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Assessment of effective factors in performance of an open stope using cavity monitoring system data: A case study

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Stope wall failures result in ground stability problems, ore dilution, revenue loss, etc. This work assesses mined primary transverse open stope (32S 204) in AngloGold Ashanti-Obuasi Mine to address stability problems mine-wide. The stope (17 m strike length, 30 m wide, 27 m high and dipping 68°NW) has the Obuasi Fissure as its hanging wall boundary. Back analysis of the performance showed that major geological structures –that is, Obuasi Fissure, Main Reef Shear, etc.-filled with graphitic materials have low shear strength. These structures significantly influence stope hanging wall stability. Undercutting of Obuasi Fissure on 32 level with crosscut and reef drive "Tee" compromised the fissure stability destroying rockmass integrity within the stope. Thus hanging wall overbreak of 2036 tonnes of wall rock diluted mined ore. An underbreak of in-situ ore accounted for 5439 tonnes ore loss. These reduced planned stope grade of 8.94 to 7.97 g/t (11% loss) on completion of mining activities. Limited geotechnical data prevented conventional rockmass classification, estimation of design rockmass strength and rockmass rating for the stope. However, stope height (30 m), dipping 68° is unlikely to present serious rockmass stability and/or overbreak conditions in mined out walls. Blast holes (rings) deviations were minimal, reducing overbreak during blasting.

Key words: Stope performance, cavity monitoring system, dilution, overbreak, underbreak, tonnage, grade.

INTRODUCTION

Geologic factors, including attitude, geometry and strength of both ore and waste rocks of a particular deposit, mainly influence the selection of the mining method (Hartman, 1992) even though other parameters (environmental, physical, etc.) are also considered. Open stopes design and stope performance analysis are based on strength and deformation characteristics of jointed rock masses, interlocking of rock blocks and surfaces condition between these blocks (Hoek and Brown, 1980a, b). According to Cepuritis and Villaescusa (2006), open stope performance is assessed using actual outcome versus the planned budget, in terms of the final tonnage, volume and grade of materials mucked. Factors responsible for overbreak include: (i) primarily large scale

geological structures, (ii) development location and undercutting, (iii) rock mass characteristics, iv) drill and blasts processes and (v) probably stope geometry (Clark and Pakalnis, 1997). Faults often result in low stress zones (Hafner, 1951; Anderson, 1942) near stopes. Interaction between rockmasses, stress, and explosive product, provide fair idea from which the extent of blast damage could be estimated. Blast damage normally persists to a depth of several metres into stope hanging-walls. Blast damage is quantified in terms of physical damage to the integrity of hanging wall rockmass exposures.

To maintain steady continuous production ground stability becomes an essential requirement especially in open stoping. The stability of stope walls (un-reinforced) and crowns (Villaescusa, 2004) as well as the stability of any exposed backfill masses determine the success of open stoping. Therefore the maximum ore (tonnage and grade with minimal dilution) extracted from open stope(s) generally measure the performance of the stope. Cepuritis et al. (2010) noted that back analysis of open stope performance is essential in dilution control process as assumptions and refined geotechnical parameters are checked to improve subsequent stope design processes for ore production optimization with minimal dilution. Mine dilution, according to Tatman (2001) is the contamination of ore with below cutoff-grade waste wall rock. Dilution can therefore decrease the grade of hoisted ore compared with mining reserves grades (Planeta et al., 1990).

AngloGold Ashanti-Obuasi Mine (AGA-OM) currently operates mechanized underground gold mine, using mostly open stoping method (transverse and longitudinal) with few sublevel caving method to mine the orebodies (quartz/sulphide mineralizations). The performance of the open stoping method operational in AGA-OM can be assessed by the ability to achieve maximum ore extraction with minimal dilution (Cepuritis et al. 2010). Cavity monitoring system (CMS) data of an open stope, provides excellent information to review the assumptions made during design stages and performance of the stope during mining to understand essential factors influencing the performance. To assess the performance, actual outcome after completely mining stope(s) is reconciled with the planned design parameters, in terms of the final volume measured by CMS survey to calculate tonnage and then estimate grade of ore extracted from grade control samples during mining.

The objective of this study is to investigate open stope performance determination procedural as practised in AGA-OM, using CMS data from earlier mined out 32s 204 Stope as a case study. This is to identify if additional beneficial information from the acquired CMS data after mining a stope could be used to enhance subsequent stope design. This is to maximize ore production from complex orebody geometries from stability view point. An attempt to mitigate negative effect the downward trend of

gold price had had on mining operation world-wide in recent times.

Study area location

Obuasi and its environs (latitude 5° 35' N and 5° 65' N and longitude 6° 35' N and 6° 90'N) in Adansi South Municipality, is about 56 km south of Kumasi, in the Ashanti Region of Ghana (Figure 1).The administrative capital of the municipality is Obuasi with hilly landform, and semi-equatorial type of climate (Dickson and Benneh, 1988). The vegetation is predominantly degraded and semi-deciduous forest.

Geological settings

The AGA-OM is located on the Ashanti Belt, one of numerous NE-striking belts of metavolanic rocks separated by intervening basins of meta-sedimentary rocks of the Birimian Supergroup in SW Ghana (Figure 1). The meta-sedimentary rocks consist of interbedded greywacke and phyllite and the meta-volcanic rocks comprise mafic to intermediate volcanic and volcaniclastic (Leube and Hirdes, 1990).

Within the consolidated concession, Birimian Supergroup and Tarkwaian Group occur with the Tarkwaian interpreted as uncomfortably overlying the Birimian in the southeastern corner of the concession. The Tarkwaian rocks include quartzites, conglometrates, argillaceous sandstones and phyllites while the Birimian rocks comprise of grewackes, phyllites, schists, volcanic and pyroclastic rocks (Leube and Hirdes, 1990). Both the Tarkwaian and the Birimian rocks were intruded by basin type plutonic age suite diabase, amphibolite, epidiorite, norite and granitic rocks (Davis et al., 1995). Metamorphism of Tarkwaian/Birimian sedimentary rocks was low-grade order, characteristic of relatively low-temperature and low-pressure conditions (Leube and Hirdes, 1990). Magmatic event constrained by U-Pb dates on metamorphic titanic from granitoids in the Obuasi Mine to 2105±5 Ma (Davis et al., 1995) suggested that the Birimian and Tarkwaian might be coevally formed. The Tarkwaian/Birimian contact is poorly exposed on the concession.

