

THE TETREM SMALL MINE

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INTRODUCTION

After a long decline of gold production in Ghana since 1986 the gold mining sector is recovering and mainly foreign investment led to substantial exploration efforts as well as to the opening up of new mines and/or rehabilitation and expansion of the existing mines after their divestiture (privatization). Foreign junior exploration and mining companies and established international mining houses are currently exploring the country in search of large, therefore commonly low-grade gold deposits which are amenable to highly mechanized open pit mining.

Beside such orebodies of the disseminated type or in granites, Ghana is endowed with many occurrences of gold quartz veins and blanket reefs, which vary in width from a few centimetres to several metres and have striking length from a few 10 meter to several kilometer. Whilst such larger orebodies with ore reserves of above 1 Mio. tons are a target for industrial mining operations, the smaller "stringer"-occurrences are not of interest, if not only as indicators in prospection for possible adjacent disseminated mineralization.

Such restricted orebodies could offer investment and employment opportunities for Ghanaian owned mining groups and companies and should therefore contribute also to economic development of the hinterland. Some of these orebodies as well as the ubiquitous gold bearing alluvials are currently mined by artisanal small scale miners, commonly called "galamsey", which use manual labour only and employ simple handtools in their operations. A visit to such operations will convince any observer that in almost all cases the operations are either inefficient, including the skimming of

orebodies, or dangerous, or environmentally unsound or a combination of these. This is due to the long oppression of small scale mining activities first by the Colonial Government from about 1920 ongoing, later by the Ghanaian authorities after Independence. Only in 1986 under the Minerals and Mining Law, PNDC-Law 153 was the small miner legally recognized and a basis laid for his operations.

Vast experience in narrow vein mining, which was available among African small scale miners before and around the advent of this century has been lost. To prove this point, mention should be made of the African engineer T.B.F. Sam, who managed and operated singlehandedly the Adja Bippo Mine near Tarkwa before 1900, the only mine of all the mines in operation since 1880 which was profitable and declared dividends (Rosenblum, 1974).

Today it can still be assumed that there will be no re-juvenation of skilfull vein mining by African small scale miners as long as it has not been proved to Ghanaian investors that such operations could be viable. The proof can be presented best by a few reference operations applying durable, locally available machinery, simple and sound mining and extraction methods, basic safety standards and bearable working conditions as well as in their effect on the environment controlled operations. Such reference mines could also provide the necessary information in respect of day-to-day running problems, required amounts of investment, minimum working capital, maintenance and repair needs and problems with unskilled labour drawn from the surroundings of such small mines, to mention only a few areas of missing experience.

THE TETREM SMALL MINE

The Tetrem Small Scale Mining Group (TSSMG) was formed in early 1990 and obtained a Small Scale Gold Mining Licence on 15th July 1991 over an area of 25 acres (about 100.000 m²) adjacent to the Village of Tetrem, Ahanta West District, Western region of Ghana. Tetrem is situated about 2 km after the Axim-Elubo Junction on the road between Agona Nkwanta and Tarkwa. The licence area was chosen on the basis of a favourable geological report by the then Gold Coast Geological Survey Geologist Tom Hirst, written in 1932, and after inspection of the overgrown workings, which included at that time one old, fallen in shaft.

HISTORY

Hirst reported in 1932 as follows: "This prospect lies in flat ground in an open valley at the head of the Fankoba river." ... "It is exposed in three recent trenches over a length of 80 feet (24.38 m), and was seen 93 feet (28.35 m) to the south-west in another trench. The reef strikes 37° magn. and dips vertically. It is 2 feet (0.61 m) thick at the extreme N.E. end, 1' 6" (0.46 m) in the two trenches southwest of the above, and only 6" (0.15m) in the last trench 93 feet (28.35 m) to the south-west. The total length as seen is therefore 173 feet (52,73 m). The quartz, which lies in granite (with little ferromagnesian material) is dense, greyish and somewhat greasy in appearance, and in places rich in pyrite mostly altered to cubes of limonite. It is smoky in places due to indeterminate inclusions. It is, however, everywhere rich in visible gold, associated with pyrite, or limonitic material. One large block on the surface, almost a yard square, gave

coarse visible gold almost every time it was broken with the hammer, whilst very rich tails of gold were obtained by dollying over the whole extent of the reef. It was extremely difficult to obtain samples free from visible gold. A sample from the extreme N.E. end gave 25.6 dwt. per ton (39.8 g/t) whilst another taken 80 feet (24.38 m) to the south-west gave 17.6 dwt. (27.4 g/t) (Ariston assays)." (Metric measurements in brackets inserted by the author).

Hirst's report showed that the vein had been discovered and worked to a little extent by African miners. Later on, the Ahanta Mining Company sunk two shafts and dug several trenches in the area but their findings are not known today and they resolved not to start a mining operation. The area was sporadically mined by Galamsey-miners over the years, which resulted in various rock-heaps scattered around the main trench on the vein and broken rock scattered at the water-pools of the Fankoba river.

GEOLOGY AND DESCRIPTION OF THE ORE

The orebody is a vertical dipping quartz vein which filled a shear zone related fracture at the contact between the undifferentiated greenstone and tholeitic basalt in the West and the belt granitoid of quartz dioritic composition in the East (Loh et al., 1995). Several volcanic-intrusive events had consecutively opened the fracture and the quartz shows clear laminations which also contain the greater part of the gold and the associated sulphides. The quartz consists of the bluish smokey, grey-white-greasy and the pure-white, hungry looking types. The latter is mostly barren but contains sporadically isolated specks of gold visible to the naked eye. The vein is distinctly

frozen to the hanging wall in the west and at the footwall to the east a thin black-brown leader defines the contact. Samples taken of the hanging and footwall showed values below 0.2 g/t, i.e. no mineralization.

