



Introduction of drill and blast utilizing pneumatic rock-drills in a Rwandan artisanal underground mine

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Synopsis

The authors were tasked with introducing safe and efficient drilling and blasting practices to a small artisanal mine located in the Eastern Province of Rwanda. The current mining methods employed at the mine are inefficient as rockbreaking is conducted utilizing hammer and chisel, or by drilling and blasting with no understanding of the rockbreaking process. The paper discusses the drill and blast cycle, as well as the associated activities necessary to support the mining cycle. The challenges and critical aspects in implementing drill and blast activities are discussed, as well as some of the technical aspects, *i.e.* drilling pattern, initiation and timing of the blast, and handling of misfires. Ancillary issues such as ventilation, training, communication, effective planning, logistics, and human relations are also examined.

Keywords

rockbreaking, drill and blast cycle, drilling pattern, blast timing, misfires, artisanal mining.

Introduction

Rockbreaking accounts for 60% of the direct mining costs in small-scale mining. When drilling and blasting practices are poorly implemented the process becomes inefficient and dangerous. The introduction of a proven drill and blast system to artisanal mining offers a number of challenges, with safety being considered the backbone of any sustainable operation and a standard that cannot be compromised.

The paper is based on a project at an artisanal cassiterite (tin) mining operation in a small village in the Kayonza District in the Eastern Province of Rwanda. A drill and blast programme utilizing industry best practice was introduced at an underground development end; a raise developed at an inclination of 23 degrees. The underground working is accessed via a 17 degree incline shaft with an operating length of 105 m and serviced by a single drum winder. The winder operates at a speed of about 3.5 m/min hoisting two 1 t wagons from the bottom of the incline to surface for tipping (Figure 1).

The drill and blast programme was implemented over a three-week period. As part



Figure 1 – 1 t hopper being hoisted out of the Incline shaft

of the work, a number of drilling patterns and cuts were investigated and a recommended pattern for the 1.2 m by 1.5 m raise was established. As part of the implementation process a time study was conducted of the development-related work cycle from the entry examination to the initiation of the development end.

The challenges that arose with dry drilling in the schist and mica-quartz rock, as well as the issues of dust and ventilation, are discussed. The paper highlights the challenges and importance of the use of direction and grade lines, proper blast-hole marking and drilling, and the use of the burn cut.

As part of the implementation programme, the actual cost of development was established, thereby allowing future mine

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plans to include accurate estimations of the cost of small-scale development activities. Recommendations are made for further work to ensure that the drilling and blasting operations remain safe and are conducted as efficiently as possible.

Scope of work

The initial scope of work for the team of mining diplomates (National Higher Diploma in Mining Engineering) from the University of Johannesburg was to assist with the implementation of drilling and blasting in one of the underground workings on the mine. The raise was selected as high-priority development end, as the raise was urgently required to (1) to provide a second egress for workers in the G1 gallery and B1 stope and (2) improve the ventilation of both working places, as all workings were dependent upon natural ventilation and the use of compressed air for ventilation.

Current drilling and blasting practice

Currently, incline and tunnel development is through the use of pneumatic rock-drills (without air legs). The initial incline development work appears to have been excavated and supported in a reasonable manner. The final 30 m of development in the horizontal haulage was poorly developed by contractors. The tunnel was not excavated straight, nor blasted to the intended width or height, with a blasting advance of the order of 0.50 m.

Flat end development is blasted utilizing six to nine blast-holes without creating a free-breaking point, *i.e.* no wedge or burn cut. Smaller ends are blasted utilizing four or fewer blast-holes. Holes are charged with a single 25 mm cartridge of emulsion (SuperPower90) and three 100 g cartridges of ammonium nitrate (ANFO). Each blast-hole is charged utilizing a capped fuse with the miner required to cut the safety fuse and secure the blasting cap to the end of the fuse. No ignition cord is utilized, the timing of the round being a function of the fuse length and the order of initiation of the fuses. No blast procedures appear to be in place other than strict control of the explosive issued by the 'magazine master'. All explosives are stored in the basement beneath the mine offices.

