

The gold estimated as hoisted by the individual shafts focuses the attention of management where necessary. The gold produced in the metallurgical plant correlates with the estimated gold hoisted. Previously the system of using stop-belt sampling did not correlate with the gold produced.

Analysis of three months' samples taken indicates that the grade distribution is log-normal (Graph 60-61) and corresponds with that found for example in the continuous sampling exercise.

3.2. VALUATION METHODS.

Several audits were done on the valuation methods used, which revealed that they were acceptable and no changes were required. **It was not the purpose of this thesis to investigate the validity of current valuation systems in use as it is an established science.** The best valuation method will not be able to accurately estimate the grade of the ore in situ if the largest bias is introduced at the sampling stage. It was therefore decided to rather focus efforts on sampling procedures.

3.3. ASSAY METHODS.

The method of assaying the samples was traditionally the fire clay method. The Aztec method was introduced during 1995 but was not fully operational by the time of completing this thesis. It is, however, not anticipated that the values as determined by the new

method will be significantly different from the traditional method. The reason is that the fire clay assay is still the reference method and that the grade determination by the Aztec method is adjusted by means of regression to correspond.

3.4. CONCLUSION.

Skin sampling is a very important component of the in-situ grade estimation of the ore underground. Rudimentary methods are used to obtain samples that are not always truly representative from the panels due to the hardness of the rock and the tools used.

Compressed air-powered angle grinders that cuts the perimeter of the sample to a depth of 2cm were introduced to ensure that the samples taken are more accurately done. The comparison of individual grades obtained from adjacent samples were meaningless, because of the great variance of gold deposition in narrow reef bands.

The methods employed in evaluating the sampled grades were audited and found to be in order. **The focus of grade estimation therefore remains on underground sampling techniques and relative density allocations of the ore.**

A task group was formed in Anglo American Corporation, Gold Division, to explore the previously mentioned issues further.

CHAPTER 4.

REEF TYPES ON WESTERN HOLDINGS MINE.

4.1. BASAL REEF

The Basal reef, in which most of the mining in Freegold takes place is essentially a pebbly, siliceous quartzite with a variable amount of conglomerate, heavy minerals and minor amounts of phyllosilicates in the matrix. Durable pebbles consist of chert, smoky quartz, white quartz and siliceous quartzite. The less durable pebble fraction comprises shale and a small proportion of porphyry and shist fragments. The placer sediment is distinguished from the overlying and underlying sediments by its greater maturity and by the relative high proportions of contained heavy minerals, particularly gold and uraninite. Pyrite is ubiquitous in the reef and is by far the most abundant detrital mineral. After pyrite, the most commonest detrital minerals are chromite, zircon and leucoxene. The important economic minerals, gold and uraninite, are present in relatively minor amounts.

The zone on the whole has a fairly well developed bottom contact consisting of a variable pebble layer with different degrees of hydro-carbon and mineralisation present. Followed by one or more additional upward fining sequences of deposition.

Gold mineralisation is dependant on the presence of adequate micro and meso scale structural alterations that have taken place. These includes, bedding plane fractures, permeability of the original conglomerate, fluid alterations and fresh hydro-carbons. The later act as an catalyst for gold precipitation from the fluids.

The facies model constitutes four different facies namely, the Loraine facies, the Geduld facies, the Steyn facies and the Black Chert facies.

4.1.1. Loraine Facies.

This facies is characterised by erratic gold grades with the main occurrence along the north-eastern fringes of the Freegold in shallow remnant channels. It consists generally of a white to grey siliceous quartzite with a purple tinge.

4.1.2. Geduld facies (lower portion)

It is a oligomictic small pebble conglomerate which has pebbles of approximately 1 cm in diameter. It consists predominantly of Black Chert, white or milky quartz pebbles, subrounded to angular. The pebbles are more abundant at the base of the unit. Limited carbon, mainly of a flyspeck nature occurs on the contact. It is usually well mineralised and may contain up to 40% Pyrite.

4.1.3. Geduld Facies (upper portion)

It consists of a grey to dark grey siliceous gritty quartzite, pyrite foresets and mineralised zones with buckshot pyrite. The top 5cm consists of a coarse sago texture known as "TOP OF BASAL".

4.1.4. Steyn Facies.

It consists of a Basal scour surface, followed by a conglomerate lag. Conglomerate layers, filling depressions and forming longitudinal bars are also present. Multiple scour surfaces, overlain by conglomerate lags are sometimes present, resulting in a lens-like internal geometry. Trough cross-bedding is generally well developed in the quartzite facies. Polimictic clasts are in abundance in the proximal areas giving it the colourful appearance of the immediate footwall below it.

Kerogen seams are seldom preserved, although small fragments of Kerogen may be present in the matrix of the conglomerates. Gold concentrations may occur throughout the vertical profile of the placer, but are generally associated with the conglomerate facies.

In contrast to the proximal facies, the distal facies is essentially a trough cross-bedded quartz arenite and the percentage conglomerate is generally considerable lower. The pebble assemblage is noticeably more oligomictic although the

matrix arenite is still polymictic, which distinguishes Geduld from Steyn placers. Kerogen seams are usually present at the bottom scoured surface of the placer but may be repeated where multiple scour surfaces are present, resulting in thick carbon layers. It is in this instance where the propensity of gold loss, especially with the use of abundant water during the mining operation, is increased.

