	0.88	80.0%		
Average	0.80	74.6%	0.88	81.1%

that breakeven could be achieved by an improvement in advance per blast of less than 1%, see figure 8.

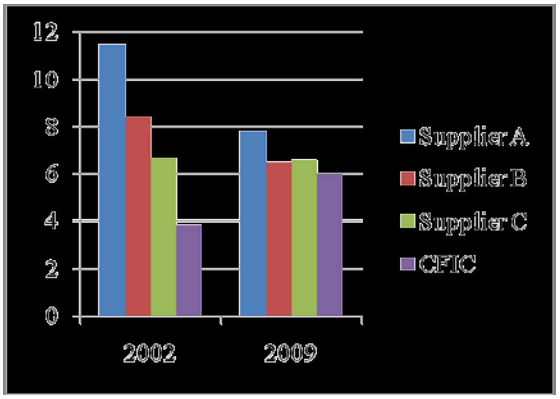

Figure 7. Relative cost of shock tubes and CFIC between 2002 and 2009

An improvement of 0.65% in advance per blast would bring the cost of shock tube initiation to break even with CFIC. For a total face length of 20,700 m this was equivalent to 0.06 m per month. The graph also shows that revenue increases by 2.5% for every percent of improvement in face advance. Note that it was shown earlier that conversion to shock tubes could conservatively provide a 5% improvement in face advance.

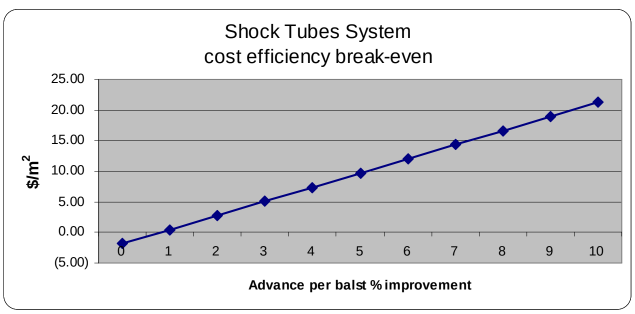

Figure 8. Effect of improved advance per blast on breakeven cost of blasting with shock tubes

2.5 *Updating the analyses*

An impact analysis was performed to determine the effect on costs of reverting to CFIC against the effect of converting fully to shock tubes. As at 2004, the relative utilization of the two methods of initiation was 34% CFIC and 66% shock tube. Taking this as the base value, the analysis showed that the effect of a decision to revert to CFIC, based on its lower cost, would be a net reduction to 96.6% of the base value despite the associated reduction in the cost of blasting accessories. However although they cost more, conversion to shock tubes was shown to be capable of generating an improvement of 1.6% over the base value. The analysis also indicated that the total cost reduction over the South African operations of Anglogold Ashanti could be as much as \$1.6/ m^2 using the 5% figure for improvement in advance per blast. Not taken into account in this calculation were the additional benefits arising from the use of shock tubes arising from the reduced likelihood of underground fires, the reduction in damage to the hangingwall of stopes, and the further reduction of the carbon footprint.

Merely converting stoping to shock tubes would not achieve management's aim of simplifying the entire blasting process throughout the mine. A way had to be found for utilising the same shock tubes into the development activity and this was the next step.

3 FROM FUSE-IGNITER CORD TO SHOCK TUBES IN DEVELOPMENT ENDS

3.1 *Dedicated project team formation*

As was noted above, Kopanang mine had been utilizing uni-delay shock tubes in stoping. Over an eighteen month period up to the end of 2007, a project team comprising members from Kopanang Mine and an explosives supplier developed a method for the use of a uni-delay shock tubes to time any development round. These uni-delay shock tubes are better known as Reef Masters® or EZ-Stopers® depending on the supplier, and the system has particular application in flat development.

During the project, two new shock tube system components were designed, tested and manufactured; namely Splitters and Starter Packs. The aim of the project was to provide an explosive initiation system which would be safer, more effective and reliable than conventional CFIC systems. Specifically, the aims were

- to develop a universally applicable system for both stoping and development
- to provide a system with greater effective blasting performance than CFIC
- to provide a reliable system
- to reduce the number of explosives inventory items

The shock tube used in stoping seemed to be a promising starting point. No specific quantitative success criteria were specified at this stage. The two systems were compared as shown in table 2.

3.2 *Determination of the optimal connecting method for development*

Kopanang mine was making use of CFIC and Tunnel Master® (TM) technology in conventional development. The system requires the use of detonating cord. Its main disadvantages are

- It is "round specific"
- It is not amenable to standardisation
- Stock control is difficult

- Timing outcome is uncertain

Table 2. Comparison of CFIC and shock tube initiation systems in development blasting

Historical performance of CFIC systems in development ends		Required performance of shock tube initiation in the development ends
Advantages	Disadvantages	
• Simple system • Easy to connect up • Low cost per unit • User-friendly • Widely available on the mine • Easy to train people on it	• Additional fuses used in cut and lifter holes • Cannot guarantee sequential firing • Cannot guarantee water resistance • Potential fire hazard • High volume of noxious gas produced • Inaccurate • Prone to cut-offs	• Reduced exposure to hazards (ie noxious gas, smoke, fire) • Improved ground conditions • Reduced wastage of explosives • Water resistant • Sequential firing • Controlled thrown rock profile • 8D cap sensitivity to obviate the need for a primer or booster • Cost per blast to be comparable with FIC systems

Trials were performed in development ends on various mines in conjunction with one of the shock tube importers using standard 200/4000 Reef Masters® (RM).They resulted in limited success. The complicated connecting up method resulted in frequent cut offs of the peripheral holes. It was noted that this problem might have been overcome had a 100/4000 RM been available, but the trial was nevertheless suspended. Kopanang mine's purpose would not be served by the introduction of an additional inventory item, so there was no interest there in developing a 100/4000 RM. Also, the 200/4000 RM had already been identified as the standard replacement product for CFIC in stoping for both gold and platinum mining in. Early in 2006, Kopanang Mine and its major explosives supplier began joint investigations for development applications of the 200/4000 RM, utilizing alternative connecting-up configurations.

Flat development ends at Kopanang are 3.0m wide by 3.8m high and are drilled using 2.3m steel. The round consists of six columns and seven rows of holes drilled on a grid with a five-hole cut drilled between the grid holes. Of the forty-seven holes, one is left uncharged. The geometry of holes and the connecting-up using CFIC is shown in figure 9. The time interval between consecutive shots when 3.0 m capped fuse is used with Duraline is approximately 4 - 30 seconds.

