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Causes and consequences of inefficient drilling and blasting in mine development headings: A case study of hard rock gold mining operations in northern Tanzania

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ABSTRACT

Despite the introduction of the perimeter blasting technique at the Tulawaka Gold Mine, the mine continued to suffer from an overbreak of mine development headings, with an average overbreak of 24% every 22 m, which is approximately twice the acceptable 10% overbreak. The causes of this problem include ineffective drilling practices and uneven and excessive charging of explosives, resulting in a slightly high powder factor of 3.94 kg/m^3 instead of 3.8 kg/m^3, as per the design. The problem of overbreak resulted in ore dilution, a longer mine development cycle time and additional costs of approximately US$ 45 per cut, especially in mucking and hauling processes. This study proposed and recommended new drill and blast designs to solve this problem. Compared to the existing design, the proposed new drill design has a total of 12 fewer drill holes; this is a significant number of holes, which significantly reduces drilling costs. The proposed new blast design consumes approximately 25 kg less ANFO than the existing practice. Moreover, the study showed that large drives in the Star and Comet in the Geita Gold Mine suffer the most from the problem of ineffective advance per cut. One of the causes of ineffective advances per cut is the low amount of explosives used per cut compared to the planned amount.

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INTRODUCTION

In any mining operation, drilling and blasting are crucial tasks. Typically, they account for 30–40% of mining-related costs (Manguye, 2023; Verma *et al.*, 2016; Ndibalema, 2008). When mining companies do not reach the target stope/drive size or the expected advance metres per cut, they face longer drilling, blasting, and hauling cycle times, resulting in increased labour, drilling, blasting, and hauling costs, as well as delays in the mining and production schedule. Moreover, in case of overbreaks, the companies face increased costs for mine support to ensure safe working environments and more dilution of ore, reducing the companies' profits (Manguye, 2023; McFadyen *et al.*, 2020; Segaetsho and Zvarivadza, 2019; Singh 2018; Verma *et al.*, 2016; Bennett, 2009). Most of times, geological factors (ground conditions), drilling and blasting factors, or both contribute to poor blasting performance that prevents the intended advance metres per cut or the intended stope/drive size from being achieved. Drilling and blasting factors, as well as drive geometry, are controllable, while geological factors are not (Manguye, 2023; Segaetsho and Zvarivadza, 2019; Singh 2018; Verma *et al*., 2016; Bennett, 2009; Germain and Hadjigeorgiou, 1997; Ibarra *et al*., 1996).

The Tulawaka Gold Mine and the Geita Gold Mine, two gold mines in northern Tanzania, are the case study utilised to assess the reasons behind inefficient drilling and blasting in mine development headings in hard rock mining operations. In 2005, the Tulawaka mine began as a conventional open pit with two merged pits: East Pit (Main pit) and West Pit. But in 2008, it switched to underground mining with the West Pit as access for underground mining (State Mining Cooperation, 2016; Mining Technology, 2014; Bennett, 2009). Geita Gold Mine is an ongoing open pit and underground mine, where historically underground mining took place between 1934 and 1966, and more recently open pit and underground mining took place at Geita Hill, Lone Cone, Nyankanga and Matandani/Kukuluma open pits from 2000. Recent underground operations began at Star and Comet in 2016 and at Nyankanga in 2017 (AngloGold Ashanti, 2024, 2022). The Star and Comet underground mine is the case study in this study.