4.1.5. Black Chert.

The Black Chert facies, hereafter referred to as the BCF, is the laterally most extensive facies which is present over the northern half of Western Holdings Mine No.1, No.4 and No.8 Shafts.

It typically consists of two upwards fining depositional cycles. Gold occurs mainly in a carbon rich scattered lag at the base of the upper cycle. In the BCF dominated area, the distribution of gold is controlled mainly by the presence of the economically important lowermost cycle.

4.2. SAAIPLAAS REEF.

The Saaipplaas reef on No.7 and No.9 Shafts Western Holdings Mine is situated at the base of the Leader quartzites (Harmony formation) and consists of the Lower Western facies horizon and the Upper Pyrite horizon.

It is a complex of siliceous quartzite channel bodies which accumulated from a network of semi-perennial, sandy, braided streams. The litho facies of the Saaiplaas placer consists predominantly of coarse grained, pebbly, siliceous quartzites with subordinate conglomerates and shale drapes.

Conglomerate beds are commonly well mineralised with gold, especially when the matrixes consist mostly of Pyrite. In contrast, the lower parts of horizontally laminated, siliceous quartzites are commonly characterised by numerous pyritic laminae and sporadic kerogen laminae, both of which contain high concentrations of gold and uranium. These sediments are a common and laterally extensive facies within many of the channel placers and therefore, provide a major economic horizon.

4.2.1. Western Facies Horizon.

The Western facies horizon varies between a pebbly quartzite and a loosely packed conglomerate of white and smoky quartz and is 1 to 2,5 metres thick. This facies is a known gold carrier with prominent coarse grained Pyrite, often in the form of buckshot.

4.2.2. Pyrite Horizon.

The Pyrite horizon consists of a lower and upper unit and directly overlies the Western facies horizon.

The lower horizon is generally a trough cross-bedded unit approximately 1,5 metres thick. The Pyrite is in the form of stringers along the crossbeds together with the occasional buckshot. It is also a gold carrier but in certain areas barren. The upper unit is a dark grey horizontally laminated quartzite with Pyrite stringers and buckshot. This is an important gold carrier, especially on the overlap area at Western Holdings Mine. It is overlain by the waxy brown Leader quartzite.

4.3. LEADER REEF PLACER

The Leader reef placer is developed over the entire Welkom Goldfields however, it is not of sufficient interest in the northern part of the Goldfields. The Leader placer unit forms the base of the Dagbreek formation and lies unconformable on the Harmony formation. The upper contact is defined by the change from siliceous quartzite to argillaceous quartzite. Although the external geometry of the placer is generally sheet-like, individual beds are lenticular and pinch out over short distances, in sections transverse to the palaeoflow. Vertical profiles through the placer are composed of heterogeneous assemblages of massive gravel, matrix-supported gravel, gravel lags, planar cross-bedded, trough cross-bedded and horizontally bedded quartzites. These characteristics are interpreted as the product of shallow, braided-stream deposition.

The Leader placer is clearly the coalescence of two different placer deposits. An older, oligomictic, texturally more mature deposit, termed the Alma and a younger polymictic Bedelia placer.

4.3.1. Alma Placer.

The Alma placer is a light grey, pyritic, pebbly, siliceous quartzite which generally has a trace of Kerogen on the bottom contact. The pebbles are composed of vein quartz and chert. Detrital Pyrite on foreset and bottomset stringers are fairly common. The Bedelia placer has eroded into the Alma and channels may be single or multiple. The gold content is practically confined to conglomerate facies and in particular to clasts supported conglomerates.

4.3.2. Bedelia Placer.

The polymictic Bedelia placer contains pebbles composed of quartz, chert, silicified yellow shale and yellow quartz porphyry. Facies changes take place over short distances and within 5 metres. A quartzite bed can grade to a pebbly quartzite to a matrix supported conglomerate. The pebble size in the Bedelia generally decreases from west to east. Trough cross bedding is well developed and measurements indicate an unimodal direction of transport to the east. Foreset planes are usually difficult to identify in the siliceous quartzite, unless fine grained Pyrite is present.

CHAPTER 5.

GOLD LOSS EXPERIMENTS

5.1. INTRODUCTION.

Unaccounted for gold is defined as the difference between the gold called for and the gold accounted for. This calculated gold loss can be described as the **theoretical gold loss** as it includes both the **real gold loss** and an **apparent gold loss** which is brought about by estimation errors (Graph 3). The **real gold loss** can only be determined by underground experimentation. A systematic approach was therefore used to determine the experimentation necessary to determine the **real gold loss** and distinguish between fact and fiction.

Gold loss can essentially take place on surface and underground. Most of the experimentation was conducted underground as this is the area where the major portion of the gold 'loss' takes place. However, limited experimentation was done on surface to ensure that gold loss due to plant efficiency is accounted for. Gold loss due to gold theft is not taken into consideration in this thesis as it is confidential information and, the extent of gold loss due to criminal activity, at this point in time, does not significantly explain the unaccounted gold.

