

Metallurgical Characterisation of some Gold Ores in the Birimian, Ghana

George M. Tetteh¹; Eric M. Kinyua²

¹Department of Geological Engineering, University of Mines and Technology, P. O. Box 237, Tarkwa, Ghana.

²Mining and Mineral Processing Engineering Department, Taita Taveta University, P.O. Box 635, Voi Kenya.

Corresponding Author: George M. Tetteh

ABSTRACT: Gold ore in the Birimian Supergroup, Ghana contains microscopic to fine grained gold inclusions in pyrite and arsenopyrite and as traces of fine disseminations in quartz veins. The ore contains sulphidesulphur ranging from 0.01 to 0.65%, sulphatesulphur from 0.02 to 0.33% turning out gold recoveries up to 98.7% on cyanide leached for 24 hours after roasting for 4 hours. Lime and cyanide consumption range from 2.3 to 6.0 kg/t and 0.2 to 0.4 kg/t respectively on leaching for 36 hours. Ores with moderate quartz content have the highest natural pH, exhibit the lowest lime consumption while samples with very high quartz and with lowest natural pH exhibit the highest lime consumption. Ore composed of very high quartz generally has the lowest cyanide consumption of 0.2 kg/t at gold head grade of 1.4 g/t. The yield stress values of the ore slurries at both natural and leaching pH are <30 Pa suggestive of two types of hydrothermal ores containing either low quartz or higher quartz which conform to the two types of gold mineralisations in the Birimian where the higher gold grade is generally hosted in quartz veins and lower grade mineralisation associated with disseminated arsenopyrite and/or pyrite.

KEY WORDS: Birimian, Hydrothermal, Gold ore, Quartz, Gold recovery, Lime, Cyanide

Date of Submission: 26-01-2019

Date of acceptance: 09-02-2019

I. INTRODUCTION

Birimian rocks form a substantial part of the Man shield [1] which occupies the southernmost third of the West African craton. The rocks were folded, metamorphosed and intruded by granitoids [2]. The Birimian rocks in Ghana also host numerous gold deposits in other parts of West African craton including southwestern Niger, Côte d'Ivoire, Burkina Faso and Mali (Figure 1) (Anon., 2011). This paper is a case study from the Ashanti gold district which hosts a hydrothermal gold deposit that exhibits variations in gangue minerals, and textures. The ore deposit shows variety of mineralisation styles which ranges from quartz-veins containing free milling gold to arsenopyrite-pyrite rich graphitic shear zones comprising refractory ores. Since the deposit lies within the strongly deformed graphitic shear zone and contains disseminations of pyrite and arsenopyrite, it is carbonaceous and sulphidic [4].

From the run-of-mine (ROM) pad, the ore is transferred for processing at a plant which comprises a gyratory two-stage grinding crusher with semi-autogenous (SAG) mill and ball mill then to a flotation circuit for concentration of sulphides. Subsequent facilities are a regrinding mill, flotation concentrate, bio-oxidation circuit to oxidise sulphides in flotation concentrate. These processes are followed by a three-stage counter-current decantation unit for solid/liquid separation, carbon-in-leach (CIL) circuit for cyanidation of bio-oxidation residue, and an elution circuit for gold desorption from carbon.

There is usually low gold recovery from the ore associated with sulphides at the refractory plant (at about 81%). The oxide plant periodically also produces low recovery as the ore, though not sulphidic, represents a typical hydrothermal gold deposit having complex gold mineral association. Currently, cyanide leaching is the most commonly employed method for gold dissolution from ore. Other lixiviates such as halides, thiosulphate, thiocyanate, and thiourea can be used as alternatives to cyanide leaching [5].

In practice, there are three major ways of applying carbon adsorption technology to recover the dissolved gold [5]: carbon-in-pulp (CIP) processes the slurry introduced to carbon and passes through several stages of carbon adsorption. After adsorption is completed, the loaded carbon is recovered from the slurry by a vibrating screen, washed, and sent for gold elution. In carbon-in-leach (CIL) process, the activated carbon competes with gold preg-robbing minerals, and preferentially adsorbs the gold. For carbon-in-clarified solution (CIC), the adsorption is carried out after filtration of the pulp leached for the activated carbon and administered in clarified pregnant solution. The influence of a major gangue mineral, quartz on lime and cyanide consumption was undertaken on the ore.