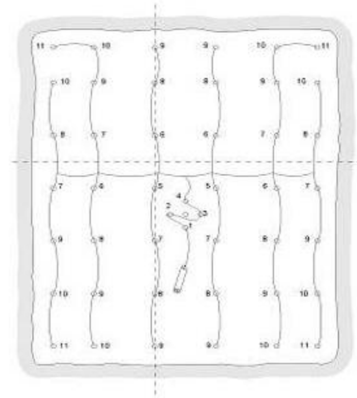

Figure 9. Conventional CFIC connection

In the case of a 200/4000 RM shock tube, only 20 delays are available. So having 46 holes to be blasted sequentially in a flat development end, it was necessary to find a new way of connecting-up the round to ensure sequential firing of all holes.

3.2.1 *Branch connection method*

The first attempt was a branch connection method. It was developed by the explosives supplier as an approximation to the CFIC method, and is shown in figure 10. Two trial blasts were conducted with less than satisfactory outcomes. The first one produced an advance of only 1.5 m with long overburdened sockets in the lower half of the face. The second blast was cut off at the cut. The extent of the thrown rock profile and the concussion produced were both excessive. Connecting-up proved to be difficult; it required stretching of the shock tube.

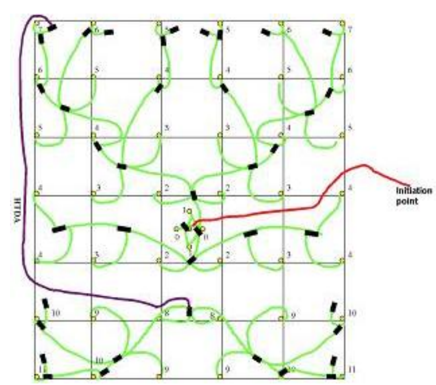

Figure 10. Initial Reef Master® branch type connection method

This method was not pursued further.

3.2.2 *Spiral connection method*

Given that the number of single connections exceeded 46 in this design, a 100/4000 RM was provided by the supplier for the purpose. Having specially manufactured for the project 100/4000 ms delay sets spiral connection was tested as shown in figure 11.

The 100/4000 RM had been used in an experimental spiral connecting-up for a blasting in both haulage and return airway. The outcome was exciting in that an advance of 2.3 m was achieved without sockets, and thrown rock accumulated within 5 m of the face. Being the best achievement so far, it was agreed that this should be set as the criterion against which all future methods of connecting-up the blast would be measured.
This method is commonly used in Canada with good results because breaking and advance is totally dependent on sequential firing.

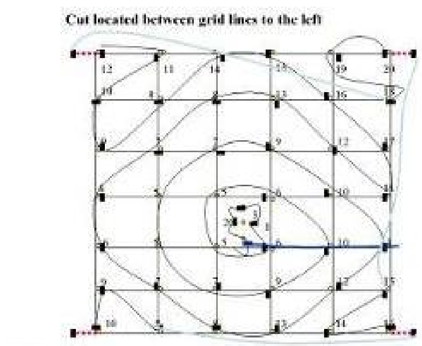

Figure 11. Flat end connected up in the spiral pattern with use of 2 extenders

3.2.3 *The half-spiral method and "the splitter"*

If the CFIC method was to be emulated, a means of overcoming the need to stretch the shock tubes was required and a method of initiating both upwards and downwards from a horizontal input had to be created. This led to the innovation known as the "splitter" and shown in figure 12. It consists of a 1.5 m length of shock tube equipped with 100 ms or 200 ms minidets in a connector block at both ends. Its function is to receive a single signal and transmit it in two different directions. It is also used to simplify the tie-up at the cut. It should be noted that the out-of-hole delay can be varied according to requirements.

Figure 12. The Splitter

The first connecting-up using Splitters is shown in figure 13. The work had taken the project into the third quarter of 2006 when the first test blast using Splitters was conducted.

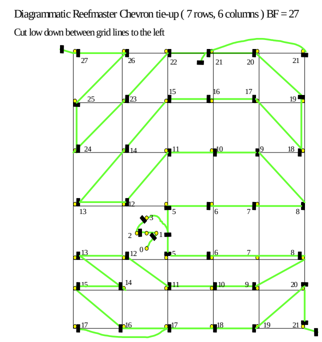

Figure 13. Diagram of the Reef Master® connection in ½ spiral

Connecting-up was much simplified as can be seen in figure 13, but the advance achieved for the 2.1 m drilled round was only 1.9 m. Rock was thrown 14 m back from the face and concussion was again observed to be excessive. In some instances 2 holes were blasted on the same delay as number of holes 1/2 spiral exceed 20. Some other combination with the single splitter were also tried but without bigger success.

In practice this arrangement proved to be difficult to connect up and to check. The advance achieved by the blast was approximately 1.7 – 1.9 m. Although the cut broke well, long sockets were observed on one sidewall. Some reduction in the distance of thrown rock was achieved.

3.2.4 *Blast simulation and timing evaluation*

Further discussions with mine management resulted in the creation of a project team to evaluate different timings. Splitters with different delays and Reef Masters® with 200/4000 ms delays would be used. Quantitative objectives were also set at this point. They were:

- advance per blast must be at least as good as that achieved with CFIC
- excavation profile must be at least as precise as that achieved with CFIC
- concussion (as measured by displaced equipment such ventilation columns) must be reduced

To establish absolute data for CFIC blasting against which the project results would be compared, a series of parameters was collected from specific CFIC blast events conducted for the purpose. Measurements and photographs were used to also establish the spread of thrown rock and the range of fragmentation achieved. A number of tests were conducted to establish the average time between consecutive shots which would produce:

- full advance
- correct thrown rock profile
- minimal concussion

Electronic test blasts were conducted using four different delay intervals of 3 m Smartdets™ for the blastholes excluding the cut, namely 50 ms, 75 ms and 100 ms. In all cases, cut holes were provided with 200 ms delays, and the rest of the round was set to the stated delay. In all cases sockets observed were minimal, and faces were blasted solid and square with no misfires or cut-offs. Results of electronic test blast are tabulated in table 3.

Table 3. Parameter-setting Smartdet ™ blast results

Delay period (ms)	Time for shots to go off (ms)	Average advance (m)	Rock throw (m)	Notes
50	2800	2,0	14.1	Only 44 holes out of 47 blasted
75	3950	2.21	14	Most thrown rock within 8m
100	5000	2.4	9	Most thrown rock within 4m with good fragmentation

It was concluded that an interhole delay period of at least 75 ms was necessary for the achievement of the required advance and thrown rock profile.

3.2.5 *The quarter spiral method or "spider" cut*

The redesigned connecting-up was a joint effort by the supplier and the Kopanang team to simulate the effect of the spiral, but using the 200/4000 RM. This gave rise to the so-called "quarter spiral" or "spider cut", a nickname assigned to it by the development crews, and shown in the diagram below. Connections for the cut and Splitters showing the links between quadrants can be seen in the diagram and photograph below.