5.2. Gold Loss On Surface.

5.2.1. Residue value.

When the Mine Call Factor (MCF) is at unacceptable levels, several explanations for the poor performance can be forthcoming. One of these would be that the gold is 'lost' in the plant. Although it is known by most persons that the gold loss in plant residues is included in the **accounted for** portion of the MCF equation, pressure is put on the metallurgical plants to explain the gold 'loss'. An experiment was conducted to confirm that the residue values are measured accurately.

The ore mined at Western Holdings Mine is delivered to two different metallurgical plants, namely the Welkom and Western Holdings plants. These plant treats ore from underground sources as well as a lesser amount from surface waste dumps.

The Welkom Plant traditionally treated ore from 1 shaft underground sources and the 1 shaft waste dump, resulting in an acceptable Mine Call Factor (74 percent). Due to a decreasing Mine Call Factor at the Western Holdings plant it was decided to re-route the ore from No's 2 and 3 shafts to the Welkom plant as it was suspected that this ore was the cause of the poor plant performance (Graph 4). The ore was finally re-routed to the Welkom plant on 3 July 1994. Initial calculations showed improved bottom line results with the processing of 1,2 and 3 shafts and the 1 shaft dump ore in the Welkom plant and introducing a higher grade waste from 5 shaft dump into the Holdings plant. The actual result was

extremely disappointing because the Mine Call Factor of the ore delivered to the Welkom plant dropped well below expectations (Graph 4). It was thought at the time that it was *conclusively proved* that in whichever plant the 2/3 shaft ore was processed, the Mine Call Factor decreased.

A suspected contributing factor of the low MCF at that time was the commencement of the 2 shaft, shaft pillar extraction which had a very high grade. The reef contained visible gold, in a flaky form, which was suspected of getting lost in the residue. A hypothesis was that the milling process caused the solid gold particles to deform so that it was not all extracted in the metallurgical process. The residue values leaving the plant increased from 0.13g/t to 0.23g/t due to the higher grade through-put.

The validity of the residue values was questioned as it was speculated that gold is lost to the slimes dam and that it was not detected in the residue value. This was despite the fact that check samples done by the Survey Department of the mine compared to those taken by the plant personnel, proved to be similar. The assistance of Anglo American Research Laboratories (A.A.R.L.) was contracted to evaluate extraction procedures and residue value validation in the plant. (15,26,27,28)

The mineralogy of No's 2/3 shafts' ore was found to be different when compared to other shafts in the same geographical area in the sense that it contained large amounts of sericite, pyrophyllite and carbonaceous

reef. Shale was mined up to a thickness of 5 metres and it was perceived that the shales adversely affect the gold extraction process.

An investigation was conducted to determine the mineralogy of the shales using a PIMA (portable infrared mineral analyser). Sixty-seven samples of the khaki shale were taken throughout the mine for analysis. The highest levels of pyrophyllite occurred in the 2 shaft, shaft pillar area.

The average rock density used was 2 780kg/m³ and several density samples indicated that this value is probably overstated, resulting in the over-estimation of underground ore grades (Table 5.1.). The contents of the table illustrates the potential affect on the gold accounting, if the density of the reef/ore is stated incorrectly. In the case of this example the gold content would be overstated by 13.6 percent.

Assumed reef Specific Gravity (kg/m ³)	Actual reef Specific Gravity (kg/m ³)	Tons mined as % of base	Overall grade Au (g/t)	Overestimation of Au as % of base
2 780	2 780	100	5	0
	2 600	98.4	4.75	6.52
	2 400	96.6	4.47	13.6

Table 5.1. Specific density.

Other elements of the metallurgical process which could affect the leaching process in the plant included cyanide addition and residence time of the ore.

The investigation by A.A.R.L. concluded that the residue values obtained were determined accurately and that the gold was not lost through residues as was suspected. It was however, recommended to increase the residence time of the ore in the milling section so as to obtain a finer grind. This was to ensure a decreased residue value as the gold extraction would increase. Obviously the slowing down of the milling section had a cost aspect attached to it, but a viable slowdown rate was implemented. Diagnostic leaching done throughout the plant indicated that the chemical addition was at an acceptable level and that no changes had to be instituted.

Conclusion.

The experiment resulted in the conclusion that the ore from No. 2 shaft affected the plants' performance historically because the gold was not there in the first instance and that blame could not be apportioned to the plant. The shaft was then destined to be closed down as it was not economically justifiable.

A subsequent experiment was conducted to finally determine to what extent the ore from 2 shaft affects a plant's performance due to it's mineralogy. The ore was diverted to a dedicated milling stream of the President Brand Carbon-in-pulp plant where the pulp value and the go-

belt sampling result at the shaft head were compared. The results are discussed under 5.3.6.

5.3. Underground Gold Loss.

5.3.1. Comparison between the gold loss in a stope with conventional cleaning vs. removing the ore using a waterjet.

5.3.1.1. Introduction.

One of the possible causes of the prevailing 65 percent Mine Call Factor was the then, June 1994, recent introduction of approximately thirty waterjets in the underground workings of the mine. A statistical analysis (linear regression) was done on the then six mines in Freegold using waterjets. The results indicated that the correlation between the number of waterjets used and the Mine Call Factor was statistically insignificant ($R^2=0.14$) (Table 67-74). At Saaiplaas Mine the Mine Call Factor actually improved with the increased use of waterjets. ($R^2= 0.17$, X Coefficient 0.24) (Graph 63-68).