INITIAL EXPLORATION, ORE RESERVES

The Tetrem Small Scale Mining Group undertook an exploration and sampling exercise before first operations were started in 1992. On the basis of 30 samples taken along the reef in the open trench near the surface an average grade of 11.48 g/t with an average width of the ore of 1.015 m over 122 m striking length was established. The reclamation of the old shaft to a depth of 11 m and subsequent sampling allowed the ore reserves to this depth be pegged at 3500 t with a gold inventory of at least 1200 oz. Further reserves were to be expected in the extension into depth and at both ends of the vein being assumed to be open ended (Barko, 1992).

FIRST OPERATIONS

A group of Small Scale Miners from Tarkwa was engaged in mining the locality for about 18 months in 1992 and 1993. They employed the usual galamsey methods of operation by digging the trench to depths of about four metres, digging short shafts and extracting ore through chiselling and breaking with crowbars. The ore was then handpicked and sorted and mainly the oxidized rock, clearly distinguishable through its reddish appearance, and other high grade material was broken with hammers, crushed in mortars and

sluiced with the usual sluice-boards equipped with cocoa-bags for gold dust recovery. The operations of this group were quite unsystematic, caused erratic holes and pits as well as dumps of broken rock mixed with soil all over the area. Due to the fact that this group of miners resolved to hide the gold produced for their own purposes and failed to account for gold production as well as supplied tools and inputs the leaders of the TSSMG dissolved this group and stopped their operations. The area was lying idle up to the time of the present operation described in the following.

START PHASE OF THE PRESENT OPERATION

Mobilization

The present operations at Tetrem were started in early September 1995. The mobilization phase included clearing of the site from overgrowth, re-cutting of the borders of the licenced area, felling of trees near the open trench for safety reasons and enlarging an existing pond in the valley for process-water supply. The erection of a fenced shed for tools and materials, the building of a latrine, acquisition of accomodation from a farmer for outside workers and having the access road with rock fillings made passable was part of the mobilization. Because the area was quite unsafe due to the rock and soil heaps placed nearby the trench causing falls of the sidewall of the trench a major exercise of clearing and sorting those heaps was begun. At the same time the trench was also excavated to remove all loose material fallen or pushed in and to expose the vein.

From the mixed up materials the rock boulders and pebbles were first sorted out by hand and the remaining soil was sieved through a

grizzly with 2 cm openings. This grizzly had been produced locally from round 1.3 cm building steel welded on a frame of 1 by 1.5 m dimension. About 120 t of rock of the size between 50 cm and 2 cm size was recovered in this way representing the rejected rock of all previous galamsey operations.

After panning tests had shown some gold in the soil a test run was carried out with the water-powered Prospector II equipment, supplied by the Small Scale Mining Project of the Minerals Commission. Several m³ of the soil were washed but gave a total return of 0.3 g of gold only. The supplied Honda-pump to power the Prospector II turned out to be unsuitable and got damaged beyond repairs (broken connecting rod with subsequent damage of the crank-housing). The only gain from such an operation would be the broken rock of size below 2 cm washed out with the sieve of the Prospector II with sieve-openings of 6 mm. This material assayed 2.13 g/t, being therefore very marginal.

The total cost of the mobilization phase up to end of the year 1995 amounted to 4.71 Mio. Cedis.

First ore treatment tests

Because no experience with quartz vein ore of the type found at Tetrem was available, the extraction of the ore was first tested with a pilot-plant consisting of a laboratory type jaw-crusher with 10cm by 7 cm opening, a hammermill obtained through conversion of a hammer-cornmill (change of hammers) and a pulverizer (Make Brown, USA) with 20 cm diameter carbon-steel grinding plates. All three machines were mounted on a small trailer and powered by an

5.97 kW (8 hp), slow rpm (850 rpm/min) one cylinder diesel engine of Indian make (Indo-Agro International) as usually used in Ghana for powering Corn-mills.

Rocks sorted out from the heaps were reduced with hammers to a size less than 5 cm, crushed by the crusher to less than 10 mm, transferred into the hammermill for grinding to a size below 2 mm and then milled in the pulverizer to 100 % of the fines below 1mm. The overall capacity of the plant was about 1 t per 8 h shift. The fines were then sluiced over a sluice box of 7.3 m length (two times 12 feet) equipped with corduroy for gold recovery. The water supplied to the sluice amounted to 30 l/min. From the concentrates obtained, an average gold content of 10.5 g/t of sluicable product was won with tailings assaying 1.26 g/t, resulting in a recovery of 89%.

The tests proved an extreme hardness of the ore, subsequent mortar tests with experienced small miners from Tarkwa revealed, that with the given material they were only able to produce about 12 kg of sluicable product per day instead of their usual output of at least 50 kg sluicable product from Tarkwa blanket materials.

Beside the extreme hardness also considerable abrasiveness of the ore was experienced, resulting in heavy wear of the steel plates in the small crusher, the hammers and linings of the hammer mill and on the plates of the pulverizer. In fact, the pulverizers' carbon-steelplates were reduced by more than 1.5 cm in thickness after milling only 2 tons of ore.

The results clearly showed the cause why this deposit had not extensively been mined by galamsey miners in the past. The hardness and abrasiveness of the ore simply prevented treatment

with the conventional comminution methods available to galamsey miners.

MINING OPERATIONS AND RESULTS

During the initial clearing of the heaps near the main trench and during excavation of the loose material out of the trench care was taken to deposit the resulting sieved soil material as far away as practical from the trench to prevent the material from running back during rains. The main trench, the north end and the southern tail of the reef were completely dug out to the extent of the previous galamsey operations. In the course of this work, the second old shaft of the Ahanta Mining Co. was re-discovered, which had completely fallen in over the first 5.5 m (18 feet) from surface and had to be re-timbered because of the loose saprolite in the shaft walls. The remaining of the shaft was found closely timbered to a depth of 12 m and the timbering still in very good condition after being about 60 years under water.