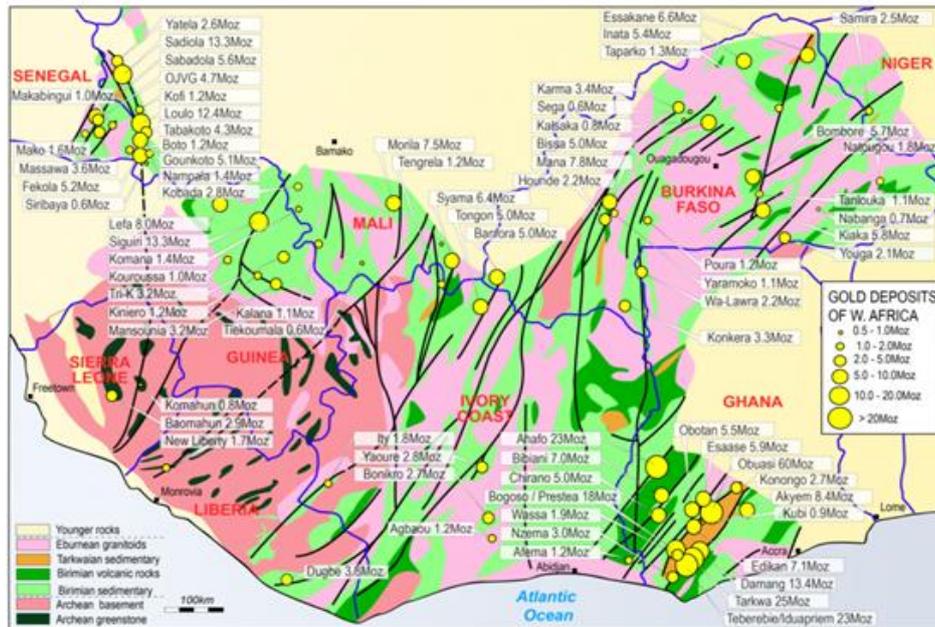


Figure 1: Geological map of West Africa showing major gold deposits [3].

II. MATERIALS AND METHODS

Determination of lime consumption starts with ore slurry preparation using 500 g of sample ground to particle size of 80% passing 106 μm screen at 50% pulp density. The natural pH of the slurries were first measured and slurries conditioned to leaching pH of 11 using lime. The pH conditioning process involved weighing small amounts of lime and adding it to the slurry, stirred for about 5 minutes, and the pH checked. This procedure was repeated several times until pH 11 was attained on each slurry. The slurries were then left to stand for 24 hours after which pH drop was adjusted by adding a known amount of lime. The various weights of lime added to each slurry were summed up and the lime consumption in kg/t of each ore determined.

The leaching kinetics of the ore samples was conducted by bottle-roll leaching test. A 1 kg of each sample ground to particle size of 80% passing 106 μm sieve was leached for 36 hours at 50% pulp density using 1000 ppm cyanide concentration. During the leaching period, small amounts of slurry samples were picked at 2nd, 8th, 24th, and 36th hour of leaching. After filtration, the resulting clear solutions were analysed for free cyanide and dissolved gold by silver nitrate titration and atomic absorption spectrometry (AAS) respectively. On completion of leaching in the 36th hour, 50 g of solid tailings of each sample was washed thoroughly to remove the entrapped aurocyanide complex, dried, and then analysed for gold by conventional fire assay. The results of the test were analysed to determine the gold recovery at various leaching time intervals and the curves of gold recovery against leaching retention time plotted for the leaching kinetics determination from the graphs.

For carbon-in-leach (CIL) test, 1 kg of each sample ground to particle size of 80% passing 106 μm was leached for 24 hours at 50% pulp density, and pH 11 using 1 kg/ton cyanide concentration in the presence of 25 g/l activated carbon of +1 mm particle size. On completion of leaching after 24 hours, the slurries were screened using a 1 mm screen to recover the loaded carbon. The loaded carbon was washed, dried, and analysed for gold content in its ash; aqua regia digested for AAS analysis. The slurries were filtered and the resulting clear solutions and solids analysed for gold content by AAS and fire assay respectively.