Structurally the Birimian rocks on the mine generally strike between 015° in the south and045° in the north, with dips also ranging from 60° to 90° NW respectively. These rocks have been subjected to tight folding due to initial deformation episodes (Sinclair et al., 1978) with fold axis plunging 30° to the north and a strike of 60°. The northern part of the mine is found several graphite-altered faults that are approximately parallel to the general strike of rocks on the mine. These include Ashanti, Obuasi, Main Reef and Cote D'or. These NE- striking faults represent pre-mineralization structures that are
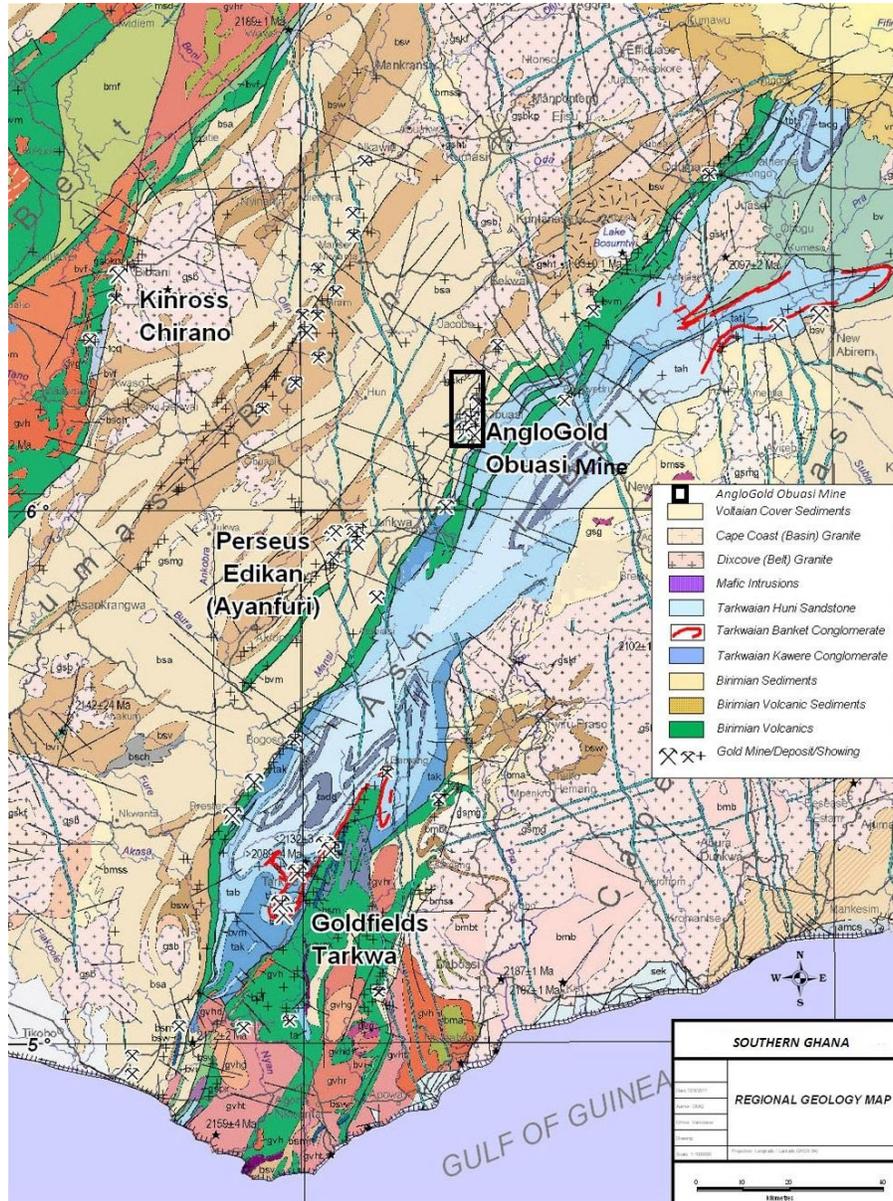
Figure 1. Location and geology map of AngloGold Obuasi mine and its environs.

apparently locally reactivated during mineralization. Two main joint sets striking 310° and 290° and dipping 75° SW and 72° NE respectively, well developed in more competent meta-greywacke and meta-volcanics, are identifiable within the mine.

Mineralization style

Mineralization on the mine is hosted by graphite-altered faults locally called fissures and greywacke/phyllite rocks adjacent to fissures that form part of the Birimian Supergroup (Amanor, 1979). Two styles of mineralization

are developed in the vicinity of the mineralized fissure; (i) auriferous quartz reefs are localized along the fault surfaces and also in fracture arrays that emanate from the fault surfaces and (ii) wallrock mineralization associated with the mineralized fault surfaces. Hence two shear zone related mineralization are delineated as quartz lodes characterized by massive to laminated quartz veins and sulphide lodes characterized by disseminated arsenopyrite alteration and gold mineralization (Farrington, 1941). The sulphide lodes consists predominantly of disseminated arsenopyrite, however variable amounts of pyrite, pyrrhotite, chalcopyrite sphalerite occur adjacent to the main shear zones. Recent
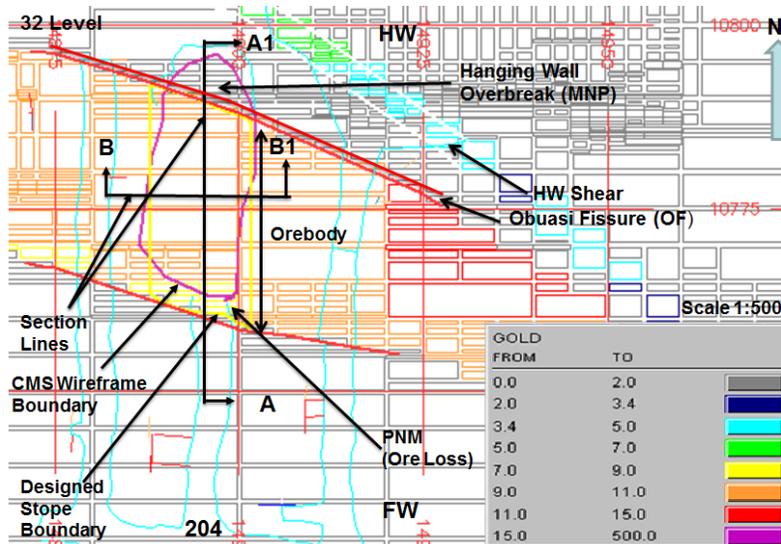
Figure 2. Block 1 32s 204 Stope; 32 level plan showing section lines (A – A1 and B – B1).