According to data gathered in 2009, during the development of mine headings, Tulawaka Gold Mine encountered issues with overbreaking and non-smoothness of the walls of the mine drives/openings. The mine experienced about 30–40% overbreak of development openings at the start of underground mining, as perimeter blasting techniques were not used to minimise overbreak and leave clean-cut solid walls. Perimeter blasting technique is a method designed to minimize over break and leave clean-cut solid walls of underground drives by drilling the holes at the perimeter of the drilling pattern close together and loading them with very light continuous explosive charges and firing them simultaneously at or near the last delay (Bennett, 2009). Following the implementation of perimeter blasting methods, the degree of overbreak was reduced by almost 50%. In 2009, the mine experienced an estimated overbreak of 15–20%, above the 10% tolerable overbreak. There were two new perimeter blasting techniques introduced: low-energy ANFO and low-impact power cord (yellow chord). Because operators often charged the entire face with low-impact ANFO instead of charging the perimeter holes with low-energy ANFO and other charged holes with high-energy ANFO, which resulted in poor blasting, the low-energy ANFO technique was abandoned. Using the low-impact power cord technique, perimeter holes are charged with high-energy ANFO, while the primer is coupled with a low-power cord, which is left unconnected outside the hole. The power of the ANFO in perimeter holes is reduced by this cord. Although an overbreak of 10% or less could not be achieved, the issue of operators charging the entire face with low-impact ANFO was resolved with the employment of this technique. Because the mine was still experiencing the overbreak, it was imperative to review the existing perimeter blasting technique. Optimisation of drilling and blasting parameters was required to manage the overbreak and produce clean-cut solid walls (i.e., smooth walls), in ore development headings (Bennett, 2009). Currently, the Star and Comet underground mine in the Geita Gold Mine is not making sufficient progress in the development headings (Manguye, 2023). As a result, the targeted advance metres per cut are not achieved. It is common for the actual advance metres per cut to differ from the anticipated advance metres per cut, typically by more or less than the permitted deviation of 0.1 m (i.e., ± 0.1 m). Therefore, it is critical to identify the root causes of the inability to reach the targeted advance metres per cut, identify the drive size that is seriously problematic, and choose the most effective means of resolving these issues (Manguye, 2023). Overbreak problems at the Tulawaka and Geita mines significantly increase operational costs, including drilling and blasting costs as well as labour and material costs. The problems also cause delays in the mining and production schedule due to longer drilling, blasting, and hauling cycle times. Moreover, they lead to more dilution of ore, reducing profits (Manguye, 2023; Bennett, 2009).

METHODS AND MATERIALS

Locations and geology of the study areas

The study areas are the Tulawaka Gold Mine (now the Stamigold Tulawaka Gold Mine) and the Geita Gold Mine (GGM) in northern Tanzania (see Figure 1). The study areas are located in the Lake Victoria Goldfields of Tanzania (Manguye, 2023; Bennett, 2009). In Tulawaka, the mining area consists mainly of sequences of volcanic sediments, greywacke, and ferruginous sandstone. In the transitional contact between the sediments and ferruginous sandstone, fine to coarse biotite and pervasive sericite alterations sometimes occur. Numerous felsic and mafic dykes, ranging in width from 5 to 50 metres, intersect the mining sequence. Quartz veins, which are found in favourable areas with moderately steep north-dipping high-strain zones, are associated with gold minerals.

