Mine Call Factor issues at Iduapriem Mine: Working towards a Mineral and Metal accounting protocol

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DECLARATION

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ABSTRACT

The theory of Mine Call Factor (MCF) compares the sum of metal produced in recovery plus residue to the metal called for by the mines evaluation method expressed as a percentage. This MCF concept is well known in underground scenarios and therefore this report highlights the MCF issues and the variable components affecting it from a surface mine perspective. The MCF investigation established the relationship between actual measurements and reporting against measurement protocols. Such measurements include "tonnage, volume, relative density, reconciliation strategy, and truck tonnage determination, sampling and assay standards. This study investigated how these measurements are conducted on Iduapriem Mine according to the mine's standard operating procedures (SOP). An improvement of documents towards a metal accounting protocol based on the AMIRA protocol is recommended. The mine's current quality control protocol was further expanded to reflect current practices. The mine to mill reconciliation compared production estimates from various sources (resource model, grade control model, pit design, plant and stockpile, truck tally, stockpile and plant feed, plant feed and plant received) in the period 2009 and 2010. Reconciliation factors expressed as a percentage were statistically analysed for discrepancies for tonnages and grades. It was realised that there is more confidence in mass (tonnage) measurement compared to grade. A generic mine to mill reconciliation path was suggested to be used by the mine.

DEDICATION

I DEDICATE THIS REPORT TO ALL THE ANGELS WHO'S NAMES THE REPORT WIL NOT BE ABLE TO CONTAIN. I AM FULL OF GRATITUDE FOR THE SUPPORT, ENCOURAGEMENT AND GUIDANCE THROUGH OUT THIS JOURNEY OF BEGINNING AND COMPLETING.

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LIST OFACRONYMS

MCF – Mine call factor

BME – Basic mining equation

BFE – Basic financial equation

BCM – Bulk cubic meters

CRM – Certified reference material

SRM – Standard reference material

DTM – Digital terrain model

AARL – Anglo American Research Laboratory

SAG – Semi Autogenous Grinding

LIST OF DEFINITIONS

Scats – These are oversized rock materials which are unable to grind in the SAG mill. This material is separated from the steel balls and is passed again into the SAG mill for further grinding.

Standards: These are materials whose expected/best value has been determined by an accredited laboratory. Standards are used to check the accuracy of the analytical laboratory.

Blanks: They can be pulp or course. They are usually devoid of metal content (waste) and are used to check on equipment cleaning effectiveness at the analytical laboratory.

Duplicates: These are used to check for the quality of sampling splitting and are used for checking for the precision of the analytical laboratory.

Density: defined for any substance i.e. solid, liquids or gas.

Bulk density: Used only in cases where the particles of matter are loosely packed with space for air within

Pulps: These are usually fine material also used to check for the precision of the laboratory.

Ore reserve: blocks of ore estimated as payable or unpayable based on current sampling of stope faces related to current pay limit.

Not in reserve: ore sent to the mill from stoping of reef that was not available for blocking when the ore reserve estimation took place.

Discrepancy: the tonnage discrepancy is either a shortfall or excess. Shortfalls is given a plus (+) sign because it is added to the surveyors calculation of tonnage to equal the mill tonnage figure; an excess is given a minus (-) sign because it has to be deducted from the surveyors figure to equal the mill measurement of ore sent to mill.

Mill bin difference: The difference between the tonnage in the mill bins at the beginning of the month and that left in the bins at the end of the same month

Delimitation error: This error occurs when the geometry of the increment is incorrectly sampled

Quality assurance (QA): it describes the methods that a laboratory uses to ensure the quality of its operations

Quality control (QC): it describes the operational techniques employed by the laboratory to fulfil requirements for quality.

Platform scale: A complex lever scale whose platform (element for receiving load) is supported by four or more (rare in most cases) main bearings.

Lixiviant: A liquid medium used in hydrometallurgy to selectively extract the desired metal (in this case gold) from the ore.

Adsorption: This is the process where solid is used for removing a soluble substance from water. In this process active carbon is the solid

Leaching: A widely used extractive metallurgy technique which converts metals into soluble salts in an aqueous media.

1 Introduction

The mine call factor (MCF) at AngloGold Ashanti Iduapriem Mine from 2006 to 2012 was averaged 100% (AngloGold Ashanti, 2013). The difference of “gold called for” and “gold produced” is 5 890 ounces at an average price of 1 098.38 US$/Oz over this period. The loss in monetary values is equivalent to US$ 6 469 432. This excludes gold lost to Scats and plant inventory change. The total revenue obtained over this period is US$ 1 230 658 000. The percentage of revenue loss is 0.7% which is not very significant. Actual MCF target was achieved in 2010 and since then has been declining to a minimum of 97% in 2012. This MCF investigation established the relationship between actual measurements and reporting against measurement protocols. Such measurements include “tonnage, volume, relative density, reconciliation strategy, truck tonnage determination, sampling and assay standards. The purpose of the study is to investigate how these measurements are conducted on the mine according to the mine’s standard operating procedures (SOP) and recommend improvement documents towards a metal accounting protocol for the mine.

1.1 Problem Definition

The in situ gold estimated is usually different from what is finally produced at the process plant. There are several variables which contribute to this variance. Until measurement points that can lead to variance are put in place to solve variances at the measuring points, it would become difficult to establish where the actual causes are.

1.2 Relevance of the research

Mineral losses arise from issues such as inaccurate grade control results, significant movements during blast, poor mining supervision, complicated crushing cycle, inaccurate weight meter readings and recoveries, poor assay strategy, and residue estimation. Metal losses can be reduced to the minimum if there is compliance to existing protocols.

The researcher believes this will bring about opportunities for improvements which can lead to minimum metal losses and hence improve plant recovery of gold and therefore serve as a good standard for addressing industry MCF problems.

The very low and high MCF requires being investigated. This is both a problem and an opportunity to do an investigation on the MCF, however this falls outside the scope of this research.

1.3 Objectives

The objectives of the research are:

- To develop an ore flow chart from the pit to the crusher;
- Establish various measuring points along ore reconciliation model;
- Analyse variances over time at those measurement points;
- Investigate causes of those variances;

- To see if measures can be put in place to minimize variances at those points which should lead to minimizing overall mine call factor variance

1.4 Expected outcome

The expected outcomes are to identify which measurement points have the highest variance and to minimise the overall effect of mine call factor variance.

1.5 The methodology employed

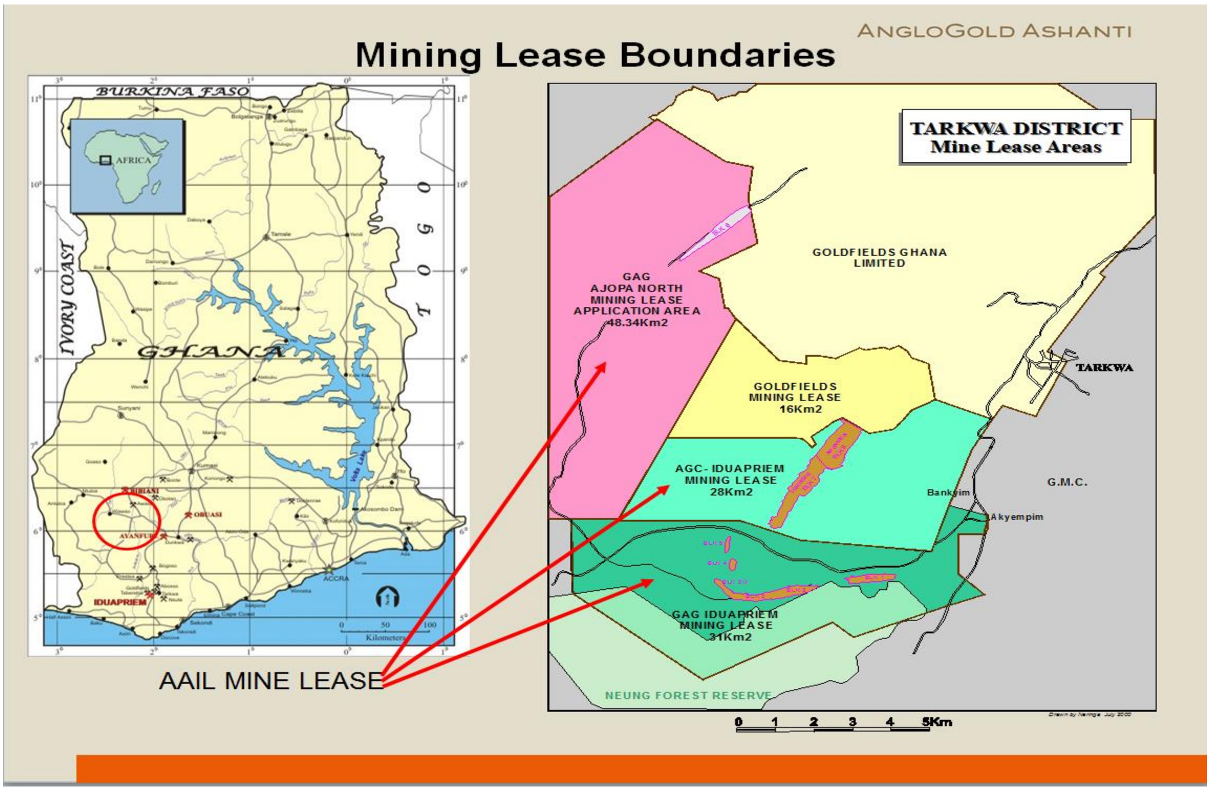
To conduct a literature review in order to understand the variable components affecting the MCF. To conduct a visit to the mine and obtain a physical feel of the ore flow from the pit to the plant. Other data required includes the reconciliation data for resource model , Grade control (GC), Ore Control polygon (perimeter), Actual mining /truck count, stockpile, mine delivered to mill , mill received, mill accounted for and bullion i.e. the gold itself in ounces/tonnes or kilograms; mine call factor data (based on the mine's method of calculation) from 2006 to 2012. To obtain standard operating procedures for measuring volume, tonnage, relative density, grades, method of evaluation, grade control procedures, sampling and assaying.

The report consists of five chapters. First chapter defines the problem, objectives, methodology, expected outcomes, background information about Iduapriem mine, production reports and the MCF issues with specific reference to periods between 2006 - 2012. The second chapter discusses a literature review on the theory of metal and ore accounting fundamentals. The objective of metal accounting, its complex nature, reconciliation and balancing, statistical concepts for determining discrepancy and the AMIRA code of practice. Chapter three highlights the fundamentals of metal accounting procedure at Iduapriem mine. Details of how quantities such tonnage, sampling and assay strategy as well as the reconciliation strategy are discussed. Chapter four then focuses on developing a metal accounting protocol suitable for Iduapriem with focus on metal accounting from source to mill. Conclusions and recommendations are made in the final chapter five.

1.6 Overview of Iduapriem mine

AngloGold Ashanti has two operations in Ghana namely Obuasi (which operates both surface and underground mining) and Iduapriem (which operates only surface). These two mines were formally owned by Ghanaian based Ashanti Goldfields. Ashanti Gold fields and South–African based AngloGold combined in April 2004 which gave birth to AngloGold Ashanti. Iduapriem gold mine is solely owned by AngloGold Ashanti since September 2007. It comprises the Iduapriem and Teberebie concessions which are approximately 110 Km^2 (InfoMine, 2012). The Iduapriem mine is located in western region of Ghana (Figure 1.1).

Figure 1.1 Iduapriem Mine lease boundaries



(Anglogold Ashanti, 2011)

1.7 Geology and Mineralisation

The geology of Ghana falls within the West African Craton. Besides South Africa, the West African Craton is the second largest in Africa where lower Proterozoic rocks are extensively preserved. Proterozoic rocks are made up of belts of metamorphosed volcanic and sedimentary rocks exposed in Ghana, Burkina Faso, Niger and Cote d'Ivoire (Scribd, 2013). For the purpose of this report, the geology and mineralisation of Iduapriem is obtained from (InfoMine, 2012) and explained below.

Iduapriem mine is situated within the tarkwain group of rocks which is part of the West Africa craton (sheltered by metavocanics and metasediments of the Birimian supergroup). The Birimian terrane is made up of north-east and south-west volcanic belts which are separated by basins.

The mineralisation of gold is hosted in the Proterozoic Banket Series conglomerates, which are developed within these sediments. There are beds of quartz pebble conglomerate, quartzites, breccia conglomerates, and grits within the Banket Reef Zone (BRZ). The Banket series outcrops in the mine lease area and it forms part of curved ridges that extend southwards from Tarkwa, westwards through Iduapriem and northwards towards Teberebie.