It was hypothesised that the increased gold loss due to the use of waterjets occurred because the gold was deposited into footwall cracks due to the increased pressure of the water. It was also thought that the water usage due to the use of waterjets increased compared to conventional water usage and that this also increased the rate of gold loss.

A stope was therefore selected to conduct an experiment to quantify the 'increased' gold loss due to the use of waterjets. A part of the experiment was to seal the strike gully with a cementitious-based product to limit the areas of potential gold loss. The 'reduction' in gold loss due to the sealing of footwall cracks in a gully was to be determined.

5.3.1.2. Overview of the experiment.

The stope selected for the gold loss experiment was a high grade Basal reef stope situated at Western Holdings Mine 4 shaft (Diagram 1). It had a stoping width of 110cm and an average grade of approximately 25g/t. The gold distribution was restricted to approximately 5mm of the carbonaceous layer in the reef.

Conventional mining was practised in the stope as follows: The supported face of the working place was marked off on a staggered pattern with a burden of 0,6m. Holes were drilled at 70 degrees to strike, at a depth of 0,9m, using percussion machines. The drilled holes were charged with explosives and blasted. The working place was subsequently made safe, watered down and the blasted rock removed from the face into the strike gully, using a scraper. The ore was scraped along the 25m long strike gully into the cross-cut, mechanically loaded into hoppers and transported to the main tips from where it was delivered ultimately to the surface metallurgical plant.

The gold loss in a carbonaceous basal reef stope was determined on a daily basis over an eighty (80) shift period. The reconciliation of the gold from the face to the tipping point took place under varying conditions; *conventional* cleaning using scrapers, introducing a *waterjet* to assist with scraper cleaning and the *sealing* of footwall cracks in the strike gully.

5.3.1.3. Reconciliation method.

A Grade Officer was assigned to reconcile the gold from the blasted face to the final tipping of the ore into the hopper during the eighty shift period.

This procedure was repeated on a daily basis, giving account of the location of the tonnage and the gold content thereof.

In-situ skin sampling of the face was done six times during the experiment. The sampling method on the face entailed chipping out samples, 7cm high, 2cm below the reef contact and 5cm above, 10cm wide, and at a depth of 2cm and at 5m intervals along the length of the face. This grade was then used to determine the **gold called for** from the stope.

The grade (g/t) of the blasted ore remaining in the face, advanced scraping gully, cross cut and hoppers was determined by taking a representative number of grab samples from the pile. Although grab samples is not a very popular method for determining grade of the ore,

it is nevertheless still used in industry for diagnostic purposes. The objective of using grab samp^l during this experiment was not to determine absolute values but, rather to determine trends for comparative purposes.

5.3.1.4. Experimental procedure.

The experiment was conducted in four phases.

Phase one consisted of the accounting of the gold from the face of the stope to the hopper, while conventional mining, as described previously, were conducted. The results were reconciled to determine the real gold loss in the working place for comparative purposes.

The reconciliation of the gold was done over seventeen production shifts. The gold content of the ore blasted was estimated by measuring the actual face advance, face length and stoping width. The tonnage was calculated using a density of 2 790kg/m³. The face grade, as per Grade Officer's report was used to estimate the amount of **gold called for**.

The tonnage in the face, behind the barricade, advanced scraping gully and cross-cut was determined by using the volume of the ore remaining in the stope. A representative number of grab samples was taken from the aforementioned sources to determine the grade of the ore in transit. The gold content of the ore loaded into the hoppers was estimated, using a hopper tonnage factor and grab samples to determine the grade.

The tonnage was then reconciled to determine the gold **accounted for**. The difference between the gold **called for** and the gold **accounted for** was taken as the **apparent gold loss**. The **apparent gold loss** amounted to 27.7 percent of the gold broken. (Graph 94 & 95., Table 61 -66)

Phase two included the reconciliation of the gold from the face to the tip in the stope, using conventional mining methods, but with the addition of the use of a waterjet to assist with the removal of the ore from the face to the gully. The waterjet had a rated capacity of 37kW and the water was released at a pressure of 7MPa.

Initially problems were experienced in getting the waterjet operational. The waterjetting phase was monitored for thirteen shifts. The progressive **apparent gold loss** amounted to 25.2 percent of the gold broken since the first shift. It was concluded that there was a 2 percent net gain of gold if compared to phase 1 of the experiment. It was therefore deduced that the rate of gold loss was not significantly different when compared to phase one. This meant that there was not an increase in gold loss with the use of waterjets, provided that the run-off water is not 'lost'.

Phase three included the removal of all the dirt from the strike gully footwall by means of a vacuum cleaner and sealing it subsequently with a cementitious-based product to prevent the downward migration of smaller particles of gold into the cracks. Thereafter conventional mining with the use of a waterjet was practised. The gold was accounted for from the face to the hoppers and the real gold loss was determined.

Preamble.