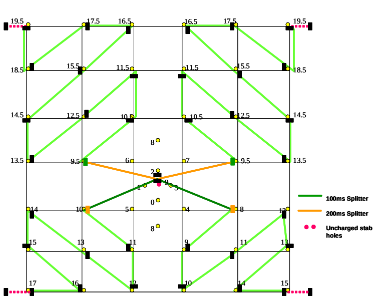

Figure 14. Diagram of 200/4000 ms Reef Master® connection in quarter spiral format

Figure 15. Photograph of Reef Master® connection in quarter spiral – chevron format

A single connector block to the first four holes of the cut is attached to Splitters (shown in red in the diagram). This method was applied over a period towards the end of 2006. Average advance exceeded 2,0 m however, the thrown rock pile profile was not ideal. Beginning in January 2007, after trying a number of variations, it was found that consistent results were obtained if the top and bottom of the face were offset by 100 ms, with the 100 ms Splitter being connected to the bottom two grid holes, splitters connections are shown in figure 16. At this point, the team was confident that the 200/4000 RM could be utilized in development. It had the following advantages

- It was simple to connect
- Faulty connections were easy to find
- 100% break was obtainable
- User acceptance had been quickly achieved

Figure 16. Photograph of spiral connection of a burn cut with Splitters

A number of additional questions now arose. These were

- whether locating the cut on the grid lines would improve results
- how practical were the tie-up formats, especially with respect to training of mine employees
- whether Splitters were required at all, or if so, whether both 100 ms and 200 ms Splitters were needed
- what length of Splitter was needed

The answers to these questions depend on local circumstances and are different for each mine. They need to be determined by iterative means.

3.3 *Validation of the blast design*

For the purposes of conducting trials on timing, a test rig was constructed from two sheets of expanded metal mounted 2 m apart in a development end as shown in figures 17 and 18.

Figure 17 Shock tube timing test rig front view

Shock tubes were suspended between them and tied up according to the pattern to be tested. A number of blasts were conducted using various combinations of 200 ms and 100 ms Splitters in conjunction with an 8D detonator to evaluate the sequence of the blast and the potential for cut-offs caused by shrapnel.

Figure 18. Shock tube timing test rig, side views

None occurred in the more than ten test blasts. Timing looked and sounded correct, and photographs showed the expected sequential burning geometry. However, the test blasts were successfully videotaped with a commercial camera. The recordings were later used for the training of the development crews.

Figure 19. Photograph of the shock tube blast in progress on the test rig

Having shown that the system worked in flat ends, wider application into other types of development was pursued. The test rig was utilized to develop and design rounds for other types of development end. Rounds were designed for a drag round in flat development, burn cuts in raise/winze development, travelling ways, boxholes and boxhole connections.

3.4 *Quantifying the results*

During the trial, 76 blasts were evaluated. The following blast parameters were measured:

- drilling geometry
- explosives use and consumption
- advance per blast
- end geometry before and after the blast.
- thrown rock profile
- fragmentation

Blasting efficiency was calculated by dividing the advance achieved by the 2.3 m drilled length of the round.

Table 4. Summary of trial blast results

System	Number of blasts	Average advance per blast (m)	Blasting efficiency %
FIC	37	2.02	88
Reef Master®	39	2.09	91
Variance		0.07	3.0

The combined results indicate that advance per blast obtained with Reef Master® Shock tubes is better than when CFIC is used. An average of 2.09 m versus 2.02 m translates into a 3.0 % improvement. This shows the effect of perfect sequential firing, no cut-offs and better accuracy of timing. Figure 20 shows the full advance achieved. Greater consistency of results was achieved using RM as displayed in figure 20.

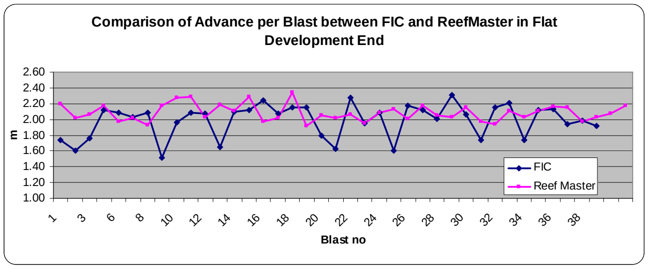

Figure 20. Comparison of the advance per blast between CFIC and shock tube technology

3.5 *Face geometry after the blast*

Full advance and sidewall control were successfully achieved while concomitant benefits arising from the precision of the timing are that fracturing is more precise and less barring is required.

3.6 *Thrown rock pile profile*

Containment of the thrown rock pile close to the face facilitates cleaning and minimizes damage to equipment. Results of blasts with RM are shown in the following illustrations figures 21 and 22.

Figure 21: Thrown rock pile

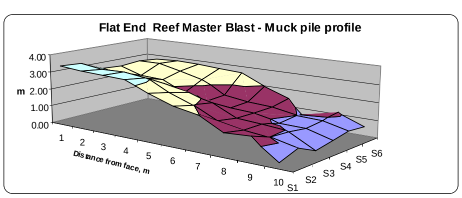

Figure 22 Corresponding 3-D image of thrown rock pile

3.7 *Fragmentation*

During the trial, photographs of the fragmented rock were visually analyzed. Using a tennis ball for reference, it was clear that shock tubes yielded coarser and more uniform fragmentation than blasting with CFIC. This can be attributed to the more accurate timing and sequential firing of the shot holes. And can be seen in Figures 23 and 24.

Figure 23. Fragmentation – Reef Master® blast

Figure 24. Fragmentation – CFIC blast

3.8 *Development of explosives accessories*

Referring to the initial aim of the project, one desired outcome was a reduction in the number and variety of explosives related items required to be held in stock by the mine. Starter packs were introduced to minimize wastage of accessories. The packs consist of one Master Starter, one extender and a one metre length of igniter cord. Following their introduction, stock control systems prevented the ordering of Master Starters and extenders as separate items.

Figure 25. Reef Master® starter pack and splitters;

Figure 26. Starter pack in the box and boxed 3.0 m Reef Master® shock tubes.

In this way, the number of items required to be supplied and stored as blasting accessories has been reduced from fourteen to six, with concomitant savings in holding costs and space requirements. This initiative will help save approximately $26,400 per annum or 6.8 % of the explosives accessories budget for development.