A 600 g of sample ground to particle sizes of 80% passing 106 μm was roasted in a gas furnace at 750 $^{\circ}\text{C}$ for 2 hours and for 4 hours to decompose the sulphides and carbonaceous matter constituents. The effect of roasting on gold recovery was then determined by cyanide leaching of 500 g of each of the roasted samples at

50% pulp density using 1000 ppm cyanide strength for 24 hours. The dissolved gold concentrations in clarified leach solutions were determined using AAS and the remaining gold in the solid tails determined by fire assay technique.

Petrographic studies of the prepared thin and polished sections involved determination of mineral composition and textures of the rocks at Geological Engineering Laboratory, University of Mines and Technology, Tarkwa, using Leica DM2700P polarising microscope with modal compositions estimated by point counting. The samples were grouped based on quartz modal composition.

III. RESULTS AND DISCUSSIONS

The main rock types of the mine are the various chlorite-schists being metamorphic versions of volcanic rocks and sedimentary rocks in the area. These rock types include chlorite-schist, carbonate-chlorite-schist, quartz-chlorite-schist, chlorite-amphibole-schist, and quartz-veined-chlorite-schist.

Chlorite-schist rock was greenish-grey, weakly foliated, and strongly silicified and comprised of quartz (2–15%), plagioclase (20–40%), chlorites (40–50%), epidote (5–8%), sericite (8–10%), pyrite (0.5–8%), arsenopyrite (0.5–3%), magnetite (0.5–3%), and traces of gold. The plagioclase was altered to sericite while the surfaces of opaque minerals were corroded by carbonates alteration. Gold occurred as fine-grained disseminations and was closely associated with fine-grained pyrite and arsenopyrite.

Carbonate-chlorite-schist was dark-grey, strongly sheared, and silicified and comprised of quartz (28–30%), plagioclase (15%), chlorites (25–30%), carbonates (10–16%), epidote (3–10%), sericite (5%), pyrite (2–4%), arsenopyrite (0.5–1%), magnetite (0.5%), and traces of gold. The plagioclase was strongly altered to sericite. Gold was fine-grained and closely associated with fine-grained pyrite and arsenopyrite.

Quartz-chlorite-schist was brownish green to greenish-grey, strongly sheared, and weakly foliated and comprised of quartz (20–35%), plagioclase (15–30%), chlorites (18–41%), carbonates (2–6%), epidote (3–5%), sericite (5–7%), pyrite (0.7–6%), arsenopyrite (0.3–0.5%), magnetite (0.2–1%), and traces of chalcopyrite, pyrrhotite, limonite and gold. The plagioclase was strongly sheared and altered to sericite while pyrite was partially replaced by limonite, an evidence that the ore was undergoing weathering and alteration processes. Gold occurred as fine-grained particles in pyrite, arsenopyrite, and quartz.

Chlorite-amphibole-schist was strongly sheared, partially oxidised and contained quartz (48%), plagioclase (20%), chlorites (10%), epidote (2.8%), sericite (3%), pyrite (2.8%), arsenopyrite (0.2%), and traces of gold. The plagioclase feldspars were altered to sericite and pyrite was corroded by gangue minerals. Gold occurred as fine-grained inclusion in sulphides and quartz vein.

Quartz-veined-chlorite-schist was a fractured glassy quartz vein with graphitic schist with chloritic schist filling the fractures. It comprised of quartz (80–85%), chlorites (3–10%), epidote (3–4%), sericite (1%), pyrite (0.8–1%), and traces of gold. The gold occurred as fine grained disseminations in quartz vein and in pyrite.

The gold in the ore samples occurred as fine-grained particles in pyrite and in arsenopyrite and as traces finely disseminated in quartz veins and in other gangue minerals. Plagioclase in the ores was altered to sericite while pyrite was partially corroded by other minerals or altered to limonite, pyrrhotite, or magnetite, an indication of weathering and oxidation processes. Hence the ores were partially oxidised.

The mineralogical composition of the ores varied from one sample to the other with quartz exhibiting the highest variation of 2–15% for chlorite-schist ore to 80–85% for quartz-veined-chlorite-schist ore. Quartz may influence comminution properties of the ore depending on its modal percent and gold association. Quartz is the hardest mineral found in the samples. Thus, the ore samples were characterised into five (5) groups based on the quartz content, namely Very Low Quartz (QVL), Low Quartz (QL), Moderate Quartz (QM), High Quartz (QH), and Very High Quartz (QVH) (Table I).

