

known fact that the workers are reluctant to install these barricades on a routine basis, resulting in the recontamination of previously-cleaned areas. Management action is continuously required to instil the principle of installing barricades.

Gold lock up.

The underground lock up of gold bearing ore in a conventional mine amounts to approximately 8.6 percent. In a pillar mine this figure can increase to approximately 32.5 percent. The infrastructural layout of pillars will therefore dictate the ore lock-up capacity.

Description	30m face length	15m face length
East siding	5m	5m
Stoping width	1m	1m
Tonnage blasted ($0.8\text{m} \times 2.78 \text{ t/m}^3 \times$ face length)	66.7t	33.3t
Gully length	30m	30m
Gully volume ($1.8\text{m} \times 2\text{m} \times \text{length}$)	108m^3	108m^3
Broken rock in gully	64.8m^3	64.8m^3
Face advance per month	9m	6m
Monthly production	750t	250t
Lock-up %	8.6%	32.5%

Table 1.2. Gold lock-up comparison, conventional vs. pillar mining.

The consequence of this lock-up is that the rate of ore removal is delayed in pillar mining when compared to concentrated mining. The lock-up will have a negative influence on the MCF as the tonnage called for cannot be hoisted timeously.

Where reef is left underground it directly affects the gold accounted for. Examples of this also include where reef is left in the footwall or hanging wall either due to mining constraints or pure negligence. In this case the gold will be **called for** if the surveyor is not aware of the incident. However, where the responsible person is aware of reef left in situ it can still be recovered economically subsequently. This gold lock-up amounts to approximately 30kg/month on Western Holdings Mine. It is determined by the survey department on a monthly basis and sweepings are not paid unless there is no gold remaining in the worked out area.

In the instances where reef remains in the worked-out areas without the surveyors' knowledge, it will probably be lost forever. It is known that gold remained as reef in the hanging in certain areas where undercut mining was practised in previous years. This gold 'loss' is part of the historically **unaccounted gold** and is coined as 'dirty mining'.

Tonnage discrepancy is the difference in tonnage received by the plant to that calculated as available to be delivered by the surveyor. It is known as a shortfall if the surveyor's calculated tonnage is more than that accounted for in the plant' and as excess vice versa. It is also known as a negative and positive tonnage discrepancy respectively.

Tonnage discrepancy can be augmented by the removal of ore remaining in worked-out areas due to dirty mining done previously. The gold from this operation is not **called for** and therefore understates the **theoretical gold loss** from current mining operations. Tonnage from old areas is used as a survival strategy by some mines in the prevailing economic situation. Some of the shafts on Western Holdings Mine has been surviving on this strategy for some years. Unfortunately this tonnage will be depleted at some point in time. The affect on the Mine Call Factor can clearly be seen on graph 2 as tonnage from old areas has been reduced since October 1994.

Waste rock to reef.

In most cases the waste rock mined underground is tipped into the reef passes and is also treated in the metallurgical plant. The downside of this arrangement is that the gold in residue increases due to the additional tonnage to be milled. It, however, rules out the possibility that reef is tipped into the waste passes resulting in a definite waste of expensive efforts.

Reef to waste.

In the cases where reef is tipped as waste, it neutralises the mining effort as it is finally tipped directly onto the waste dump. It is possible that this gold can ultimately be recovered when the waste dump is treated in the metallurgical plant. However, it is not good business

by any means if this loss is allowed by the management of a mine. The reef contamination should be terminated at the source.

Other potential areas of gold loss.

In some circles it is suggested that an appreciable amount of gold is lost during the blasting operation. An experiment was conducted to validate this concept but no evidence to back up this suspicion has been found to date. Gold was found in insignificant volumes in dust which gathered in airways underground. It was found that it was not viable to recover this gold. It will be discussed to some extent in Chapter 5.

1.3. Surface Control.

On surface, the following aspects can play a role in determining the Mine Call Factor.

1.3.1. Ore accounting.

The method of accounting can have a significant influence the Mine Call Factor as indicated in the following section.

1.3.1.1. Grade allocation.

The valuation of the gold present in the ore mined has a big influence on the accuracy of the calculated gold loss in a mine. The method of sampling, interpretation and subsequent valuation is questioned. There

are at least four areas of bias in the method of determining the values of the gold. These are the physical sampling method, the splitting of the pulverised sample, the assay bias and finally the valuation bias.

The **sampling bias** is introduced by the position of the sample taken, the actual chipping of the sample and the collection of the fines of the sample. The hardness of the rock plays a role in the accuracy of the samples taken. These aspects were experimented with and recommendations made.

The influence of stoping width on the important g/t concept affects the decisions on grade control and selective mining.

The preparation of samples for **assaying** should be done separately for high and low grade samples. This is to avoid the over-estimation of the lower grade samples. Aztec assay systems were introduced during 1995 in Freegold to augment fire clay assay. This aspect is not further explored in this thesis as there is a linear relationship between the two methods mentioned.

Several experiments have been conducted to evaluate the effectiveness of the awarding of grade values on the reef plane. The results of the aforementioned will be discussed in Chapter 3.