The south end of the reef was opened up by an open trench started at the bottom of the valley along reef strike to ascertain the reef width in depth and to allow later gravity dewatering of the main workings. The reef was found to be uneconomically low in grade and in thickness, below 2 g/t and thickness not exceeding 20 cm (8") over the first 25 m striking length from the South end, average thickness being less than 10 cm.

Also further north, in the main trench and at the north end of the vein, the ore-widths found at a depth of about 4 m were considerably lower than previously established near surface during

the exploration. At the north end the vein thinned out to below 15 cm and dives downwards. For the remaining workable area of 75 m striking length with reef widths of above 25 cm an average weighted width of 38.4 cm has been determined, which probably will be reduced further due to the fact that in the rediscovered South shaft at 12 m depth the vein is only 16 cm wide. The available ore reserves from the previously mined out level down to the 10 m level was recalculated to be 360 tons only, together with the rocks won from the heaps and rubble therefore less than 20 % of the originally assumed ore reserve to this depth.

Mining from the four meter level further down by digging and throwing the materials out of the trench became too laborious. Therefore, the trench opening was partially closed at about 3 m depth by scissorlike wooden support to create a platform for deposition of waste material on top and at the same time to support the walls to prevent caving in.

The mining is carried out in a kind of underhand stoping with slicing first a portion of the hanging or footwall - whichever is found softer to be excavated with pickaxe and shovel. In a second step, after removing the waste, the ore is separately broken in with chisels and crow bars, when necessary reduced in size with sledgehammers and then hoisted with buckets to the surface. Ore transport to the treatment plant is effected in half oil drums and by a trolley on car-tires as in common use in Ghana's markets for transportation.

In eight months of operation, about 240 tons of ore have been won and about 200 m³ of waste materials have been moved, resulting in a total output per man and shift of about 0.38 t including all accessory works like timber cutting, timbering, transport of ore to mill

etc.. The overall mining production rate in respect of ore is 0.125 t per man and shift.

Dewatering of mine workings

Water is the enemy of any miner and this was experienced to the extreme at the operations at Tetrem. During the dry season, when operations were started, not much difficulties by occasional rains were experienced. The two shafts acted as dewatering sumps and by pumping them out with an electric pump, driven by a generator-set, the workings were kept dry.

Conditions changed dramatically with the advent of the rainy season in May 1996. The area became flooded during torrential rains and the workings filled up to the brim with water, which caused the saprolitic material in the walls, which in the dry season was only to be mined by heavy breaking with pickaxe, to become soft, greasy and muddy with the result of the collapse of a part of the workings. The amount of water in the ground became so excessive, that even day long pumping did not lower the water table in the main trench. Only the south end was kept somehow dry due to the gravity dewatering of these workings through the existing tunnel towards the valley. Within the tunnel, heavy re-timbering was necessary to prevent the caving in of the tunnel due to excessive pressure from the saprolite in the walls. In fact, every stull of the previously set timber support had to be supported by posts with bracings mounted at the top and foot.

Safety aspects in mining operation

Every worker is personally protected by a helmet, safety boots, leather-gloves and, where necessary, with safety goggles to prevent eye damages during chisseling of rock or breaking of boulders with the sledge hammer.

It has been found that constant reminders are necessary to enforce the wearing of these safety items. Virtually every day some workers were found to flout these rules due to comfort, negligence or because they had taken the helmet or boots home and left them there. The helmet and boots became rather a status symbol in the adjacent village than an accepted protection equipment during work.

During shaft reclamation or in the trench at unknown areas with the possibility of underlying old stopes it was made obligatory that workers were secured by belts and ropes kept as short as possible and tightened to firm posts outside the workings. This safety measure was also constantly flouted by the workers due to the restrictions imposed by the ropes during shovelling.

Work in shafts or deep trenches necessarily needs a permanent observer at surface to enable instant alarming of the foremen or other workers in case of a danger or an incident.

It was made a rule that during heavy rains and one day after heavy rains no work was permitted in unsupported trenches or pits with depth of more than 1.8 m (6 ft.). Likewise, every morning an inspection of the workings has to be carried out first before the workers are allowed to start. The inspection has to concentrate on rock-falls overnight, visible cracks or fissures in the benches or trench walls and any timber work damaged.

Discussion of mining operation

Whilst the restriction of Small Scale Mining to a depth of 10 m is beneficial for alluvial operations to prevent collapse of pits with instant burial of workers, in hard rock, especially vein mining this restriction is inappropriate because it leads necessarily to bad mining practice. For a vein like that found at Tetrem from the surface to a depth of at least 8 m a safety pillar should have been left to prevent inrush of surface waters and the collapse of sidewalls made up of unconsolidated materials like the saprolite. This would render the deposit for small scale mining useless, because only some two or three meters of ore would be left for mining within the given limit. Consequently, the mining started right at the outcrop and with advance into depth all difficulties with wall instabilities, surface waters etc. are met with.

ORE TREATMENT AT TETREM

The present treatment plant was installed in Nov./Dec. 1995 and was fully operational at the beginning of 1996 being therefore in the Western Region the first plant of this kind in small scale mining. The plant consists of a locally obtained crusher, locally produced hammer mills, sieves, and sluice boxes.

The crusher, a Parker 14 x 7 inches (35 x 18 cm) jaw crusher made in 1956 was reconditioned with new bushings, toggle plates and linings, new crusher and side plates. It is powered by a 5.9 kW (8 hp) slow speed, one cylinder diesel engine of Indian make

commonly used for powering cornmills. The transmission is done by a flatbelt of 10 cm width from the pulley of the engine to the flywheel of the crusher, giving a transmission ratio of 4:1, i.e. the crusher is operated with about 200 rpm. Despite some doubts at the beginning the engine turned out to be sufficiently strong to drive the crusher even when fully loaded with rock.