3.9 *Development cost comparison*

Development blasting analysis showed that the use of uni-delay shock tubes did not result in costs higher than those for conventional CFIC system initiation. Also, the trend in actual costs is positive, see figure 27. Comparative statistics are tabulated below. For each method of initiation, the following parameters were common:
Total number of holes blasted per round = 46
Included number of perimeter holes = 12
Cost of primers = $ 3.67
Cost of perimeter hole explosives = $8.58
Cost of ANFEX = $14.59 per blast
Assumed wastage of explosive = 5%
Total cost per round of explosive = $27.74

Table 5. Comparative analysis of development costs

	CFIC	**LP**	**Uni- delay shock tube**
Advance per blast	2.05 m	2.15 m	2.15 m
Fuses	46 + 10		
Duraline (m)	23		
Number of shock tubes		46	46
Explosives cost (R/m)	136.30	129.96	129.96
Initiation system costs (R)			
Fuse	235.40		
Duraline	32.20		
Shock tube		257.40	283.50
Cordtex		35,88	N/A
Startup system per blast	4.98	4.04	4.04
Assumed wastage	10%	5%	3%
Total initiation system	299.84	312.19	296.17
Total per blast	579.25	591.60	575.58
R/m blasted	282.56	275.16	267.71
Relative values	1.00	0.97	0.95

Note: Double fuses are used in cut and lifter holes with CFIC adding 10 fuses per blast.
This calculation is idealized as regards advance per blast. The year-to-date development explosives cost for Kopanang mine is $31.6/m and include the effects of inaccurate drilling, poor timing and other inadequacies. It also demonstrates that potential savings of the order of $4.6/m or about 14% are possible.

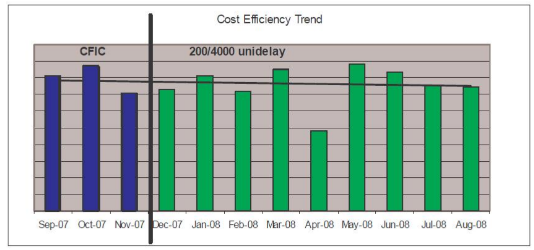

Figure 27. Graphical summary of cost efficiency trends

3.10 *Environmental impact*

Moving away from CFIC towards RM initiation can have a dramatic impact on the volume of “greenhouse” gas created by blasting. A conservative estimation demonstrated that CO_2 and NO_2 emissions would be reduced from over 100 t/a to below 2 t/a on a mine the size of Kopanang.

3.11 *Training and implementation*

Prior to mine-wide implementation, a three module training course was developed. In addition to simple and straightforward procedures for the use of Reef Masters® in development, it also covered basic blasting theory as well as charging up procedures and disposal requirements for old explosives. All development crews were exposed to the programme. For completeness, those aspects of blasting practice which are common to all explosives were included. This served to reinforce the safety effort.

3.12 *Conclusion*

The work described above successfully met the aim of providing a versatile and reliable blast initiation system for development. It showed that the single delay 200/4000 ms shock tube used in stoping could be applied to the development environment. The breaking capabilities of a shock tube system which are attributable to its timing properties improved the advance per blast and the geometry of excavations was improved by 3%. This has the effect of improving ground conditions and of reducing hazards in the working places. The stockholding for explosives accessories has been reduced and simplified from fourteen items to six. Unit cost reduction for blasting of the order of 5% has been shown to be achievable.

4 FROM ANFO EXPLOSIVES TO PUMPABLE BULK EXPLOSIVES IN NARROW REEF MINING

4.1 *Circumstances applicable to narrow reef mining.*

Where accessibility is not a problem, such as in opencast or massive operations, or where rock strength is low, such as in collieries, the transition from dangerous nitro-glycerine based and difficult to handle packaged explosives, to a non-detonable and safe means of rock breaking has been a relatively simple process. A means of utilizing the benefits of a pumpable emulsion that only becomes explosive when actually mixed in the blasthole has been sought by miners in the narrow tabular hard rock environment for many years. Such an innovation carries challenges and opportunities in equal measures. This section covers the reasons for needing to change from conventional explosives and it details the potential benefits accruing from the use of pumpable emulsions or watergels. A major drawback has hitherto been the difficulties associated with delivery of a pumpable product of reliable consistency and quality to the underground workings where it is required. The section describes the activities which were undertaken in attempting to overcome this obstacle. Development of a suitable forms of this explosive for use in stopes is also described. Work in this field has been conducted by several different organizations over a period of about twenty years, but results are only now sufficiently encouraging to warrant their publication. The latest development from the project realistically meets the vision of an "explosives-free" underground environment through their replacement by non-explosive components.

4.2 *Conversion from Anfex to Kubela Explosives at Kopanang Mine*

Low cost and simple logistics have helped to maintain the use of ANFO type explosives in South African mines. Low Mine Call Factor on Kopanang Mine and also low breaking efficiencies prompted the need to increase advance per blast and gold recovery. Work done between 2001 and 2004 by Kopanang Mine and AngloGold Mining Technologies (AMT) into determination of the impact of explosives type on the fragmentation distribution and rock breaking efficiencies revealed that watergels type explosives produce a greater quantity of coarser material than standard Anfex. Based on that, decision was taken in July 2004 to convert whole mine's stoping operations from Anfex to Kubela 420 dry watergel explosives.
Initial tests conducted between April and June 2004 showed good performance of the Kubela cartridges in terms rock breaking and user friendliness.
The results achieved are presented in table 6.

Table 6. Summary of Kubela 420 versus Anfex test results

4.3 *Explosives consumption*

	Anfex	Kubela	Variance
Average Hole Depth (m)	0.90	0.88	-0.02
Drilling Angle (º)	78.86	76.71	-2.15
Advance (m)	0.67	0.78	0.11
Socket Length (m)	0.23	0.10	-0.13
Charge up per Hole (Mins)	0.67	0.47	-0.20
Potential Advance (m)	0.88	0.86	-0.02
Blast Efficiency (%) Act	75.8%	90.7%	14.9%
Explosives Per Hole (g)	0.77	0.35	-0.42

The conversion began in August 2004 and was completed in a period of six weeks. Explosives consumption statistics for period January to October 2004 show the salient facts and trends. The results are shown in figure 28. The quantity of explosives boxes and bags was reduced by approximately 2000 per month; that is by 24 %. The number of trips required for transporting explosives in the shaft was materially reduced and thereby also part of the transport cost.
Over the same period, explosives efficiency showed a similar improvement. In volume terms, the performance was 53% better than with Anfex.
The success of this work paved the way for the change from cartridged emulsions to bulk supply.

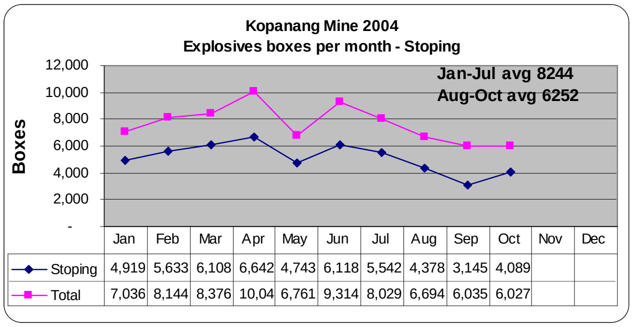

Figure 28. Consumption of explosives at Kopanang mine between January and October 2004

5 BULK SUPPLY OF EMULSION EXPLOSIVES TO DEVELOPMENT AND STOPING

5.1 *Introduction*

Pumpable watergel and emulsion explosives are already in use as a bulk product in opencast and massive mining. Initiation can be achieved by any available means; a conventional detonator, shock tube or by electronic means. Thus the technology already existed; the task was one of adaptation for its use in narrow tabular deposits. The success that was achieved came through a synergistic relationship between the explosives supplier, operational management and technical expertise at the mine, none of whom could have achieved it alone. Work continues in a number of areas. These include the improvement of portability of pumps and the minimization of maintenance requirements without compromising their robustness. Also, a means of correctly measuring the correct quantity, consistency and uniformity of the explosive charge into the blasthole is required.