Condition of mining equipment and infrastructure

The incline shaft and horizontal haulage are equipped with a set of rails that needs to be rehabilitated, and in some sections replaced. The initial installation of the rail lacks sufficient sleepers and ballast, with many rail joints (fishplates) incorrectly connected. Rail switches need to be either removed or replaced; no switches were found in good working order within the underground workings. The tracks should be properly graded and reinstalled on line and grade.

The drill-related equipment is old and the rock-drills, pusher legs, and lubricators all require replacing. It was pointed out to the mine management that the use of hydraulic oil as a lubricant is poor practice—hydraulic oil is not intended to be used for pneumatic rock-drilling as it expands as it heats up, causing damage to the lubricators and the rock-drill and pusher leg seals. Taper rods (1.2 m) are in short supply with only two taper rods available for the development.

Small-scale development mining

The plan to transform the current 'artisanal' development practices into 'small-scale' development focused on the proper marking and drilling of the blast-holes and the proper charging and timing of the blast round. The initial development blast pattern for the 1.2 m by 1.5 m raise was based on a 15 blast-hole pattern. Based on the rock conditions and observed fragmentation the blast round was modified to a 13-hole pattern. Subsequently, the 5-hole burn cut was reduced to a 4-hole burn cut (Figures 2 and 3) with no adverse effects on the blast advance or fragmentation. During this period the local foreman was identified as 'champion' and was trained in the marking-off of the development round, taking line and grade, the proper methodology of charging the blast-holes, and the correct sequencing (timing) of the blast-holes utilizing safety fuses.

The supply of service water for dust allaying purposes posed a problem for the initial 10 days of the implementation programme until a storage tank was installed on surface and used to gravity-feed water to the rock-drill. The availability of qualified rock-drill operators remained a problem throughout the training period; however, 15 workers were subsequently

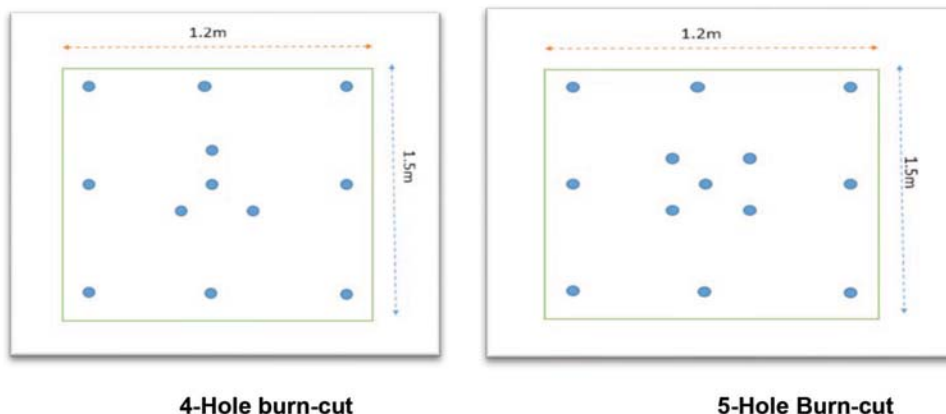


Figure 2—Development round utilizing a 4- or 5-hole burn cut

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Figure 3—Underground rock-drilling and completed development round (11 holes) utilizing a 5-hole burn cut

trained on a week-long course provided by Atlas Copco. The biggest hurdle, and one which still remains, is the concept of creating a free-breaking face through the use of a burn cut. When left unsupervised the miner would attempt to charge the ‘free hole’ or do away with the burn cut completely. Figure 4 depicts the total time for load and haul activities based on the time study results shown in Table I. The analysis of the data highlights the following issues that affected the overall efficiency of the load and haul cycle.