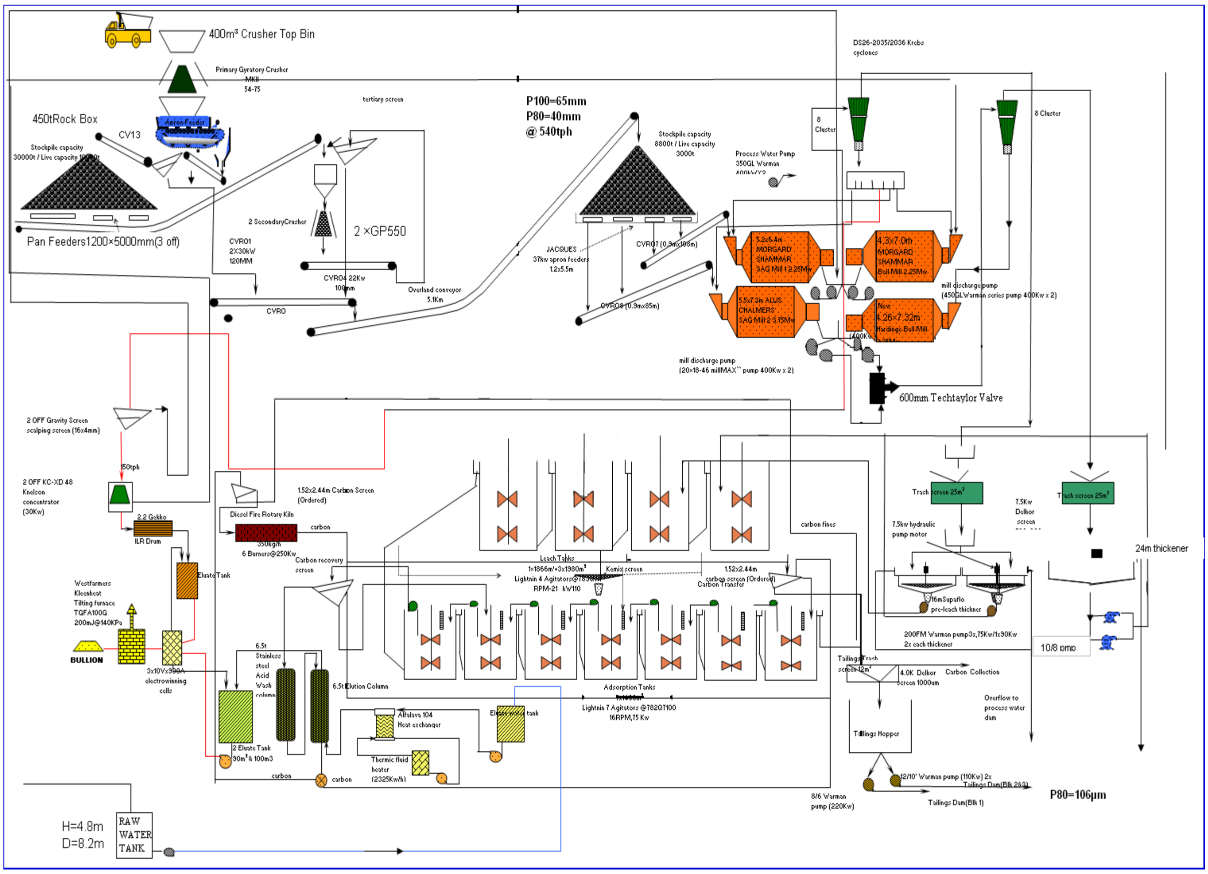
The size and amount of quartz pebbles within the conglomerate units defines the gold content. Reports of mineralogical studies indicate that the size of gold particles ranges from 2 to 501.

Iduapriem gold mine contained 55.4 million tonnes of ore as proven and probable reserves at a grade of 1.43g/t as at December 2011.

1.8 Processing Plant

The method of extraction by the processing plant is cyanide leaching. Cyanide is relatively cheaper than the use of other lixiviants and can be disposed of naturally, in a cost effective manner. It consists of a 3000t live capacity, a two circuit milling (classification and thickening), Gravity circuit, 4-stage leach circuit and 7-stage CIP circuit, a 6 ton AARL elution and carbon regeneration circuit and a tailings storage facility. The CIP tanks are each of capacity 1460m^3 (Baidoo, 2013). He further explained that CIL which is the acronym for Carbon In Leach involves a simultaneous combination of leaching and adsorption in the same tank. If the ore being treated is suspected to be significantly carbonaceous, there is the possibility that as leaching takes place, the naturally occurring carbon in the ore adsorbs the gold ions from the solution and prevents the process from benefiting from the gold leached as it will end up as tails. The activated carbon usually has a higher activity than the naturally occurring carbon and so it is able to preferentially adsorb a significant proportion of the gold ions. CIP is the acronym for Carbon In Pulp which involves leaching of gold into solution in one tank after which the leached slurry containing the gold ions are transferred into a separate tank for activated carbon adsorption. The issue with these two acronyms is that they are often used interchangeably.

Figure 1.2 Iduapriem CIP Plant



(Anglogold Ashanti, 2012)

1.9 Sampling and assaying procedure after mill fence

Hammer sampling is done on the conveyor belt prior to milling to determine the head grade of the ore. Other sampling points include the new leach feed, cyclone overflow and underflow, mill discharge and tails sampling also done after milling. The in-house assaying deals with only gold solution analysis whiles the solid samples are handled by SGS laboratory contractor and they use mainly aqua regia and fire assay depending on Iduapriem's specifications (Baidoo, 2013).

1.10 Mining and operation

The Iduapriem mine is regarded as a conventional open pit mining operation. The units under mining are drilling, blasting, loading and hauling. Moolman mining Ghana Limited, which is a subsidiary of the Aveng Group of South Africa, has been contracted by Iduapriem mine to do mining. According to (Yamoah, 2013), loading and hauling of material is executed using two Liebherr 9350 (Figure 1.2) and one Liebherr 984 excavator with a fleet of caterpillar 785 150-tonne capacity haul trucks (Figure 1.3).

Figure 1.3 Leibherr 9350 excavator (Iduapriem Mine)

Source (Photo taken by researcher on the mine)

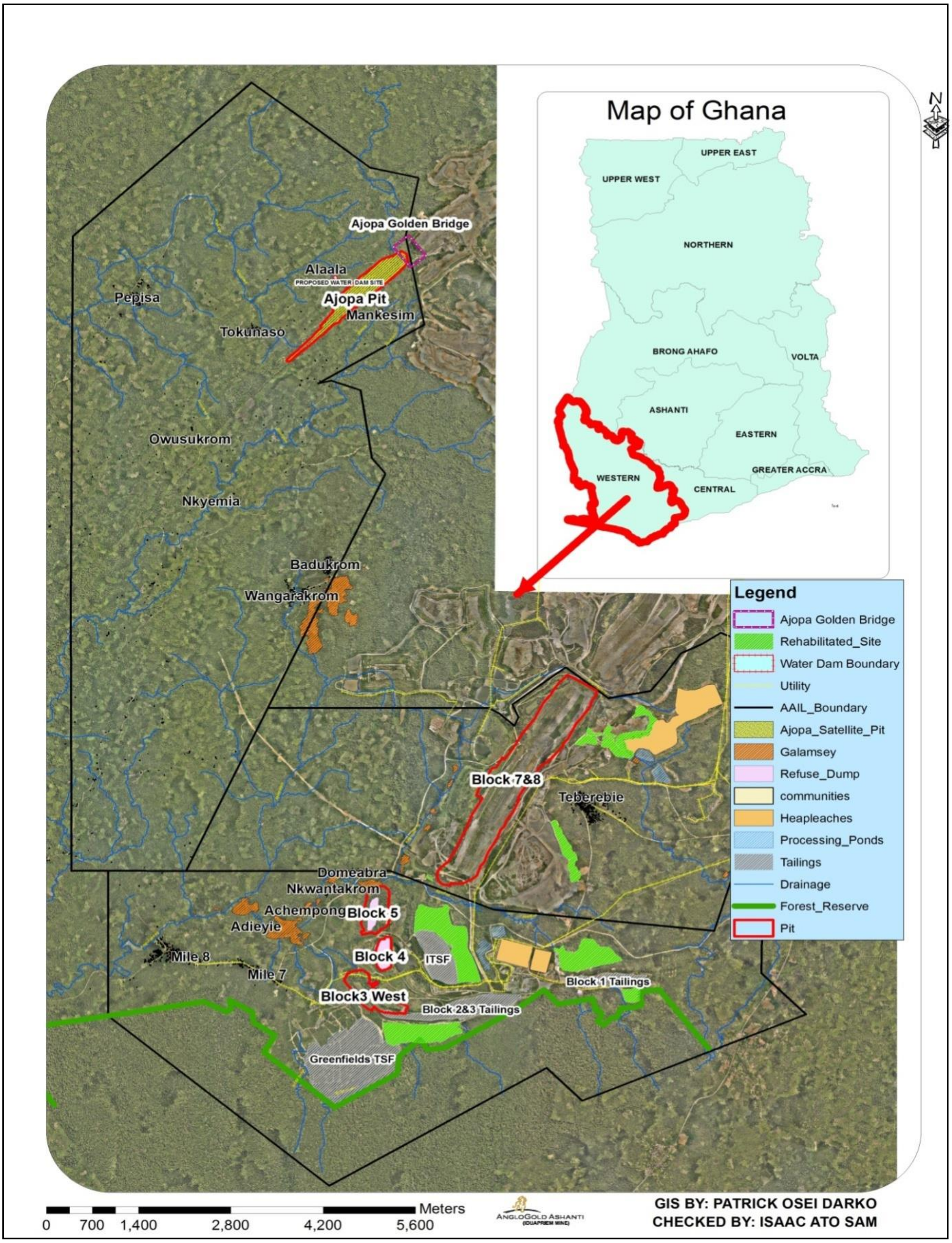
Figure 1.4 Caterpillar haul truck (Iduapriem Mine)

Source (Photo taken by researcher on the mine)

There has been a renewed interest in evaluating and considering low-grade Mineral resources in the Tarkwain conglomerates that was below limits of the existing pits because of increases in gold price. It is planned in 2012 to determine if the resources are economically sufficient to support underground mining (InfoMine, 2012).

The Ajopa project was to begin in 2010 but due to some constraints it was scheduled to start in mid-2012 and expected to cost $12m. Ajopa is said to host an ore reserve estimated at 4.9 Mt at grade of 2.05g/t, which is approximately 363,000 oz of gold (InfoMine, 2012). Ajopa is located within the AngloGold Ashanti Iduapriem Limited boundary with an area size of 48.34Km^2. During the time of researcher's visit to the mine, the Ajopa pit, block 7 and 8 were the active pits (Figure 1.4).

Figure 1.5 Iduapriem Regional map



(Anglogold Ashanti Iduapriem Mine, 2012)

1.11 Iduapriem production reports and MCF issues

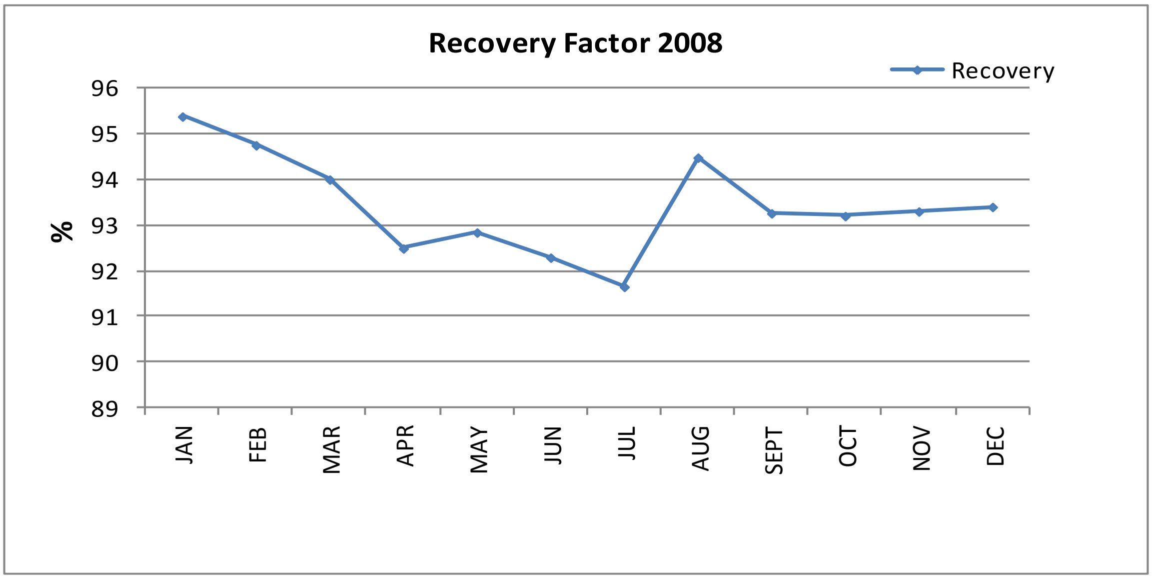
The MCF trend is analysed from the period 2006 – 2012 and therefore the production reports over this period is discussed below.