A number of stopes throughout AAC were visited by personnel from the Anglo American Research Laboratories (AARL) in the early 1970's to evaluate the loss of gold into footwall cracks (16-22). Several samples were chiselled from the footwall while the cracks were mapped and analysed. It was found that some gold was lost and that most of it was trapped in the zone 60-65 cm below the footwall. More recent investigations on the mine indicated that the quantity of gold trapped in the cracks cannot be viably recovered. It was therefore decided to seal the footwall of a gully to prevent the gold from being lost in the first instance.

The sealant had to be of an extremely fine texture so as to counteract the migration of minute gold particles into footwall cracks due to gravitational and other forces. AARL was consulted to determine the size distribution of the gold particulate. A cementitious-based product was used to ensure the best possible seal.

Due to the cost of the product it was decided to restrict the sealing of the cracks to the strike gully only. The ore would continuously be moved over this scraper path and would therefore be the best result indicator. The strike gully measured approximately 15m in length at that stage.

The gully area to be sealed was initially waterjetted and then vacuum cleaned during an off-weekend. It was surprising to note that the fines

vacuumed did not have the high grade that was anticipated. In this case the grade amounted to 11.9g/t which was about fifty percent of the prevailing face value. (It is generally believed that the values at the bottom of gullies are significantly higher than that sampled on the face, due to the placement theory of solids.)

The cementitious product was mixed in the cross-cut and pumped to the gully. Here the product was applied to the footwall as a coating. Special care was taken to seal the footwall cracks that were apparent. The product was allowed to dry and mining proceeded as before.

Results.

The experiment was conducted over a thirteen shift period. The productivity increased with the use of the waterjet, as was the case during phase two of the project. The rate of gold loss increased during this period and progressively amounted to 39 percent since the start of the project. However, with reference to graph 94, it appears that that the gold blasted could be over-stated. If, the **gold blasted** line is adjusted, it can be deduced that the rate of **gold blasted** and **accounted for**, are following similar trends as was found during the previous phases of the experiment.

The scraper did remove some of the coating on the footwall due to abrasion. The cracks, however, remained sealed. Further sealing was done during the ensuing weekend with equally good results.

Although the product applied seems to perform the function it was intended for (sealing of footwall cracks) it appears not to have any measurable influence on the gold loss.

Phase four included conventional mining, as described previously without the use of a waterjet and the gully still sealed.

The working place was monitored over a thirty-seven day period, but for comparative purposes, only the results of the first fifteen shifts were used to determine the gold loss. The progressive gold loss amounted to thirty-seven percent. At this stage the **gold called for** amounted to 22.8kg and the **gold accounted for** to 14.5kg, resulting in a **calculated gold loss** of 8.3kg. However, this **gold loss** could not be found inside the stope. Even if the grade was doubled to 50g/t, the unaccounted tonnage would equate to approximately 160tons.

The panel grade increased to 37.3g/t during shift 33 (Table 61), which was approximately fifty percent higher than that determined before and after this instance. If, this grade is normalised (Graph 94), it can be concluded that the gold loss appears to be similar if compared to the previous phases of the experiment.

Gold content of the mud.

The grade of the mud was determined in the cross-cut by sampling it at 15m intervals. The water handling system in the cross-cut was poorly designed, resulting in the water flowing in sub standard drains,

depositing the mud on the footwall due to the decreasing velocity of the water. In this instance it was hypothesised that the carbonaceous nature of the reef, combined with the increased use of water, would liberate gold, and the deposition thereof in the cross-cut will be according to the placement theory. It was suggested that the gold will be concentrated in the mud closest to the source and decrease proportionally with the distance travelled and the decreasing speed of the water.

Description	Grade (g/t)
Average stoping grade	25.0
Mud at slope exit	10.1
Slope exit + 30m along cross-cut	24.9
Slope exit + 40m along cross-cut	15.5
Slope exit + 50m along cross-cut	11.6
Slope exit + 70m along cross-cut	5.1
Slope exit + 90m along cross-cut	18.8
Slope exit + 100m along cross-cut	13.7
Slope exit + 120m along cross-cut	6.5
Slope exit + 130m along cross-cut	15.9

Table 5.2. Grade distribution of mud.

The average grade of the working place was 25g/t and that of the mud at 13.6g/t which equated to approximately 50 percent of that prevailing at the face. (Several other samples were taken underground which confirmed

that mud had approximately the same grade as that prevailing in the surrounding panels. It is generally assumed that mud has extremely high grades but, unfortunately, this appears not to be the case.) There were 32 tons of mud in the cross-cut which amounts to 436g of gold.

Gold liberated as fine particles during blasting.

Gravimetric dust sampling pumps were used during blasting to determine the gold content of the dust during the blast. The samples were analysed and the resultant grade amounted only to a trace. It was therefore concluded that an insignificant amount of gold is conveyed with the dust during blasting operations. Blasting barricades were used in the stope to prevent the blasted rock from being thrown into previously cleaned areas.

Samples of dust in return airways were also taken to determine the gold content. It was found that the content was insignificant and the volume of dust did not warrant further investigations. Examples of the grade is as follows: 4.7g/t, 9.7g/t, 1.6g/t, 2.9g/t, 1.8g/t, 3.5g/t, 2.9g/t, 8.9g/t and traces.

5.3.1.5. General.

It was decided at a very late stage of the experiment to introduce a vacuum cleaner to vacuum the fines in the areas that were previously swept and passed. Only certain areas were vacuumed at random due to time

constraints. Unfortunately no significant volumes of fines nor high grades were discovered.