5.2 *Early work*

Work on a pumpable explosives system had begun in 1988 on AngloGold's Vaal reefs operations. By 2003 and in spite of more than ten years research and testing, there was still no pump or reticulation system suitable for the mine wide distribution of bulk explosives to the stoping production areas. A similar situation pertained to the non-explosive chemical components of bulk explosives; although they had been in use for some time in other mining sectors. However, equipment for bulk loading in large excavations like tunnels or gullies was readily available in the market. Kopanang Mine management recognized this as the starting point for introduction of this concept into the development workings.
Two types of explosives were chosen for test namely:

- Pumpable emulsion explosives
- Pumpable watergel explosives

5.3 *Bulk explosives concept*

Bulk explosives consist of a mix of two non-explosive components which only become a detonatable explosive compound after they have been intimately mixed in the blasthole. An immediate consequence of this is that the risks, requirements and regulations applicable to the transport and handling and storage of explosives cease to be applicable. They possess the following attributes:

- Variable explosive strength
- Short charging up times
- Safer application (2 or 3 non-explosive components)
- Lower cost than cartridged explosives.

The two types of pumpable bulk explosives are the emulsion known as R100G, and the watergel known as RIOFLEX.

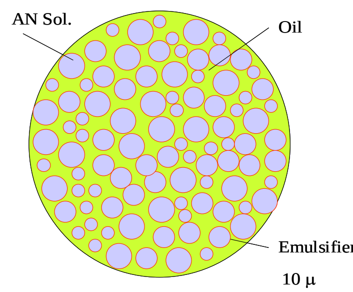

Figure 29. Schematic diagram of emulsion explosives structure

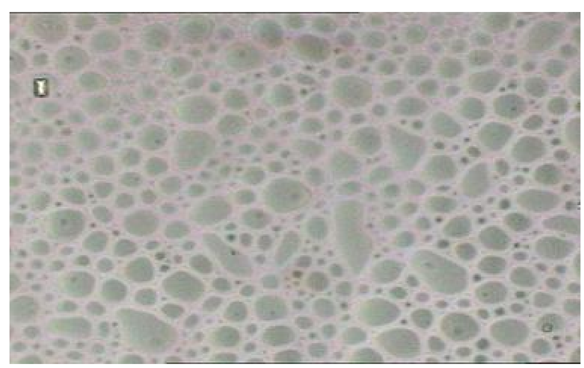

Figure 30: Microscopic image of base emulsion; droplet size 1 – 2 microns

R100G is a mixture of a non-detonable emulsion and a gassing solution in the ratio of 97%-98% base emulsion to 2%-3% gassing solution. The base emulsion is manufactured by blending an oxidizer (ammonium nitrate solution) and fuels (oils) with various pH regulators and the addition of catalysts to stabilize it. The gassing solution is a mixture of sodium nitrate and water; the quantity used directly affecting the final density of the product. Schematic structure of bulk emulsion is shown in figure 29 and its microscopic image in figure 30.

During the sensitization process, sodium nitrate from the gassing solution reacts under pressure with the ammonium nitrate in the base emulsion to produce microscopic nitrogen bubbles, making the final product cap sensitive explosives.

RIOFLEX is a member of the ammonium nitrate family of bulk watergel explosives. This product consists of two main components, namely the RIOFLEX matrix and a sensitizer called a crosslinker. The crosslinker consists mainly of a water solution of sodium nitrate. The matrix is a fully crystallized suspension of ammonium nitrate salts and soluble and insoluble fuels in an aqueous solution, with additions of gelatinizing chemicals. Schematic diagram is shown in figure 31.

Comparable properties of both explosives are given in Table 7.

Equipment used during the trial was as similar as possible for each explosives type. For the emulsion R100G a charging unit referred to as the U 117E from African Explosives Limited (AEL) was used.

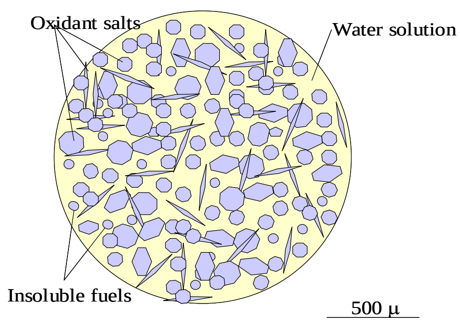

Fig 31. Schematic diagram of watergel explosives structure

In the case of Rioflex watergel, the equipment was provided by UEE Dantex.

Table 7. Comparison between pumpable emulsion and pumpable watergel explosives

Parameter	Emulsion R100G	RIOFLEX
Density g/cm³	1.25	1.15
VOD m/s	4500	3800
Gassing rate (minute)	25 min	1 min
Critical Diameter (mm)	22 mm	25 mm
Critical density	1.29 g/cm³	1.15 – 1.20 g/cm³
Sensitivity	8D	6D
Energy (Bulk) kJ/kg	2.78	3.136
Waterproof	Yes	Yes

Fig 32. Emulsion charging unit U 117E

Fig 33. Watergel charging unit

A comparison of the operational parameters of the units given in Table 8.

Parameter	Emulsion unit	Watergel unit
Base tank capacity	1400 litres	1400 litres
Sensitizer tank capacity	60 litres	60 litres
Flow rate	30-40 kg/min	21-26 kg/min
Dual density arrangement	Not available	Mechanical switch gear arrangement, low and high density application
Relative cost	1.0	0.8
Working pressure	28 – 30 bar	20 bar
Power medium	525 v – Electric drive	525 v – Electric drive
Mixing arrangement (lance/unit)	On line unit modernizer or static mixer	Lance with static mixer
Main pump	Orbit 0504- Electrical speed reducing gearbox – chain driven	Orbit 0504-Electrical speed reducing gearbox- chain driven
Secondary pumps	2 x Cat 2ES 20/35 high pressure pumps	M 83 Orbit pump
Bursting disc pressure	56 bar	20 bar – min of 14 bar
Electrical safety PLC	40 bar	None – no high pressure pumps
Additional tanks	Hose lubrication and flushing 60 litre capacity	None

Table 8. Comparison of the operational parameters of the bulk explosives charging units.