The Mine Call Factor may react to fluctuations in grade of the ore mined, especially if the change takes place shortly after the current sampling. It is for this reason that graphs of the MCF, for averages

over a 3 or 6 month period etc. are kept. The fluctuations balance out over a period of time.

It is only at the lower grade mines of the Freegold stable, such as Freddie's and Saaiplaas, where there is a definite correlation between the stoping grade and the Mine Call Factor (Graph 16, 35, 39, 43, Table 1 & 51). This is due to the fact that the reef mined is more homogeneous than on the other mines in the Freegold stable. There is no statistically significant relationship (Table 11, 21, 31, 41) between the previously-mentioned parameters on the other mines and the graphs does not indicate any significant trends (Graph 16, 35, 39, 43). This is mostly due to the fact that some carbonaceous reef and relatively higher grade ore is being mined.

1.3.1.2. Tonnage allocation

Tonnage allocation due to various tramming methods in use can be distorted. Tonnage factors are being allocated on an ad hoc basis to the various containers in which ore is being transported. The tonnage will be valid provided that the correct volumes are being adhered to. However, this is not always the case. Recently a 20 000 ton discrepancy out of 320 000 tons hoisted per month was recorded on Western Holdings Mine between that tonnage that was claimed as delivered to the plant and that actually received by the plant. It was concluded then that these tons amounted to no more than "fresh air" and were as such reallocated proportionally to the different sources of ore. The obvious method to resolve such a problem is to have proper audit systems in place.

When incorrect assumptions are used, emphasis can be placed in the wrong areas. This can be ill afforded as each business unit has to make a profit to ensure survival in the short and longer term. Improved methods for tonnage accounting in the form of shaft-head belt weightometers were installed on the mine to counteract this problem.

Belt weightometers.

Belt weightometers with go-belt samplers were installed at shaft heads during July 1995. These have now been operational for three months (Jan. 96) and some teething problems are still being experienced, but the tonnage, and ultimately the gold apportionment, will be done using this equipment. In the past the tonnage hoisted in most cases was accounted for by the number of skips hoisted and the allocated skip factor. The skip factor in turn was allocated by weighing skips filled with ore. The procedure of confirming the skip factor would be repeated on a monthly basis.

The allocation of underground tonnage remains a contentious issue as the sender invariably sends more tons than the receiver receives. The tonnage is mostly calculated using a hopper factor and the number of hoppers trammed.

1.3.1.3. Relative density of the ore.

The relative density of the rock mass is generally applied as 2 780kg/m³. This density figure is used in the calculation of tonnage from

all sources underground. The gold called for from underground sources is also determined using this figure. Experiments were conducted to confirm the factors used⁽³¹⁾

A collection of 129 rock samples from President Brand Mine, from all applicable lithologies was tested at Anglo American Research Laboratories for density determination. The results were as follows:

Lithology	No of samples	Mean t/m ³	Std Deviation
Footwall (UF1)	11	2.655	0.007
Basal conglomerate	16	2.754	0.134
Basal reef quartzite	12	2.663	0.023
Sandbar quartzite	4	2.663	0.033
Laminated quartzite	8	2.719	0.107
Khaki shale	12	2.812	0.056
Waxy brown L quartzite	9	2.667	0.039
Leader reef quartzite	10	2.663	0.015
Bedelia conglomerate	16	2.728	0.057
Bedelia hanging wall	5	2.698	0.049
'A' footwall conglomerate	5	2.684	0.009
'A' footwall quartzite	5	2.684	0.015
Witpan reef (lower)	5	2.804	0.053
Uitsig reef (upper)	5	2.896	0.199
'A' hanging wall quartzite	5	2.660	0.012
Middle reef quartzite	1	2.660	0

Table 1.3. Specific density results.

The recalculated density of the reef at President Brand Mine amounted to 2 712kg/m³. St Helena Mine uses 2 700kg/m³ and Harmony Mine uses 2 717kg/m³. These mines are contiguous to that of Freegold. When the new density of 2 712kg/m³ is used it results in a decrease of 2.5% which affects the tonnage calculation proportionally.

The gold estimated during the sampling process can be overcalled due to the use of an overstated specific density. According to Harrison⁽³¹⁾, if samples are taken in the carbon-rich Steyn facies (specific gravity of 2 660kg/m³), the gold will be over-estimated by 4.3%. This will vary for the different reef types and is part of the cause of **apparent gold loss**.

This error is biased towards higher grade reef as the portion of the rock in which the gold is located ,especially carbonaceous reefs, is normally at a lower density to the rock that will ultimately be mined. The lower grade reefs normally has the gold distributed throughout a bigger portion of the rock. Therefor the extrapolation of the gold content of the sampled portion to the rock that will ultimately be mined, will be more accurate as the densities will be similar in this instance.

1.3.1.4. Transit time of rock from underground to the surface.

Traditionally it is assumed that it would take a volume of ore broken approximately 14 days before it would manifest itself as gold at the plant. It is for this reason that the production month-end and the milling month-end is 14 days apart so as to attempt to reconcile the

gold broken/accounted for. This figure is acceptable during the full production period of a mine. The lag, as was previously mentioned, increases when a mine is in the orderly closing down phase of its life. As was previously stated, the tonnage lock-up capacity underground in a pillar mine is at least three times more than in a conventional mine. The tonnage lock-up capacity of a mine is a function of the average panel length of the underground stopes. It is therefore obvious that the tonnage produced can take longer to be transported in a pillar than in the case of conventional mining.