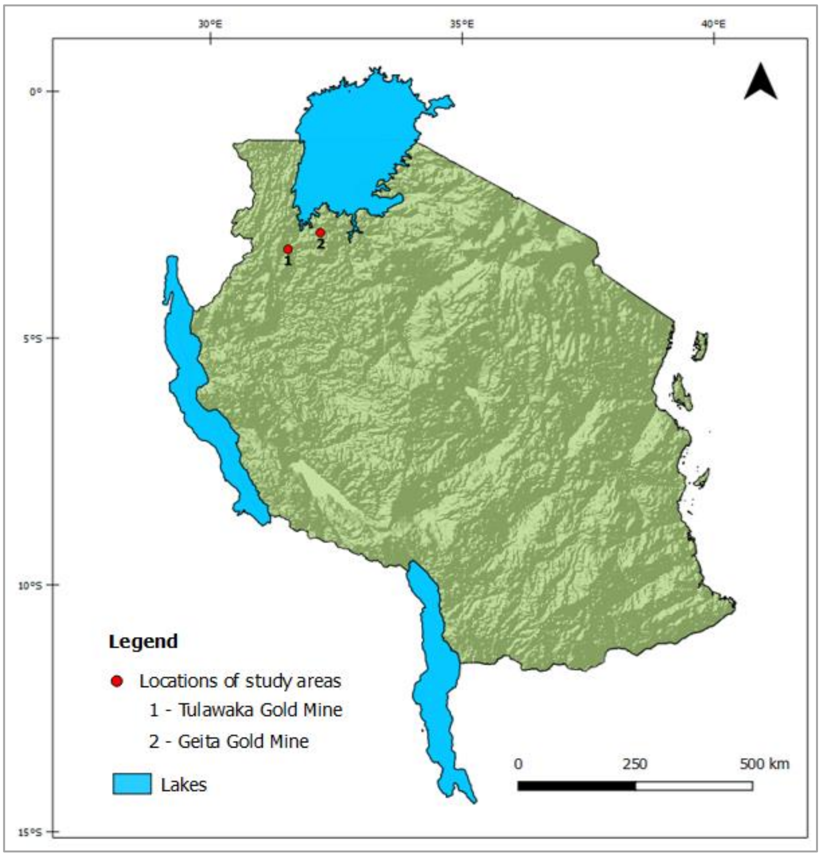

Figure 1. Map of Tanzania showing the locations of the Tulawaka Gold Mine and Geita Gold Mine in northern Tanzania.

The thickness of the quartz veins carrying gold minerals varies from 60 cm to 4 m. The specific gravity of the quartzite in the region is 2.7, and its tensile and uniaxial compressive strengths range from 10 to 20 MPa and 150 to 300 MPa, respectively. Block-like in shape, the rock type has a joint plane orientation that's dips into the mine face and an intermediate joint plane spacing face (Bennett, 2009). At Geita, the geology of the mine area consists of low metamorphosed Nyanzian chemical sediments, including graphite-bearing sediments and oxide-bearing sediments, intruded by post- and syngenetic igneous rocks, including diorites, quartz-feldspar porphyrites, lamprophyres, gabbroic dykes and other granitoids, pyroclasts and volcanic rocks (Manguye, 2023).

Fieldwork, data collection and data analysis

Fieldwork at Tulawaka Gold Mine took place in April 2009, during fieldwork of the undergraduate final year project of George Bennett, graduate of the University of Dar es Salaam, while fieldwork at Star and Comet in the Geita Gold Mine took place from March to April 2021, during fieldwork of the undergraduate final year project of Samson Manguye, graduate of the University of Dodoma. Data were collected at critical times when studies were needed to investigate the causes of overbreak and low advance metres per cut for in Tulawaka Gold Mine and Star and Comet underground mine in the Geita Gold Mine, respectively. For Tulawaka Mine, headings with drive size of 4.0 m x 4.0 m will be examined, while for Star and Comet, headings with three different drive sizes: 5.5 m x 5.5 m, 5.0 m x 5.0 m, and 4.5 m x 4.5 m will be examined.

Two types of data were collected: primary and secondary data. Primary data were collected through direct measurements, which included measurement of hole diameter, hole length, spacing, burden, number of holes drilled, advance metres per cut, number of holes charged, and amount of explosives used per cut. Secondary data were obtained from mining production reports, these data include design drive sizes, actual drive sizes, percentage of overbreak, percentage of ore dilution, design and actual advance metres per cut, and designed volume and tonnage of the blasted material for different development headings. In addition, designed drilling and blasting parameters were also obtained as secondary data. The drilling parameters included drill hole length, hole diameter, burden, spacing, and total number of holes per cut, while the blasting parameters include explosive type and density, explosive consumption per cut, and explosive consumption per hole. Furthermore, discussions/consultations with mining engineers, surveyors and blasting crew were also used to collect qualitative and quantitative hand-on information on the drilling and blasting practices in the study areas. Information on the nature of rock mass, orientation of joint plane, tensile and uniaxial compressive strengths of the rock masses was collected. To analyse the effectiveness of drilling and blasting practices, a comparison was made between actual and designed drilling and blasting parameters collected. This helps to identify parameters that fall inside or outside the designs; the parameters that fall outside the designs are those that contribute to overbreak and low advance metres per cut. The analysis to determine the ideal drilling and blasting parameters used several mathematical formulae, equation 1 to 16, which are listed in *Appendix A* (Massawe, 2007). The mathematical formulae were used to calculate the number of perimeter holes, perimeter spacing, perimeter burden, hole spacing, hole burden, percentage overbreak, theoretical tonnage per cut, actual tonnage per cut, theoretical powder factor, actual powder factor, explosive used per hole, and explosive used per cut.