To obtain a maximum amount of fines from the ore, the crusher was installed on a flat concrete platform to allow crushed material to build up under the crusher in order to retain the ore as long as possible between the crushing plates. It was observed that the moving jaw of the crusher, the "stock", pushes the crushed material from the outlet opening to the front of the machine thereby separating coarser material from the fines. The coarser material is concentrated on the outside of the resulting heap and can easily be picked with shovels and immediately recirculated.

This mode of operation produced an average of more than 30% of fines below 1mm size per crushed ton of rock. The unusual operation of the crusher, which lead to wear on the stock which is permanently pushing the material, necessitated the installation of wear plates made of spring steel at the lower end of the stock to prevent the wearing of the lower plate holder and the end of the stock.

The material obtained from the crusher is loaded by shovel on a double deck sieve of local production, which is vibrated by a rope connected to the moving crusher stock. The sieve with 6 mm square opening separates the oversize from the fines, which are further separated by a sieve of 1 mm square opening in mill entry product (-6mm to + 1mm) and fines of -1mm, which are sent without further

comminution to sluicing. The oversize of +6mm is recirculated into the crusher and for each batch of rock at least three recirculation cycles are performed before sieving the oversize through a sieve with 12mm square opening and sending the resulting fraction of +6mm to -12 mm to the stockpile. It was found that this material consisted mainly of white quartz with an average grade of only 1.81 g/t gold, but caused excessive wear on the crusher and mill when milled further down.

Of prime importance to mention is the fact that from the crusher fines of below 1 mm size the majority of the gold is obtained. It was proved in several tests, that an enrichment of the gold takes place in these fines. The explanation is found in the nature of the ore which is laminated and the gold is concentrated along the lamination fissures. During crushing the ore tends to break along those laminations and the gold contained therein is liberated. As a general rule, from one ton of laminated ore about 60 % of the total gold was won from the crusher fines, and about 40 % from the consecutively milled portion of the ore, although the crusher fines represented only up to 400 kg from that treated ton of ore. In their returns from crusher fines and mill fines more balanced gold recovery was only observed with the more dense, grey-white quartz-ores with less ore non-existent lamination. In such ores the gold is obviously uniformly distributed.

The mill entry product of - 6mm and + 1mm is milled in a hammer mill to 80% below 1mm fines, which are send to sluicing. The mill was built from a locally obtained palm kernel crusher through lining the inside of the mill housing with spring steel and fabrication of a rotor, which is holding three pairs of hammers manufactured also

from flat spring steel. Running this mill under production conditions necessitated the reconstruction of the mill shaft with improved bearings and dust seals. The mill is powered by the same type of diesel engine as used for the crusher, transmission also by flatbelt of 10 cm width with a transmission ratio of 2.5 to 1, i.e. the mill is operating at about 2000 rpm.

The fines from the mill are caught in small drums, in which a type of skirt is hanging from the outlet of the mill to prevent the dust from escaping. The drums are then transferred to the sluicing unit.

The sluice consists of two wooden sluices of 45 and 50 cm width and 3.65 m length each, used in series and equipped with corduroy. The corduroy is held in place by frames fitted into the sluice boxes. The boxes are set at an angle of 8° and 9° respectively, which is easily effected by wooden wedges of these angles used together with a spirit level to adjust the inclination of the sluices. The water supply of 50 l/min is obtained from a 1 m³ watertank, put on a timber crib of round timber obtained during the felling of trees during mobilization. The height of the watertank above the sluice hopper is about 5 m and the water supply is effected from the pond in the valley with a 5-hp-Hondapump. The water is pumped via 5 cm diameter plastic hoses about 15 m high into the tank. During the sluicing, the re-starting of the pump for a duration of about 5 min. after about every 30 min is required to operate the sluice continuously.

The fines are suspended in a hopper with the water fed by a waterhose and then run through both sluiceboxes. The tailings flow freely into a tailing pond from where they are regularly backed onto a tailing heap for eventual further treatment in the future.

Depending on the type of fines treated the corduroy is washed according to a set-out schedule regularly and the concentrates collected in plastic buckets for further extraction.

To allow uninterrupted milling and sluicing operation during rains and especially to keep products dry, which is of prime importance for the headmaterial of the hammermill, the whole treatment plant as well as the sluice boxes have been roofed by using the local "roofing sheets" made from bamboo-leaves.

Safety aspects in the treatment plant

As in mining, all workers have to wear protective equipment like helmets, safety-boots and leather-gloves. The wearing of goggles at the crusher and the mill is obligatory to prevent eye damages from fly-rock out of the crusher during crushing and from splitting rock with the sledge hammer. Wearing of tight fitting shirts is enforced to prevent workers being pulled into the machines by rotating parts.

The wearing of dust masks and their permanent replacement is of highest importance to protect the workers from silicosis. It took some time to convince the workers about the necessity and there was no other method than indictment of recalcitrant workers to enforce the use.

Due to the running of the machinery with flat transmission belts, special attention has to be paid to the belt connectors, which can inflict serious wounds when breaking off. Daily inspection and if a break is found, immediate replacement is necessary. In addition,

wooden poles have been placed in front of the pulleys or flywheels to catch any off-flying belt.

The operation of the generator-set and running of electric tools or the electric water pumps for dewatering the shafts was restricted to the presence of the engineer. It was found that the workers had had no exposure yet to electricity and the dangers involved. Good earthing and all electric motors and tools being equipped with earth wires is essential. The generator-set is equipped with a warning-device in respect of insufficient earthing and permanent controls are essential to prevent electrocution.