1.3.1.5. Gold produced allocated.

The tonnage used for calculating the apportionment of gold produced can either be tonnes broken underground or tonnes delivered to the plant, depending on the requirements of the concern.

Gold broken.

The tonnage required from the different mining activities is calculated by the Survey department. It includes the square metres mined, as measured by the surveyor, as well as the stope widths. Tonnage from reef development, as well as tonnage from previously worked areas where ore remained underground, is included as tonnage available for milling. Estimates of falls of ground reported are taken into consideration in this instance.

Gold hoisted.

The tonnage hoisted from the underground section of the mine was not normally taken into consideration in gold produced allocation. It is, however, a fact that the tonnage hoisted must be in excess of that calculated as broken to ensure that no ore remains locked up underground. In the case where the ground was broken and not all removed from underground, gold can be apportioned unfairly because it is assumed that all the gold broken was hoisted to surface. With the advent of surface, shaft head based go-belt sampling and weightometers, cognisance is taken of the gold hoisted and subsequent allocation of the gold produced. This has been the case at Vaal Reefs, and Freegold is scheduled to allocate gold on this basis as from April 1996.

Gold hoisted will be calculated using the tonnage hoisted as per weightometer and the mean grade as determined using the go-belt system. This method will ensure that gold is apportioned correctly to the individual shafts. Emphasis will therefor be placed on the shafts that does not hoist it's gold in an effort to improve the Mine Call Factor.

1.4. Conclusion.

Factors affecting the Mine Call Factor were examined in detail and, where possible, full scale experiments were conducted to prove/disprove certain perceptions. The validity of the accounting method was explored and recommendations made to improve the Mine Call Factor.

The purpose of this thesis is to address the Mine Call Factor holistically so that the difference between **real gold loss** and **apparent gold loss** is understood so that the **apparent gold loss** can ultimately be minimised.

CHAPTER 2.

STATISTICAL ANALYSIS.

2.1. Introduction.

Classical statistical analysis in the form of multiple variable regression was used to determine the relationship of various influences on the Mine Call Factor (MCF). The results of this analysis was used to direct experimental efforts for this thesis. Information pertaining to the period 1992 to 1995 from all five mines in the Freegold stable which includes Western Holdings, Freddie's, President Brand, President Steyn and Saaiplaas were analysed (Table 1-60).

It is accepted in Freegold, that the gold broken underground will remain in transit for approximately 14 days before it manifests itself as gold produced in the metallurgical plant. The period that is known as the **milling month** follows the **measuring month** normally by 14 days. It is for this reason that the statistical analysis has been carried without off-setting the variables from one another by longer periods.

It is reasonable to assume that in the case of pillar mining, the transient time, between the production of ore and delivery to the plant, extends beyond 14 days. The ore lock-up capacity of pillars are dictated by the underground in-stope infrastructure. (Table 1.1).

Inconclusive tests were conducted on Western Holdings Mine to determine the resident time of ore between the stope face and the metallurgical plant. However, Freegold has no intention at this stage to alter the period between the milling and measuring months.

The correlation co-efficient (R^2) diminishes when analysis, using single independent variable regression, is off-set by one month or more to the dependant variable (**MCF**). The most meaningful results were obtained comparing the results as were recorded, month by month. Three monthly moving averages of the Mine Call Factor was used for graphical trend analysis.

2.1.1. Variables.

Information on related issues were obtained from an Anglo American Corporation, Gold Division, database called IRIS. The period analysed includes April 1992 to September 1995. The variables considered were determined using the following logic.

$$\text{MCF} = \frac{(\text{Gold called from stopes} + \text{gold from reef development})}{(\text{gold produced} + \text{residues})}$$

$$\therefore \text{Gold called for (GCF)} = f[(\text{area mined (m}^2\text{)}, \text{stoping width (m)}, \text{grade (g/t)}, \text{relative density (kg/m}^3\text{)}, \text{development volume (m}^3\text{)}],$$

The Mine Call Factor was established as the dependant variable and the independent variables that could influence it were used for the statistical analysis. All the mines in the Freegold stable were analysed with particular reference to Western Holdings mine.

2.1.2. Statistical analysis, comparison between different mines.

The analysis were conducted using both single and multiple variables regression analysis to determine the statistical significant relationships.

2.1.2.1. Correlation coefficient (R^2) of regression analysis between the Mine Call Factor and single independent variables.

A single relevant independent variable was selected and the relationship between that and the Mine Call Factor (MCF) was determined. The analysis was conducted accepting that the measurement of the independent and dependent variables were 14 days apart, as described previously.

The number of observations were 42 with 40 degrees of freedom in all cases except Free State Geduld where it was 32 and 30 respectively. This is due to it's incorporation into Western Holdings Mine which then accounted for the combined production results.