The country report from Iduapriem Ghana (2006) says production at Iduapriem declined by a 4% which was below 200 000 oz. The decline was attributed to mechanical problems with an old crushing plant and a gear box failure at SAG 2 mill during the last quarter of 2006 (Anglogold Ashanti, 2006). The MCF was however slightly above 100% although the "gold produced" was less the "gold called for" by 580 ounces (Appendix A).

In 2007, Production was just below 185 000 Oz, a 6% decrease on the previous year. This reduction was again recognised due to frequent mechanical problems and aged crushing plant and gearbox failure on SAG 2 mill, just like the previous year. Also a fire outbreak which occurred on one of the Volta River Authority's (mandated by law to generate, transmit and supply electricity) sub-station affected production (Anglogold Ashanti Iduapriem, 2007). The impact on the MCF was little because it increased by 1.8% and there was still a theoretical gold loss of 510 ounces (Appendix A).

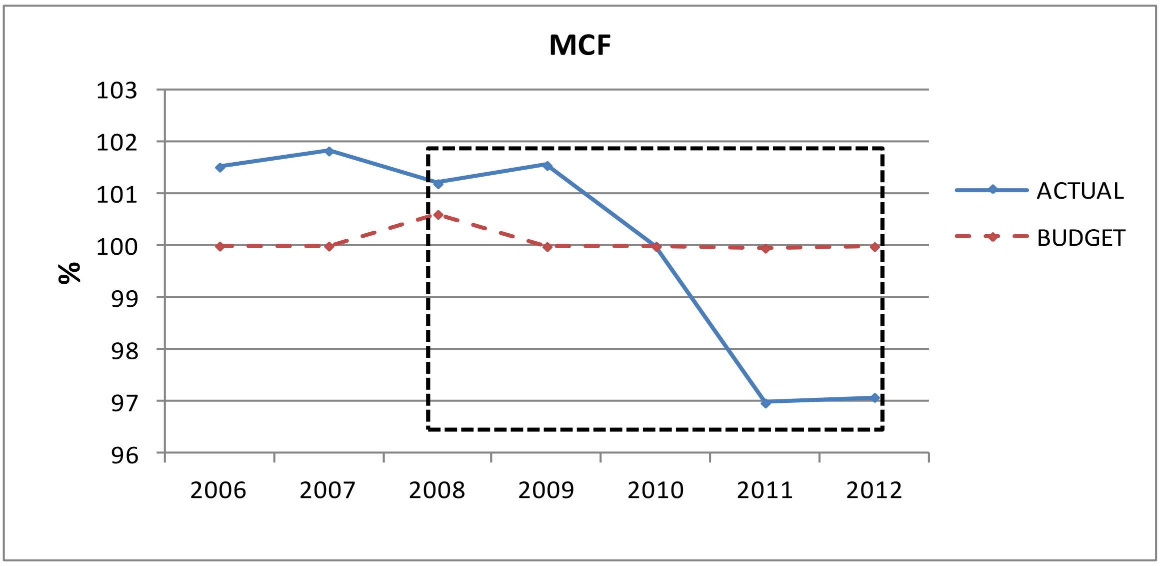
It was reported in 2008 that production increased by 8% to 200,000 ounces despite a decline in the grade mined. A scutter crusher was commissioned in the first half of the year and an improvement of blast fragmentation resulted in an increase in crushed tonnage by 26%. Recovered grade however declined by 5% which was due to reduced head grade and lower recoveries in the first half of the year (Figure 1.6); however recoveries were improved in the fourth quarter because of mechanical upgrades of the hydraulic flow path in the leach section (Anglogold Ashanti, 2008). MCF dropped slightly by 0.6% from the previous year but was still above a 100% and recorded a gold loss of 960 ounces. Figure 1.7 shows the general trend of the actual MCF for the same period. A proper MCF investigation will be discussed in chapter 3 that may have led to the decline since 2008.

Figure 1.6 Recovery factor



Source (AngloGold Ashanti, 2013)

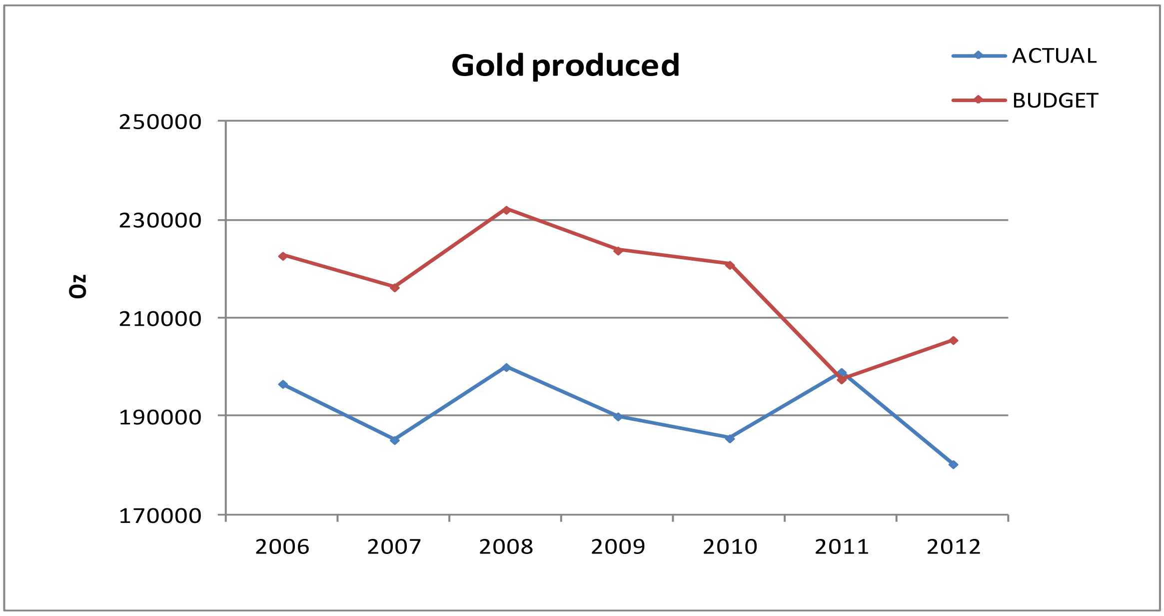
Figure 1.7 MCF 2006-2012



Source (AngloGold Ashanti, 2013)

In 2009, there was a decline in production of gold ounces by 15% with respect to the budget of 223,730 ounces. The MCF was achieved once in 2010 and has since been declining. The actual, and budgeted gold produced over the same period 2006-2012 is in Figure 1.8. Actual gold produced is below the forecast for throughout this period and was 0.7% above the budget in 2011.

Figure 1.8 Actual and budget gold produced



Source (AngloGold Ashanti, 2013)

1.12 Mine call factor and its significance

Storrar (1981) defines the MCF as *the ratio, expressed as a percentage, which the specific product accounted for in recovery plus residue bears to the corresponding product called for by the mine's measuring methods*. The formula is mathematically expressed below:

$$MCF = \frac{SUM\ OF\ METAL\ PRODUCED\ IN\ RECOVERY\ PLUS\ RESIDUE}{METAL\ CALLED\ FOR\ BY\ MINES\ EVELUATIONS\ METHOD}\ \text{X } 100\%.$$

If sampling, assaying and tonnage measurements in a mine and plant are perfect and there is no mineral lost at any stage during handling and processing, then the MCF should theoretically be 100%. Shortfalls and excesses are tonnage discrepancies which should balance out over time; these conditions should enable the use of correct densities and accurate measurements for calculating tonnages of rocks loaded (Storrar, 1981).

1.13 Apparent and real losses

Losses are further classified as either “Apparent or Real” (Cawood, 2003). He further explains that apparent losses result from issues such as over-estimation of metal contents, inappropriate sampling standards and incorrect relative density applied. These will occur during geological modelling, evaluation models, and block models. Real losses are actual, physical metal losses like gold locked in crevices in the plant as well as during trammings, hoisting, extraction, and processing. In addition there are some instances where gold losses (real or apparent) is inevitable e.g. gold locked in crevices, gold lost due to spillages, gold lost due to absorption in the plant etc. All of these prevailing factors tend to keep the MCF on average below 100%. Cawood (2003) investigated Underground face sampling on narrow reefs and he f ound that MCF is not a reliable indicator when used to identify the location of gold loss because it covers a wide range of activities, starting at the work face to the final gold product accounted for in the plant. These ranges of activities involve a wide area of technical disciplines; a condition that is a breeding ground for conflicts amongst the various disciplines.

This is further emphasised by (Storrar, 1981:265) who stresses that an “abnormally high MCF which persists over a long period is unsatisfactory if the cause is not determined”. Such irregularities might undermine the confidence of sampling procedures and therefore sampling methods must be reviewed.

Storrar, (1981) says a persistent low MCF of below 80% gives rise to a concern at all management levels because mine sampling may be exaggerating the content of ore treated. However, if sampling, assaying and extraction procedures are all good, then it becomes a major problem in detecting where and how mineral losses are occurring. This should not cause personnel to manipulate the ore accounting procedures simply to achieve a MCF that is perceived as acceptable.

(Springett, 1993) indicates that the MCF should not just serve the purpose of simply balancing the books and overlooking the real problems which when dealt with can increase profitability. Such an approach is dangerous and counter-productive. A better way forward is to identify causes and use the information obtained to solve the problem.

Various factors are known to have contributed to a low mine call factor. (Clarke & Harper, nd) in their poster session articulates the potential physical causes for the loss of gold in a producing mine ranges from gold loss due to blasting, to gold thefts by syndicates which operate on many mines. (De Jager, 1997) also declares that the mining industry is sometimes faced with various myths with regards to the possible causes of unaccounted for gold. The MCF issues vary from mine to mine and therefore it is rather appropriate to follow a systematic scientific approach to distinguish between fallacies and facts and hence an investigation into the mine call factor should not concentrate on “finding, fingering and fixing” losses (Cawood, 2003:203). The investigation should begin by analysing the various components of MCF. Even though the variable components of MCF vary from one mine to the other, depending on the ore flow diagram from underground or surface to the smelter, (Storrar, 1981) suggested a systematic examination of the following nine variables for typical underground gold mines at that time.

- Surveyor's measurements and calculations;
- Current sampling of ore sent to mill;
- Rock packed underground as waste;
- Rock sorted as waste on surface;
- Ore picked on surface from waste;
- Tipping of ore and car factors;
- Accumulation of ore and sweepings;
- Losses or theft of gold in plant;
- Assay bias and allowance for silver content in gold assaying.

The reasons for a MCF being less than perfect may be classified into errors in estimating the quantity and quality of mineral expected from the resource. A similar approach is recommended for surface mines in addition to Storrar's nine variable components to be examined in order to trace potential losses. Table 1.1 below lists areas of potential ore losses for surface mines.

Table 1.1 Areas of potential losses from a surface Mine

Category	**Type of ore Loss**
Resource /Reserve	Failure of ore body to show sufficient robustness
	Wrong interpretation of ore body geometry
	Inappropriate grade estimation algorithm
	Incorrect truck tonnage factor
	Incorrect use of in situ and soil densities
	Mistakes with down hole survey
Survey	Wrong drill hole coordinates
	Incorrect control for markout
Sampling	Insufficient sampling to confirm continuity
	Gold loss from poor sampling prior to assay
	Loss of sample
	Incorrect ore geometry interpretation
	Applying wrong cut off criterion
	Poor quality control methods and analysis
Blasting	Excessive heave
	Excessive back break
	Excessive flying rocks
	Toe undercuts
	Blasting of large boulders
Material movement	Digging beyond contact
	Trucking to wrong destinations
	truck spillages
Clean up	Grading across ore zones
	Dozing ramps on ore faces
	Running trucks across ore zones
ROM pad	Ore left on rom pad
	Spillages from trucks
	Stockpile pad supercharging

Source (Developed by researcher from Iduapriem grade control manual)

Cawood (2003:203) has outlined a seven step approach on how to do a MCF investigation namely "Review the existing protocols and establish compliance, Literature Survey, Understanding the process of ore flow to the smelter, Identifying and quantifying risks, Introduce tests and observe responses, Distinguish between long term and short term research topics and Adjusting existing protocols as a consequence of research outcomes".