- There was not a dedicated crew for cleaning, hence there was a variation in the time taken to fill the wagons
- The shovels used were of poor quality
- The rails were not installed to standard, which led to wagon derailments. Insufficient sleepers were used to support and maintain the gauge of the track; material was allowed to accumulate between the rails resulting in the wheels of the wagons running on the dirt rather than the rail; fishplates were often missing or improperly installed, allowing for large gaps between the rail and an uneven elevation of the rail. All these factors, as well as others, contributed to the derailments

- The tipping area on surface was too small, necessitating the wagons to be partly offloaded utilizing shovels
- The absence of proper equipment extended the period of the entry examination; for example, no barring tools were available, which led to either no barring being done or barring utilizing a 2 kg hammer. The mine had no procedure in place for removing misfires (scraper wire), which resulted in delays in removing misfires and the dangerous practice of utilizing a hammer and chisel to remove misfires.



Figure 4—Loading and hauling

Table I
Load and haul cycle

Activity	Time	Day														
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Loading of the first wagon	Minutes	no loading and hauling	20	17	no loading and hauling	10	15	17	no loading and hauling	no loading and hauling	15	17	10	15	23	23
Loading of the second wagon			15	15		13	20	21			22	20	15	18	12	18
Coupling			1	1		1	1	1			1	1	1	1	1	1
Moving of the wagon from v9 to bottom of incline		4	4	20	16	14	20	21	15	13	4	4				
From bottom of incline to tipping point		21	20	19	20	25	20	23	21	22	19	23				
Emptying of first wagon		18	15	19	21	25	20	18	19	18	22	19				
Emptying of second wagon		30	20	15	20	28	15	24	22	21	25	30				
Total time for loading and hauling		0	109	92	0	97	113	131	0	0	113	124	103	108	106	118

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Table II

Drill and blast cycle		Day														
Activity	Time	1	2	3	4	5	6	7	8	9	10	11	12	13	14	
		Marking of the face	minutes	1	3	2	3	2	3	no drill and blast	no drill and blast	2	4	3	6	3
Drilling	minutes	304	63	138	167	286	316	93	148			320	150	90	85	161
Charging up	minutes	40	42	39	49	50	45	37	35			36	25	36	33	30
Total	minutes	344	108	179	219	338	364	132	187			358	181	129	120	194
Blasting	exact time	18:10	15:59	15:41	16:45	17:20	17:08	14:15	15:03			16:15	16:03	14:24	10:56	18:13
Advance	metres	0	0.6	0.75	0.74	0.65	0.8	0	0	0	1.1	0.6	0.7	0.8	0.83	0.7

Figure 5 depicts the total time for the drill and blast activity over a 15-day period, based on the time study results presented in Table II. The drill and blast cycle was also delayed due to many contributing factors, which include the following.

- The lack of a dedicated rock-drill operator, which resulted in the miner conducting the drilling on his own. The University of Johannesburg team also assisted with drilling operations on numerous occasions due to a lack of a drill operator
- The rock-drill machines were not serviced regularly and therefore did not operate efficiently, *i.e.* over-use of hydraulic oil (incorrect lubricant), inability to receive water for dust suppression due to damaged water connections on the drill
- There was a shortage of drill steel and drill bits. When the drill steel became stuck in the face, time was wasted in finding additional drill steel and bits to continue drilling, and in two instances blasting had to take place with the round being only partially drilled as there was no drill steel to complete the development round
- There were numerous leakages in the service columns, which had a direct impact on the efficiency of the drilling. In addition, the compressed air column reduced from 75 mm to 50 mm in diameter, which also affected the compressed air pressure. One day was lost when the compressed air column collapsed in the haulage due to incorrect suspension of the column
- The lack of skilled drill operators impacted directly on the drilling cycle and at times led to drill steel becoming stuck in the face. In addition, the only available drill operator worked at a nearby (1.5 km away) surface operation as a supervisor, which often led to unnecessary delays in the commencement of drilling activities
- Delays were also experienced due to a shortage of, or delay in the delivery of, fuel (diesel) to operate the compressor, as well as a shortage of hydraulic oil.