5.4 *Project implementation process*

The tripartite project team, consisting of the two explosives suppliers and representatives from AngloGold Ashanti, agreed that a suitable site for the trials would be in development ends on 53 level. For the first month, the emulsion unit would serve the return airway (RAW) and the watergel unit would serve the haulage and connecting cross-cut. In the following month, the location of the units would be swapped between the working-places.

Test criteria were set as follows:

- The period of the trial would be two months (September and October 2003)
- Each supplier could have the same opportunities to blast in each working-place
- The same initiation system could be used at both locations
- Only one variable would be introduced at a time and it would be done at both locations simultaneously

Desired outcomes were determined as follows:

- The safety requirement was that no incidents or accidents related to explosives handling should occur
- Each blast was to achieve 100% advance, that is 3,0 m/blast
- Over break was to be reduced to 0%, this would reduce the total number of hoppers of broken rock generated per blast to 20
- Eexplosives used had to be able to be initiated with 8D detonators only, no primers were to be used.
- Only pumpable bulk explosives could be used.
- The quality of the blast would be measured by the presence of barrels on the sidewalls
- The lowest possible cost per metre of advance was required

5.5 *Test results*

Results of the trial are shown in the illustrations below.

Fig 34. Blast with ANFO and CFIC showing large number of sockets and poor condition of sidewall (base case).

Fig 35. Clean break after blast with pumpable watergel, some barrels can be seen at lower left

Statistics from the trial are tabulated below.

The average advance per blast achieved with the watergel RIOFLEX system outperformed the emulsion R100G by 11.4% being 2.99 m/blast against 2.66 m/blast.

During the trial, the watergel system showed greater reliability and availability, which can be attributed to the following factors:

- The system is mechanically sound
- It requires very minimal servicing
- The system is simpler having no PLC controls

The availability of the respective systems was measured by calculating the ratio of the number of meters blasted to the total meters surveyed at the end of the measuring month. The results are shown in Table 10.

Both products are sensitive to an 8D detonator; however the watergel RIOFLEX produced a clean break whereas the emulsion R100G product broke only 1.40 m from a 3.0 m hole leaving sockets 1.60 meters long.

Table 9. Summary of the comparative blasting trial results

Number of blasts	Advance blasted m	Metres drilled m	Advance per blast m	Blasting efficiency %	Explosives kg/m
September 2003 Emulsion					
10	25.19	30.00	2.52	84	68.52
September 2003 Watergel					
11	32.67	32.89	2.97	99	60.47
Oct 2003 Emulsion					
9	25.31	27.46	2.81	92	No data
October 2003 Watergel in the RAW					
10	29.93	29.87	2.99	100	61.0
October 2003 Watergel in the haulage					
5	15.15	14.95	3.03	107	60.30
Summary Emulsion					

Number of blasts	Advance blasted m	Metres drilled m	Advance per blast m	Blasting efficiency %	Explosives kg/m
19	50.5	57.46	2.66	87	
Summary Watergel					
26	77.75	76.91	2.99	100	60.67

Table 10. Availability of the charging-up units

	Survey meters measured	Emulsion		Watergel	
		m	%	m	%
September 2003	8.,7	25.19	31.1	32.67	40.5
October 2003	70,.	25.31	35.8	45.1	63.4

The objective for smooth blasting was not fully achieved and only a few barrels on the sidewalls were observed during the test period. However, towards end of the test the number of barrels per blast reached about 30 % of the total number of the perimeter holes (see figure 35). This could be attributed to better utilization of the dual density facility by the crew. The project team concluded that changes would have to be made to the blast design before the desired objective could be achieved.

5.6 *Conclusions and recommendations from the trial*

The results of the trial indicate that the pumpable watergel product provided on average 11.04% better advance per blast than the pumpable emulsion product and leaves no sockets on the face after the blast.

Use of the watergel product would provide a 14 % reduction in cost per meter blasted. A reduction in the time required for the charging up process was also observed, being only about 30 min for the 51 hole round of 3.0 m holes.

To obtain a smooth blasting effect, more work needs to be carried out on the blast design. This work needs to include both the drill rig supplier and explosives suppliers.

Based on the achieved results system has been introduced across all high speed development ends.

6 STOPING APPLICATION

6.1 *Introduction*

After the success of the trials in development ends, Kopanang Mine management and the project team turned their efforts towards the stoping application of pumpable explosives products. The following benefits were anticipated:

- Less blast-induced hangingwall damage, and thereby enhanced workplace safety.
- Reduced volumes of fumes generated by the blast, thereby enabling reduced re-entry delays.
- Removal of constraints and restrictions arising from the many Regulations relating to storage, transport and handling of explosives
- Improvement in the utilization of hoisting and transport infrastructure.
- Improved labour efficiency in transportation and handling of non-detonable constituents, being simple and safe to handle
- Good fragmentation and thereby faster cleaning of stope faces and of tipping, with a reduction in the quantities of fines and mud and an improvement in the Mine Call Factor.

Attempts to introduce pumpable emulsion into the narrow reef production environment had begun in late eighties and early nineties of the previous century with varying degrees of success. Most of the work was carried on Vaal Reefs mines. Both batch delivery and continuous supply methods were attempted at various times. Some of the factors which had to be taken into consideration when developing a stoping application were:

- Pumping and distribution equipment was cumbersome
- In-stope pumps were lacking in robustness and of questionable reliability. Sabotage, vandalism and other expressions of resistance to change had to be addressed
- Poor drilling discipline would negate any benefits accruing from improvements in advance per blast, fragmentation and reduced hangingwall damage

6.2 *Project development stages*

The project was developed in a series of stages, beginning with batched supply and followed by an attempt to provide continuous supply. Initially, this was attempted from outside the stope; later versions involved the use of a header tank at a strategic point inside the stope. Neither system was successful, The key element in the system was the pump, and its attributes of mass and manoeuvrability in the cramped working spaces..

6.2.1 *Batched supply*

Between 1988 and 1992, batched supply of emulsion was provided. The in-stope pump available at that time was called the ACE porcupine pump provided by AECI, the precursor of AEL.

Figure 36. ACE porcupine pump in use from 1988 to 1992

The pump system had the following attributes

- Mass exceeding 60 kg
- Emulsion tank capacity 25 l
- Sensitizer tank capacity 3 l
- Compressed air power source
- Emulsion was supplied in 200 l drums requiring transfer into the pump's tank
- Total gross mass exceeded 90 kg

Potential for breaking efficiency and cost reduction were demonstrated by the tests. Its cumbersome construction and low reliability, together with difficulties associated with the mixing system meant that it never warranted introduction into full production. Some mines attempted to use it niche applications such as drop-raising and longhole blasting.