Treatment of concentrates

The collected concentrates are always transferred to Tarkwa to avoid amalgamation on the site due to safety and environmental reasons. The concentrates are further concentrated by use of the traditional wooden bowl, the rejects being collected for re-circulation on the sluice boxes. The resulting gold concentrates are first treated with a magnet to remove steel powder from wear of the crushing and milling plates, then amalgamated and the amalgam separated from the concentrate through washing. The resulting tailings are kept for further treatment in the future, because it was found that they contain still 5 to 6 oz. Au per ton of concentrate, mainly locked up in sulfides but it is also noteworthy that one ton of such concentrate is only obtained after milling about 1500 t of ore.

The amalgam is pressed to remove free mercury, then retorted, providing the sponge gold for smelting to arrive at the final product.

Operational experiences with the sluice box

Tests were carried out by weighing the fines before sluicing and after sluicing by washing the corduroys of the first half and second half of the upper sluicebox as well as the corduroy of the lower sluicebox separately to keep the concentrates obtained separate. At the same time sampling of the tailings was carried out.

The results obtained showed clearly that above 90 % of the gold is caught on the first 1.8 m of the upper sluice, about 5 % on the lower half 1.8 m of the upper sluice and only less than 5 % on the lower sluicebox, here mainly only occasional specks which have rolled down from the upper sluice and a little fine gold dust, see for example Table 1. The assays of the concentrates after amalgamation showed only 1.5 % of gold locked up in the concentrates, about 100 g of concentrates per ton of fines sluiced was obtained.

Table 1 Concentrate and gold recovery distribution on sluice box

	Concentrates obtained (%)	Gold recovered (%)
Upper Sluice box		
Upper 1.8 m section	49	93.6
Lower 1.8 m section	41	4.9
Lower sluice box (3.6 m)	10	1.5
Total	100	100

The total recovery of the sluice was therefore established to be 84 % with respect to the material sluiced which was obtained from the long exposed rock heaps and outcrops of the vein. This rock had been weathered and all pyrites decomposed to limonite or even only cavities with visible gold specks left in the rock. The tailings from such materials contained between 1.26 and 1.37 g/t Au.

Recovery changed dramatically when fines from fresh rock were sluiced. The recovery dropped as low as 51%, as had to be expected with fines containing sulphides. The distribution of gold recovery on the upper and lower sluice box did not change much, still only up to 5% of the gold was caught on the lower sluice box. The tailings however, increased in grade to an average of 2.2 g/t, with single results considerably higher to nearly 5 g/t. The concentrates obtained from the corduroy increased significantly in weight per ton of fines sluiced. Roasting, pounding and direct amalgamation with consecutive assaying of the remaining tailings proved gold contents as high as 1 g/t of originally sluiced material or

up to 15 % of total gold recovery obtained from these concentrates alone .

Tests with sulphidic ores to improve recovery

The tests on the concentrates and tailings showed that considerable amounts of gold are locked in the sulphides and consequently lost. Trials were therefore started to increase the amount of concentrates obtained from the sluice box by washing the corduroy far more often than previously done. Through washing the corduroy of the whole sluice box after every 300 kg of fines sluiced, quite large amounts of concentrates are obtained, which can reach weights of up to 1 kg per ton of fines sluiced. The operation has shown that sulphides settle quite easily in the small riffles of the corduroy despite the minimal difference in specific weight to quartz rock of same particle size. Immediately when the riffles are packed, further sulfides will float over and are lost, whilst gold is still settling in the corduroy riffles, replacing already settled sulphide particles.

The free gold is extracted from these concentrates by using the wooden bowl as previously described in chapter Treatment of Concentrates. The rejects of this concentration are roasted/dried, pounded in mortars and sieved to 100% of the material below 0.3 mm. Direct amalgamation of these concentrates resulted in recoveries of up to 1g per ton of fines sluiced.

In a second attempt to minimize losses, the tailings were allowed to flow into an aluminium basin, which was emptied after every 300 kg of fines sluiced, resulting in about 30 kg of concentrated tailings. Assays showed values as high as 30 g/t of such material. These tailings are then dried and re-milled to about 80 % below 0.5 mm

and sluiced, which resulted in gold recovery of between 4 to 5 g/t of these tailings.

The decision was therefore taken to construct a buddle for further tailings concentration. Through sampling and assaying the core and the outer third of the deposited material it has been proved already that the core contains grades about four times higher than the average tailings. The tests have not yet been concluded and further results are awaited.

Amalgamation and smelting problems with sulphides

After entering into fresh rock the quality of the amalgam won as well as the recovery from the smelting of the final product changed. The losses during smelting increased from only a few percent at the beginning to nearly 50 % in one smelting operation, where it was found that the concentrates had contained galena as accessory mineral which came as a complete surprise and caused difficulties in retorting as well as smelting. The fineness of the gold also reduced considerably from nearly 23 karat at the beginning (two times re-smelting) to as low as 16.8 karat. Gold fineness of between 18 and 19 carat from fresh rock concentrates could only be obtained by re-smelting four or five times and treatment of the product with sulphur, salpeter and nitric acid during the smelting processes. This processing caused higher losses and reduced the recovery rate.

Results of ore treatment at Tetrem

During six months of operation, a total amount of 140 t of ore has been treated and 120 t of sluice fines extracted.

The average recovered grade, i.e. final, smelted product, was 6.14 g/t, ranging for single ore batches from 1.4 g/t to 10.5 g/t. The average head grade of the ore was calculated to be nearly 10 g/t (9.94 g/t) which is 1.5g/t lower than the average grade found during initial exploration. The "picking of the eyes" by the initial "galamsey"-operation could account for this. Anyway, the grade of the ore is, as usual with such quartz veins, erratic and patchy and at Tetrem with clearly lower grades at the northern and southern end of the exposed vein. Ore batches with grades below cut-off-grade were milled because preliminary pounding and panning with sample-tyre had shown them to be carrying but the sampling obviously did not represent the whole batch of ore. In the opposite, ore that was assessed very marginal through pounding and panning of samples turned later out to be even above average in grade. The ore-grade control in a small scale operation like Tetrem is one of the major problems discussed further below in the chapter "Profitability".