The following factors were determined as influencing the advance (Figure 6) of the development round:

- The blast-holes were not drilled to the full length of the jumper, which slowed the rate of advance
- The lack of knowledge of grade and direction lines by the rock-drill operator caused holes to be drilled off-line
- The change from the 5-hole burn cut to a 4-hole burn saved explosives and increased the size of the fragmentation, but had no effect on the rate of advance
- The schist absorbed the majority of the blast energy, therefore in cases when the dip of the raise needed to be increased, additional holes were required to ensure the full break of the top blast-holes
- The intersecting of mica quartzite made chiselling redundant.

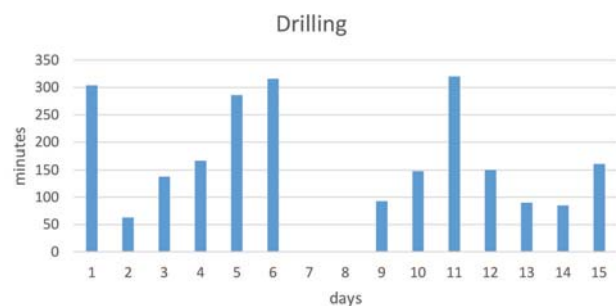


Figure 5—Drilling times

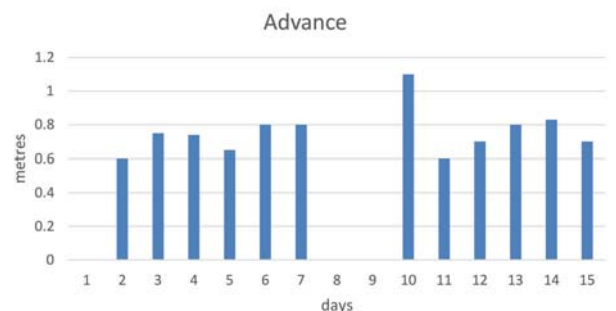


Figure 6—Advance of the raise

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Environmental conditions

The underground working is accessed using an incline shaft at an angle of approximately 17 degrees spanning a distance of 105 m, with a horizontal haulage having planned dimensions of 2 m by 2.5 m. The haulage continues horizontally for 130 m with crosscuts developed on cassiterite veins along the haulage; from vein no. 4 to vein no. 10. The gallery is not equipped with a mechanized ventilation system, *i.e.* no ventilation fans or columns to provide air. During the time of the project, the no. 4 vein stope and raise, which had holed through to surface, was used to provide fresh air to the underground workings. The natural ventilation, however, becomes less effective with the increasing distance in the haulage.

Natural ventilation is the only means of suppressing the dust that is generated by the dry drilling and blasting. The lack of ventilation created a dusty environment and necessitated the use of dusk masks during drilling operations. A fan is planned to be purchased, but to date has not been procured. In the interim, the holing of the development of the raise into the B1 stope will temporarily improve the ventilation conditions by providing fresh air to the gallery, as well as provide a secondary or emergency escape route.

Logistics

The load and haul cycle commences at the beginning of the working shift. Two 1 t wagons are used to clean the blasted rock. A tramming crew comprising four workers (cleaning crew) pushes the wagons from the tipping point to the shaft entrance, where the wagons are connected to the winch rope and sent down the incline at a speed of 3.5 m/min. Once the wagons reach the bottom of the incline they are uncoupled from the winder rope and pushed by hand to the working place by the cleaning crew. The cleaning crew then loads the blasted material into the wagons using hand shovels. The

loaded wagons are pushed to the bottom of the incline, where they are coupled together and connected to the winding rope. The winch operator is then instructed to start the winch and the wagons are hauled to the top of the incline shaft. Once on surface, the cleaning crew disconnects the wagons, pushes them to the tipping point, and unloads the material to either the ore tipping point or the waste tipping point.

One-ton wagons are used for cleaning ends in the incline. A time study conducted on the loading and hauling of the development rock indicated an average cycle time of 110 minutes using two 1 t wagons for cleaning a 1.2 by 1.5 m raise. For the entire cleaning cycle two trips (four wagons) are required to clean the 1.5 m by 1.2 m end, requiring a total of 181 minutes (Table III).