studies had established that these sulphide lodes are epigenetic in nature and that they are associated with the fissures (Oberthür et al., 1994). The association of gold with sulphides in the sulphide lodes implies that the deposition of gold was controlled by alteration reactions in the wallrock, whereby the formation of sulphides destabilizes gold complexes in the fluid and caused gold to precipitate with the sulphides. According to Amanor (1979) quartz lodes which may be massive or laminated with mineralized schist referred to as smoky quartz are restricted to fissures and tend to be high grade while the vitreous and white (milky) quartz carry variable amounts of gold.

Gold is present in the quartz as free-milling grains and often in patchy high grade concentrations visible to the naked eye. These quartz lodes have a mineral inventory that is somewhat distinct from the sulphide lodes. The veins consist of quartz with distinct selvedges of carbonate. Sub-microscope gold is typically found along quartz grain boundaries, or as inclusions along fractures within the quartz (Farrington, 1941), representing late-stage introduction subsequent to the emplacement and tectonisation of the quartz. The association of high-grade free-milling gold within the quartz veins and no correlation between gold and sulphide abundance within these veins, indicates that gold was deposited in the quartz lodes due to chemical changes within the fluid conduit rather than the wallrock reaction mechanisms (Amanor, 1979) that dominated in the sulphide lodes.

MATERIALS AND METHODS

The Block 1 mineralization is localized along two extensive graphite-altered faults known as Obuasi Fissure (OBF) and 12/74

Fissure, both striking within030°-040° in a left-stepping flexure that acted as dilatant sites for the mineralization. The OBF is more strongly sheared than the 12/74 Fissure and it is to the west of the latter. The OBF dips steeply (68°-72°NW) while 12/74 Fissure has a comparatively gentle dip (53° - 58°NW).The OBF, 12/74 Fissure and other prominent fissures on the entire mine intersect at depth to form the Main Reef Shear (MRS). The Block 1 mineralization is a broad zone of high-grade ore shoots that generally plunge 40°– 50° NE, and more associated with the OBF than 12/74 Fissure as the latter splays away, especially to the south; the ore shoots extend from 29 Level to 41 Level. The OBF vary in width from a few centimetres to about 2 m and characterized by broken, highly sheared graphite/graphitic schist materials with sheared auriferous quartz reef lenses having banded structure (graphite at the contacts and partings within).

These quartz reefs have varied geometries, with the widest veins produced at intersections where spurs splay off the OBF. On the other hand, the 12/74 is at most a metre thick with graphitic infill without quartz lenses. Two sets of joints ($310°/75°$SW and $290°/72°$ NE) classified as shear joints (Sinclair et al., 1978) are prevalent in the ore shoots. The OBF in-filled with auriferous quartz reefs that mark the 32S 204 Stope hanging wall; the associated sulphide lode (arsenopyrite disseminated) footwall rock of the Block 1 mineralization is illustrated with grade models on 32 Level plan (Figure. 2) and 32N1 Level plan (Figure 3). Figure 4 is a cross section (looking north) drawn along section line A-A1 shown on 32 Level plan, and Figure 5 is a longitudinal section (looking west) drawn along B-B1 also shown on 32 Level plan.

Further west of the OBF is another graphitic fissure (flatly dipping) referred to as Hanging Wall (HW) Shear (Figures 2 and 3). The two fissures (OBF and HW Shear) intersect somewhere north of 32S 204 stope. The sulphide lode, which occurs in widely spaced jointed metasediments (mostly greywackes and phyllites), is considered the principal Ore body in Block 1.The ore shoots (quartz reef and sulphide lode) dip approximately 68°NW (Figure 4). Where widths of ore shoots from foot-wall to the hanging-wall exceed 10 m, mostly in the northern section of the Block 1, transverse open stoping mining method is practiced using primaries and secondaries adjacent to backfilled panels. The 32S 204 stope is a primary transverse stope in Block 1 Upper, with 32N1 Level as the
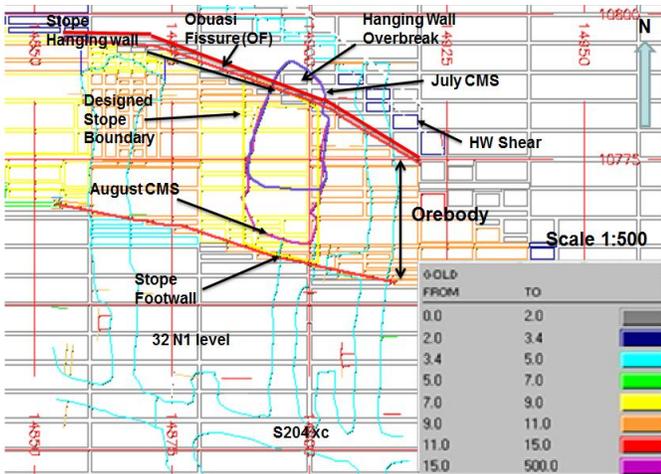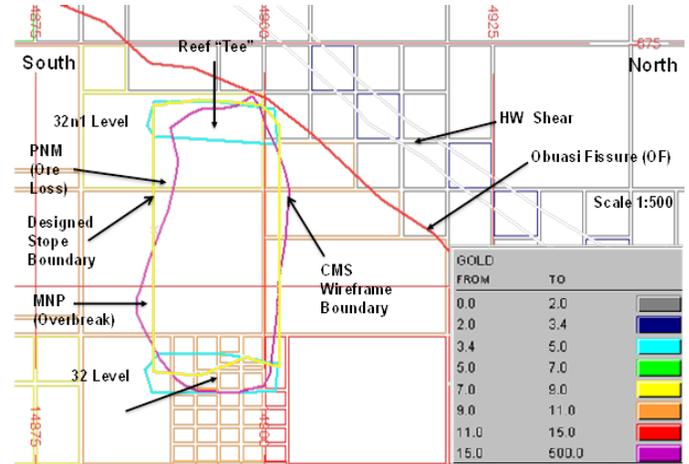
Figure 3. Block 1 32s 204 stope; 32N1 level plan.