The correlation coefficient (R^2) of 0.1121 is statistically significant at a confidence level of 95% for Free State Geduld while the figure valid for the other mines equates to 0.0927.

The results of the analysis is tabled below and the variables of statistical **insignificance** are shaded.

Description	Fred	FSG	WH	PB	PS	SP
MCF vs. Stope grade (g/t)	0.4362	0.0899	0.0007	0.0459	0.0794	0.341
MCF vs. Hoisted grade (g/t)	0.69226	0.18499	0.0350	0.2264	0.2433	0.4166
MCF vs. Gold called for stopes (kg)	0.4512	0.0262	0.3282	0.0059	0.0419	0.4804
MCF vs. Total tons hoisted	0.1566	0.0133	0.6585	0.0483	0.3105	0.0002
MCF vs. Ore hoisted tonnage discrepancy	0.2432	0.0112	0.2453	0.224	0.4385	0.0126
MCF vs. Reef tons hoisted	0.3463	0.1333	0.6499	0.0296	0.3137	0.0003
MCF vs. Sweepings	0.02848	0.0721	0.0534	0.1174	0.0011	0.0312
MCF vs. Trammings width	0.2915	0.0314	0.1988	0.0883	0.1065	0.0228
MCF vs. Stoping width	0.0109	0.0007	0.0239	0.089	0.1379	0.0013

Table 2.1. Statistical relationships between the MCF and relevant variables.

(Detail: Freddie's Table 1-10. .

FSG	Table 11-20.
WH	Table 21-30.
PB	Table 31-40.
PS	Table 41-50.
SP	Table 51-60.)

Discussion of the statistical analysis.

Freddies and Saaiplaas Mines are known as relatively low grade operations and operates in the 800cmg/t to 1 100cmg/t grade ranges. The other mines in the Freegold stable operates in the 1 200cmg/t to 2 000cmg/t ranges. The correlation co-efficient between the MCF and the stope grade, in the case of Saaiplaas and Freddies Mines, is statistically significant (Table 2.11), whereas in the case of the higher grade mines it is insignificant. The characteristics of the reef mined at these two mines include bigger conglomerates and the almost absence of carbonaceous reef.

Western Holdings in particular has a statistically significant correlation co-efficient between MCF and total tons hoisted (0.6585), ore hoisted discrepancy (0.2453) and reef tons hoisted (0.6499). The focus of Western Holdings Mine is the removal of ore from previously worked out areas that remained behind. This gold is not **called for**, but it is **accounted for**, resulting in the under-stating of the poor prevailing MCF.

Sweepings versus MCF has a statistically insignificant relationship other than at President Brand. The reason why the aforementioned mine has a different characteristic compared to other mines remains unexplained. However, the relationship at the other mines is somewhat surprising as traditionally focus was placed on this parameter to ensure a good MCF. The net worth of sweepings per sé is discussed in Chapter 5.

The results of this statistical analysis was used to determine the experimental strategy. **Stope grade**, in the case of the higher grade mines, **hoisted grade** the case of Western Holdings Mine and **sweepings** were the parameters to be further investigated by experimentation. The detail of the experimentation is described in Chapter 5.

2.1.2.2. Multiple variable regression analysis results:

A model was developed to investigate the possible measurable variables that affects the Mine Call Factor. A best fit model was developed and compare between the mines. The dependant variable was taken as the Mine Call Factor while the independent variables included are described below.

Gold called for = $f(\text{gold from stopes, gold from reef development, gold from old areas})$.

Gold called for = $f(\text{m}^2 \text{ mined}, \text{ grade, gold from o/a, tonnage hoisted, tonnage hoisted discrepancy, area swept, hoisted grade, reef metres, stoping width})$.

Multiple variable regression analysis were used to determine the correlation between the variables. It is concluded from the results that each mine has it's own characteristics. The results can be used to forecast the Mine Call Factor accurate to some degree. President Brand Mine appears to be an exception to this rule although the R^2 of 0.435 is statistically significant at a confidence interval of 95%. The results are published in the table below and the detail input/output is also described in attached tables 1 to 60.

Multiple variable regression results. (42 observations, 33 degrees of freedom, except for Free State Geduld where it is 32 and 22 respectively.) The correlation coefficient (R^2) in all cases is statistically significant at a 95% confidence interval.

DESCRIPTION	Freddies	Free State Geduld	Western Holdings
R Squared	0.8049	0.5988	0.7538
Constant	159.47	164.99	98.39
VARIABLES.			
Stoping width (cm)	-0.465	-0.645	-0.254
Gold called for o/a (kg)	0.0853	0.0727	0.0585

Area mined (m ²)	0.0006	0.0003	0.0008
Area swept (m ²)	-0.000042	0.0001	0.000011
Stope grade (cmg/t)	0.0594	0.0671	0.0258
Ore hoisted discr (ton)	-0.000093	-0.0005	0.000058
Total tons hoisted	-0.000065	-0.000014	-0.000083
Grade hoisted (g/t)	-14.962	-12.151	-7.66
Reef metres	-0.0064	-0.0105	0.0125

Table 2.2. Multiple variable regression results.