The average recovery at the treatment plant was found to be 74% with single recoveries from nearly 90% to as low as 50% in heavy sulphidic ores. The recovery in gold extraction also varied between 98% and 51 % giving an average recovery of 84%. Therefore the overall recovery of the operation was 62%.

The plant utilization, here defined as time of plant worked to available working time in a one-shift per day operation was around 40%, this mainly due to non-operation of the plant when the engineer was not present. The average output of the crushing and

milling plant was just above 2t/day with single outputs ranging from 0.5 to 3.3 t/day. The actual long term capacity of the plant per month in one shift/day operation can be estimated at 50 t.

At the sluice, depending on the frequency of washing of the corduroy and depending on the availability of water in the pond for pumping, at an average 2.2 tons of fines per day have been treated, with peak outputs of nearly 4.5 tons per day. During the dry season, the sluicing was hampered by non-availability of water in the pond. In March and April, sluicing was only possible by using the water pumped from the two shafts and recirculating this water from the pond where all treatment water is caught. After about 5 to 6 days this water reserve was exhausted and it took about two weeks for the shafts to refill.

Technical problems with plant machinery, maintenance, repairs

Due to the extreme hardness of the ore, the crusher jaws as well as mill hammers and linings suffered extensive wear. Crusher plates made of manganese steel with teeth-heights of 4 cm were found to be worn flat after crushing of 30 to 35 tons of ore. The side-plates showed washing out to half their original thickness of 2.6 cm. The wear and therefore the steel consumption per ton of ore crushed is estimated to reach one kg/ton of ore.

In the absence of any new spares to be obtained locally it became necessary to rebuild the crusher jaw plates. Flat spring steel from obsolete lorry springs is cut into short pieces and welded on the worn crusher jaw plates, resulting into a lined but flat plate face. This is not a disadvantage for a small mine operation because the final

product should be fines and not aggregates like in quarries. The re-lined flat plates showed lifetimes of 20 to 25 tons crushed before the spring steel is worn off and new lining is necessary. The side-plates are also rebuilt with flat springsteel.

Similar wear is observed on the linings and at the hammers of the hammermill. Production experience has shown that a new set of hammers is worn after milling of 3 to 4.5 tons of millhead. The hammers, consisting of 6 cm wide flat spring steel are then reduced by 3 to 4 cm in length, reducing the output of the mill due to the increased gap between the hammers in rotation and the mill lining. The lining of the mill, made also of flat spring steel has to be replaced after every 25 to 30 tons of milling. The average steel consumption in the mill has been found to be around 0.5 kg per ton of milled material.

Major problems were encountered with the bearings of the crusher and the mill. The outer lead bush bearings of the used crusher wore out after about 80 tons of crushing. There are only two places in Ghana (Tema Shipyard and Drydock Ltd. in Tema, Ghana Railway Corp. in Takoradi) where such bearings can be recasted and where the original material for such bushings is available.

The roller bearings of the mill were first too small dimensioned and it was found in addition that due to the abrasiveness of the dust even completely sealed bearings were damaged through dust entering the rollerrace. Re-construction of the shaft and the bearing assembly of the mill has solved these problems but in any construction of similar equipment special attention should be paid to this construction detail because of the high cost of bearing replacement.

The set of rollerbearings for the mill cost presently above 200.000 Cedis.

The decision to power the crusher and the mill at Tetrem with Indian made one cylinder diesel engines of the "Lister-type" was based on the very competitive price (below 1 Mio. Cedis per engine), the apparent robustness with service weight of 340 kg and the assumption, that these engines can be repaired virtually everywhere by any fitter or blacksmith because of their widespread use in the hinterland of Ghana for powering corn and oil mills, cassava graters etc. This assumption was a misconception because almost all fitters called upon turned out to perform trial and error jobs. They were unable to put the engines back to reliable operation after some minor faults had developed like worn connecting rod bearing, injector blockage, oil or diesel leakages. There was no other option than to train a worker in doing minor repairs and maintenance and to transfer faulty engines to Accra for repairs and overhaul. Due to these circumstances a spare engine was acquired to change the whole engine rather than to carry out major repairs on site with the resulting down times and production loss.

A second problem posed is spareparts which are not available everywhere. In addition, too many fake parts are on the market, all bearing the Lister brand but being produced somewhere else, which do not last or, even worse, cause additional damage when built in. This necessitates having a stock of spareparts available at the mine to secure fast repairs.

Handtools like pickaxes, shovels, spades, chisels, crowbars, hammers, sledge hammers need continuous servicing in respect of their wooden parts and handles and access to a blacksmith for

sharpening and hardening of the picks, crowbars and chisels is indispensable. At Tetrim, one worker with carpentry background was given responsibility to carry out hand tool service daily.

All the above proves the need for a small workshop at the mining site comprising of a electric welding machine, two hand angle grinder for cutting steel, electric handdrill and standing drill, vice, thread grinding equipment for threads from 4 mm to 12 mm diameter and for pipes from 3/8 to 1.5 inches (9.5 mm to 38.1 mm).

Beside the usual tool sets with spanners, screwdrivers, files, hacksaw etc., universal pulleys, a set of crowbars, a chain pulley with 3 t lifting capacity and at least two big pliers for pipes up to 3 inches are indispensable.

Access to a turning lathe and a shaping machine is necessary for manufacture of spareparts. Giving such work out to machine-shops takes usually too long a time and is expensive.