1.14 Summary of MCF issues at Iduapriem

The actual MCF from 2006 to 2009 was above the budget. The target was achieved once in 2010. Subsequent years recorded a decline of actual MCF below the budget. The average MCF throughout this period is 100%. Theoretical loss in ounces is equivalent to 5 890 and revenue of US$ 6 152 044.80. This excludes gold lost to Scats and plant inventory change. Although the average MCF is 100 %, it is expected, based on the theory of MCF, that there will be totally no gold loss recorded; however this shows that a MCF is not a reliable indicator to determine losses. Compliance to standard operating procedures and constantly checking practices against best practices are important. A protocol for measurements such as tonnage and grade measurements are all very critical in attaining maximum recovery of gold and hence acceptable MCF. An understanding of why and where the variances occur between measuring points along the ore flow diagram is also critical in addressing MCF problems which is what the study is about.

1.15 Conclusion

In this chapter, the trend of MCF at Iduapriem mine from 2006 to 2012 was discussed. The unaccounted gold with respect to the total revenue is not very significant; however this remains a concern to management which is faced with significant drops in feed grades. The report therefore aims at analysing the measuring points along the ore flow diagram, studying variances over time at the measuring points and investigating causes of these variances which should lead to minimising an overall MCF variance.

The mine was visited by the researcher and the data (BME BFE from period 2006-2012, reconciliation data from mine-mill,) was obtained. The use of spread sheets is very important in keeping track of the various ore mined and from different sources for accurate metal accounting. The second chapter gives insight into the theory of metal accounting. The complexity of metal bearing flow of material, systematic approach towards material movements from source to plant or stockpile and an ore balanced sheet which should lead to a MCF calculation is discussed.

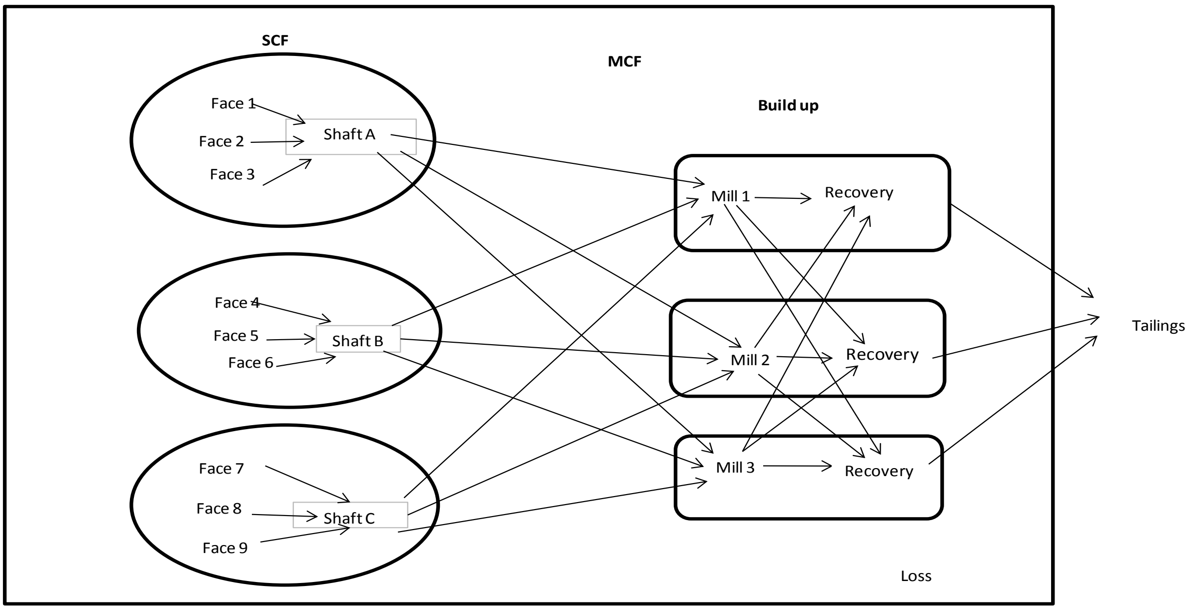
CHAPTER 2

2 Theory of Metal accounting

Metal accounting is an essential aspect of Mineral resource management. The objective is to balance the physical metal content (Hills, 2000). The flow of materials bearing metal is complex in nature. Figure 2.1 shows the complex nature of the material flow in an Underground mine. A similar principle can be applied to that of an open cast mine (Figure 2.2). These complexities according to Hills arise from:

- Different and multiple mining faces;
- Multiple shafts/pits, which sometimes come from different mines;
- Multiple plants with transfers of inter plant materials; sometimes regional refinery receiving metal bearing from several unrelated mines;
- Loss of production as a result of dilution of material and inefficiencies such as transfer or waste to plant and ore to waste dump;
- Several storage points e.g. bins and stockpiles.

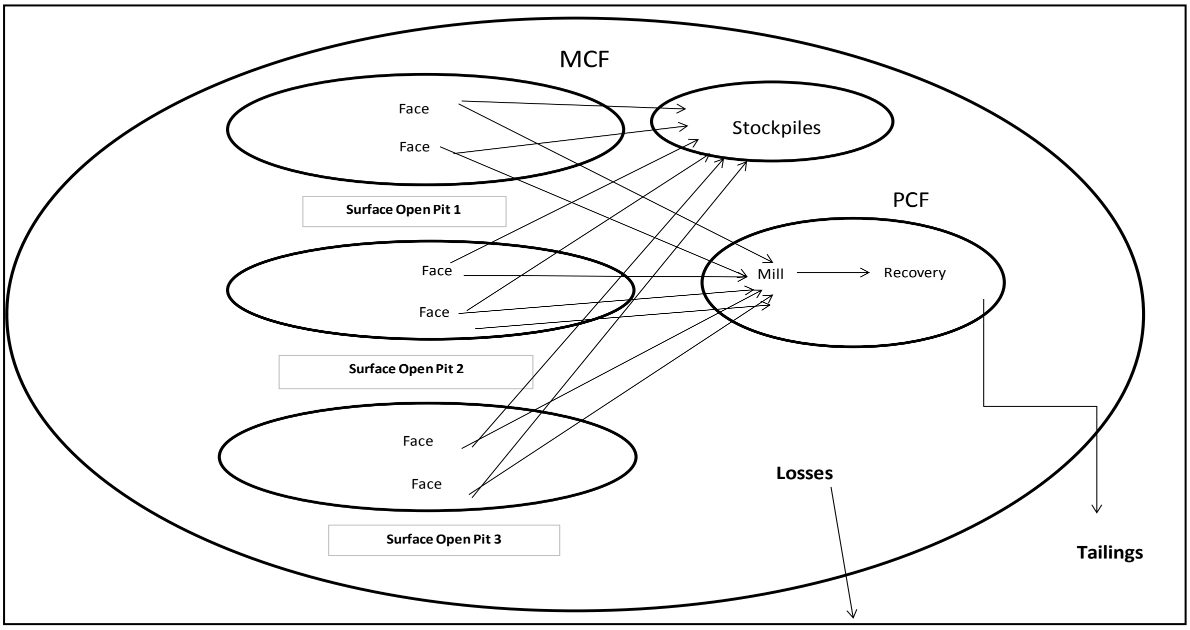
Figure 2.1 Underground material flow



Source: (Hill, 2000)

An open cast flow (Figure 2.2) may be less complicated if there are fewer faces to mine and less stockpiles. The complexity varies from mine to mine and the nature determines what the flow pattern would be.

Figure 2.2 Open cast material flow



Source (Concept based on Hills)

The complexity of material flow however requires a systematic and safe approach in determining the metal content (product of measured tonnage and sampled grade) between any two points along the flow. (Janisch, 1973) defines ore accounting procedures as a systematic approach for recording and representing of ore valuation, mine grade and ore treatment statistics of a mine. The main aim is to present statistical data in a format that is easy to efficiently track and monitor the flow of materials from underground or surface to the plant for treatment which will assist management to examine performance and analyse potential profitability. These statistics are used to generate reconciliation and efficiency factors such as the MCF which is used by management for planning purposes. The surveyor measures and compiles quantity or tonnage records respectively, the sampler measures the quality of ore and waste rock mined and disposed of in or out of the mine. Treatment statistics are compiled at the ore processing/metallurgical department. Records of these are kept in a spread sheet to serve as a good management tool for examination of actual performance against planned performance from which the MCF is determined. The conversion of physical losses into monetary values emphasises areas where there is greatest monetary losses (Storrar, 1981). Management can therefore focus on specific areas and identify specific measures to implement for further improvements.

2.1 Ore flow balance sheet

It is a good practice to generate a standard ore flow balance sheet indicating all sources of metals such as gold. An explanation of all recoveries and losses along the flow must be accurately captured in the standard ore flow sheet. An example of an ore accounting sheet showing the sources of ore is in table 2.1

Table 2.1 Ore flow balance sheet

Category	%	Mass	grade	content(g)
Pay ore reserve				
+ Not in reserve				
+ Unpay ore reserve				
+ Other sources stoping				
+ Stope reclamalation				
+ Stope Development				
Total broken in stopes				
- Sorted and packed in stopes				
Trammed from stopes				
+ Reef development				
- Reef in ballast				
- Development ore sorted				
total reef trammed				
+/- Discrepancy				
Hoisted as reef				
+ Shaft bins (beginning of month)				
Total in shaft bins for the month				
- Shaft bins (end of month)				
Total from shaft bins				
+ Ore from reef dump or stockpile				
To surface sorting plant				
- Sorting				
+ Ore from reef picking plant				
To mill bins				
+ Mill bins beginning of month				
Total in mill bins for the month				
-Mill bins (end of month)				
- Tons milled				
+ Tons milled				
+ Slimes treated				
Total tons treated (gold called for)				
Mine Call factor				
Recovery/ Residues				
Recovery plus residues (gold accounted for)				

Source (Cawood, 2012)

An application using the ore flow balance sheet in an underground scenario is demonstrated using the following extract (Table 2.2) from the records of a mine for one month's production and result is displayed in Table 2.3. The following are calculated:

- The survey discrepancy (Is it a shortfall or an excess?)
- Mine call factor
- Plant recovery factor

Table 2.2 Mine production results for one month

Metallurgical Plant	
Tons treated	280 000 t
Residue value	0.55 g/t
surface sorting	9% of the tonnage received at the sorting plant
Gold recovered	4 360 kg
Mill bins	
Beginning of the month	7 400 t @ value of 12,36 g/t
End of month	9 500 t
Density of solid rock	2,78t/m3
Mined from:	
Pay ore Reserves	51 340 m2 @ 2 127 cm-g/t @ 106 cm
Not in Reserves	17 554 m2 @ 1 740 cm-g/t @ 101 cm
Unpay Block	13 620 m2 @ 745 cm-g/t @ 96 cm
Pillar mining	9 428 m2 @ 1 602 cm-g/t @ 105 cm
Stope reclamation	4 000 t @ 15,00 g/t
Waste from Other Sources	6 000 t
Gully Vamping	3 774 t @ 7,45 g/t
Gully waste	15000
Waste packed in stopes	13 770 t @ 1,34 g/t
Reef develpoment	12 000 t @ 4,72 g/t
Average dimensions of development end	1 000 m
advance @ 2,4 m wide and 1,8 m high	
Heagear Bins	
Beginning of the month	2 800 t @ 15,80 g/t
End of month	3 100 t
Surface Stockpile	
Beginning of the month	3 500 t @ 13,80 g/t
End of month	3 100 t
Surface Stockpile	
Beginning of the month	3 500 t @ 13,80 g/t
End of month	3 100 t

Source (Course notes on role of mineral evaluator in MRM by Cawood)

Table 2.3 Ore balance sheet

Category	%	Mass(t)	grade(g/t)	content(g)
Pay ore reserve		151289	20.066	3035770.783
+ Not in reserve		49288	17.228	849120
+ Unpay ore reserve		36349	7.760	282083.3854
Pillar mining		27520	15.257	419876.5714
+ Stope reclamalation		4000	15	60000
waste from other sources		6000		
Gully Vamping		3774	7.45	28116.3
Gully waste		15000		
Total broken in stopes		**293220**		**4674967.04**
sorted in stopes		13770	1.34	18451.8
Trammed from stopes		**279450**		**4656515.24**
+ Reef development		12000	4.72	56640
total trammed		**291450**		**4713155.24**
- Reef in ballast				
- Development ore sorted				
+/- Discrepancy		18350		
Hoisted as reef		309800	15.214	**4713155.24**
+ Shaft bins (beginning of month)		2800	15.8	44240
Total in shaft bins for the month		312600	15.219	**4757395.24**
- Shaft bins (end of month)		3000	15.219	45656.38
Total from shaft bins		309600	15.219	**4711738.86**
+stockpile (beginning)		3500	13.8	48300
Total		313100	15.203	4760038.86
- stockpile (end)		3100	15.203	47129.10
Ore to surface sorting plant		310000	15.203	4712909.76
- Sorted	9	27900	0.65	18135
+ Ore from reef picking plant		_		
To mill bins		282100	16.642	4694774.76
+ Mill bins beginning of month		7400	12.36	91464
Total in mill bins for the month		289500	16.533	4786238.76
-Mill bins (end of month)		9500	16.533	157061.38
- Tons milled				
+ Tons milled				
+ Slimes treated				
Total tons treated (gold called for)		280000	16.533	4629177.38
Mine Call factor	97.5			
Recovery				4360000
Residues			0.55	154000.00
Recovery plus residues (gold accounted for)				4514000.00

- The discrepancy is a shortfall of 18 350.0
- MCF is 97.5%
- The plant recovery is 96.6%

A typical metal balance sheet bears two sides namely the “budget/planned and the actual” (Storrar, 1982:226). The source of the metal that is called for is on the left side and the actual is on the right side. The surveyor measures area to calculate volume of rocks mined from various sources. These are converted to tonnes by applying the relative density factor of the rock for that particular area.

