

Gold Ore Treatment at Tetrem Small Scale Mine

By

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ABSTRACT

This publication is the continuation of a previous presentation of the operation and cost analysis of the partly mechanised Tetrem small scale gold mine which discussed in detail the mining, safety and environmental aspects of the venture.

At Tetrem, various experiments with different machinery for crushing and grinding the ore in preparation for gravity separation (sluicing) were carried out and the results are presented. The trials led to the construction and application of suitable, locally obtainable equipment for such milling operations. Continuous weighing and sample checks at the different stages of the comminution and gravity separation provided interesting production and recovery data which could be of help for similar operations.

Trials for improved recovery of gold from sulphidic ores were carried out and showed the restrictions of gravity separation when confronted with such „fresh ores“.

The amalgamation process, recirculation of tailings, special handling of tailings for recovery improvements are described. An overview about the results obtained during six months of operation is presented.

Improvements for the locally produced machines and problems with maintenance and repairs of the equipment as well as strategies to solve them are discussed.

The experiences obtained in the marketing of the final gold product are critically outlined.

1. INTRODUCTION

In a previous publication „Operation and Cost Analysis of a Partly Mechanized Small Scale Gold Mine in Ghana“ the operations of the Tetrem Small Scale Mine were described in detail. In the course of the operation various tests to find the best way of extracting the ore, to enhance recoveries, to save gold from tailings were carried out which results could be of interest for other, similar operations. The aim of the following is to present the data and experiences collected.

2. DESIGN AND OPERATION OF A PARTLY MECHANIZED PLANT FOR GOLD EXTRACTION AT TETREM

2.1 Preliminary pilot-tests

Because no practical extraction experience with quartz vein ore of the type found at Tetrem was available, the milling of the ore was first tested with a pilot-plant consisting of a laboratory type jaw-crusher with 10 cm 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 left by the previous galamsey-miners 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 pilot-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 at Tetrem 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 basket 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 grinding 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. Consequently, the size and design of the Tetrem plant had to match these extreme properties of the ore.

2.2 The ore treatment plant at Tetrem

The 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 consisted 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. The crusher plates were produced locally by welding lorry-spring steel on the baseplates. The crusher was powered by a 5.9 kW (8 hp) slow speed, one cylinder diesel engine of Indian make commonly used for powering cornmills. The transmission was effected 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 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.

This Parker Crusher was obtained from a local quarry at the cost of 3 Mio. Cedis (or 2000 US\$) on as it where it is basis. For the Tetrem operation this was the easiest way to obtain a suitable crushing equipment. Later investigations were carried out whether such crusher could be produced locally. In the industrial area of Tema suitable workshops were identified who could manufacture copies of such crushers out of steelplates for ship-building. Production of fly-wheels, the crusher axle and the bearings are also practicable. Invoices obtained for a similar crusher produced locally ranged from 4.0 to 6.0 Mio. Cedis (or 2,600.- to 4,000.- US\$).

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. When producing a crusher locally, this feature should be included into the design of the stock.

The material obtained from the crusher was loaded by shovel on a double deck sieve of local production, which was vibrated by a rope connected to the moving crusher stock. The sieve with 6 mm square opening separated the oversize from the fines, which were further separated by a sieve of 1 mm square opening in mill entry product (- 6mm to + 1mm) and fines of -1mm, which were sent without further comminution to sluicing. The oversize of +6mm was recirculated into the crusher and for each batch of rock at least three recirculation cycles were performed before sieving the oversize through a sieve with 12mm square opening and discharging 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 and therefore marginal, but it caused excessive wear on the crusher and mill when milled further down. Therefore

the gold content of this fraction could not pay for the milling cost. This stockpile material was later sold to a building contractor who used it for terazzo work.

Of prime importance to mention is the fact that from the crusher fines fraction 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. 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 was milled in a hammer mill to 80% below 1mm fines, which were send direct 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 was 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 4 to 1, i.e. the mill operated at about 2000 rpm.

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

The sluice consisted 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 was fixed in place by frames fitted into the sluice boxes. The sluice-boxes were set at an angle of 8° and 9° respectively, which is easily effected by wooden wedges cut to these specific angles and used together with a spirit level to adjust the inclination of the sluices. The wedge will be placed on the top of the side wall of the sluice and the spirit level will show level, when the sluice is set at the appropriate inclination.