COST ANALYSIS OF TETREM SMALL MINE

Investment costs

The primary exploration, mobilization and acquisition of equipment demanded a total investment of 22.769.000 Mio. Cedis (at average of 1500 Cedis to 1 US\$) which is broken down in Table 2:

Table 2 Break down of investment for Tetrem Small Mine

Item	Costs (Cedis)
Acquisition of Licence, payments to local authorities etc	900.000
Cost for 55 Samples	1.204.000
Machinery and accessories (Crusher, mills, 3 diesel engines, pumps, Prospector II, generator-set)	12.470.000
Handtools (shovels, pickaxes, pans etc)	330.000
Workshop equipment (welding, grinding etc.)	1.285.000
Shed, roofs over plant, foundations	910.000
Safety equipment (helmets, boots, etc)	970.000
Mobilization (3 months)	4.700.000
TOTAL	22.769.000

With an average work force of 18 workers (engineer, two foremen, one mechanic/welder, 12 workers and two security/watchmen) the average investment per working place amounts to 1.265 Mio. Cedis.

Operation costs

During six months of operation average working costs per month of 2.041 Mio. Cedis have been experienced, which are broken down in Table 3.

Table 3: Operation costs at Tetrem according to cost items and cost centres

Cost items	%
Labour Cost	71
Energy	7
Consumables, spares	21
Cost centres	
Mining	45
Milling, Extraction	36
Overhead (Security, Admin.)	19

Profitability of the operation, discussion

As at time of writing (End Sept. 1996) the Tetrem Small Mine is not profitable. The gold returns just cover the operation cost, but do not contribute to investment repayment including payment of interest on investment. There are several reasons:

- Ore grade is lower than expected
- Recovery is lower than expected
- Ore width is less than expected, therefore higher mining costs
- Low utilization of the treatment plant
- Ore batches below cut-off grade were processed in the treatment plant

As already mentioned earlier, the average ore grade was found to be around 10 g/t instead of 11.5 g/t. The recovery was estimated at 80 %, but turned out to be 62%. The originally calculated cut-off grade of 6 g/t was therefore set too low and should be rather set at 8 g/t given the experienced recovery and the actual cash returns for gold with about 18 to 19 carat. This cut-off grade would definitely restrict the available ore reserves further.

At Tetrem it came as a complete surprise to have run into ore with fresh sulphides and galena as associated mineral at a level only 4 to 5 m below surface. It was originally assumed that the free milling properties of the ore met with at surface will continue at least to a depth of 10 to 15 m.

An increase of the overall gold recovery through better recovery of sulphides and better extraction of gold from sulphides will enhance the economics of the operation.

The considerably reduced ore width of 0.384 m compared with 1.014 m has caused higher mining costs, because more waste rock had to be handled to excavate the minimum necessary mining width of 1.2 m in presently mined areas where the hanging and footwall has to be supported by timbering. With increased mining depth it is to be expected that the hanging- and footwall will become more stable allowing operation with a reduced mining width of about 1 m.

The low utilization of the treatment plant was caused due to absence of the engineer and could be easily rectified by having a resident engineer on site.

The treatment of ore batches below cut-off is caused due to the in-availability of reliable assaying facilities. Carrying out reliable assays at the mining site is beyond the capabilities of a small mining operation. The estimation through pounding and panning turned out to be unreliable, as described earlier. Because of the patchiness of the ore, every batch of 3 to 5 tons should be assayed. Sending assays outside to available laboratories turns out to be prohibitive expensive, one assay costs 17 US\$ plus batch fee of 20 US\$ for the four or five samples to be sent. The services of the Small Scale Mining Centre in Tarkwa could only be used, when stockpiling larger amounts of ore to be able to send the by them required minimum number of 10 samples, which will still attract 100.000 Cedis assay-charge.

Considering an operation far away in the hinterland, the ore might have been milled already when the assay results become available at the mine. Further research and trials are necessary to find a lasting solution for this ore grade control problem.

The following graph has been developed on the basis of investment and cost records at Tetrem which can not be reproduced here. With an initial investment of 22.77 Mio Cedis, an interest rate of 50 % per annum by quarterly annuity and an investment repayment period of three years the reader can obtain from the graph the cut-off grade as well as the break-even grade at different recovery rates for an operation with similar ore (hardness, abrasiveness), one shift

operation and ore to waste ratio of 1:1. The gold returns were entered with 15.000 Cedis per g of gold of 18 carat. Gold of 21 carat fineness would give figures similar to a 10 % better recovery.

The minimum required proven ore reserves for three years operation are between 720 t for 20t per month and 1800 t for 50 t per month plant capacity.

From the graph can be learned that at Tetrem only enhanced recovery and full utilization of the plant will secure profitability if the average ore grade remains around 10 g/t. Further ore reserves will have to be identified.

As a general observation for similar quartz mining operations it can be stated, that at least an average grade of 12 to 15 g/t for the initially proven reserves is desirable to secure the success of the small mine. It is of interest to note that over 60 years nothing much has changed because in the 1930th quartz veins with less than 10 dwts. (15.5 g/t) were considered marginal or uneconomical as various progress reports of geologists of the Gold Coast Geological Survey had stated.

MARKETING

After extraction of the gold from the concentrates won through amalgamation and smelting, in small scale mining operations very often fast cash is needed to pay workers, inputs and repairs.

Unfortunately, all legal buying outfits in the Tarkwa area do not respond to quick change of gold produce into money. The Precious Minerals Marketing Corporation (P.M.M.C.) drills a hole in the gold bar, dissolves the drill-chippings in aqua regia and tests the sample

with AAS. This protects their buying interests but at the expense of waisting considerable time for the seller. When the deal is concluded, a cheque is issued which can be cashed at the bank, but of course only during banking hours.

Private buying agencies act here faster by weighing the product dry and submersed in water for determination of the fineness of the gold. They have physical cash readily available, but also only during their office hours and alàs, all too often they do not open their offices during their regular office hours. In addition, it is of interest to note that their base price is quoted on 23 carat gold whilst the official quotation of P.M.M.C. is for 22 carat. Even with a slightly better price offered, the customer will loose money if he is not aware of this difference, which is not revealed by the buyer. The buyer only quotes the base price and his buying price according to the fineness of the gold he determined. The unaware seller is loosing up to 4 %.