5.3.1.6. Conclusion.

The **theoretical gold loss** in this experiment amounted to 37 percent of the gold mined in the stope. The **real gold loss** is significantly less because there was no major lock-up of ore remaining in the stope. Selected areas that were previously passed as swept were vacuum cleaned to determine the gold left behind. The volumes of gold were insignificant and do not explain the **theoretical gold loss**. The only problem remaining can therefore be the gold estimation procedure and in particular the sampling method. Experiments were conducted in this regard and is discussed in Chapter 3.

The gold loss in this stope was directly related to the gold called for in the stope. A linear regression analysis done between the **gold called for** in the stope and the **gold accounted for** in the stope confirmed that the correlation coefficient (R^2) amounted to 0.95. The rate of gold loss was very similar between the different phases of the experiment. It is, therefore, concluded that:

- i. The use of a waterjet in this particular stope did not increase the rate of gold loss.
- ii. The sealing of footwall cracks at this stage does not seem to be a viable proposition, because it has not been conclusively proved that a significant amount of gold is lost in the cracks.

- iii. The rate of production increased with the introduction of the waterjet.
- iv. Attention must be given to the complete water handling circuit in the case of waterjets, not just to get the water from the stope to the cross-cut.

5.3.2. Quantifying Stope Service Water Usage Underground.

5.3.2.1. Objective of the experiment.

To determine the underground in-stope usage of service water for different activities.

5.3.2.2. Background.

The excessive use of water in the mining process underground has a negative influence on the gold recovered. The consumption of water on Western Holdings was quoted as up to 4 ton of water per ton of rock broken. The industry norm is set as 1 ton per ton. This experiment was conducted to quantify the water consumption for different activities in the mining cycle of a stope.

A site was selected where conventional mining and waterjet cleaning was done. The site selected was at Western Holdings Mine No.1 shaft 55b-30 stope. It was planned to conduct the experiment over a 1 month period, but was terminated after 2 weeks because the instrumentation was wilfully damaged.

5.3.2.3. Experimental procedure.

A magflow water flow meter was installed in the service water feed pipe of 1/55b-30 stope. The water flow meter (4 to 20mV output) was hardwired to relay the information to the existing fire detection computer on surface. The information was recorded and real time graphs reflecting water consumption continuously were available. The information was subsequently analysed to determine the water usage for different mining periods such as day shift, night shift and off shift (leakage) (Graph 70-72).

The volume of rock removed during July 1994 was measured by the Survey Department (8 429 tons).

5.3.2.4. Results.

The results of the experiment were analysed to determine the following (Table 75):

- i. The volume of service water used and rock produce expressed as a ratio of ton of water per ton of rock ($\frac{tw}{tr}$). OVERALL RATIO 1.19 ton per ton.
- ii. The $\frac{tw}{tr}$ ratio determined for day shift and night shift. DAY SHIFT = 0.62 ton per ton and NIGHT SHIFT = 0.43 ton per ton.
- iii. The $\frac{tw}{tr}$ ratio for waterjet cleaning deduced to be the same as for night shift = 0.43 ton per ton.

- iv. The tw/tr ratio for drilling deduced to be the same as for day shift = 0.62 ton per ton.

5.3.3. Mud Loading And Old Areas.

5.3.3.1. Introduction.

Management has a number of options available to influence the Mine Call Factor. One method of ensuring a better than normal Mine Call Factor, is to ensure that tonnage from sources other than that called for is added to the plant throughput (Graph 2). This option is called positive tonnage discrepancy and will ensure an improved Mine Call Factor if the tonnage is significant in relation to the hoisted tonnage called for. However, the tonnage must come from somewhere. It could essentially only come from two areas, namely ore that was mined and subsequently remained underground on a previous occasion and mud left underground.

There is a limit to the amount of material that can reach the plant in this fashion and it must cease at some point in time. At the moment the additional tonnage that can be added to the ore mined, without having to be called for, is assisting survival (refer to graph 2, April 1990 to October 1994). In real terms the Mine Call Factor remains as unacceptable as the gold called for and that accounted for does not come from the same sources. The additional tonnage, although beneficial, contaminates the MCF as such. In some cases the gold from the old areas and mud could be called for as an internal arrangement, as gold could be

split between different units on a hoisted tonnage/grade relationship determined using a surface go-belt arrangement.

There is a serious drive on to install barricades in the current working areas, so as to prevent previously swept areas from being decontaminated. This is aimed at preventing the re-creation of areas where the ore is mined and subsequently left underground.

Mud in cross-cuts and haulages is generally seen as a potential lock-up of gold. One of the reasons why this viewpoint is strongly supported is that the mud clean up teams underground were reduced to insignificance due to labour cut-backs in recent years. Obviously mud creation must be reduced at its source and this will ultimately lead to less mud being accumulated. However, severe cost cut-backs, at times, inhibit proper designs, resulting in the accumulation of mud in haulages and cross-cuts.