Table II shows gold recoveries in ore samples after leaching for a period and the respective head grades of gold after carrying out bottle-roll leaching kinetic tests. The ores reported lime and cyanide consumption ranging from 2.3 to 6.0 kg/t and 0.2 to 0.4 kg/t respectively on leaching for 36 hours (Table I). The amount of lime consumed by an ore is dependent on its natural pH whereby samples with lower natural pH require high lime consumption to condition them to leaching pH. Ore with moderate quartz content (QM) has the highest natural pH of 8.35, exhibits the lowest lime consumption while very high quartz ore (QHV) with the lowest natural pH exhibits the highest consumption (Table I). However, this trend is not the case for all the samples since the rate of acid generation when the slurries were left for 24 hours while conditioning the pH, was not the same because sulphidesulphur content and distribution of the ore components was not the same.

Cyanide consumption is dependent on gold content in the ores as well as the amount and type of cyanicides present. Sample with very high quartz composition (QVH) exhibited the lowest cyanide consumption of 0.2 kg/t despite having the second highest gold head grade of 1.4 g/t. The low cyanide consumption could be attributed to mineralogical composition which was predominantly quartz (80–85%) with low sulphidesulphur

content (0.09%). Quartz does not significantly react with cyanide during cyanidation [5]. However, sulphide minerals readily react with cyanide during cyanidation leading to increase in cyanide consumption.

The yield stress values of the slurries at both natural pH and leaching are <30 Pa. These indicate that the ores have good flow-ability properties at a particle size of 80% passing 106 µm sieve, 50% pulp density of slurry at slurry temperature of 28 °C. The processing plant could under such conditions experience low energy consumption in pumping and mixing of slurries in tanks and impellers. There is also high rate of oxygen mass transfer during leaching fast kinetics in carbon adsorption processes, among others [5]. Low yield values were primarily attributed to the mineralogy as the ores were partially oxidised and altered with little or no detectable clay minerals. Since the yield stress values of the ores at leaching pH were higher than at natural pH, lime used to condition the slurries to the leaching pH is also a viscosity modifier (Table I).

Two types of ore could be identified as low in quartz, and higher in quartz composition with other types grading in between. There is maximum recovery of 52.3% after two hours leaching and complete leaching after 36 hours (Table II). The recovery was significantly increased to 95% after roasting for two hours and 98.7% after 4 hours roasting (Table II, Fig. 2). According to [7, 8] two types of gold mineralisation occur in the Birimian. Higher grade is generally hosted in quartz veins while lower grade sulphidemineralisation is associated with disseminated arsenopyrite and/or pyrite [2, 9]. Since head grades are distinctly different for the five categories of ore based on quartz content, treatment should be further studied on these modalities.

Roasting was carried out to decompose both sulphides and carbonaceous matter before leaching. The resulting calcine reported improved gold recoveries ranging from 68.4 to 95.0% and 80.0 to 98.7% after roasting for 2 hours and 4 hours respectively (Figure 2). After roasting for 4 hours, the majority of ore exhibited gold recoveries exceeding the 81% achieved by the bio-oxidation plant, which only oxidises sulphides. Carbonaceous matter in the ores not acted upon by the current biooxidation process at the plant continues to serve as preg-robbbers. Therefore to increase recovery, suitable microbes which also degrade carbonaceous matter should be incorporated at the plant in addition to the current biooxidation circuit which only oxidises sulphides.

IV. CONCLUSIONS

The predominant gangue minerals in the ore include quartz, plagioclase, chlorites, amphibole, and carbonates while the major gold-bearing minerals are pyrite and arsenopyrite. Gold occurs as fine-grains in pyrite and arsenopyrite and as traces of fine disseminations in quartz vein with highest variation in the ores which contain sulphide and organic carbon ranging from 0.01 to 0.65% and 0.11 to 0.51% respectively. Sulphate sulphur in the ores ranges from 0.02 to 0.33% due to partial oxidation processes. The gold recoveries are up to 98.7 % on cyanide leaching for 24 hours after roasting to decompose sulphides and carbonaceous matter. Thus majority of the ores exhibited recoveries exceeding 81%.