6.2.2 *Continuous supply*

During the period from 2000 to 2003, the development of a continuous supply system was attempted on Great Noligwa mine. The key elements were

- Delivery of emulsion from surface to the crosscut in 1.5 t cassettes
- Use of a transfer pump from the cassette to a silo in the crosscut
- Reticulation of two non-explosive products from the crosscut directly into the shot hole (pumping over 200 m distance)

This approach foundered on the high pumping pressure required together with problems of leakage from the columns and wastage of product. The objective had been to eliminate the manual transport of product into the stope. Some research and development work had been undertaken by AEL in this direction, however the main obstacle was the high pressure required to mix and shear emulsion with the gassing solution at the charging end of the pipeline. It was thought that in-line booster micro pumps might be required to make this concept feasible but it never materialized due to complexities surrounding the pumps and piping. As no significant progress had been made, the project was suspended until the suppliers could complete characterization of the emulsion flow over long distance and change its formulation accordingly.

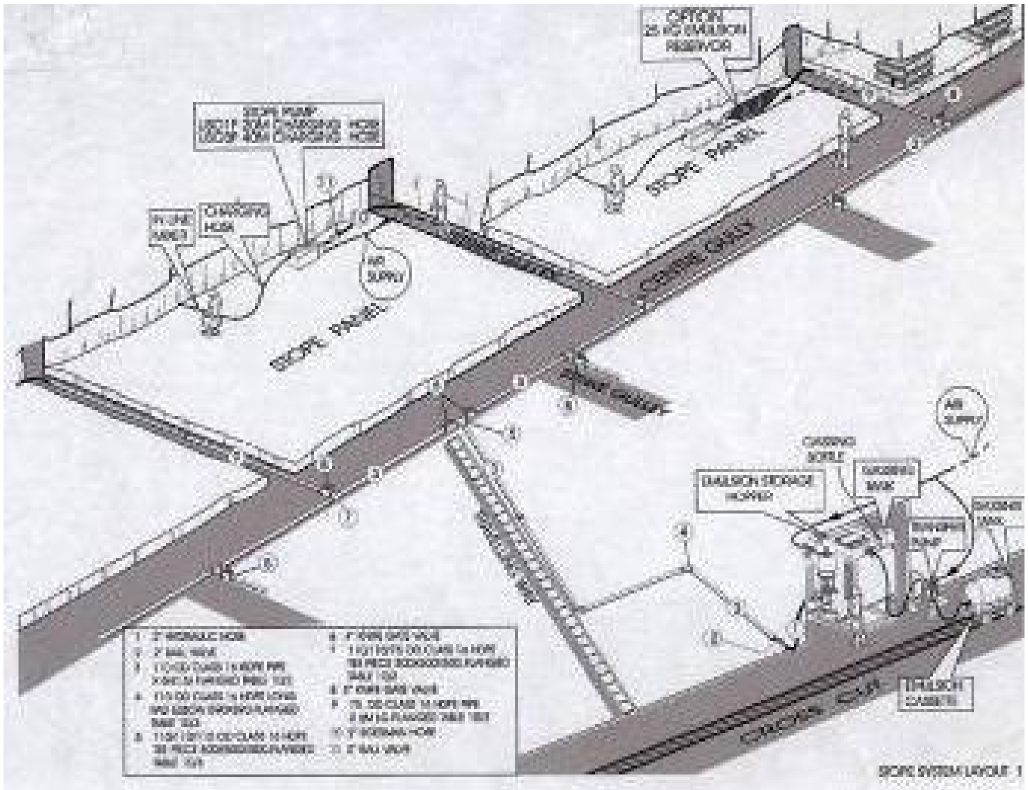

Figure 37. Schematic of the stope layout of a continuous bulk explosives supply system

6.2.3 *Reflection and review*

A number of useful lessons were identified from the discontinued projects. They could be classified into organisational issues, technical issues and logistical issues. The organizational issues identified were

- Lack of a documented project plan
- Lack of defined responsibilities and relationships
- Inadequate project staffing

Technical issues requiring attention included

- Lack of definition of a successful outcome
- Lack of specified test criteria and protocols
- Ineffective monitoring and control

Logistical issues which remained unresolved were

- Maintenance responsibility
- Delivery of product to workplaces
- Stock control and inventory management

On Kopanang Mine suggestions were made at that time to replace a reticulated supply system with a batch option, but they were not acted upon, and the project was suspended in April 2002.

6.3 *Research and development off the mine*

Work on the development of a suitable pump and studies of the flow of emulsion in narrow pipelines were continued by supplier organisations. Interest was maintained at Anglogold Mining Technologies (AMT), the successor to Technical Development Services (TDS) as AngloGold's research and development unit was known. With the long-term benefits in mind, Kopanang mine management prepared a user requirement specification document based on the issues gleaned from the earlier projects and presented it to all product developers. It contained the following salient points, most of which affected the pump:

1. The system must be cost-neutral when compared to existing systems.
2. The pump should be universally applicable to any blasting application.
3. The pump had to be provided with a variable or adjustable batch counter system to match the charge mass required.
4. Total mass of a dry (that is empty) pump could not exceed 25kg.
5. The pump had to allow for one-man operation at the stope face.
6. The cost of a pump had to be reasonable in order to limit capital outlay.
7. The output from the pump had to reliably meet the required density and consistency specifications.

8. The supplier of the pump had to have acceptable credentials and be well established in South Africa.
9. From a safety perspective, the mixing and shearing could only be allowed to take place in the in-line mixer, placed in the vicinity of the blasthole.
10. The pump had to be capable of pumping any new bulk explosives which might come onto the market, albeit after approved modifications and re-licensing.

In early 2003, the emulsion supplier produced the fifth version of their pump, known as the US05 in early 2003.

6.4 *Development of improved pumps*

6.4.1 *US 05 pump*

The new pump (figures 38a and 38b)did meet the Kopanang prerequisite for one man operation; however it fell short in other aspects, having a dry mass of over 34 kg, problematic gassing facility, inaccurate batching and the portable reservoir was very expensive. Its use was discontinued in the middle of 2004 , and the experience gained resulted in the construction of the more efficient and lighter US 06 model.

Figure 38a: The US05 pump disassembled

Figure 38b. The associated portable reservoir, developed in 2003 -2004

6.4.2 *US 06 pump*

Figure 39. The US 06 pump used between 2004 and 2006

Similar to its predecessors, the US06 figure 39 was a piston pump with a separate batching unit for delivering a fixed mass of gassed emulsion into the blasthole. It was robust and reliable and had a maintenance interval of approximately six months. Failures on the pump were restricted to the frame and body. Minor repairs arising from blocked valves were resolved by the operators on site. Although it was the best pump so far, the size, weight and batching mechanism were limitations that prevented the US06 pump from being recommended for use on a large scale in this mining environment. Development work to supersede it with something that better met the customer specification eventually resulted in the US08 pump. The omission from the numbering sequence of US 07 was due to the fact that this version of the pump was bigger and bulkier, being designed for stationary applications.