DESCRIPTION	Pres. Brand	Pres. Steyn	Saaiplaas
R Squared	0.435	0.602	0.73
Constant	241.85	95.07	185.6
VARIABLES.			
Stoping width (cm)	-0.556	-0.018	0.166
Gold called for o/a (kg)	-0.185	-0.019	0.001
Area mined (m ²)	-0.004	-0.0009	0.0001
Area swept (m ²)	-0.0003	0.0000576	0.041
Stope grade (cmg/t)	-0.0813	-0.0158	0.0000176
Ore hoisted discr (ton)	-0.0002	0.000059	-0.0003
Total tons hoisted	0.0006	0.0002	-25.5
Grade hoisted	11.17	2.722	0.0164
Reef metres	0.0061	-0.028	0.0115

Table 2.3. Multiple variable regression results.

2.2. Western Holdings Mine.

The MCF of Western Holdings mine was relatively constant between October 1986 and October 1993 at approximately 80 percent (Graph 2). However, as from then onwards the MCF dropped alarmingly to as low as 56 percent (Graph 9) in April 1995 when some of the ore from 2 and 3 shaft were transferred from the Welkom metallurgical plant to President Brand carbon-in-pulp plant and the 1 shaft ore re-directed from the Welkom plant to the Western Holdings plant (Graph 4). The subsequent lock-up in the President Brand Plant exceeded the release of the previous lock ups in the Welkom plant, whereas the release of gold from the Welkom plant in the 1 shaft case exceeded the subsequent lock-up in the Western Holdings plant. The combined effect of the gold lock-up was partly the cause of the unacceptable Mine Call Factor. It has been recovering to 67 percent in January 1996 but, it is far from being acceptable.

It is important to note that the mining method employed has remained without major changes during this period. The geographical spread of the remaining pillars mined has increased dramatically in recent years. One of the innovations that did take the industry by storm for a short while was the re-introduction of waterjets used for stope cleaning in 1992. It was soon blamed for the gold losses experienced (Graph 62-68). Many of these machines were subsequently removed from the underground workings of several mines. Unfortunately the gold loss did not decrease significantly in these instances. An experiment was conducted to confirm/dispute

this perception and the findings were that this method did not increase the gold loss when compared to conventional scraper cleaning. The results are described in Chapter 5.

2.2.1. Production information.

The stoping grade on Western Holdings Mine has increased from 10.8g/t in 1992 to 13.3g/t in September 1995. The tons hoisted decreased from 404kt/month to 328kt/month for the same period. This 328kt/month includes the areas taken over from the former Free State Geduld mine as from April 1995. It certainly indicates the tremendous cut back in production experienced due to the poor gold production. The prevailing MCF results in unattainable high stoping grades required to be mined at so as to remain in business.

The gold called for (kg) dropped from 3 486kg in April 1992 to 2 893kg in September 1995 (-17 percent) and decreasing. The gold produced decreased from 2 800kg to 2 050kg, which equates to a drop of 27 percent while the head grade increased from 9.06g/t in April 1992 to 9.13g/t in September 1995. The stoping grade increased from 1 515cmg/t to 1 754cmg/t during the period in question, (15.8 percent). It is not surprising that the Mine Call Factor dropped by 14.8 percent during the same period.

2.2.2. Statistical analysis results for Western Holdings Mine.

Statistical analysis was done for Western Holdings Mine using regression analysis. The Mine Call Factor was used as the dependant variable and the independent variables as listed below. There were 42 observations and the Degrees of freedom amounted to 40. A correlation coefficient (R^2) of above 0.0927 is statistically significant at a confidence interval of 95%.

The shaded areas indicate correlation coefficients of statistical insignificance.

Independent variable	REFERENCE				
	R Squared (R^2)	Constant	x coefficient	Graph number.	Table number.
Stoping grade (g/t)	0.000724	78.79782	-0.27809	16	21
Hoisted grade (g/t)	0.035017	94.09295	-1.99939	19	22
Gold called for stopes (kg)	0.0328189	43.88189	0.009195	14	23
Total tonnage hoisted (tons)	0.658473	27.36071	0.000119		24
Ore hoisted discrepancy (tons)	0.245317	71.16691	0.00017	22	25
Reef tons hoisted (tons)	0.649987	24.40517	0.000131	20	26
Sweepings (m ²)	0.053405	63.61374	0.118066	18	27
Tramming width (cm)	0.1987837	23.4154	0.327647		28
Stoping width (cm)	0.0239727	23.30919	0.382211		28

Table 2.4. Regression analysis of Western Holdings Mine.

It is surprising to find through analysis that there is no statistical significant relationship between the Mine Call Factor and the estimated **stoping grade** (Graph 16 & Table 17). The estimation of in-situ grade is therefore questioned and experimentation was conducted to prove/disprove this finding.

It appears from underground observations that the sampling of lower grade wider channel width reefs can be done more accurately compared to the narrow channel width high grade reefs. This conjecture will be discussed at length in Chapter 3. It is suggested that during the physical chipping of the sample, some softer portion of the carbonaceous reef is over-sampled in the case of higher grade ore. Sampling in the bigger conglomerate reef types is more accurate than in the reef types such as the Geduld facies where the gold is distributed throughout the channel and is not contained only in the 5mm carbon layer such as in the Steyn facies.