To determine the quality of the ore, there is sufficient sampling of all the areas mined and the grades are determined and expressed in terms of grams of gold per tonne of ore. The mass of the material constantly supplied to the plant is constantly checked by the use of electric weight meters (Figure 2.3) which measure the tonnage over specific lengths and time periods.

Figure 2.3 Electric weightmeter for mass measurement at plant (Iduapriem mine)

Source (Photo taken by researcher on Iduapreim Mine)

Janisch ore flow balance sheet is designed in such a way that at the left hand side of the sheet, survey measurements and figures are recorded in the forward direction and on the right hand side the figures are calculated backwards from recovery and residue data.

The ore balance sheet normally indicates the various sources of the rock which are usually from the ore reserve, not in reserve (NIR), development sections and other sources. This clearly indicates the tonnes generated by the mine which has a known metal content that is determined when sampling values are obtained. The metal content for the entire tonnes from the different sources are calculated by using a weighted average calculated. The quality and quantity of the ore sent to the stock pile and plant are also recorded in the ore balance sheet. A tonnage discrepancy known as the shortfall is determined by the difference between the tonnes sent to the plant and the tonnes generated by the mine and sent to the stockpile. Shortfalls are usually assigned zero metal content. The reasons for short falls based on an investigation undertaken by a group of mines according to Storrar (1981) are:

- Undermeasuring of stope areas and widths
- Using incorrect especially low rock densities for tonnage calculations
- Allowances for sag and scaling of walls of excavations of headings
- The tearing up of foot walls by scrapers
- The measuring of cross-sectional areas of headings
- Allowance for track ballast
- Estimation of waste packed under surface sorting spillage in drains

In surface mines, short falls can be as a result of:

- Underestimation of dilution
- Not recording losses at production areas, temporary stockpiles, spillages
- Incorrect truck tonnage factor
- Incorrect use of in situ and soil densities

The tonnage discrepancy can also be an excess when the surveyors calculated tonnage is greater than the tonnage accounted for by the plant. Tonnage discrepancies are usually shortfalls because of large parts of waste trammed as ore. It is therefore necessary to take steps to minimise such possible errors.

The metal recovered from the smelter is accurately accounted for by working backwards from the residue and recovery data. The mine call factor is then calculated based on the weighted average of the contribution of various sources of metal content carried through on both sides of the ore flow balance sheet. In the ideal situation, the MCF should be a 100% but in reality, it can be below 100% in one month and more than that in the next month. This is usually attributed to the fluctuations in the throughput velocity in the plant. If it is other reasons such as accuracy of measurements (mass measurements, sampling, analysis etc.) along the ore flow sheet, then it is a problem which needs to be dealt with as quickly as possible so as to satisfy the standards required for auditing purposes.

The mathematical illustration of the ore flow and its mineral content from the in-situ reserves to the final product in the smelt house according to (Janisch, 1973:237) is shown in table 2.4.

.

Table 2.4 Janisch ore flow balance sheet

Survey				Plant			
	Metric tons	Value (g/t)	Content(g)		Metric tons	Value	Content
Mined From:				Mill recovered	T_Z	V_Z	$C_Z=T_ZV_Z$
Ore Reserve	T_1	V_1	$C_1=T_1V_1$	Mill discard(residue)	T_R	V_R	$C_Z=T_RV_R$
NIR	T_2	V_2	$C_2= T_2V_2$	To mill	$T_m=T_R+T_Z$	$V_M=C_M/T_M$	$\mathbf{C_M=C_R+C_Z}$
Developmment	T_3	V_3	$C_3= T_3V_3$				
Other Sources	T_4	V_4	$C_4= T_4V_4$	from unmilled stockpile	T_a	V_a	$C_a=T_aV_a$
Hoisted	$T_L=T_1+T_2+T_3+T_4$	$V_L=C_L/T_L$	$C_L= C_1+C_2+C_3+C_4$	From HMS plant	$T_H=T_M-T_a$	$V_H=C_h/_H$	$C_H=C_m-C_a$
				HMS. * discard	T_b	V_b	$C_b=T_bV_b$
To Stock pile	T_5	V_5	$C_{5=}T_5V_5$	To HMS. Plant	$T_h+ T_b$		$C_H+ C_b$
Shortfall	$S=Tp-(T_L+T_5)$	zero	Zero	Hand - sorted waste	T_w	V_w	$C_w=T_wV_w$
To Plant	Tp	Vp=Cp/Tp	$\mathbf{Cp=C_L-C_5}$	To Plant	$Tp= T_h+T_b+_w$	$V=C/T_p$	$C=C_h+C_b+C_w$
				MCF=100 C_M/Cp			

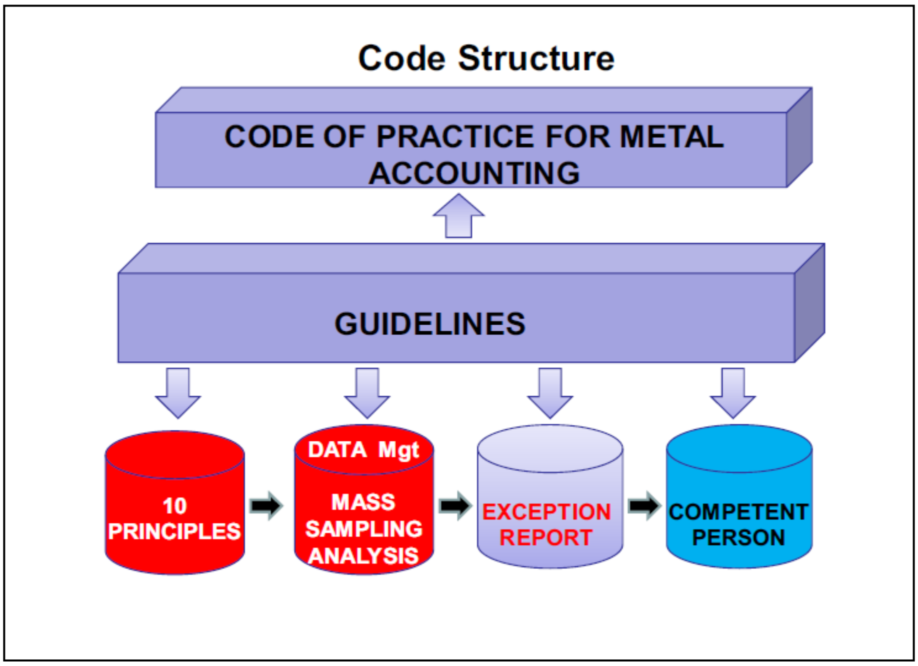
Source (Janisch, 1973:237)

2.2 AMIRA P754 code

Mass measurements and sampling are usually complex across the various measuring points and they need higher accuracy requirements in compiling useful ore flow balance sheet. Specialist equipment and advanced sampling and analysis techniques are also required.

The AMIRA P754 Code was developed similar to the SAMREC Code (for reporting of exploration results, mineral resources and mineral reserves) for metal accounting and reconciliation to solve the problem of a lack of standardization from mine to mill reconciliation. The primary aim was to provide auditability, transparency and good corporate governance (AMIRA International, 2005). It focuses on all activities within the areas of mass measurement, sampling, sampling preparation and analysis to be carried out accurately and with an acceptable level of precision. The code, however does not yet address the ore accounting in mines from the surface or underground operations as its starting point but rather focuses on the metal accounting in the metallurgical treatment plant. The structure of the code with ten guiding principles for reporting is illustrated by (Gaylard et al, 2009) and shown in Figure 2.4.

Figure 2.4 Structure of AMIRA code



Source (Gaylard et al, 2009)

The ten guiding principles for reporting are:

1. The account system must be based on accurate and precise measurements of mass and metal content and the system must be a full "check in – check out "system using the best practices defined in the code.
2. The source of all input data into the system must be clear, transparent and understood by all users of the system. The design of the system must incorporate the outcomes of a risk assessment of all aspects of the process of metal accounting.

3. The procedure for accounting must be well documented and user friendly for easy application by plant personnel and avoid the system from becoming one person dependent. It must incorporate clear controls and calculation procedures which are spelt out in the code.
4. There should be regular internal and external audits and reviews of the system to ensure compliance with all aspects of laid down procedure. Assessments of associated risks and recommendations must be included in these reviews and recommendations made.
5. In order to meet operational needs, accounting results must be made in time to facilitate corrective action or investigation. A competent person must sign off when the plan and resulting action is completed.
6. When provisional data is used in order to meet reporting deadlines such as month ends, clear procedures and levels of authorisation for replacement with provisional data should be laid down.
7. There must be sufficient data generated to allow for data verification, handling of metal/commodity transfers, reconciliation of metal/commodity balances, accuracy measurements and error detection, should not be bias.
8. The expected precision levels for metal recoveries, based on raw data, over a reporting period should be stated in the company's report to the audit committee. If there is any bias that is detected which is considered relevant to results, it should be reported to shareholders.
9. In – process inventory figures should be verified by stock takes at defined intervals. Stock adjustments require procedures and authority levels. Unaccounted losses or gains should also be clearly defined.

10. The system must be able to detect biases or errors which may occur as quickly as possible to remove them from measurements of mass, sampling and analyses, as soon as they are detected.

Reasons for a company or an operation's inability to comply with standards set out in the code must be reported in an exception report. Competent persons are however required for both internal and external audits of the entire metal accounting system.

The code however addresses metal accounting within the plant section and therefore mass measurements, sampling and analyses conducted on the mine level are not in the code. This poses a significant shortcoming because the probability of loss increases as one approaches the production area.

2.3 Mass measurements

The aim of mass measurements required for metal accounting purpose is to establish the mass of a material at a specific time or mass of material flow over defined time periods to defined accuracy suitable for mass and metal balancing. The purposes of mass measurements are classified into three areas namely (JKMRC, 2008:78):

- "Measurements required for custody transfer and Primary metal accounting e.g. measurement for input and output streams to plant operation. This measurement has to be of a high quality".
- "A measurement for Secondary metal accounting is for management control and includes internal company transfers and plant performance. It may also be used for cross checking primary accounting. Measurement for environmental monitoring also falls in this area".

- “Measurements for plant control only”

The accuracy requirement for primary accounting is the strictest, which requires a certified weighing equipment and the establishment of the distribution of random error (i.e. gross and systematic errors have been removed) model associated with the mass measurement.

2.4 Accuracy and precision of mass measurement

The terms accuracy and precision have been used interchangeably over time. Measurements are termed accurate if the average of a number of the same measurements is close to the true value which in most cases is unknown. Precise measurements on the other hand have a standard deviation about their mean value which is lower than a defined dispersion or a probability density distribution of a particular time (JKMRC, 2008).

Static measurements establish the “true” value by the use of calibration and certification which make use of test weights which have been certified for use in custody transfer applications and can be traced to national and international standards. The mass measurement equipment is adjusted so that it records this value (JKMRC, 2008). Figure 2.5 shows the mass measurement of a truck on a weighbridge at Iduapriem mine with a certificate of verification (Figure 2.6) showing an error or +/- 20kg.

Figure 2.5 Truck on weigh bridge (Iduapriem Mine)

Source (Picture taken by researcher on the Mine)

Figure 2.6 Certificate of verification (Iduapriem Mine)

Source (Picture taken by researcher on the mine)

There is a more complex calibration and certification requirement for mass measurement by conveyor belt weigher or by electromagnetic flow meter. Random and systematic (bias) errors are possible throughout the mass measuring chain. Bias errors are caused by factors such as the spillages on a weigh platform and can be reduced by good housekeeping (JKMRC, 2008). Illegitimate errors also arise due to equipment failure or mistakes in readings. These can be easily seen and eliminated. The objective in mass measurement for metal accounting is to reduce the random errors to minimum levels and to eliminate systematic errors. Specific examples of error sources are discussed further under “Mass measurements” in the book “Introduction to Metal Balancing and Reconciliation” published by JKMRC.