Operating costs

Explosives costs (Table IV) are based on the June 2016 cost from suppliers in Uganda and include transport costs from Uganda and within Rwanda, as well as import duties of 34%. Explosive costs equate to US\$2.76 per charged blast-hole (US\$56.95 per metre). No estimate has been made for spillage, misfires, or other wastage.

Compressed air costs are estimated on an average of 3.0 hours to drill 12 blast-holes and based on a compressor fuel consumption rate of 6.84 litres of diesel per hour at a fuel cost of US\$1.19 per litre. Lubricating oil is based on a consumption of 0.25 litres of hydraulic oil per working face. One litre of hydraulic oil retails at US\$4.87. Drill bit consumption is based on an average bit life of four weeks and a drill bit cost of US\$23.36.

Labour requirements are based on a single working shift consisting of a miner, foreman, driller, four cleaners, and a quarter (¼) salary for the incline winch operator. Salaries used to estimate labour costs are based on June 2016 rates. Labour equates to US\$885 per month or US\$44.21 per developed metre.

Table III

Loading and hauling cycle times

Activity	Time	Total time (min)
Loading first wagon	09h25-09h45	20
Exchanging wagons	09h45-09h50	5
Loading second wagon	09h50-10h14	24
Coupling	10h14-10h20	6
Moving wagons from V9 to entrance	10h20-10h37	17
Moving wagon from entrance to tipping point	10h37-10h50	13
Emptying wagons	10h50 -11h05	15
Moving wagons to entrance	11h05-11h10	5
Wagons down incline	11h10-11h23	13
Wagons moved to V9 entrance	11h23-11h34	11
Loading of 1 st wagon	11h34-11h55	21
Coupling wagons	11h55-12h01	6
Loading 2 nd wagon	12h01-12h26	25
Total time to clean		181 minutes

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Table IV

Explosive cost, 1.2 m drill-hole

Item	Unit cost	Cost per hole* (US\$)
Detonator	0.30 each	0.45
Safety fuse (1.8 m per hole)	0.34 per metre	0.93
Super90 emulsion	85 per case [#]	0.69
ANFO (300 g per /hole) (3 sticks)	1.3 per kg	0.59
Total cost per blast-hole		2.66

*Includes transport charges US\$3725 Uganda and US\$3725 Rwanda and 34% duty
[#]188 cartridges per case

Table V

Summary of operating costs

Description	Monthly Cost (US\$)
Development End	1.2m by 1.5 (4.5t)
Development Blast Holes	11 holes
Consumables:	
Explosives and Accessories	585
Diesel	488
Lubricator	24
Drilling Bit	23
Labour	717
Sub total	1838
Contingency (15%)	276
Total	2114
Cost (US\$/m)	US\$106/m
*Based on 24 shifts per month	
Exchange rate based on RWF780:US\$	

The operating costs of US116 per metre for a 1.2 m by 1.5 m development end is shown in Table V and is based on actual operating costs as experienced during trial mining in June and July, 2016. A contingency of 15% has been applied to cover unbudgeted costs. Operating costs are based on single shift/blast mining activities with no night shift cleaning. It should be noted that the potential does exist for two blasts per shift, which was achieved on one of the working shifts during the project.

Capital expenditure

Prior to trial mining, five rock-drills, pusher legs, and assorted tapper rods and drill bits (41 mm) were purchased from Atlas Copco for a cost of US\$57 000 (excluding any duties). As part of the trial mining programme, air (25 mm) and water (19 mm) columns were ordered but due to cash constraints were not purchased.

Challenges

Safety in any mining operation is a prerequisite and a non-negotiable standard that cannot be compromised in any way. The underground mining is currently carried out by contract miners. The mining operations are adversely affected by a lack of a skilled management and labour, outdated and poorly maintained equipment, air and water services, and inadequate logistics system.

The shortage of a skilled workforce is aggravated by the use of contract mining. There is minimal transfer of knowledge and skills to the company employees, as the motivating force behind the mining operation is production. Thus, all work conducted is based on the immediate realization of production and decisions are made to support current production with no consideration to medium- or long-term requirements.