RESULTS AND DISCUSSION

Development headings in the Tulawaka Gold Mine

Based on the comparison between actual and designed parameters in headings with designed drive size of 4.0 m x 4.0 m, the results indicate that of the 11 different drilling parameters examined, almost 73% (*n* = 8) of the actual drilling parameters do not match the designed parameters (see Table 1). This is a significant percentage indicating that the drilling practice is ineffective. Ineffective drilling practices contributing to the failure to achieve the designed drive size of 4.0 m x 4.0 m in the study area. For example, in Table 2, about 83% (*n* = 5) of the six (6) headings with a designed drive size of 4.0 m x 4.0 m show large actual drive sizes; large drive sizes than the designed size indicate overbreak. The results in Table 2 also show that the mine continued to suffer from overbreak, with an average overbreak of 24% every 22 m, which is approximately twice the acceptable 10% overbreak. Drilling parameters that contribute greatly to overbreak include perimeter hole spacing, perimeter hole burden, lifter burden to perimeter hole, and hole length; all these parameters, except hole length, have values greater than the desired values. In addition, ineffective drilling also causes the actual advance metres per cut to be lower than the design, values range from 2.7 – 2.9 m compared to 3.0 m as per the design (see Table 2). The average advance metres per cut is 2.8 m. Over time, this leads to higher drilling and blasting costs. The raw drilling data are given in Table B.1 and Table B.2, while the raw overbreak data are given in Table B.3 in *Appendix B.*

Table 1. Comparison between planned and actual drilling parameters for advance per cut in 4.0 m x 4.0 m headings at Tulawaka Gold Mine

Parameter	Designed Value (m)	Range of Actual Value (m)	Remarks
Reamer hole length	3.2	2.75 – 3.12	Out of the design
Burn hole length	3.2	2.71 – 3.13	Out of the design
Perimeter hole length	3.2	2.76 – 3.11	Out of the design
Perimeter hole burden	0.6 – 0.8	0.9 – 1.6	Above the design
Perimeter hole spacing	0.6	0.8 – 1.2	Above the design
Lifter spacing	0.6 – 0.7	0.6 – 0.7	Within the design
In face burden	0.8 – 1.0	0.9 – 1.0	Within the design
Regular blasthole spacing	0.96 – 1.2	0.9 – 1.1	Within the design
Lifter burden to perimeter hole	0.8	0.3 – 0.4	Extremely below the design
Lifter burden to cut hole	1.0	0.3 – 0.4	Extremely below the design
Advance per cut	3.0	2.7 – 2.9	Below the design

Table 2. Actual drive size, actual advance, actual and design tonnage, and percentage overbreak in different 4.0 m x 4.0 m headings at Tulawaka Gold Mine

Heading Name	Width (m)	Height (m)	Advance (m)	Volume (m^3)	Actual tonnage (t)	Design tonnage (t)	Overbreak (%)
Level 4 West	4.5	4.5	29.9	605	1635	1292	27
Level 5 East	4.3	4.7	36.6	740	1997	1581	26
Level 6 East	4.4	4.6	20.4	413	1115	881	27
Level 7 West	4.0	4.0	4.7	75	203	203	0
Level 5 East 1	4.2	4.6	9.4	182	460	406	13
Level 4 East 120	4.3	4.6	33.0	653	1762	1426	24
Average	**4.3**	**4.5**	**22.33**	**444.67**	**1195.33**	**964.83**	**24**