2.5 Measurement of density and moisture

A precise method of measuring density or bulk density is by laboratory tests whereby volumes of substances are accurately determined and weighed and the density calculated. A simple method of determining volume is the displacement of liquid (AMIRA International, 2005). In order to determine moisture, the weight of a material is determined on an electronic balance and then later on dried under defined conditions in an oven for a specific period of time and the loss in weight due to moisture is calculated. This is applicable to stockpiles, materials from conveyor belts and bulk ore. The degree of uncertainty depends on correct sampling and sample handling. The bulk density determination for Iduapriem mine is conducted by the rock and soil mechanical laboratory of the University of Mines and Technology (UMAT), Tarkwa based on the BS: 812 standard of density measurement. The calculations are based on “Archimedes’ Principle” (Figure 2.6). The result of the test conducted by researcher is shown in table 2.5

Figure 2.7 Insitu test procedure and calculation (BS: 812)

ROCK DENSITY TEST (ARCHIMEDES' PRINCIPLE)

INTRODUCTION

The density of rocks depends on a lot of factors including the composition of the mineral grains that constitute the rock, its origin, porosity and degree of weathering. The density of rock is used in the estimation of tonnages of material mined, blasted, hauled or sold. Very important decisions are based on the tonnage factor, that is, the tonnes per cubic metre of material, for example the amount of gold expected from a deposit. Since water contributes to the weight of rock, the dry density of the rock is used in these calculations. For other analyses like rock slope stability the bulk density of the rock is used. Universally the average density of rock is 2.7 tonnes/m^3 or 2700 kg/m^3.

APPARATUS

Waxing pot and laddle
Electronic balance
Thread
Hammer
Beaker
Distilled water
Paraffin wax
Oven

PROCEDURE

Break the rock core specimen into pieces weighing between 80 to 150g. For each sample given, prepare three specimens for the density test and extra three specimens for moisture content test. Label the specimens and weigh them on the electronic balance to the nearest 0.01g. Put the moisture content specimens into the oven for at least 20 hours.

Coat the density specimens with paraffin wax and weigh the waxed specimens. Pour some amount of distilled water into a beaker until it is about 2/3 full and place it on the electronic balance. Tare the weight (for the balance to read 0.00g). Tie a piece of thread around a specimen and gently lower it into the beaker until it is fully submerged but making sure it does not touch any part of the beaker. Take the reading on the electronic balance.

For the oven dried specimens, remove after 20 hours, cool them in a dessicator and weigh them.

CALCULATIONS

The method is based on Archimedes' principle. The specimen is coated to prevent water absorption. The density of the parafin wax is 911 kg/m^3.

Let the mass of rock specimen $= M_1$ g
Let the mass of waxed specimen $= M_2$ g
Let the mass of diplaced water $= M_3$ g

Since the density of water is 1g/cm^3, the mass of displaced water = volume of waxed sample.

Volume of waxed sample $= M_3$ cm^3
Mass of wax coating $= M_2 - M_1$
Volume of wax $= (M_2 - M_1)/0.911$

Volume of specimen, V $= M_3 - \left(\dfrac{M_2 - M_1}{0.911}\right)$ cm^3

Bulk density, ρ_b $= \dfrac{M_1}{M_3 - \left(\dfrac{M_2 - M_1}{0.911}\right)}$ tonnes/m^3

Moisture content, w $= \dfrac{\text{Mass of wet sample} - \text{Mass of dry sample}}{\text{Mass of dry sample}} \times 100\%$

Dry density of rock $= \dfrac{100\rho_b}{100 + w}$

Source (University of Mines and Technology, Tarkwa)

Table 2.5 Relative density test result of sample from Iduapriem mine

Specimen Number	1	2	3
Container Number			
Mass of container + Wet Specimen (g)			
Wet Specimen + Wax (g)	77.3	119.7	124.9
Mass of container (g)			
Mass of Specimen (g)	75.9	116.8	122.8
Mass of wax (g)	1.4	2.9	2.1
Water displaced (cm^3)	29.7	47.4	49.2
Volume of Wax	1.537	3.183	2.305
Volume of Specimen (cm^3)	28.163	44.217	46.895
Bulk density (t/m^3)	2.695	2.642	2.619
Average Bulk Density	2.652		

The bulk density test result (2.65 t/m^3) agrees with the value used in metal accounting (Refer to appendix A).

2.6 Sampling and analyses

All samples in a process stream must have the equal probability of being sampled. This requires a correctly designed mechanical sampler (Figure 2.7) which must take the sample whilst it is in motion. The accuracy of such a sampler must be confirmed with stop belt samples (AMIRA P754 code).

Figure 2.8 Belt sampler at plant (Iduapriem Mine)

Source (Picture by researcher on the Mine)

The code advises that the sampling of bins and stockpiles for metal accounting purposes are avoided because of the difficulty of obtaining accurate and representative samples.

2.7 Storage of samples and labelling

Before delivery to a laboratory for testing, all samples must be stored in suitable sealed containers to avoid significant changes in their physical or chemical parameters. This is very necessary as far as moisture content is concerned. Samples must be labelled uniquely so that there is no ambiguity in identifying samples (AMIRA International, 2005).

2.8 Sample preparation and analyses

The AMIRA P754 code further explains procedures for preparation of samples, moisture determination, reduction of samples ahead of sample division, sample mixing, sample storage after its preparation and analysis. The analysis for metallurgical accounting must be performed using extensive Quality assurance (QA) and Quality Control (QC) systems and checks. All analysis of samples must conform to methods approved by international standards such as the ISO standards.

2.9 Spangenbergs sampling checklist

Although it has not been adopted as a standard for sampling practice for Gold mining in Africa, (Spangenberg, 2012) developed a check list which can be used by each mine in the quest for correct sampling practices. Refer to Appendix B for details of the check list.

2.10 Survey compliance code

The Mineral Resource Management (MRM) department is responsible for measuring tonnages. In South Africa, the compliance code for mine surveyors is the South African Mine health and Safety act 29 of 1996 which gives regulations for the accuracy requirements for surveying, mapping and representations for mine plans. The code of practice developed by the Government of Australia under the department of mines and petroleum resources "provides guidance on meeting requirements in the Mines Safety and Inspection Act 1994 and Mines safety and Inspection Regulations 1995 relating to surveys at mining operations" (Government of Western Australia, 2013). These codes give accuracy requirements for distance and angular measurements depending on the type of instrument being used.

2.11 Reconciliation between planned and actual in the coal industry

The theory of metal accounting as previously discussed in the case of gold, is applicable to the coal industry. A major difference is that coal is referred to as a commodity. An extensive proportion of the deposit is directly shipped or processed into saleable material and then shipped. Cawood (2012) in his course notes "The role of the evaluator in the Mineral resource management" explains that the ultimate objective of reconciling between planned with actual in the coal industry is to have confidence in coal measurements, accounting and reporting process through:

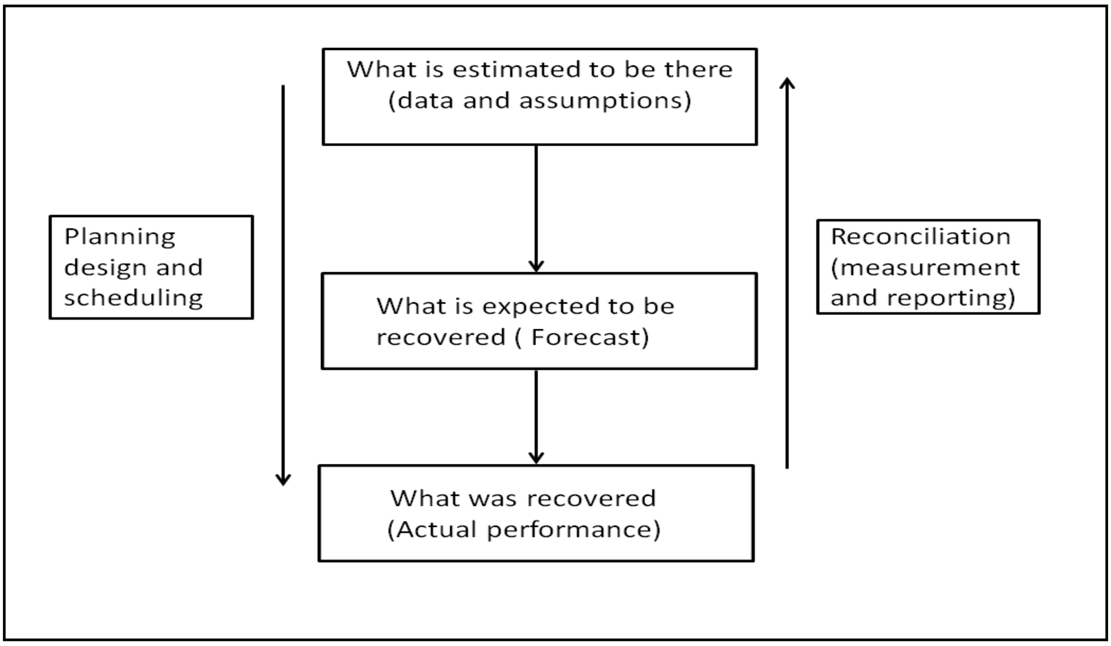
- Formalising the mine planning process,
- Promote integration between mining practice with broader business objectives, so that Cooperate objectives are not dissociated from actual mining practice,
- Integration amongst all departments ; Mine planning, Technical inputs from survey, Geology, Mining, Processing and Finance,
- Reconciliation of coal from the Mine Plan to customer.

(Clarkson et al, 2002) examined the entire reconciliation process in various sites and noted that very few sites managed to do a rigorous reconciliation of the predicted production schedule and actual production. The problem of calculating in-situ tonnes from volume and in situ RD was critical.

2.12 Mine planning and reconciliation process

The mine planning process involves the development of resource and reserve statements. The applications of the modifying factors (economic and marketing criteria) are used to detect possible mining areas and then blocks are scheduled with the coal quantities and qualities generated for each block. Modification factors are then applied to estimate Coal Resources and Reserves. These Reserves become the foundation of the Resource Development plan and provide a standard for reconciliation of mining recovery and utilisation of resource (Cawood, 2012). Figure 2.8 shows the basic planning and reconciliation cycle.

Figure 2.9 Basic planning and reconciliation cycle



Source (JKMRC, 2008: 520)

The reconciliation compares what was recovered with what was expected to be achieved so that the planning and production processes can be continually refined and improved. This appears to be basic but the entire reconciliation process can be a complicated one. Complications can come from “inadequate definition of reserves, inadequate monitoring of production performance, difficulties tracking coal through stockpiles and blends and inconsistencies between the time when measurements are taken and when results can be applied” Source (JKMRC, 2008:520). Highly sophisticated tools are developed and used for modelling, planning, measurements and reporting by the industry to tackle some of these challenges.

2.13 Statistical methods

There are numerous statistical methods applied in analytical laboratories, ranging from the analysis of variance method, to the method of applying the student’s t-test for comparing results from two analysts and the method of calculating precision. The procedure which is commonly applied is the “Shewhart Control graphs” and CUSUM (cumulative sum) graphs (JKMRC, 2008:198).

2.13.1 Shewhart (control) graph

This method is commonly used to analyse measurements in a QC or a certified reference material (also known as a standard reference material). The steps in making these graphs from (JKMRC, 2008:198) are:

- The accepted value(best value) of the measurement is fixed at the origin of the y-axis;
- x-axis is divided into integers to indicate the sequence of the analysis which in most cases is the date;

- Draw parallel horizontal lines above or below the x-axis at values or ±2 x standard deviations (SD) called “controlled limits” and ±3 x standard deviations (SD) also called “action limits”;
- The measured values are then plotted on the graph;
- Any result outside the control limit (±2 SD) should be reanalysed and result outside the action limit (± 3SD) is an indication of a major problem which should be immediately investigated;
- If a series of more than 5 sequential results are above or below the x-axis, there is a possibility of a biased situation.