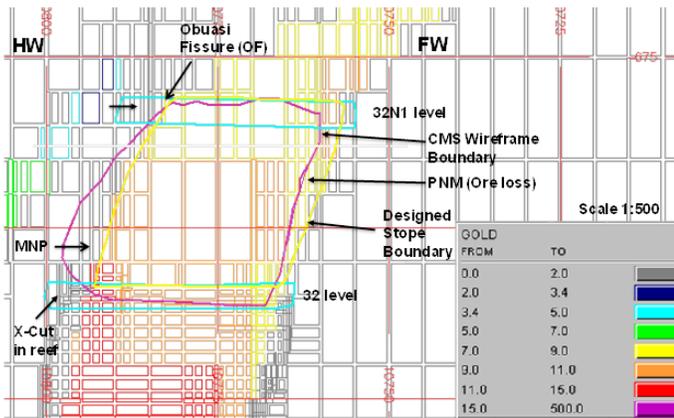**Figure 5.** Longitudinal Section along B – B1 of Block 1 32s 204 Stope, showing lateral Overbreak and Underbreak.

Figure 4. Cross section along A – A1 of block 1 32s 204 stope (looking North).

The various tonnages were calculated from the respective void volumes using densities of 2.89 and 2.64 g/m^3 for sulphide and quartz ores, respectively. Tonnages and grade values of dilution and ore loss were calculated using the designed and final stope void model volumes in conjunction with a block model of reserve grades and densities. CMS field data normally have some limitations, viz. treatment of "blind-spots" or shadow areas, and the presence of broken material or rill in the stope and stray points significantly affect the accuracy of the final void shape. In every shift, the materials mucked from the stope were sampled at every tenth bucket and the samples obtained at the end of the shift were sent to the on-site analytical laboratory for analysis to get the actual grade of the ore cleaned from the stope. The weighted cumulative grade is the achieved grade for the stope on completion.

RESULTS

Summarized statistics of planned information (tonnes, grade, ounces and dilution) and the stope reconciliation report are presented in Table 1, and the broken ore analysis in Figure 6. It shows that over 11,600 tonnes of ore with an envelope of 155 tonnes of waste at a grade of 8.94 g/t was planned to be mined to produce about 3380 ounces of gold; this excludes stope development materials. However, under 10,500 tonnes of ore and over 2,000 tonnes of waste materials were mucked from the stope; the achieved grade was 7.97 g/t and 3,228 ounces of gold obtained (Table 1). Thus, the variance was -1,142 tonnes of ore, with a corresponding grade of 0.97 g/t and 154 ounces (5%) of gold lost; this increased dilution from the under 1% planned to 16%.From the reconciled planned and mined tonnages, it was found that 920 tonnes of ore was mined that was not planned for, and over 5,500 tonnes of ore planned to be mined was lost (Table 1). Approximately 4,000 tonnes of ore was obtained from stope development activities, whereas a little over 2,000 tonnes of waste was mined that was not planned for; yet, part of the planned waste was not

upper level and 32 Level as the lower level (Figure 4). The dimensions of the stope are: strike length, 17 m; width, 30 m; height, 27 m, and average dip of 68° NW. On completing the mining operations in 32S 204 stope, the survey department conducted the CMS underground survey using Optech V400 (an auto scanning laser instrument) to pick the final excavated void of the stope, orientation and volume. The instrument as modern graphical interface and therefore uses real-time data streaming and built-in post-processing mechanisms to verify collected data and geo-references while in the field.

The Surpac software was used to process the survey data on a desk top computer. Redundant points were filtered out of the data, and wireframe models of final void volumes of mined-out stope(s) generated. The final void volumes were checked by slicing through sections using reasonable slice intervals to reduce redundant points; the choice of slice interval and filtering level (Cepuritis and Villaescusa, 2006) severely affect the precision and accuracy of final void models. The final void model (3-D) was then compared with designed stope boundaries (orebody contacts and in-place development), and the end product was accessed and displayed in 2-D sections (Figures 2, 3, 4 and 5).The void model was used to determine the stope over-breaks and under-breaks.

Table 1. 32s 204 Stope reconciliation report.

Stope survey reconciliation report					
Obuasi mine - MRM department					
Block	Block 1				
Stope	32s204				
Geologic model	1rb0607m				
Stope status	There are no materials left to be mucked				
		Tonnes		**Grade (g/t)***	**Ounces****
Stope tonnes	**Ore**	**Waste**	**Total**		
Total tonnes planned	11612	155	11767	8.94	3382
Total tonnes mined	10470	2115	12585	7.97	3228
Tonnes variance	-1142	1960	818	-0.97	-154
% Variance	-10	1265	7	-11	-5
Reconciliation tonnes (Mined x Planned)					
Mined not planned	920	2036	2956	3.47	330
Planned not mined	5439	155	5594	8.76	1576
Total	6359	2191	8550	6.93	1906
Development tonnes	3853	1251	5103	7.10	1165
Total tonnes mined	14323	3366	17688	7.72	4393
Planned dilution (%)	1				
Unplanned dilution (%)	16				
Ore loss	884	tonnes			

*Total tonnes planned grade = IS Grade with planned dilution (rings grade); ** Ounces = total tonnes × Total tonnes grade / 31.1034768.

mined. Of the approximately 90% total ore mined (Figure 6), 82% came directly from the planned stope leaving 7% behind not mined, and about 7% coming from unplanned areas. Though 5% gold loss was budgeted, about 9% was lost on completion of stoping (Table 1).