The water supply of 50 l/min was 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 was about 5 m and the water supply was provided from the pond in the valley with a 5-hp-Hondapump. The water was pumped via 5 cm diameter plastic hoses from the valley 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 importance of the watertank lies in the even water pressure during sluicing. Only with a virtually constant water flow optimal settlement conditions in the sluice boxes can be maintained.

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

Depending on the type of fines treated the corduroy was washed out 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 had been roofed by using the local "roofing sheets" made from bamboo-leaves.

A flowsheet of the treatment plant is given.

2.3 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, who had been recruited from the nearby village, 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.

2.4 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 had 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

Having measured the weight of the fines sluiced in each batch operation and the gold obtained from the respective concentrates, having determined the remaining gold in the tailings as well as in the concentrate-residues after amalgamation by respective assays, the total recovery of the sluice was established to be 84 %. This 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 rock 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.

2.5 Tests with sulphidic ores to improve gold 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 the heavier gold is still settling in the corduroy riffles, replacing already settled sulphide particles.

As a general experience it was observed that the recovery of the free gold did not change much along the sluice, but due to loss of sulphides the overall gold recovery was reduced, sometimes dramatically.

The free gold was extracted from these concentrates by separation in the wooden bowl as described in chapter Treatment of Concentrates (see below). The rejects of this concentration process were roasted/dried on an open iron plate fired by charcoal, 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, i.e. between 10 to 15 % additional gold recovery from one ton of fines.

In a second attempt to minimize losses, the tailings discharged at the end of the sluice 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 taken showed values as high as 30 g/t of such material. This indicated that a gravitational enrichment process had taken place within the bowl. These tailings were 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. A buddle is an old stationary concentration device which consist of a circular pond or tank with a slightly conical bottom, the peak of the conus situated in the centre of the circular tank. Through a feeding pipe the tailings are discharged right on the top of the conus and the tank will gradually fill up with tailings deposited in layers within the tank. The build-up process is controlled by raising a weir at the outlet of the tank, where the process water is discharged. The buddle at Tetrem was capable to receive the total tailings from one day sluicing, i.e. up to 4 tons. The tailings were allowed to settle over night, then the inner core of about 50 cm diameter

was dug out and returned to the mill, the rest of the tailings was backed to the tailing heaps.

Through sampling and assaying the core and the outer third of the deposited material in the buddle it was proved that the core contained grades about four times higher than the average tailing grade. The tests have therefore shown that such simple device can help to improve recoveries. The core material had to be sun-dried and was then remilled and again sluiced and showed recoveries of about 2 to 3 g/t of remilled fines.

2.6 Treatment of concentrates

All collected concentrates were always transferred to Tarkwa to avoid amalgamation on the site due to safety and environmental reasons. The concentrates were further concentrated by use of the traditional wooden bowl, the rejects being collected for re-circulation on the sluice boxes. The resulting gold concentrates were 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 were kept for further treatment in the future, because it was found that they contained 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 was pressed to remove free mercury, then retorted, providing the sponge gold for smelting to arrive at the final product, the marketable gold ingot.

The amalgamation tailings mentioned above were later tested for further extraction by oxidising them in a oil drum with caustic soda, which was easily obtainable locally. For about one month the tailings were exposed to a strong caustic soda solution and daily stirring of the tailings took place. Then the tailings were dried/roasted, pounded and again amalgamated. About 8 g of gold from estimated 60 kg of amalgamation tailings were obtained but this return did not justify the work involved, especially the quite nasty fumes experienced during drying and the difficult handling of the acidic materials during amalgamation.

2.7 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 carat at the beginning (two times re-smelting) to as low as 16.8 carat. 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.

2.8 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.

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.

3. 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 was 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 was necessary. The side-plates were also rebuilt with flat springsteel.

Similar wear was observed on the linings and at the hammers of the hammermill. Production experience had shown that a new set of hammers was worn after milling of 3 to 4.5 tons of millhead material. The hammers, consisting of 6 cm wide flat spring steel were 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 above 200.000 Cedis (or 140 US\$).

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 or 650,- US\$ 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.

4. 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 wasting 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 .

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