According to the drilling design, the total number of holes drilled per drill face is 45, including 17 perimeter holes, 5 reamer holes and 23 regular blasthole, but the results show that the number of actual holes drilled is always smaller than 2 to 10 holes or larger than 2 to 11 holes (see Table 3). This also shows the ineffectiveness of the drilling practice, which also contributes to the overbreaking. The diameter of the reamer holes drilled in the centre of the drill face and not filled with explosives to create a free face is 102 mm, while the diameter of other holes is 45 mm. Based on the analysis of 13 different drill face, almost 70% (n = 9) of the drill faces were charged with larger amounts of explosive, on average about 214 kg, than the planned amount of 183 kg (see Table 3). This overcharging of explosives resulted in a slightly high powder factor of 3.94 kg/m^3 instead of 3.8 kg/m^3, as per the design. This shows the ineffectiveness of the blasting practice, which also contributed greatly to the overbreak. The type of explosive used was ANFO with a density of 0.9 g/cc, while a power gel with a diameter of 32 mm was used as the explosive cartridge. The designed amount of ANFO per hole was 4.575 kg (1.525 kg/m). In addition, the nature of the rock in the ore drive, which is fair and poor in some areas, also leads to overbreak (Singh 2018; Verma *et al*., 2016; Bennett, 2009).

Furthermore, according to field observations and interviews with shift supervisors, mining engineers, and mine operators, the perimeter blasting technique used at Tulawaka has several drawbacks. These include the fact that it is time-consuming, leading mine operators to take shortcuts to save time by plugging a few perimeter holes with yellow cord, and that perimeter holes are heavily loaded with explosives, similar to other blastholes, resulting to the overbreak (Bennett, 2009).

Table 3. Number of holes drilled, holes charged, and amount of ANFO used per cut at Tulawaka Gold Mine

Date	Heading Name	Number of				Total ANFO Used (Kg)
		Total Holes Drilled	Perimeter Holes	Total Holes Charged	sockets	

05/04/2009	Level 5 East	35	17	30	12	200
07/04/2009	Level 5 East	42	18	36	13	175
	Level 7 West	43	18	36	11	225
08/04/2009	Level 6 East	48	17	42	10	225
	Level 7 West	41	18	35	13	175
	Level 7 East	50	19	44	8	250
09/04/2009	Level 6 East	56	17	48	10	200
	Level 7 East	51	18	43	6	250
10/04/2009	Level 5 East	43	18	37	14	150
	Level 7 East	47	18	41	12	250
11/04/2009	Level 5 East	39	16	33	10	150
	Level 7 East	50	18	44	11	275
	Level 6 East	48	17	42	13	250
	Average	**46**	**18**	**40**	**11**	**213.5**

In addition, overbreak results in additional/unbudgeted costs of about $358.8 per 22.3 m advance in the mucking and hauling activities (see Table 4), due to the increase in blasted materials. The mucking and hauling cost of blasted materials at Tulawaka is $1.56/t. The performance of the mine development cycle is lowered when there is a rise in blasted material because it lengthens the time needed for mucking and hauling as well as for scaling and supporting the tunnel to ensure safety. Considering an average of 2.8 m for actual advance metres per cut, the unbudgeted cost per cut comes to $45.05. Moreover, overbreak contributes to ore dilution; in the four headings this study examined, the percentage of ore dilution varies from 7 – 29% (see Table B.4 in *Appendix B*).

Table 4. Mucking and hauling costs of blasted materials at Tulawaka Gold Mine.

Design Tonnage (t)	Actual Tonnage (t)	Budgeted Cost ($)	Actual Cost ($)	Unbudgeted Cost ($)	Advance (m)
965	1195	1505.4	1864.2	358.8	22.3