6.4.3 *US 08 pump*

Many of the shortcomings of its predecessor were addressed in this model and it met the prerequisites set for it by Kopanang mine as regards dry mass (it was less than 23 kg) and variability of charge size (it could be set between 450 g and 900 g). This model used bagged emulsion and was well received by the operating crews. More than 1600 test blasts were carried out with it.

Having satisfied all the design criteria, the next step was to industrialize the design. A specialized pump manufacturer, RUMEG, was introduced for this purpose. The US 08 has been in use since 2007. It has

Figure 40: Bagged emulsion and sachet of the gassing agent

been well accepted by the operating crews, being reliable and accurate in batching and having a mass of less than 15 kg; a superior pump for the purpose.

Figure 41: The RUMEG US 08 pump in use since 2007

Figure 42: Charging-up in progress

Some of the desired criteria remain unfulfilled; nevertheless progress towards meeting them is acceptable. Performances achieved so far are:

- blasting efficiency (with primers) is 100%
- powder factor achieved is 2.7 t/m^3 versus target of 2.2 t/m^3
- pump maintenance cost is acceptably low
- total cost is approximately 16% (inclusive of maintenance) higher than target expressed in unit cost per square metre terms, at this stage

6.5 *Conclusion*

Since October 2007, one entire production section (10 000 m^2 per month output) has been converted onto pumpable emulsion explosives. Results are carefully monitored and analysed. It now appears certain that Kopanang Mine has made the first real breakthrough in the application of pumpable emulsion explosives in narrow reef stoping. The following observations have been made

- The US08 is the first pump to fulfil the requirements for narrow reef stoping
- The system has been well received by both the workforce and supervisors alike
- Significant improvement in ground conditions and a reduction in fall of ground incidents have been reported
- Cost performance is about 10% above target, but is showing a falling trend
- The pump's reliability has been high, and maintenance requirements are low
- Plans have been made to expand the use of pumpable emulsion explosives across the entire mine
- Test of the pump in the development environment have shown that it also has the potential for implementation into this area of operation

7 PRIMERLESS BLASTING

A key requirement of the Kopanang Mine vision of an "explosives free" underground environment was primerless blasting. During 2005 and early 2006, the R100G emulsion was re-formulated with this in mind. Underground testing of the revised product (abbreviated to "SDS" for "small diameter stoping") commenced in July 2006 with extremely promising results from both cost-effectiveness and production points of view. Socket-free stope faces with a blasting efficiency of 100% were achieved. Further testing continued to address the emulsion's performance in development, and a detailed fragmentation analysis was performed for the stoping application. This was completed by year-end. However, the breakthrough was achieved lay in the higher sensitivity of SDS which eliminated the need for primers in a ready-made blasting agent.

The completion of the above, in conjunction with the release of the latest pump and an improved bagging facility ensured that AEL had the necessary resources to expand the emulsion operation in 2007.

8 CHALLENGES FOR THE FUTURE

Test site results have been achieved by close supervision and committed personnel. Successful translation of these into the wider operational environment requires that the following elements be developed to a high degree of reliability and completion:

- system components,
- product quality
- procedural specification
- creation and delivery of training material
- acceptability by operators

Challenges that will therefore have to be met include:

1 Guaranteeing that a quality product is delivered to every blast hole every time.
2 Managing the pump maintenance costs and logistics of having a large fleet of explosive pumps underground.
3 Minimizing waste and ensuring that a fixed mass of emulsion is batched per blasthole.
4 Safe and environmentally sound destruction of used emulsion bags.
5 Creation of logistical system for transport of large volumes of emulsion between supplier manufacturing facility and the face.

9 OPPORTUNITIES

Depending on local circumstances, some of the benefits will apply to other mines. Improved cost-effectiveness will create opportunities for exploitation of lower grade deposits. As yet unquantified outcomes include the environmental effects arising from the reduction in the production of noxious gases by blasting, the reduction in pollution from run-off of rain water from surface dumps of broken rock from mines.

10 CONCLUSION

Perseverance and a basic belief that there is a better way of doing things have brought the work of achieving an "explosives free" mining environment closer than it ever was. Whereas previous trials have been abandoned for lack of any realistic hope of success, the continued co-operation between the explosives supplier and Kopanang personnel holds real promise of a favourable outcome. Close collaboration with minimal concern for petty personal or corporate secrecy has been the key element in enabling the work to reach this point, and it will continue in the same way.

The indications are that pumpable emulsion and shock tube initiation is a feasible element of narrow reef mining, offering a number of opportunities to reduce overall mining costs. The chequered history of the development of pumpable emulsions for use in narrow tabular deposits has reached its current stage through the combined efforts of people from different backgrounds and with differing motivations. The success achieved to date is testimony to the synergy that common purpose creates.

If the bagged delivery system proposal had been pursued earlier then perhaps the current position might have been achieved sooner.

The vision of the "explosives-free" environment for narrow reef mining has thus been realized through the development of an initiator, that is the shock tube, a pumpable product which only becomes an explosive when mixed in the blasthole and a suitably lightweight and robust pump suitable for use in narrow reef stoping for use in conjunction with similar products used in development tunnels. Benefits associated with these developments include improved

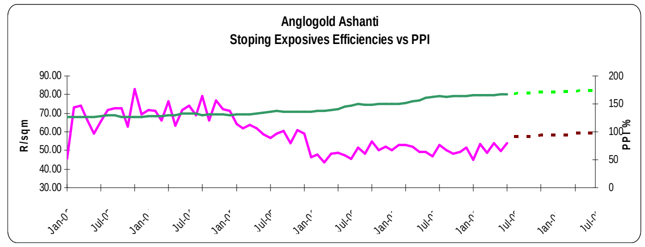

Figure 43. Unit explosives cost versus PPI

workplace safety, cost savings, and reduced creation of pollutants.

In figure 43, the beneficial effects of the successive interventions aimed at reducing the cost of explosives on AngloGold Ashanti mines can be seen and they are:

- Replacement of CFIC or EDDs by shock tubes in 2003 – 2005
- Removal of ANFO from stoping in 2005-2006
- Incoming successful implementation of bulk pumpable explosives into narrow reef stoping

The relatively flat graph of unit explosives costs against a rising PPI means that in real terms, unit explosives costs have been reduced.

11 ACKNOWLEDGEMENTS

The authors are grateful for the permission granted by their respective principals, AngloGold Ashanti and AEL, to publish this paper. Thanks are also extended to the following collaborating institutions

- African Explosives Ltd (AEL)
- MAXAM Dantex SA
- SASOL Nitro
- Lakho Mining Services
- CSIR-Miningtek
- RUMEG (Pty) Ltd
- ICTUS Pumps

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