DISCUSSION

The main physical criteria for assessing 32s 204 stope are over-break and under-break volumes. About 3000 tonnes of overbreak (920 tonnes of ore and 2036 tonnes of waste) was recorded (Table 1). The OBF, marking the hanging wall boundary (Figures 2 to 4) of the entire Block 1 ore shoot including the strike length of the stope, is the major geological structure –fault/shear zone infilled with graphitic materials.

The OBF zone is about 1 m thick and the graphitic materials within deteriorate quickly with moisture on exposure (Suorineni and Tidzi, 1990). This significantly influenced the stability of the hanging wall (Villaescusa and Cepuritis, 2005). Some of the materials in the OBF zone (graphite and carbonaceous rocks) have been

reported to have low compressive strength of about 5 to 100 MPa (Suorineni, 1998). Slippage occurs along the OBF when the shear stress in the fault plane exceeds the shear strength (Ryder, 1988; Hafner, 1951). The stress parallel to the fissure (Mathews et al., 1980) possibly controlled hanging walls stability in the stope. It is very likely that, the reduction in stress parallel to the fissure and not reduction in radial direction or reduction in confinement (Kaiser, et al., 1997) might have caused rockmasses relaxation in the stope. Hence allowing slippage along the OBF (Figure 4) to cause excess shear stresses to be relieved and/or redistributed elsewhere as mining retreated towards the foot-wall.

Sheared surfaces of the OBF consist of graphite making cohesion almost negligible, and shear strengths controlled only by friction. Friction in this instance was low because the infilling materials (especially graphite) of the OBF provided smooth surfaces for easier movements of wedge rocks on either side of the fissure. This contributed to the extent of observed overbreak (Figures 2, 3 and 4) in the stope. The platy nature of graphite and low frictional angles (Hoek and Brown, 1980; Deere, 1973) possibly accounted slippage. Also graphite
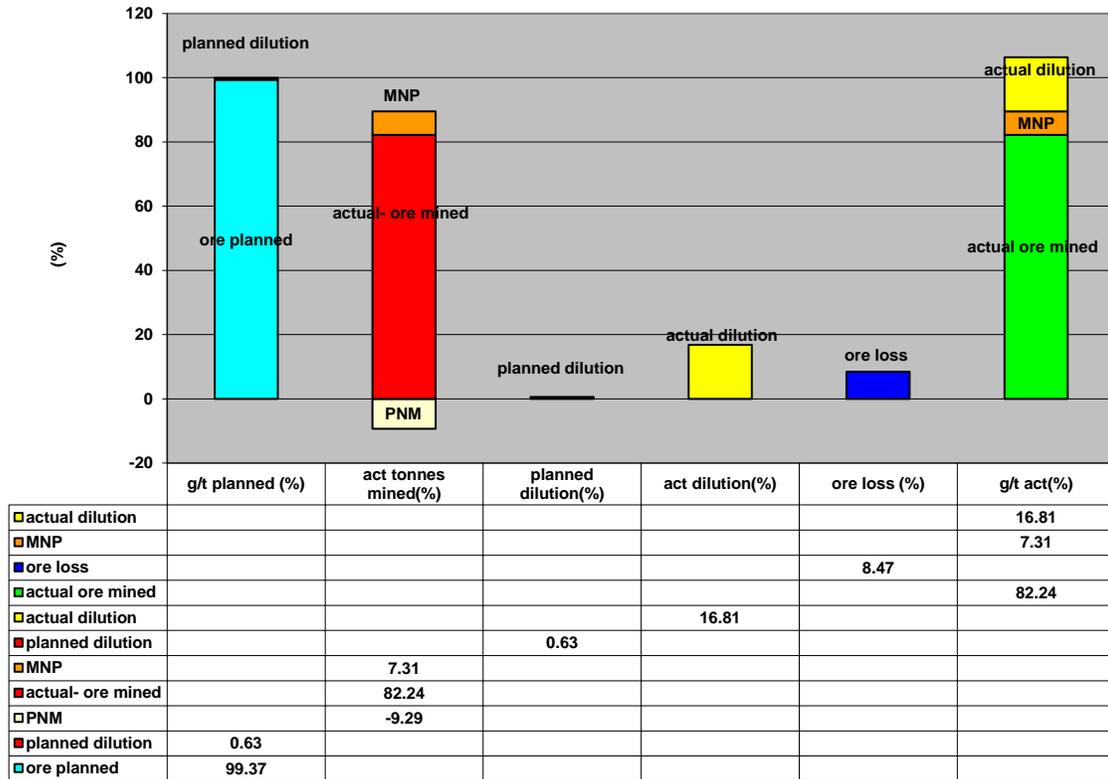
	g/t planned (%)	act tonnes mined(%)	planned dilution(%)	act dilution(%)	ore loss (%)	g/t act(%)
▢ actual dilution						16.81
▢ MNP						7.31
▪ ore loss					8.47	
▪ actual ore mined						82.24
▢ actual dilution				16.81		
▪ planned dilution			0.63			
▢ MNP		7.31				
▪ actual- ore mined		82.24				
▢ PNM		-9.29				
▪ planned dilution	0.63					
▢ ore planned	99.37					

Figure 6. 32s 204 Stope performance graph.

deteriorated quickly on exposure to moisture (Suorineni and Tidzi, 1990) to facilitate slippage that resulted in the observed breakage. With the continuous blasting, rock fracturing increased stresses within the stope thereby making induced stress (es) to exceed unconfined strength of the intact rock (Martin and Chandler, 1994). This implies that prevailing shear stresses and low frictional properties of the graphitic infilling materials of the OBF in the stope, controlled the slippage to determine extent of overbreak. Hence, larger volumes of overbreak resulted in the hanging wall as shown by the CMS wireframe strings (Figures 2 to 4). Overbreaking due to jointing would not have been massive as observed. It is also possible that previous blast vibrations of mining activities elsewhere in the entire Block 1 ore shoot, also induced seismic activity (Villaescusa and Cepuritis, 2005) that influenced the OBF non-uniform redistribution of stresses (Suorineni, 1998) in the 32S 204 stope. This situation could induce unstable hanging wall conditions in the stope.