It is a popular belief that a good Mine Call Factor will be achieved when **sweepings** targets are achieved. A very controversial result from statistical analysis indicates the almost non-existent relationship between sweepings and the Mine Call Factor (Table 7, 17, 27, 37, 47, 57). Sweepings, however, has a positive contribution to the Mine Call Factor despite the aforementioned as the x-coefficient is positive in the linear relationship with the Mine Call Factor (Table 27). It is a perception in the mining industry that the gold

is concentrated in the fines of sweepings. Experimentation has indicated that this is not the case but that sweepings should rather be viewed as an indicator that the majority of the tonnage produced in the stope were removed (Chapter 5).

Western Holdings Mine is more sensitive to **reef tonnage hoisted**, $R^2=0.649987$ (Table 26), than to the variation to grade when it comes to Mine Call Factor. This is deduced because of the different R^2 's which are 0.000724 and 0.035017 respectively (Table 21 & 22). A reason for this phenomenon could be that the estimation of tonnage is more accurate than the estimation of grade called for. It remains intriguing that the MCF is reduced with a higher grade called for. This relationship is borne out by all the MCF/grade graphs (Graph 16, 23, 27, 35, 39 and 47). The control over the MCF using the normal avenues becomes less effective with increased grades.

The X-coefficient in the Mine Call Factor vs. total tons, ore hoisted discrepancy, reef tons and indicates it's relative importance (Table 24-26). These figures indicate that the most important contributor is tonnage hoisted discrepancy which has an x-coefficient of 0.00017 while that of reef tonnage is 0.00013 and that of total tons hoisted is 0.00011.

2.3. Conclusion.

The different mines in the Freegold stable all has different characteristics. The lower grade mine like Freddie's (Table 1) and

Saaiplaas (Table 51) are very dependant on the grade mined to ensure an acceptable Mine Call Factor, whereas a higher grade mine like Western Holdings (Table 21 & 24) must concentrate more on getting the tonnage produced from underground to surface.

The **theoretical gold loss** is investigated by means of full scale experiments underground to verify certain perceptions of the gold loss saga.

Mining legend has it that some of the possible areas for the gold unaccounted for are as follows:

- I. The fine particles of gold are trapped in the footwall.
- II. The fine particles of gold are lost during the blast.
- III. The fine particles of gold are blasted into the packs.
- IV. The fine particles of gold are washed away with the water.
- V. The gold is stolen.
- VI. The gold is concentrated in the mud.
- VII. The gold is lost in the plant.
- VIII. The residue values in the plant are not measure accurately.
- IX. The mineralogy of the ore is peculiar.
- X. Shale causes the gold to be lost.
- XI. It is in the sampling method.
- XII. It is in the valuation method, and many more.

Some of these causes were further investigated to resolve the

quest for the lost gold and are discussed in detail in subsequent chapters.

C H A P T E R 3.

GRADE ESTIMATION.

3.1. SAMPLING METHODS

3.1.1. Current Sampling Method Practised At Western Holdings Mine.

3.1.1.1. Introduction.

Skin or in-situ sampling of the ore underground remains a daunting task. It is possibly the most important part of the gold estimation procedure and this rudimentary method takes place under extremely difficult conditions underground. The validity of this procedure is highly questionable.

3.1.1.2. Description of the sampling method.

The face to be sampled is divided into 5 metre sections starting 2.5 metre from the upper limit. At the 5 metre position, the contact of the reef is determined. The face is dressed so as to have a workable surface to subsequently chip out. A section is then marked off, from the 5 metre mark, 10cm to the left, and a check sample, 10cm to the right of the 5 metre position. The bottom limit of the sample is marked 2cm below the contact and the upper limit is a minimum of 5cm and a maximum of 11cm above the contact.

In the case where a wide reef band is sampled, i.e. 30cm, the samples are marked off on the stope face according to the appearance of the reef, taking the previously mentioned dimensions into account.

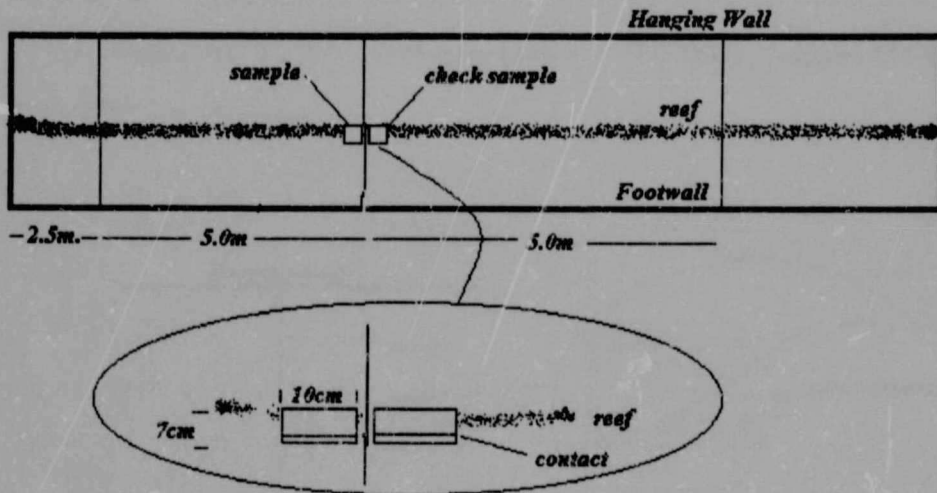


Diagram 3.1. Position of sample and check sample.