Western Holdings Mine No.2 shaft was a site where several thousand tons of mud was accumulated in cross-cuts and haulages. During 1994 it was decided to employ outside contracting firms to assist with the clean-up of these areas. It is imperative from a gold production viewpoint that significant volumes of mud be loaded and treated for it to make any significant difference to the gold accounted for. Mining legend has it that gold is enriched in the mud and that small volumes of mud into the plant will make noticeable differences in additional gold recovery. It was found after sampling approximately 10 000m of underground footwall excavations at 2 shaft Western Holdings Mine that the grade of gold in

the mud was significantly lower than was expected (Graph 24). The grade of the mud was in most cases lower or equal to the grade of the ore mined in the surrounding areas. The average grade of the mud was 5.6g/t with the lowest being nil and the highest at 60g/t. It can be seen on graph 24 and 25 that the gold grade approximates a log-normal distribution which is characteristic of that of ore in-situ. This finding was confirmed by St Helena mine who embarked on similar investigations.

In the case of No.2 shaft, the hoisted tonnage from all sources equates to approximately 30 000 tons per month, and in this particular instance it was required to add at least 4 000 tons of mud. The gold in the mud is not called for when it comes to the Mine Call Factor, as it has theoretically been called for previously. (It was called previously as gold from stopes which did not manifest itself in the plant and appeared as gold loss.) It therefore acts as a sweetener for the gold called for and the tonnage is reflected in the tonnage discrepancy as it is ultimately hoisted or pumped to the plant.

5.3.3.2. Grade distribution of mud.

One hundred and fifty six mud samples were taken over a distance of 10.2km over 4 levels at Western Holdings No.2 shaft (Graph 24). These levels were 40, 41, 42 and 43 levels. The samples were taken approximately 30m apart where the mud was spread fairly consistently. In some instances the distance between samples varied up to 280m apart. The distribution of the grade was skewed to the left as can be seen on the graph. The average grade was 5.69g/t. The average depth of the mud was

0.42m (range of 0.1 to 2m) and amounted to 21 570tons at a density of 1 700kg/m³. The available gold amounted to approximately 123kg. Although this is a significant amount it confirms that the gold is not concentrated in the mud and that it certainly only amounts to a small portion of the **real gold loss**. It, however, remains a real gold loss if not finally recovered in the plant.

It can be seen from the aforementioned that a fair tonnage of mud is required to make a noticeable difference in the plant. The shaft in question had the following gold called for in October 1995:

Source	Tonnage	Grade (g/t)	Au (kg)	MCF (%) (RF=95%)	Gold Prod. (kg)
Stoping	25 137	9.98	250.926	67.0	159.67
Old areas	2 732	5.59	15.272		14.5
Total	27 869	9.55	266.198	69.4	174.17

(RF = Recovery factor in the plant)

Table 5.3. Old area tonnage.

When the current Mine Call Factor (67 percent) and recovery factor of 0.95 is applied, the gold accounted for amounts to; $266.198 \times 0.67 \times 0.95 = 169.43\text{kg}$. Currently the mud is loaded at a rate of 4 000t/month on this particular shaft. The maximum gold that can be extracted from the mud loading operation amounts to $4\ 000 \times 5.59 \times 0.95 = 21.62\text{kg}$. Note that the positive tonnage discrepancy will now amount to $4\ 000 / 27\ 869 = 14.3$

percent which is an admirable figure. Obviously the gold from this mud is not **called for** as previously stated. The net result of the mud loading should be seen in the plant as gold accounted for. It can be calculated that this additional effort in follows: 169.43kg **Au Accounted for** + 21.62kg *Au from mud* = 191.05kg **Au accounted for**. The **gold called for** remains at 266.198kg **Au** resulting in an increase of the Mine Call Factor from 67 percent to 71.8 percent.

The additional gold bearing tonnage not **called for** improves the Mine Call Factor, but it certainly does not solve the shortfall in the Mine Call Factor by far. In this particular case the Mine Call Factor improves by 5 percent, yet where is the remaining gold? The difference between 100 percent and 71.8 percent is still unaccounted for.

The average grade of the mud at No.2 shaft was found to be 5.69g/t while the average grade mined is 9.98g/t. This disproves the hypothesis that gold is generally concentrated in mud. The grade of gold locked up in mud generally is lower than that of the surrounding working places and significant volumes of mud loading are necessary to make a noticeable difference in the gold accounted for.

Conclusion.

This also indicates that the enrichment of gold in mud remains a fallacy. The mud would generally be lower than that in the surrounding working places.

5.3.4. Determination Of Gold Loss In A Stope Mined In Basal Reef, Geduld Facies.

5.3.4.1. Introduction.

The objective of the experiment was to determine the gold loss from that estimated to be in-situ to that finally accounted for. It was anticipated that the gold loss will be less in the Geduld facies compared to the gold loss in the Steyn facies.

The stope selected for the gold loss experiment was a high grade basal reef stope in the Geduld facies, situated at Western Holdings Mine 9 shaft in the 45 level shaft pillar area. The gold distribution was contained within the big pebble conglomerate. It was mined at a stoping width of 151cm and an average grade of approximately 46.2g/t.

The blasted rock was scraped from the face into a 10m long strike gully to an ore pass using a conventional scraper. The ore was subsequently tipped into hoppers and transported to the main tips.

5.3.4.2. Gold loss experimental procedure.

Reconciliation method.