Lime and cyanide consumption range from 2.3 to 6.0 kg/t and 0.2 to 0.4 kg/t respectively on leaching for 36 hours such that ore with moderate quartz content has the highest natural pH, exhibits the lowest lime consumption while the ore with very high quartz and lowest natural pH the ore exhibits highest consumption. Also, ore with very high quartz composition has the lowest cyanide consumption of 0.2 kg/t at gold head grade of 1.4 g/t. Hence, two main types of hydrothermal ores occur in the Birimian as either being low in quartz or higher in quartz possibly conforming to the two types of gold mineralisations in the Birimian where the higher grade is generally hosted in quartz veins, and lower grade sulphidemineralisation is associated with disseminated arsenopyrite and/or pyrite.

The yield stress values of the slurries at both natural pH and leaching are <30 Pa. These low yield values were primarily attributed to the mineralogy as the ores were partially oxidised and altered with little or no detectable clay minerals. Since the yield stress values of the ores at leaching pH were higher than at natural pH, lime used to condition the slurries to the leaching pH is also a viscosity modifier.

Table I. Reagent Consumption Parameters and Yield Stress

| Rock Type | % Quartz | Sulphide Sulphur/ % | Natural pH | Lime consumption Kg/t | Final pH | Cyanide Consumption/ kg/t | Yield Stress at Natural pH/ Pa | Yield Stress at Leaching pH/ Pa |
|-----------|----------|---------------------|------------|-----------------------|----------|---------------------------|--------------------------------|---------------------------------|
| QVL | <20 | 0.01 | 8.21 | 3 | 11.03 | 0.3 | 0.92 | 3.24 |
| QL | 20–30 | 0.65 | 7.69 | 2.6 | 10.97 | 0.4 | 0.94 | 3.75 |
| QM | 30–40 | 0.29 | 8.35 | 2.3 | 11.01 | 0.3 | 1.04 | 3.98 |
| QH | 40–80 | 0.37 | 8.31 | 4 | 11.07 | 0.2 | 0.68 | 3.2 |
| QVH | >80 | 0.09 | 7.24 | 6 | 10.98 | 0.2 | 0.7 | 3.72 |

Table II Gold in solution (mg/l) and recovery % in bold after leaching time

| Leaching/hrs. | QVL | | QL | | QM | | QH | | QVH | |
|---------------|------|-------------|------|------------|------|-------------|------|-------------|------|-------------|
| 2 | 0.14 | 39.8 | 0.04 | 2.3 | 0.1 | 12.9 | 0.02 | 10.9 | 0.35 | 52.3 |
| 8 | 0.1 | 28.4 | 0.01 | 0.6 | 0.07 | 8.2 | 0 | 2.4 | 0.29 | 43.5 |
| 24 | 0.06 | 15.3 | 0.01 | 0.6 | 0.02 | 2.4 | 0.01 | 6.7 | 0.22 | 32.4 |

| | | | | | | | | | | |
|--------------------------------------|------|------|------|---|------|-----|------|---|------|------|
| 36 | 0.05 | 13.1 | 0 | 0 | 0.03 | 3.5 | 0 | 0 | 0.19 | 28.6 |
| Tailings | 0.31 | | 1.71 | | 0.76 | | 0.17 | | 0.48 | |
| Calculated Head grade, g/t | 0.36 | | 1.71 | | 0.79 | | 0.17 | | 0.67 | |
| Recovery % after Roasting for 2 hrs. | 81.6 | | 71.5 | | 77.8 | | 68.4 | | 95.0 | |
| Recovery % after Roasting for 4 hrs. | 87.5 | | 82.5 | | 87.0 | | 80.0 | | 98.7 | |