Due to the ineffective of drilling and blasting practices, which indicate that the existing drilling and blast designs are unfeasible, this study proposes new values for drilling parameters (Table 5) and new values for blasting parameters (Table 6) for a drive size of 4.0 m x 4.0 m to solve the problem of overbreak in the study area. The basis of these proposed designs is the optimisation of the systems parameters. To optimise the drilling parameters, it was determined that the perimeter holes bulge by 10 cm (0.1 m) from the established heading boundary, that five lifter holes are sufficient due to the size of the drive and that lifter burdens of 0.8 m to the perimeter hole and 1.0 m to the cut holes are appropriate. Given the proximity of the perimeter holes, a factor of 15 was used to calculate the perimeter hole spacing using equation 2 (listed in *Appendix A*); the spacing between perimeter holes is 15 to 16 times the hole diameter, while the burden between the breast holes and the perimeter holes is 1.25 times the perimeter hole spacing. Due to the drive size and when using a power gel with a diameter of 32 mm, a factor of 25 (range of factor: 20 – 40) was used to calculate the hole burden using equation 4 (listed in *Appendix A*), and a factor of 1.2 (range of factor: 1 – 1.8) was used to calculate the hole spacing using equation 5 (listed in *Appendix A*). To optimise the blasting parameters, the amount of explosive in each perimeter hole is about 80% of the calculated amount of explosive per hole (calculated using equation 15, see *Appendix A*), because a perimeter hole should be charged with about 20% less explosives than that in a regular blasthole. In addition, the amount of explosive in each cut hole is about 22% more explosive than the calculated amount of explosive per hole, while the amount of explosive in each buffer and lifter hole is about 17% more explosive than the calculated amount of explosive per hole (Bennett, 2009).

Compared to the existing design, the proposed new drilling design has a total of 12 fewer drill holes; this is a significant number of holes, resulting in significantly lower drilling costs, while the proposed new blasting design consumes about 25 kg less ANFO than the existing practice, resulting in lower blasting costs. Figure 2 shows the proposed drilling layout for a 4.0 m x 4.0 m drive size at Tulawaka Gold Mine.

Table 5. Proposed new drilling parameters for a 4.0 m x 4.0 m drive size at Tulawaka Gold Mine

Parameter	Dimension
Hole length	3.2 m
Perimeter hole burden	0.85 m
Perimeter hole spacing	0.68 m
Lifter spacing	0.95 m
In face burden	0.8 m
Regular blasthole spacing	0.96 m
Lifter burden to perimeter hole	0.8 m
Lifter burden to cut hole	1.0 m
Advance per cut	3.0 m
Total drilled holes per cut	**33** (15 perimeters, 5 lifters, 8 buffers, 4 cuts and 1 reamer)
Total charged holes	**32** (reamer remain uncharged)

Table 6. Proposed new blasting parameters for a 4.0 m x 4.0 m drive size at Tulawaka Gold Mine

Parameter	Measurement
Explosive type and density	ANFO (0.9 g/cc)
Total explosive amount per cut	189 kg
Amount of explosive per hole in cut holes	7.2 kg
Amount of explosive per hole in buffer and lifter holes	6.9 kg
Amount of explosive per hole in perimeter holes	4.7 kg