The very close proximity of the OBF to the stope, being the actual 32S 204 stope hanging wall boundary, may also have had some influence on the overbreak in the stope hanging wall. The parallel positioning of the OBF as the stope hanging wall boundary had caused serious overbreak, especially from the toe to almost the crown

(Figure 4). The OBF, with average thickness of 1m made sloughage important due to its lack of cohesion (Villaescusa and Cepuritis, 2005). Tension zone areas around the developed stope crosscuts, reef drive "Tees" and slot raise are dependent on factors that include; the distance of the OBF from the stope surface, angle between the stope surface and the OBF, stope surface dip, *in-situ* stresses, stope aspect ratio and OBF shear strength (Suorineni, 1998). In this particular case, the OBF is the stope surface. Thus, tension zones could be used to estimate the possible stope hanging wall overbreak.

This makes development location and undercutting of the OBF, very crucial in the stope performance analysis. On both 32 and 32N1 levels (Figures 2 and 3) the crosscuts were developed to intersect the OBF. The OBF was actually crossed and further developed into the hanging wall (Figure 4). Thus, on both levels the OBF had been undercut with crosscuts. Again, the reef drive "Tees" were developed in both the OBF and the sulphide orebody (Figures 2 and 3). Hence part if not all of the thickness of the OBF for the entire stope strike length had been undercut. This situation caused variations in the angle between stope surface and OBF and stope surface dip a situation likely to affect in-situ stresses and their distribution. Numerical modelling of the tension around

the stope was not done due to lack of relevant data. However, from observational analytical view point, the tension zone in the stope may have increased as the angle between the OBF and the stope face increased.

This may diminish after a certain critical angle in the course of mining. The critical angle beyond which tension reduces has not been established for the Block 1 ore shoot in general and, for that matter 3S 204 stope in particular. To reduce tension zones in subsequent stopes, the OBF factor (that is, fault factor), stope surface dip, *in-situ* stress factors, etc., must be incorporated into subsequent stope designs. This is likely to increase stope wall stability in order to minimize hanging wall overbreak. Undercutting of the OBF on 32 Level with crosscut and reef drive "Tee" (Figures 2 and 4) during stope preparatory development may have compromised the integrity of the rockmass. This instability will occur along continuous foliations in the sheared zone of OBF and/or along bedding planes parallel to hanging-wall contact (Wang et al., 2002a). Hence the stope hanging-wall may have become unstable due to the lack of cohesion of the smooth surfaces (Villaescusa and Cepuritis, 2005) and slough to the depth of the undercut (Yao et al., 1999). The rockmass may have relaxed with the undercutting of the OBF, leading to slippage in the fissure and resulting in hanging wall failure. Even though undercutting of major geological structures such as the OBF are unavoidable in order to determine full span of payable zones or for support requirements in stopes, it must be minimized to avoid overhanging of rocks. This will then reduce stress in the fault/shear zones thereby lowering the probability of slough into the mined out area.

Similarly, the request by mine geologists to drift into the country rock to expose the full width of the ore zone needs revision. Such request(s) cause undercutting of major geological structures and must be weighed against latter consequences especially when the stope in question will be delayed in mining. Where necessary, diamond drill holes must be used to explore limits of pay zones in such major geological structures. The dimension of the crosscut (4x4 m) may be considered minimal in comparison with the stope dimension (17x30x27 m). However, it may be argued that the positioning of the reef drive "Tee", slot extensions (Figures 2 and 3) and hanging wall support works tantamount to undercutting the entire stretch of the OBF in the stope. This kind of undercutting causes stress relief of otherwise competent rockmass in the stope to deform and create serious problems during mining.

Joint systems on both sides of the OBF that intersected with the fissure caused instability in the stope hanging wall during mining. Situations like this release blocks that easily fell either under gravity or blast. The attitudes of joint systems, frequency and distribution need to be seriously factored into subsequent stope designs with respect to open stope widths. Joints sub parallel to orebodies in stopes, dipping in the same directions will

cause slabbing should designed blasting rings be parallel to orebody. Again distinction between continuous and discontinuous joints in the rockmass is very crucial. This is because rockmasses consisting of discontinuous joint sets have greater self-support capacity than continuous joint sets (Diederichs and Kaiser, 1999). Designing large stopes such as 32s 204 adjacent to or on the OBF without reinforcement of the walls will make stability to become an issue. More especially in sheared and fractured grounds. Such situations ought to be addressed in order to increase production and minimize development/reduce costs. This is because better stope performances of such large stopes depend on the stability of stope walls, crown and any exposed masses in the case of secondaries (Villaescusa, 2004). In this particular stope no splay shear occurs to present significant impact on the ground conditions. In instances where splay shear(s) intersect the OBF within a stope or splay shear(s), there is change in positions with depth in the stope. Significant impact on variability of ground conditions will result.

Stope dimensions, particularly stope height are variable factors that influence overbreak. The selection of stope dimension(s) is therefore a compromise between 'acceptable' overbreak and cost to mine smaller more stable blocks. With the stope dipping at 68°, vertical stresses are shed around the orebody. The 27 m height of the stope is not likely to present any serious overbreak conditions in the mined out walls or to rockmass stability. Again this stope height will allow easy movement of drilling equipments. This coupled with less jointed orebody in the stope most likely did not cause much borehole deviation during drilling to influence overbreaks during blasting. It implies that the exposed hanging-wall height had less influence on overbreak than did the strike length of 17 m. This is because the exposed area is dependent on the strike length more than the height. In determining the exposed hanging wall area and if the exposed area will be stable as mining retreats towards the foot-wall, the stope height is considered as the length of area and the stope strike length becomes the exposed area width. Therefore the exposed stope height (maximum length 27 m) at any given time as mining proceeded, determine how much hanging wall area was exposed.

Considering the general stope dip 68° and exposed hanging wall area, not much overbreaks will occur. The 30 m stope width eliminates border effects (Vallée et al., 1992) as the stope is wide enough to reduce unplanned dilution should the stope walls fail. This may be one of the reasons why the final achieved grade dropped by -0.97 g/t (Table 1) compared to the planned grade 8.94 g/t, else it would have been a significant drop. Even though large waste material overbreak occurred on the hanging-wall, the foot-wall side of the stope had 5439 tonnes at 8.76 g/t ore underbreak (Figure 4).