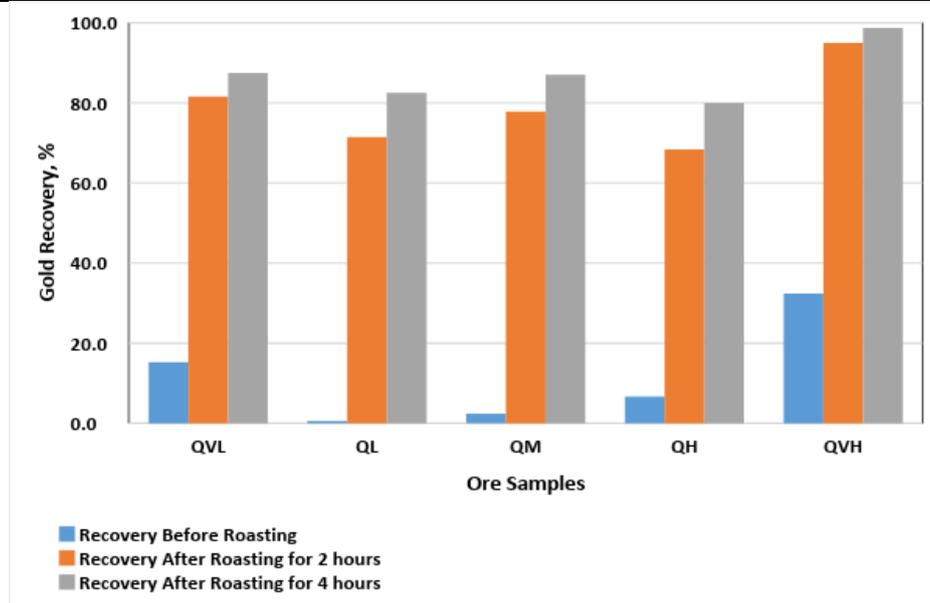


Figure 2: Gold recovery % for various ore types before and after roasting.

ACKNOWLEDGEMENT

Sponsorship by TaitaTaveta University College, technical advice from Prof. Richard K. Amankwah, and laboratory assistance from Messrs. Francis Stephen and Dickson Adjei at Minerals Engineering Laboratory and Mr Clement Benyarku at the Geological Engineering Laboratory, University of Mines and Technology, Tarkwa, Ghana are acknowledged.

REFERENCES

- [1]. Bessoles, B. (1977). Géologie de l'Afrique, Le Craton OuestAfricain. Mémoire Bureau RechercheGéologiqueMinière, Orléans, 88.
 - [2]. Leube, A., Hirdes, W., Mauer, R., and Kesse, G. O. (1990). The Early Proterozoic Birimian Supergroup of Ghana and Some Aspects of its Associated Gold Mineralisation, Precambrian Research, Vol. 46, pp. 139-165.
 - [3]. Anon. (2011). <https://avesoro.com/wp-content/uploads/2011/04/man-craton2-1024x724.png>. January 2019.
 - [4]. Oldcorn, R., Bray, C., Arthur, J. and Bourassa, Y. (2014). Technical Report on Resources and Reserves, Golden Star Resources Limited, BogosoPrestea Gold Mine, 138 p.
 - [5]. Marsden, J. O. and House, C. I. (2006). The Chemistry of Gold Extraction, Society for Mining, Metallurgy, and Exploration, Inc., Littleton, Colorado, USA, 2nd edition, 651 p.
 - [6]. Mezei, A. (2003). "Hydrometallurgy Applications of Rheology Testing", SGS Mineral Services: Technical Bulletin 2003-06, 9 pp.
 - [7]. Appiah, H. (1991). Geology and mine exploration trends of Prestea Goldfields, Ghana. Journal of African Earth Science, Vol. 13 pp. 235-241.
 - [8]. Allibone, A., Teasdale, J., Cameron G., Etheridge, M., Uttley, P., Soboh, A., Appiah-Kubi, J., Adanu, A., Arthur, R., Mamphey, J., Odoom, B., Zuta, J., Tsikata, A., Pataye, F., Famiyeh, S. and Lamb, E. (2002). Timing and Structural Controls on Gold Mineralisation at the Bogoso Gold Mine, West Africa. Economic Geology, Vol. 17, pp. 949-969.
- Oberthür, T., Vetter, U., Schmidt Mumm, A., Weisser, T., Amanor, J. A., Gyapong, W. A., Kumi, R. and Blenkinsop, T. G. (1994). "The Ashanti gold mine at Obuasi in Ghana: Mineralogical, Geochemical, Stable Isotope and Fluid inclusion Studies on Metallogenesis of the Deposit", Geologisches, Jahrbuch, Vol. D100, pp. 31-129.

Tetteh and Kinyua "Metallurgical Characterisation of some Gold Ores in the Birimian, Ghana" International Journal of Modern Engineering Research (IJMER), vol. 08, no. 12, 2018, pp 28-32