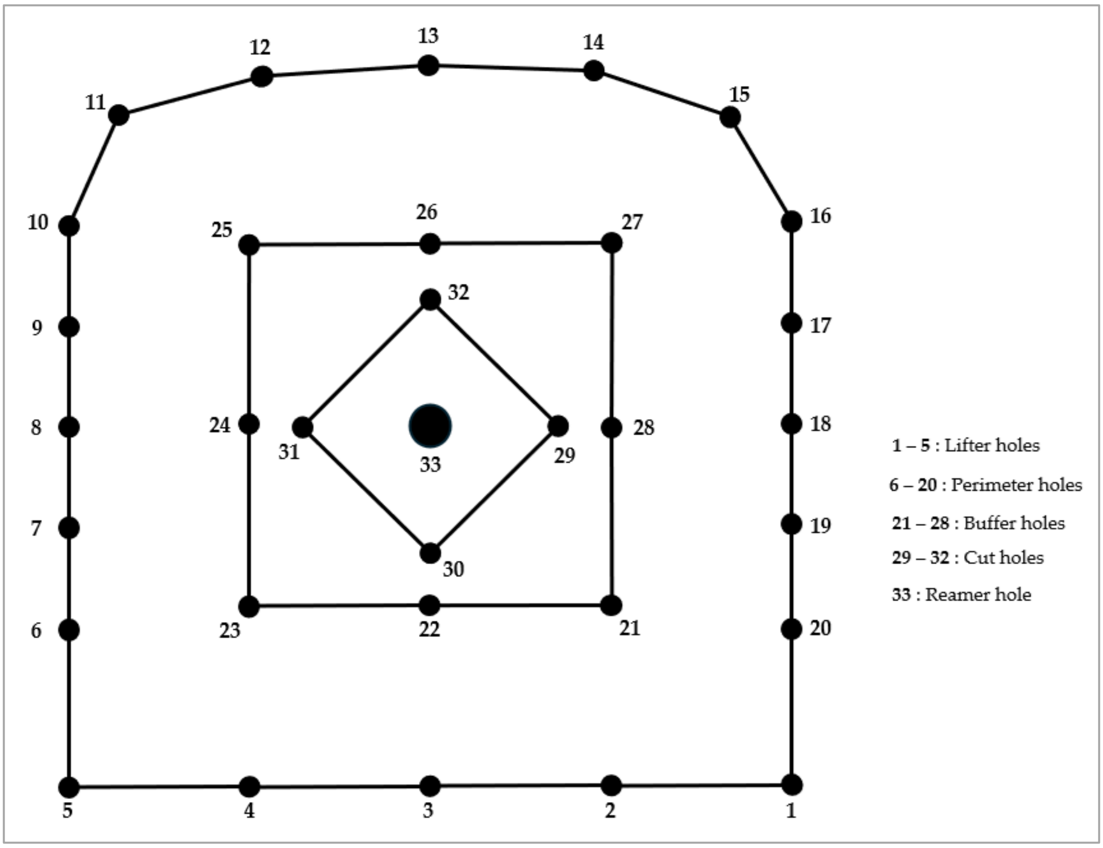

Figure 2. Proposed drilling layout for a 4.0 m x 4.0 m drive size at Tulawaka Gold Mine.

Development headings at Star and Comet in the Geita Gold Mine

Based on the analysis of eight different development headings/drives with different sizes and number of cuts, the results generally indicate that large drives suffer the most from the problem of ineffective advance per cut, especially drives of 5.5 m x 5.5 m, and 5.0 m x 5.0 m (see

Table *7*). The negative deviation values indicate that the actual advance metres were below the planned advance metres, while the positive values indicate that the actual advance metres were above the planned advance. The results show that the planned advance metres per cut were always too low for a 4.5 m x 4.5 m drive, while for drives of 5.5 m x 5.5 m and 5.0 m x 5.0 m, the planned advance metres per cut were either too low or too high. The results also show that deviations of actual advance metres from planned advance metres increase as the number of cuts increases (see

Table *7*). This problem further increases if multiple cuts are used to achieve an advance of more than 4.1 m. Therefore, the hypothesis of this study is that the ineffective advance during heading development increases as the size of the drive and the number of cuts increase. In the future, more data for a small drive size, 4.5 m x 4.5 m, should be analysed to confirm the hypothesis of this study. Table C.1 in *Appendix C* gives the raw data for planned and actual advance metres used in this study.

Table 7. Range of actual advance deviation from the target advance for different drive sizes and number of cuts.

Drive Size (width x height) (m^2)	Heading ID	Number of Cuts	Range of Actual Advance Deviation from Planned Advance (m)
4.5 x 4.5	C31021 DD ACCESS	2	-1.0 – 0.5
5.0 x 5.0	C31051 B ODN	3	-0.7 – 0.7
	C31076 A N3 SLD	5	-1.5 – 0.3
	C31076 A N4 SLD	6	-1.8 – 0.6
5.5 x 5.5	C3916 DECLINE	4	-0.8 – 0.6
	C3921 DSP	4	-1.1 – 1.3
	C3981 PSTN	5	-1.7 – 0.4
	C3981 RAE	2	0.8 – 1.0