Another key challenge for the company is the lack of well-trained and competent rock-drill operators. The company does not have training facilities where it can train its employees and ensure that they are competent in their area of work. Mining is mostly conducted on economic veins using traditional chisel and hammer techniques, an artisanal mining system that is not efficient.

The transition from chiselling to the use of pneumatic hand-held rock-drills is a major challenge, as the rock-drill operators are accustomed to drilling few blast holes, general three to five holes. The new system introduced involves creating a free-breaking point utilizing a four-hole burn cut as well as easers and sides holes, thus some 12 blast-holes are drilled to a 1.2 m length. The pneumatic rock-drill is powered using a mobile compressor unit that consumes an average of 6 litres of diesel per hour, which is not always readily available with the gallery often restricted to 20 litres per day, *i.e.* just enough for a specific shift. The lack of

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adequately trained rock-drill operators adds to the drilling time, thus increasing diesel consumption. The poor condition of the rock-drills and air legs further adds to the delay in drilling, as well as to the excessive consumption of lubricating oil. The concept of a blasting design with a free-breaking point in development ends is foreign to the workforce; an example of this is the charging crew wanting to charge all of the holes in the burn cut. The other pressing issue is the lack of equipment and tools for dealing with any misfires on the development face. There is a need to obtain tools such as a two-way blowpipe, an approved scraper wire, socket plugs, and approved explosive transportation bags for explosives and accessories.

Environmental conditions are not ideal, as the water services are inadequate with water being supplied on surface via a small tanks filled by a water tanker truck on request. Thus, dry drilling is often conducted, thereby causing the work area to be inundated by dust. The mine requires a proper water delivery system and ventilation system, as the current water pipes are small and old and no ventilation fan is available. Similarly, the compressed air system requires an upgrade, as the compressed air column is too small (50 mm diameter) and suffers from numerous leaks which reduce the efficiency of the system.

From a material handling point of view, the trackbound wagons operate on a track system that is neither maintained nor serviced. The rail has unnecessary bends with a significant portion of the trackwork unsupported by sleepers. The installation and maintenance of the trackwork is of a poor quality and consequently derailments are an issue. There are no stopping devices to arrest runaways in the incline or haulage.

The labour issue is further exacerbated by the lack of planning, leadership, organization, and an effective management system.

Conclusions and recommendations

Several challenges were encountered during the project. Critical issues encountered included the lack of service water, human resources in regard to a dedicated miner and rock-drill operator, regular supply of consumables such as diesel (compressor) and lubricating oil, and the timeous delivery

of explosives. The drilling equipment employed is old and required regular repairs, as did the compressed air column installed in the incline shaft. Explosive accessories, such as two-way blowpipes, scrape wire, charging stick, and pinch bar were either inadequate or nonexistent. Management of the development process was deficient in planning and generally the organization was poor. Blasting standards were found to be nonexistent, with no understanding of the risks associated with the drill and blasting process.

Based on the three-week trial development programme it is recommended that mine management conduct a monthly planning and production meeting with a breakdown of the objectives, stipulating outcomes and targets set for the month. Through effective planning, all the required consumable materials (exception) could be delivered to the shaft office at the commencement of each week, thus reducing the delays currently being experienced. Delivery of the explosive should be confirmed on a daily basis with the explosives delivered at a set time, for example 11 am, which is the general time that drilling commences. Regular training sessions should be held so that employees are updated on best practices for drilling and blasting. In order to transform the incline shaft to small-scale mining, equipment, and spares are required to be purchased from reputable suppliers and installed. These include columns for compressed air, water, and ventilation; new track and tipping facilities; equipment for explosives, such as scraper-wires, two-way blowpipes, and charging-sticks. The initiation system should be upgraded to at least a fuse and ignition cord system.

Dedicated people must be selected for the key positions of miner, foreman, and rock-drill operator. Currently, management continues to move personnel on a regular basis, which means that continuity of the training programme is lost and untrained personnel introduced into the high-risk operation of drilling and blasting.

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