The working place was monitored over a twenty (20) shift period. A Grade Officer was assigned to reconcile the gold from the blasted face to the final tipping of the ore into the hopper.

This procedure was repeated on a daily basis, giving account of the location of the tonnage and the gold content thereof. In-situ sampling of the face was done five times during the experimental period. This grade was then used to determine the **gold called for** from the stope.

The value of the blasted ore remaining in the face, advanced scraping gully, ore pass and hoppers was determined by taking a representative number of grab samples from the pile. Grab samples has many drawbacks but absolute values was not the objective of the experiment as the results were used for comparative purposes only.

5.3.4.3. Results of the gold loss experiment.

The gold content of the ore blasted was estimated by measuring the actual face advance, face length and stoping width. The tonnage was calculated using a density of $2\ 780\text{kg/m}^3$. The face grade as per Grade Officer's report was used to estimate the amount of gold **called for**.

The tonnage in the face behind the barricade, advanced scraping gully and cross-cut was determined by using the volume of the ore remaining in the stope. A representative number of grab samples was taken from the aforementioned sources to determine the grade of the ore in transit. The gold content of the ore tipped into the hoppers was estimated, using a hopper tonnage factor and grab samples to determine the grade.

The tonnage was then reconciled to determine the gold **accounted for**. The difference between the gold **called for** and the gold **accounted for** was taken as the **theoretical gold loss**.

5.3.4.4. Discussion of the gold loss experiment.

The gold loss determined during the experiment would be overestimated if the gold called for from the stope was determined using the valuated grade as determined by the survey department.

Description	Grade (g/t)	Gold called for (g)	Gold accounted for (g)	'Apparent' Gold loss / gain (g)	Stope Call Factor (%)
Experiment	46.16	8012	6862	-708	85.6
Survey Dept	33.85	5875	6862	987	116.8
Difference	-12.3	-2137	0	-1695	31.2

Table 5.4. Grade difference.

The **theoretical gold loss** in this case amounts to 14.4 percent. The results of an experiment previously done in the Steyn facies indicated that the corresponding loss amounted to 28.5 percent. This difference is in line with the hypothesis that the gold loss would be less in the Geduld facies than the Steyn facies, primarily due to the gold distribution characteristic. The difference is ascribed to the fact that the gold in the Steyn facies is mostly contained in a 5mm layer of

carbon, whereas the gold is distributed more broadly in the Geduld facies which is a big pebble conglomerate.

5.3.4.5. Conclusion.

- i. It is concluded that the **theoretical gold loss** in the Geduld facies (14.4 percent) is less when compared to that in the Steyn facies (28.5 percent) under similar circumstances.
- ii. The use of the compressed air driven angle grinder cutting a 2cm outer perimeter slot of in situ samples shows a lot of promise, both in accuracy and productivity.
- iii. The gold distribution in the panel varies significantly, e.g. from 13.69g/t, 23.43g/t, 16.12g/t, 11g/t and 78.81g/t. (Table 86)
- iv. The valuated grade was 26.6 percent lower than the grade determined by frequent sampling in the panel.

5.3.5. Dry sweepings.

5.3.5.1. Introduction.

It is standard practice on some mines that sweepings to be done in the carbonaceous reef type stopes should be performed with the use of a limited amount of water and hand-held tools. The reason for conducting dry sweepings is that it is generally believed that the use of excessive amounts of water will increase the potential for gold loss in the footwall cracks. However, experimentation conducted has indicated that an insignificant amount of gold remains in the footwall cracks after

being swept with the aid of water. In addition, an experiment conducted in an area where dry sweepings were performed indicated that the work completed was of a substandard condition. In this particular instance it would have been advantageous if the area was washed over with water as the fines accumulated on the footwall would have been cleaned out.

Dry sweepings include the use of wire brushes and shovels to sweep the fines, as a final clean-up, from the worked out areas into the scraper path. No water, other than that for dust allaying, is used to wash over the swept areas. Labour efficiency in the case where dry sweepings are conducted is less than when the use of a reasonable amount of water is allowed.

It is conceivable that unless sweepings are done properly, more fines can be left remaining in the worked-out areas than when water is used to do the final clean-up. It is recommended that dry sweepings are kept to a minimum on a mine as the general belief that gold is enriched in fines has not been proven to date. Sweepings is merely an indicator that the majority of the blasted ore was removed from the worked-out areas. It was previously stated in Chapter 3 that there is no statistical significant relationship between the Mine Call Factor and sweepings.

Author De Jager E J

Name of thesis The analysis of the mine call factor in gold mining with specific reference to Western Holdings Mine 1997

PUBLISHER:

University of the Witwatersrand, Johannesburg

©2013

LEGAL NOTICES:

Copyright Notice: All materials on the University of the Witwatersrand, Johannesburg Library website are protected by South African copyright law and may not be distributed, transmitted, displayed, or otherwise published in any format, without the prior written permission of the copyright owner.

Disclaimer and Terms of Use: Provided that you maintain all copyright and other notices contained therein, you may download material (one machine readable copy and one print copy per page) for your personal and/or educational non-commercial use only.

The University of the Witwatersrand, Johannesburg, is not responsible for any errors or omissions and excludes any and all liability for any errors in or omissions from the information on the Library website.