The sample and check sample are then chipped out separately to a depth of 2cm, into a collection pan using a chisel and hammer. A sample size of at least 300g is required to perform the assay at a later stage.

This part of the grade estimation procedure is inaccurate to a large degree. It is conceivable that a bias can be introduced, due to the hardness of the rock and the use of a chisel to chip the samples out of the surrounding rock. Observations have indicated that, especially in the case of carbonaceous reef, more than the required amount of this softer component is contained in the

sample. This practice can lead to the over-estimation of the gold in-situ as the gold deposition is contained primarily in the carbon zone. This is particularly so as the density of the softer rock is normally less ($2\,663\text{kg/m}^3$) than that of the surrounding rock ($2\,780\text{kg/m}^3$). The result of this is that when the gold content of the sample is extrapolated to that of the stoping width to be mined, it is an overestimation of the gold content (e.g. $2780/2663 = 1.04$.) (Table 1.2)

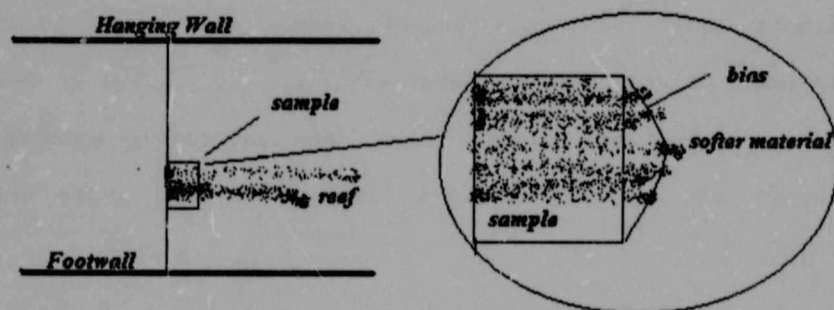


Diagram 3.2. Sampling bias that can be introduced during chipping.

3.1.1.3. 7cm vs. 4cm sampling experiment.

An experiment was conducted to compare grade values where samples and check samples were taken at different dimensions. The sample sizes were 7cm high (contact -2cm and contact +5cm), 2cm deep and 10cm wide. The check sample with dimensions of 4cm high (contact -2cm and contact +2cm), 2cm deep and a width of 15cm at least, but with a minimum weight of 300g.

Theoretically the sampled value in g/t will be less for the 7cm sample as it could contain areas of 0 g/t in the waste sections if it is assumed that the gold content was restricted to the reef contact. Conversely the g/t value of the 4cm samples should be higher than that of the 7cm sample as it contains less waste. However, the values of both samples expressed in cm.g/t should correspond as the individual g/t samples are adjusted for the height.

The results of the experiment indicated that the '7cm' had in 52 percent of the cases higher values than the '4cm' samples expressed in g/t (Graph 92 & 93). The immediate reaction is that the '7cm' samples overstates the actual value. The problem is to establish which one of the two sample methods is the correct method.

One of the possible reasons for the lower values in the 4cm sample is that the reef was in fact not restricted to the contact and that it was also distributed in the area above the 4cm sample which is seen as waste. A further experiment was therefore conducted to determine whether the aforementioned was the case. In this experiment the check sample was chipped as a 4+3cm sample. The 4cm and the 3cm sections were chipped and collected separately and individual grades determined.

A diamond blade angle grinder was used to cut the samples from the face. It was surprising to find that there were 0g/t in the 3cm

section and that the grade was confined to the contact of the reef. It certainly did not explain why the 7cm samples had higher values in this instance. The only logical explanation for this occurrence is that the variation of the grade in situ in the x,y and z horizons varies to such a degree that **no real comparison could be made.**

It is a logical conclusion that the sample size must be such that the complete channel width, at least, is taken into consideration when taking the sample. It should obviously meet the minimum weight requirements for assay purposes.

An over-estimation of the gold content of the sample can be achieved if more than the required amount of the reef contact is included in the sample. It is possible for the aforementioned to occur in the instance where the contact is of a softer texture than that of the surrounding rock.

No conclusion could be made on the basis of this experiment as the grade values varies in the x,y and z planes. The comparison of 7cm to 4cm samples in practice can therefor not be done accurately.