The 32S 204 stope was located where the ore shoot is

broad enough such that defined stope dimension does not show much geometric complexities. The stope had planar surfaces17x30x27 m (Figures 2 to 4) and could be classified as being regular (Germain et al., 1996). This stope geometry coupled with few joint planes will not have serious overbreaks. Consequently planned dilution material was just 155 tonnes (Table 1). This geometry most likely controlled surface performance by distributing stress around the stope (Laubscher, 1990; Deere, 1973, 1979). Again with a stope aspect ratio of 1.1 (largest lateral dimension divided by height) and dip of 68°, no serious overbreak is expected (Suorineni, 1998) because the stope is not too high.

In Block 1 geotechnical discontinuity data was limited and with 32s 204 extremely few joints measurements were available. This situation posed a challenge to conventional rock mass classification. Hence estimation of design rockmass strength and mining rock mass rating was virtually impossible. However, Suorineni (1998) reported that rocks in AGA-OM classified as poor to good depending on the mining block. Also rockmass qualities reduced further when graphite zones intersect stopes from studying mining operations in Block 2. The rock properties in the OBF (graphite/quartz) and the associated footwall orebody (greywacke/phyllite) are not uniform within the stope. Therefore it is very likely that gravity induced fracturing (Hoek et al., 1995) in the hanging wall section of the stope contributed to rockmass relaxation. The zone of relaxation defined an envelope within which gravity driven block failures occurred resulting in the volume of unplanned dilution.

Rockmass in the footwall section of 32s 204 stope could generally be classified as good judging from the massiveness of in situ rock. The very few joints have good surface conditions due to the roughness of unweathered rocks in the sulphide orebody. This accounted for the minimal 920 tonnes stope overbreaks (Figure 5) and about 5450 tonnes of footwall underbreak (Figure 4). However, characterizing OBF rockmass on the basis of interlocking blocky nature, it would described as sheared, lacking blockiness (close spacing of shear planes), weak schistosity and poor surface conditions due to graphitic materials (Hoek and Brown, 1980a, b). The limited geotechnical discontinuity data in Block 1 had compromised meaningful strength index estimation.

The strength of jointed rockmass depends on the properties (geometrical shape, surfaces separating the pieces, etc.) of the intact rock pieces. These properties control the freedom of jointed block pieces to slide and rotate under different stress conditions. Combining strength index and intact rock properties for this stope could have been used for estimating the reduction in rock mass strength for the OBF and metasedimentary sulphide orebody. It is thus recommended that the Geotechnical Department undertake consistent geotechnical discontinuity mapping of excavations. The data to be acquired will assist with rockmass parameters (RQD, joint sets, joint roughness, stress factor, failure mode factor, etc.) and uniaxial compressive strength determination for the different rock types. The information could be incorporated in subsequent stope designs to minimize failures. Care must be taken that strength index is not estimated from rock mass rating classification as this approach had proved to be unreliable particularly for poor quality rock masses (Marinos and Hoek, 2001). This is also true for rocks with lithological peculiarities that cannot be accommodated in the rock mass rating classification.

Drillhole inaccuracy is a major ore dilution factor (Beauchamp and Cameron, 1999) emanating from significantly damaged stope walls, when deviated drillholes are blasted. Although serious overbreaks/damages had occurred in the stope hanging wall (Figures 2 to 4) attributable to other factors other than drilling, yet it could be argued from the longitudinal section (Figure 5) that there were very minimal drill inaccuracies. The very little overbreaks laterally into ores demonstrated that accurate holes alignments (that is, little deflection from the intended direction) were drilled. Inaccurate drilling leads to poor blasts with higher vibration and damage. It is difficult to solely quantify blast-induced damage from production blast vibrations of the available overbreaks information. However, considering the observed hanging wall instability, it can be concluded that blast-induced damage probably contributed to the overbreak. The strength of the remaining hanging-wall rockmass certainly deteriorated due to blast induced crack, blast vibrations opening, shearing and extending pre-existing planes of weakness (Villaesusa et al., 2004; Connors et al., 1996; and Liu et al., 1995).This is because some blast rings were definitely parallel to discontinuity patterns within the stope. This situation did cause blast ring slabbing and weakening to reduce quality or integrity of rockmasses. The extent of blast-induced damage though not immediately quantifiable with the measured overbreak, is very important in the stope performance back analysis. Therefore mine-wide geotechnical discontinuity data base ought to be addressed with the needed urgency.

Considering Figures 2 to 4, it can be adduced that the breakups had direct relationship with high magnitudes of energy from heavy blast vibrations that occurred in the stope. This might have caused the severity of rockmass overbreaks. The reduction in rockmass quality most likely paved way for weakening or dilation of existing discontinuities in the stope. Such situations created new fractures to further cause hanging wall instability. This information is crucial for ground control, support, and dilution estimation. Down blast-holes drilled in the stope were 76 mm in diameter and the pattern was (2x2 m). This practice must be adhered to, to prevent blast-induced damages. Yu (1980) is of the view that large diameter (200 mm) blastholes cause excessive stope wall sloughage. Explosive product (high or low density),

powder factor (kg of explosives/stope tonnes), charge concentration and delay timing (Scoble et al., 1997) ought to be factored into future stope design either to minimize or eliminate damage.

The entire overbreak tonnage, mostly barren wall rock from the hanging wall (Figures 2, 3 and 4) of 32s 204 stope, diluted the ore in the stope. The planned grade of 8.94 g/t was reduced on completion of mining activities to 7.97 g/t (Table 1), representing -0.97 g/t drop in grade to compromise profitability. The term 'overbreak' is synonymous with unplanned wall dilution (Yao et al., 1999). This situation resulted in 16% unplanned dilution (Table 1 and Figure 6).

According to David (1988), smaller size machines used for ore mucking in stopes with wall failures produce better ore recoveries than bigger size machines (size effect). The bigger unmanned machines (Scooptram ST1030) used to muck ore from 32s 204 Stope, introduced much dilution from the hanging wall overbreak into the blasted ore. The operators using remote control systems for mucking could not visually differentiate between waste and ore materials in the stope at the time of mucking. Hence operators mixed the ore and barren materials together. In this particular situation the 10 tonnes ST1030 Scooptram bucket having length of 1.66 m and width 2.48 m can arguably be described as unsuitable sized equipment (Trevor, 1991) had more dilution introduced into the blasted ore (David, 1988). Thus planned dilution of 1% increased by 16% to actual total dilution of 17% (Figure 6) comprising planned and unplanned dilutions (Scoble and Moss, 1994). The achieved total dilution was calculated from the difference between CMS area and designed stope outline area divided by CMS area (Mah et al., 1995).