2.13.2 Cusum graphs

Cumulative sum (CUSUM) graphs are used to monitor possible bias in analysed values of a quality control. The steps for plotting CUSUM graphs are (JKMRC, 2008: 200):

- The origin on the y-axis is set at the accepted value of the measurement and the x-axis is divided into integers to represent the sequence of the analysis or the date;
- The assigned value is subtracted from the measurement and the difference is plotted at an appropriate point on the graph;
- The subsequent differences are plotted in relation to the last point;
- A resulting series of positive values indicate a positive bias and the negative values indicate a negative bias

The formula for calculating Bias is given as [(Mean of received results/expected value)-1 expressed as a percentage]. According to the Iduapriem’s standards, for any particular batch or results to be accepted, the blank(waste material) and standards(known grade material) within the batch submitted to the laboratory should satisfy the following criteria:

- Standards should have a Mean grade ± 2 standard deviation
- Blank grades should be ≤ 0.02 g/t

2.14 Conclusion

A metal accounting balance sheet contains figures of mass measurements (tonnage), grade (g/t) and content expressed in grams. A typical ore flow balance sheet indicates the sources of metals. An indication of all the recoveries and losses along the flow are accurately captured in the flow sheet. A tonnage discrepancy can be a shortfall if tonnage sent to the plant is greater than the tonnage generated by the mine or the opposite will result in an excess. The measurements of these quantities and qualities are based on scientific measurements and analyses which are discussed into detail in codes such as the AMIRA P754 code. Other codes available for ensuring accurate measurements include the South African Mine health safety act 29 of 1996. A reconciliation and efficiency factor are used to identify discrepancies between planned and actual figures generated from what was expected to be recovered and what was recovered. The MCF is often used to predict future production discrepancy estimates, but is not a useful tool in identifying the causes of these discrepancies.

The use of Statistical concepts such as the Shewhart (control) graph and Cusum graphs are useful in determining discrepancies and analysing QC measurements.

The next chapter focuses on the fundamentals of ore accounting at Iduapriem mine. A discussion on the details of the behaviour of MCF since 2008 as seen in Fig 1.8 will be explained.

Chapter 3

3 Fundamentals of metal accounting at Iduapriem

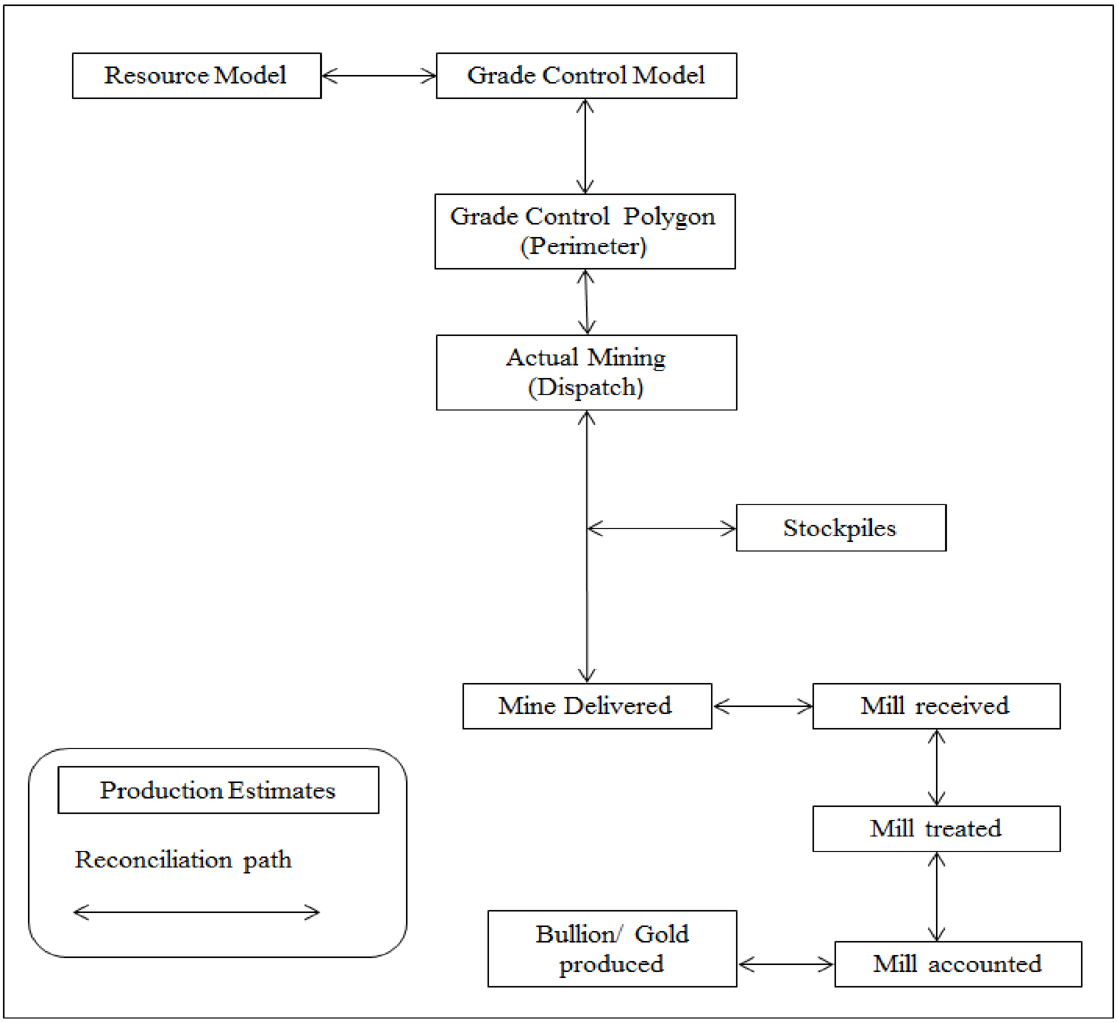
3.1 Introduction

The mine to mill reconciliation is done to compare production estimates from the mine's measurement systems to production estimates from the processing plant. The estimates from the mine include the surveyor's volume measured expressed in "bulk cubic meters" (BCM) mined, bulk density (RD) of the in-situ material expressed in tonnes per meter cube (t/m^3), and the in situ grade (grade control) expressed in grams per tonne (g/t). The tonnage factor is applied to the BCM to give the tonnage of ore and waste mined.

The total number of trucks removed from each bench is obtained from monthly statistical sheets and the number of trucks loaded from every pit is entered in the end of month pit volume spread sheet to give the average truck factor for each bench.

Reconciliation is a subset of metal accounting because it has to do with the measurement, determination and reporting of production figures over an accounting period. Figure 3.1 illustrates general mine-mill reconciliation from product to source.

Figure 3.1 Generic production reconciliation



Source (Developed by researcher on the Mine)

3.2 Mine to mill reconciliation background

The main objectives for performing mine to mill reconciliation are (JKMRC, 2008:448):

- "To obtain reliable production figures for the purpose of metal accounting reporting" and
- "To scrutinise the performance of quantity and quality estimates from resource and grade control models".

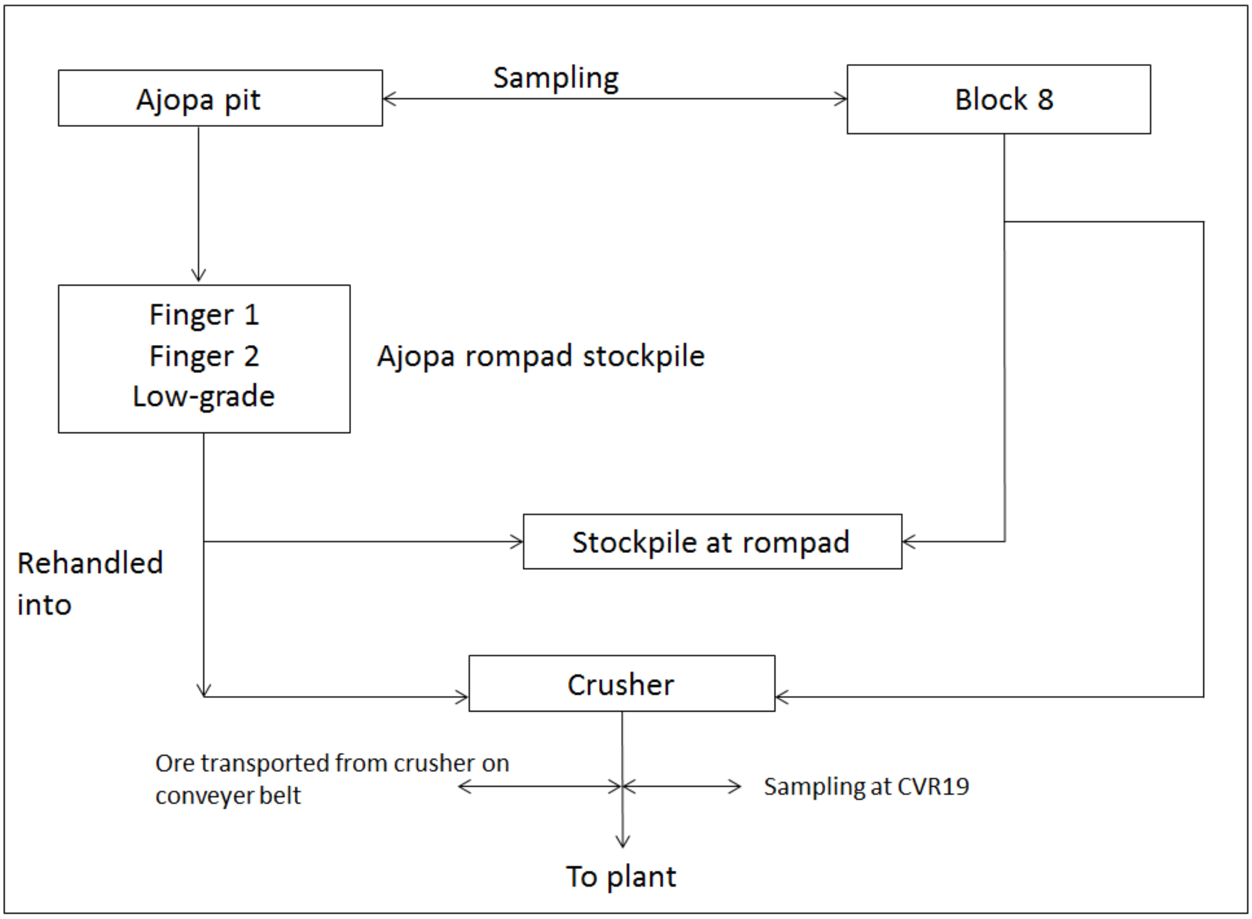
3.3 BCM measurement at Iduapriem mine

The month-end volume measurement is jointly conducted between Iduapriem and contractor (Moolman) surveyors. A print of the previous month's plan showing the status of the pit at the beginning of the month serves as a guide for identification of changes in the that current month. It is a good practice that the new survey pickup of a current month overlaps with the previous month's survey.

3.4 Survey tonnage and spot tonnage

The survey tonnage is determined based on the surveyors calculated volume and the spot tonnage is from the tally sheets compiled by a spotter. A truck tally load system of booking is used to record the movement of materials from the pit to various destinations. Figure 3.2 shows the schematic diagram of material movements at Iduapriem mine. Ore mined from two pits (Ajopa and Block 8) are sampled using the reverse circulation drilling method. Ajopa rompad stockpile contains ore from Ajopa pit denoted as fingers 1,2 and low grade. Some of this ore is re-handled into a stockpile at rompad close to the crusher while the rest is directly dumped into the crusher. The ore from block 8 is also stockpiled at rompad close to crusher or directly dumped into the crusher.

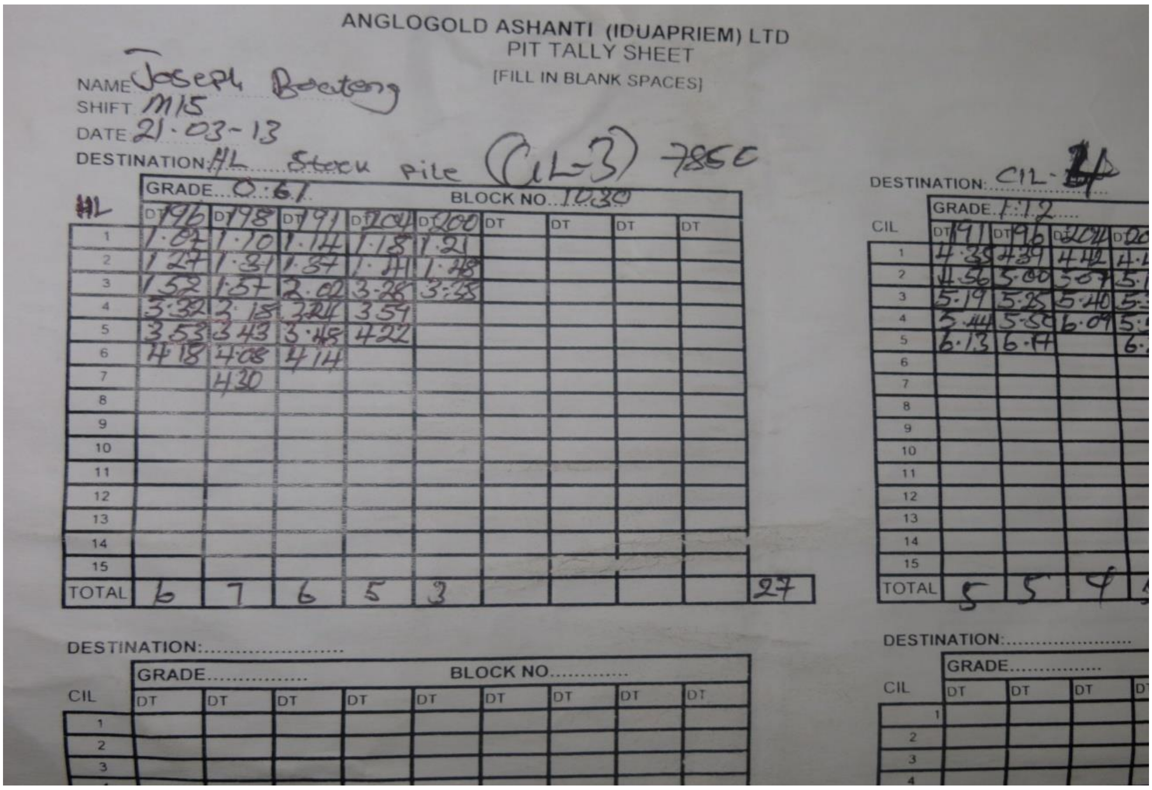
Figure 3.2 Material movement at Iduapriem



Source (developed by researcher on the mine)

The spotter counts the number of trucks for each mining block at the end of each shift and records the totals (figure 3.3). The shift geologists ensures all necessary information (pit, bench, block number, grade) is correctly entered on the tally sheet before entering into the database by the data management officer.