Since 135 kg of ANFO is designed to blast a drive size of 5.5 m x 5.5 m with a planned advance of 4.1 m, this study assumes that the planned amount of explosive for blasting a drive size of 5.0 x 5.0 m is 120 kg, 240 kg, and 360 kg of ANFO for 4.1 m, 8.2 m and 12.3 m advance, respectively, while for blasting a drive size of 4.5 m x 4.5 m with a planned advance of 4.1 m, the planned amount of explosives is 80 kg of ANFO. In some cases, drives with the same size and the same amount of actual explosive used per cut of the same advance meters produce different blast results, including actual metres per cut that are below, within or above the permitted value of ± 0.1 m (Table 8). This indicates that the amount of explosive used per cut is not the only possible cause of ineffective advance per cut, so other factors such as drilling parameters, geological factors, and geotechnical parameters can also contribute to ineffective advance per cut (Witt, 2023; Salmi and Sellers, 2021; Segaetsho and Zvarivadza, 2019; Verma *et al*., 2016). The raw data for the actual amount of explosive used at different drive sizes for different number of cuts are given in Table C.2 in *Appendix C*.

Table 8. Planned and actual amount of explosive used for different drive sizes and number of cuts

Drive Size (m^2)	Heading ID	Planned Advance (m)	Actual Advance (m)	Planned Explosive (kg)	Actual Explosive used (kg)
5.5 x 5.5	C3916 DECLINE	4.1	3.5	135	90
	C3 916 DECLINE	8.2	8.8	270	180
	C3921 DSP	4.1	3.9	135	90
	C3 921 DSP	4.1	3.0	135	90
	C3 921 DSP	8.2	9.5	270	190
	C3981 PSTN	4.1	4.5	135	90

	C3981 PSTN	4.1	4.0	135	91
	C3981 PSTN	4.1	4.1	135	96
	C3981 RAE	4.1	5.1	135	95
	C3981 RAE	4.1	4.9	135	95
5.0 x 5.0	C31051 B ODN	4.1	3.9	120	85
	C31051 B ODN	4.1	4.8	120	85
	C31051 B ODN	4.1	3.4	120	81
	C31076 A N3 SLD	8.2	6.7	240	150
	C31076 A N3 SLD	4.1	4.4	120	85
	C31076 A N3 SLD	8.2	7.2	240	155
	C31076 A N4 SLD	4.1	3.3	120	86
	C31076 A N4 SLD	12.3	12.9	360	280
	C31076 A N4 SLD	8.2	6.4	240	151
4.5 x 4.5	C31021 DD ACCESS	4.1	4.6	80	65
	C31021 ODS EXT	4.1	3.9	80	65

CONCLUSION

Impractical drilling and blasting designs, ineffective drilling and blasting operations, and the nature of the blocky rock mass. Overbreaking leads to additional/unbudgeted costs, especially in mucking and hauling processes, as the cycle time for mucking and hauling increases and more time is spent scaling and supporting the tunnel to ensure safety. Overbreaking also causes greater dilution of the ore, lowering the overall ore grade and in turn lowering the company's profit. In addition, overbreaking also increases the mine development cycle time due to the increase in the amount of blasted material.

In general, the Star and Comet underground mine at the Geita Gold Mine experience ineffective advances per cut for larger drives, especially 5.5 × 5.5 m, and 5.0 m x 5.0 m drives. The problem also increases further when multiple cuts are used to achieve an advance metre of more than 4.1 m. The study found that one of the causes of ineffective advance per cut is a lower actual amount of explosive used per cut than the planned amount. In some cases, drives with the same size and the same amount of actual explosive used per cut produce different blast results, including actual metres per cut that are below, within or above the permitted value of ± 0.1 m. This indicates that the amount of explosive used per cut is not the only possible cause of ineffective advance per cut; therefore, this study recommends further investigation into drilling parameters, geological factors, and geotechnical parameters to reveal their contribution to ineffective advance per cut.

Author Contribution

George Bennett: Conceptualisation, Methodology, Validation, Formal analysis, Investigation, Resources, Data curation, Writing – original draft, Writing – review and editing.

Declaration of Competing Interest

The author declares that he has no known competing financial interests or personal relationships that could have appeared to influence the work reported in this paper.

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