The Tetrem Small Scale Mining Group has lost hours and in the sum of all transactions days for just obtaining the cash for their gold produced.

The other available buyers, who claim to be licenced but never issue a proper receipt for the transaction - the receipt book is always somewhere else - are available at any day or night time, at the weekend, on Sundays and holidays. They also pay ready cash and at competitive price levels.

All buyers allow for themselves profit margins of up to 10 %, taking the fineness and the refining costs into consideration and comparing their prices at bank exchange rates to the world market price. Silver values contained in the sold product are not determined and are not honored. All buyers offer payment in local currency only. If a small

mining operation has to obtain foreign currency, for example to order machinery or spareparts from abroad, the only way is the change at foreign exchange bureaus with consecutive further losses due to the higher selling rates. In summary, in the presently operated marketing system the small scale operation might loose up to 20 % of the real purchase power of their gold produced .

ENVIRONMENTAL CONSIDERATIONS

Any mining operation should be planned and carried out in a way to minimize its effect on the environment. Mining is using the restricted land rессources only for a comparative short time to take out minerals contained in the land. This short term use should not spoil the future use of the land for agriculture, forestry, recreational or residential purposes. Same applies to water rессources.

At Tetrem Small Mine the following factors were especially observed to safeguard the environment:

1.

No use of mercury or other chemicals at the site, which is near the headwaters of the Fankoba river. The extraction of concentrates through amalgamation is carried out in Tarkwa under controlled conditions in a laboratory like environment.

2.

In respect of tailings disposal and water disposal from the sluices a series of ponds was created to allow settlement of slimes before the water seeps into the natural drainage. An additional advantage of

the settling ponds is that tailings can be recovered and stacked for future re-treatment. The initial planning of the tailing site will finally allow the deposition of the tailings in terraces as infilling of a small valley. The-run off of the tailings after final deposition will still pass the settling ponds for de-sliming.

3.

After abandonment, pits, trenches and shafts can pose hazards to animals and human beings when left open. Therefore, at Tetrem measures are taken to place waste material in a way that easy backfilling could be effected. This heaped material is acting in the meantime as a small protective dam to prevent water run-off into the workings from the slightly eastward sloping adjacent farmland. In addition, with advance into depth the chosen mining process involves the covering of the trenches, levelling the areas on surface and replanting. This provides the immediate advantage for the ongoing mining operation that rainwaters will not run into the workings but rather be led away, erosion is minimized and the workers use the levelled areas for planting foodstuff. Shafts should be planned in a way that they can be later easily covered or converted into water wells. This will require a concrete cover on surface to avoid contamination of the well with run-off or through fallen in animals. The water has to be chemically analyzed before handing such wells over to the public.

4.

All machine sites have been concreted to prevent oil or diesel spillage to seep into the ground. In case of leakages, immediate repair is enforced.

5.

As laid down in the mining regulations, a latrine has been built to prevent contamination of the surrounding with faeces and especially to prevent the possible contamination of the ponds at the headwater of the Fankoba with bilharzia. These ponds are used as water source by the villages nearby.

MULTIPLICATION OF PARTLY MECHANIZED SMALL MINES

Given sufficient ore reserves of at least 3000 t of proven ore at an average grade of around 15g/t for a gold occurrence the multiplication of partly mechanised small mines in Ghana should be possible if the following factors are observed and assistance will be provided.

From the experience gathered at Tetrem Small Mine the initial exploration is of prime importance to such quartz vein mining operations. Whilst in most cases no money will be available for extensive exploration, the exploration has to prove at least the minimum amount of ore reserves to enable repayment of the investment. For further continuation of the operation out of the ore development can take place. The near-surface properties of the vein can be assessed by digging trenches and sampling but the extension into depth will be only assessed properly by drilling.

Assistance for Small Mines should therefore provide cost-free or pre-financed core-drilling facilities to prove ore-grades and -thicknesses to a depth of 15 to 20 m. This could be effected by employment of Cobra-drills (Petrol-driven, portable handdrill with core-barrel).

Depending on the extent of the orebody, at least some 50 to 100 samples will have to be analyzed and costs of 2000 US\$ have to be reckoned with. The samples should not be analyzed for gold only but also for elements like silver, copper, lead, zinc, arsen and iron to allow an immediate assessment in respect of sulfides and/or refractory nature of the ore.

Tetrem's operations have shown that local equipment, which has been extensively tested under actual working conditions, can be used for similar operations. But the availability and the distribution of the machines alone will not promote the multiplication. Assistance with spare-part depots, workshop- and servicing facilities would be of prime importance. Crediting and selling of suitable equipment without securing the after sales services and a continuous engineering assistance is inappropriate.

Not all investors can afford the investment solely from equity. The necessity of loan-facilities for the small scale mining sector in Ghana through opening of mining windows at banking institutions had been postulated at every conference or meeting in respect of Small Scale Mining but until today such facilities are missing. Credit systems for imported mining equipment were abrogated because the supplied, untested equipment failed after a short time and the small miners did not repay the loans for already obsolete equipment which did not contribute to income from their operations.

With short-term financing opportunities available, operations like Tetrem could resolve seasonal problems like water deficiencies during dry season and too much water during rainy season through seasonal mining schedules, which demand higher working capital outlay. During the dry season only ore winning would be performed and all ore stockpiled for treatment in the rainy season, when sufficient water is available for high utilization of the sluice boxes. Such schedule will require working capital for at least 5 to 6 months of operation without any tangible returns.

As long as no specialized experience about the particularities of mining operations is available in the banking sector the multiplication of small mines like Tetrem and mining ventures of Ghanaian companies in general will be seriously hampered and only foreign investment will promote but also control all gold mining activities in Ghana.

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