3.1.1.4. Skin sampling using diamond blade cutting.

Skin sampling of the underground panels is a daunting task. Once the face is dressed to be as solid as possible, the sample is marked off using a wax crayon. The sample is then chipped out of

the face using a hammer and chisel. The chipped sample is deposited into a collection pan while chipping so as to ensure that the complete sample is collected. However, it is easier said than done as sampling takes place under extremely difficult conditions. The ore sampled is extremely hard as the quartz pebbles in the rock have a hardness of 7 on Mohs scale of hardness. The sample is chipped by applying blows with the hammer onto the chisel. Sometimes it happens that after several blows onto the rock nothing is achieved due to the rigidity of the ore. The very next blow of the hammer can cause a large slab of rock to be dislocated which could be larger than the marked off section. It could also happen that only half of the section being chipped is suddenly dislodged, the other half remaining in the face. This type of occurrence is frequent and certainly places a great demand on both the workload and the integrity of the sampler. It seldom occurs that a perfect sample is chipped. This type of occurrence can introduce a bias into the estimation of the grade of the ore in situ.

Another possible bias that can be introduced is that once the sample position is determined, it could be found that the sample position is not ideal in the sampler's view. In such instance the position of the sample can be moved to a perceived ideal position. This interference in random sampling certainly will introduce a bias.

After observations of samplers in action underground it was decided to introduce the use of compressed air-powered 100mm diameter blade angle grinders. Although 150mm diameter double bladed compressed air-powered angle grinders are being used in industry, these units are big and cumbersome. The new unit introduced is a much smaller version and performs acceptably underground. Teething problems were still being experienced at the time of completing this thesis. The samplers, however, committed themselves to making this innovation work on Western Holdings Mine.

Comparative samples between rock sawing and chip sampling were conducted but it is extremely difficult, if not impossible, to compare the two methods' grades with each other, the main reason being the large variation in grade from samples adjoining each other even though the continuous sampling along a face will ultimately result in a log-normal distribution.

It was concluded that despite the fact that comparative sampling remains largely irrelevant, the use of diamond blade cutting of the samples is more accurate. Samples are cut along the perimeter to a depth of 2cm and then chipped out. The process is therefore refined and should result in improved accuracy in gold estimation methods.

The use of various valuation techniques to determine the grade is not explored any further as it is an established science. No such

method can estimate the in-situ value to an acceptable degree of accuracy if there is doubt in the validity of the samples. It was for this reason that the focus of the estimation process was restricted to improving the method of taking samples.

3.1.1.5. Continuous sampling experiment.

During 1988 an experiment was conducted to determine the grade distribution in a panel⁽²⁴⁾ (Graph 73-81). Four different areas were selected where limited geological disturbances occurred. Three of the panels were selected in the Basal reef, carbon facies, and one panel in the Leader reef (Alma).

The entire length of the panel was sampled in 10cm sections with a vertical width of 7cm in the Basal reef. In the case of the Leader reef in the panel was sampled over a vertical width of 17cm. A total of 940 samples were taken and analysed. The individual sample values were extremely varied and ranged from a trace of gold to 5111 g/t. An average of 54 percent of the samples were below the arithmetical mean of the total sample results. The variance reduced once the number of samples used to calculate the mean was increased. The mean grade determined using 2m sample results ranged from 224cmg/t to 1 747cmg/t while the true mean was 570cmg/t.

The experiment was repeated during 1995 (Graph 82-84) where a panel was selected in the Basal reef, Steyn facies. Samples were again

taken along the entire face every 10cm and a height of 7cm. The results were similar to those determined during 1988. The results of this experiment indicated that the grade dispersion along the face was very erratic. The position of the samples taken plays a significant role in the ultimate determination of the face grade.

It was concluded that the **gold called for** from underground sources is largely influenced by where exactly the samples were taken. It is clearly understood that when valuation methods are applied, significantly more values are taken into consideration to determine the estimated gold in situ. However, by deduction it was concluded that additional investigations need to be conducted to ensure that improvements are effected to the estimation procedure as a whole. The difference between the gold finally **accounted for** and that **called for** from all underground sources could not fully be explained from experimentation. The issue of whether the gold was there in the first instance remains. It is suggested that a portion of the **gold called for** did not exist in the first instance. The unaccounted for portion of the **gold called for** can be split into **real gold loss** and **apparent gold loss**. Real gold loss can be found underground and on surface. The remainder of the unaccounted portion was not there in the first instance. Reference must here again be made to the influence of density adjustment on apparent gold loss (Section 1.3.1.3). Once this issue is rectified, more attention can be given to the areas where **real gold loss** occurred.

3.1.1.6. Go-belt sampling.

Go-belt samplers are installed on Western Holdings Mine as a complete unit on the conveyor belts transporting the ore at shaft head. The sampling unit consists of a former that sweeps rapidly across the moving belt so as to remove a sample of approximately 50kg of ore in one sweep from the belt. The sample is deposited into a chute which extends to a container into which the sample is deposited. A weightometer on the belt is pre-set to trigger the sampler at regular tonnage intervals, i.e. every 150 tons.

The shift's samples are collected into the container and representative samples are finally taken to estimate the grade that was conveyed by means of the belt. The tonnage recorded by the weightometer for the period in question is used to finally estimate the gold hoisted for each period.

Go-belt samplers were introduced on Western Holdings Mine during July 1995 and were installed at shaft outlets. At the time of writing the thesis, the system was fully operational and is used as a daily information system. It was planned that gold apportionment for Freegold will be done using this system as from April 1996. It will replace the system of using the calculated gold broken for gold apportionment at that point in time.

Author De Jager E J

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