It will be recommended that in situations that the barren materials are failing into stopes, medium/smaller size machines be substituted for the bigger machines to reduce dilution. Again if during mining it becomes obvious that the barren materials are failing in stopes then more grade control samples ought to be taken on shift basis to check dilution. By this procedure it is less likely that waste will be sent to the mill from lack of credible information that grade is falling during mining. When more grade control samples are collected and analyzed through assaying there is better discrimination between ore and waste. This improves the quality of the estimated values on which decision will be made. But if few samples are used the information will be poor and many mistakes will be made. Thus it is beneficial to spend more money on grade control practices for better selectivity to reduce dilution and enhance achieve grade at the end of stoping.

The thickness of ore material that ought to have been left as a skin to prevent slippage along OBF during the opening of the slot extension may have been blasted off by mining crew (Elbrond, 1994). This is because the OBF and beyond in some instances may had been drilled and blasted alongside the ore to cause dilution in the stope.

Blasting damage adversely affected the hanging wall stability causing rockmasses and OBF rupture (Vallée et al., 1992). Continuous blasting make it difficult to prevent overbreak of almost the entire stope hanging wall (Figures 2 to 4) into the stope to increase unplanned dilution. If the stope is emptied before new blast rounds are taken, dilution may increase. So some amount of ore must always be left in the stope before new rings are blasted to prevent persistent dilution from the hanging-wall. Mass blasting is an alternative measure to reduce dilution resulting from persistent ground vibrations of firing rings patterns one after the other.

Stope preparation also introduced waste materials to cause ore dilution. Stope lateral and vertical development materials (Planeta et al., 1990) accounted for about 1300 tonnes of waste (Table 1) that contributed to reduce achieve stope grade. The dilution in this stope could be attributed more to hanging wall failure and attended blast damage. Dilution dropped the grade from 8.94 g/t to over-all achieved grade of 7.72 g/t for about 14,400 tonnes of stoping and developments materials mucked (Trevor, 1991). An unplanned dilution of 16% obtained from this stoping operation could jeopardize the feasibility of mining other stopes (Elbrond, 1994) in Block 1 with lower grades and narrow ore widths. It implies that dilution severely constrain economic aspects of mining by affecting among others net present value, production cost and ore loss (Elbrond, 1994; Tatman, 2001). Therefore dilution ought to be reduced to the minimum in bulk ore production methods such as open stoping. There seems to be an inverse correlation between dilution severity and the final achieved ore grade (Planeta et al., 1990).This makes acceptable level of dilution highly dependent on grade. So a relatively higher grade 32s 204 stope can be economical with a total of 17% dilution. An erratic sulphide orebody of lower grade in Block 1 may not be mineable should projections indicate similar level of dilution.

Ore losses are equally significant just as dilution; as such ore losses can have considerable effects on the reserves and economical results of mining operation. Underbreaks constitute ore loss (Cepuritis and Villaescusa, 2006). About 5450 tonnes of ore planned to be mined (Table 1) had been left unmined. It is not immediately clear from Figure 4 if blastholes were designed to bring down this material and was not blasted. Or the assumption was that undercutting the material it will reel down under gravity given the high dip and shape of the foot-wall side of the stope. If blastholes were not taken, then in subsequent stoping operations all blastholes must be fired to avoid ore loss. This is because there is no major geologic structure in the immediate surroundings of the foot-wall side of the stope to cause sloughage. The 12/74 Fissure that occur on the foot-wall side of the Block 1 ore shoot is some distance off the payable zone of the orebody being mined. If vertical long blasthole were designed given the geometry of the stope outline, then shorter holes that will break ore to the desire boundary

limit must be incorporated in subsequent design. Then all planned ore in the stope can be blasted off.

The approximately 8.5% ore loss (Figure 6) could have compensated for some of the unplanned dilution that occurred in this stope.

The 32S 204 stope produced about 83% of the planned ore (Figure 6) and therefore could be said to have performed well. However better result would have been achieved had dilution been drastically reduced from the achieved 17% to something close to the planned dilution of 1%. The CMS data had assisted in identifying over-break and under-break, their nature, extent and morphology. It had brought to the fore that in the Block 1 ore shoot, the OBF is critical factor that needs serious attention when designing stopes due to its nearness and graphitic infills. The OBF had had control on rockmass quality and subsequent influence on the levels of overbreak in the hanging wall.

Conclusion

This back analysis of 32s 204 stope using the acquired CMS data has resulted in thorough understanding of the stope hanging wall overbreak due to the influence of the OBF . The analysis has demonstrated the need to understand relevant factors for individual stope design rather than generalizing all stope designs, a situation that can lead to poor design. This is because different major geological structures (Main Reef Shear, 12/74 Fissure, Obuasi Fissure, etc.) with varying characteristics occur in Obuasi Mine that requires specific design approach. The gravity-driven failures that occurred are attributable to undercutting of the OBF and lack of confinement. Developments for stoping must not be allowed to stand for longer time for the ground to deteriorate.

Averaging rock mass properties for stope design in different ground conditions might not always provide the best solution to stability issues. Many parameters must be factored into designing individual stopes henceforth than hitherto been done. Therefore the recommended acquisition of pieces of information mentioned above in the discussion section must not be over looked. Lastly it is very necessary AGA-OM develop a database of all CMS surveys analyses of all factors that affect open stope performance like this for all personnels involved in the design and production stages appreciate the negative effects of overbreaks and underbreaks based on the understanding derived from relevant factors reviewed on completion of the excavation. A database of this nature will inform the necessary and adequate inputs that ought to be incorporated in subsequent stope designs and mining activities mine-wide to optimize stoping and reduce dilution to the barest minimum.

Conflict of Interest

The authors have not declared any conflict of interest.

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