Figure 3.3 Pit tally sheet (Iduapriem mine)



Source (captured by researcher on Iduapriem Mine)

The spot tonnage will be computed based on the truck/bucket factor. The spot tonnage and the survey tonnage obtained from the survey digital terrain model (DTM) generated by the mine on monthly basis is expected to be the same but not always the same. This is due to human errors in entering the correct figures and operator inefficiencies. Table 3.1 shows the survey and spot tonnages over the period 2008 – 2012 at the mine. The average percentage ratio between the tonnes DTM and the tonnes spot in this period is 95%.

Table 3.1 Survey tonnage and spot tonnage

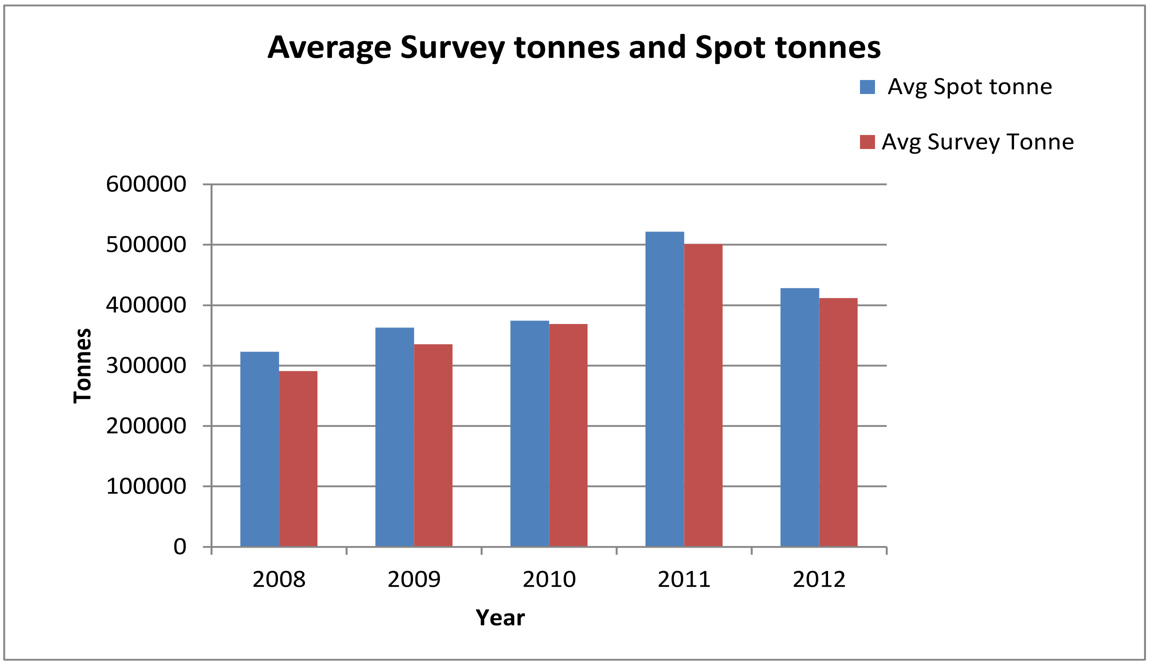
	tonnes DTM	grade 1		tonnes Spot	grade 2	TONNES DTM/TONNE SPOT
EOM JAN 2008	241710.00	1.66		255704.00	1.73	95%
EOM FEB 2008	*285009.00*	*1.91*		*278143.00*	*1.94*	102%
EOM MAR 2008	310083.00	1.84		332384.00	1.86	93%
EOM APR 2008	354642.00	1.71		383198.00	1.72	93%
EOM MAY 2008	220312.00	1.72		253541.06	1.68	87%
EOM JUNE 2008	320858.00	1.74		350419.00	1.76	92%
EOM JULY 2008	387613.00	1.87		466090.00	1.82	83%
EOM AUG 2008	184127.00	2.03		233814.00	1.84	79%
EOM SEPT 2008	232822.00	2.04		277872.00	1.88	84%
EOM OCT 2008	286472.75	1.82		299582.00	1.75	96%
EOM NOV 2008	331885.00	1.86		369141.00	1.66	90%
EOM DEC 2008	336687.00	2.00		373381.00	1.78	90%
EOM JAN 2009	166562.60	1.91		169979.00	1.71	98%
EOM FEB 2009	265677.00	1.95		308290.00	1.68	86%
EOM MAR 2009	454870.94	1.88		466850.50	1.67	97%
EOM APR 2009	325702.00	1.96		329425.00	1.73	99%
EOM MAY 2009	243491.00	1.87		285950.00	1.58	85%
EOM JUNE 2009	237155.00	1.64		240750.00	1.48	99%
EOM JULY 2009	259751.42	1.23		282272.00	1.11	92%
EOM AUG 2009	204420.80	1.11		227197.00	1.06	90%
EOM SEPT 2009	375373.10	1.73		422376.00	1.56	89%
EOM OCT 2009	656000.70	1.75		756492.00	1.65	87%
EOM NOV 2009	369342.53	1.74		372180.00	1.75	99%
EOM DEC 2009	464709.25	1.59		492033.00	1.63	94%
EOM JAN 2010	356306.86	1.58		346382.00	1.59	103%
EOM FEB 2010	250214.47	1.58		313828.00	1.61	80%
EOM MAR 2010	223569.29	1.65		207224.00	1.61	108%
EOM APR 2010	517173.00	1.57		560234.00	1.54	92%
EOM MAY 2010	585241.87	1.62		602591.00	1.60	97%
EOM JUNE 2010	609808.85	1.50		613243.00	1.49	99%
EOM JULY 2010	191165.60	1.74		192244.00	1.65	99%
EOM AUG 2010	33735.20	1.81		34631.00	1.74	97%
EOM SEPT 2010	326471.63	1.66		324480.00	1.61	101%
EOM OCT 2010	329214.38	1.62		336434.00	1.64	98%
EOM NOV 2010	567583.00	1.70		543909.00	1.66	104%
EOM DEC 2010	433924.75	1.56		418181.00	1.60	104%
EOM JAN 2011	788616.27	1.49		789175.00	1.48	100%
EOM FEB 2011	712386.00	1.52		713356.00	1.44	100%
EOM MAR 2011	676235.18	1.49		727933.00	1.45	93%
EOM APR 2011	396619.62	1.28		359676.00	1.23	110%
EOM MAY 2011						
EOM JUNE 2011	434374.70	1.21		405834.00	1.13	107%
EOM JULY 2011	183150.96	1.23		180877.00	1.22	101%
EOM AUG 2011	531095.25	1.23		631823.00	1.26	84%
EOM SEPT 2011	744311.95	1.46		735275.00	1.43	101%
EOM OCT 2011	296759.39	1.13		321550.00	1.00	92%
EOM NOV 2011	211255.83	1.33		346013.00	1.40	61%
EOM DEC 2011	535840.58	1.21		524031.00	1.22	102%
EOM JAN 2012	502809.50	0.90		659832.00	1.06	76%
EOM FEB 2012	574973.60	1.03		570978.00	1.02	101%
EOM MAR 2012	549825.90	1.03		570974.00	1.03	96%
EOM APR 2012	389624.00	1.09		437315.00	1.09	89%
EOM MAY 2012	461169.30	1.11		464356.00	1.08	99%
EOM JUNE 2012	382091.40	1.05		333849.00	1.04	114%
EOM JULY 2012	374737.10	1.26		420144.00	1.28	89%
EOM AUG 2012	343959.73	1.12		345014.00	1.13	100%
EOM SEPT 2012	378002.50	1.04		361334.00	1.18	105%
EOM OCT 2012	296759.39	1.13		321550.00	1.00	92%
EOM NOV 2012	417285.80	1.39		387910.00	1.39	108%
EOM DEC 2012	270778.10	1.51		266084.00	1.55	102%
ARERAGE						95%

Source (Iduapriem, EOM 2008-2012)

3.5 Spot and survey tonnage (DTM) analysis

A bar chart of the survey and spot tonnage is shown in Figure 3.4. The highest percentage ratio (98%) of survey over spot tonnage occurred in 2010 (the two tonnage measurements are nearly the same) and the lowest percentage ratio (90%) occurred in 2008.

Figure 3.4 Survey tonnage and Spot tonnage



Source (Iduapriem EOM 2008-2012)

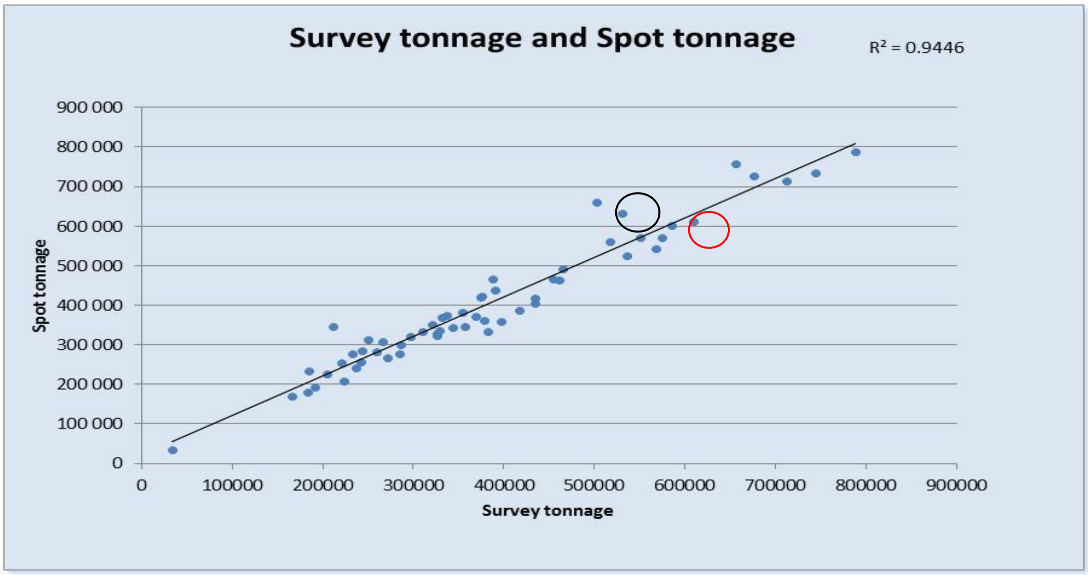
From Figure 3.4, the spot tonnages are consistently higher than the survey tonnage. The survey tonnage is a controlled measurement because it uses specialised instruments with accuracies as high as (1 mm + 2 ppm) whereas the spot system of tonnage measurement is not controlled in the sense that it is human dependent. Errors such as mistakes from spotters in doing tally sheets, as well as operator inefficiencies may arise which may not be automatically detected. The truck factor determined from scales at weighbridges has an accuracy of (±20kg).

It will therefore be appropriate that in cases where there are high discrepancies between spot and survey tonnage, survey tonnage is accepted because of the higher accuracies of survey equipment.

3.6 Correlation between survey and spot tonnage

The survey and spot tonnage show a positive linear relationship (Figure 3.5) and therefore the points which have totally deviated from the linear graph are likely to have been over or under estimated. The correlation coefficient (r), which measures the intensity of the linear relationship existing between the two variables, is equal to 0.97 which is almost equal to one (1) and therefore a very strong linear relationship exists between survey and spot tonnages.

Figure 3.5 Linear relationship between survey and spot tonnage



Source (developed by researcher based on Iduapriem data)

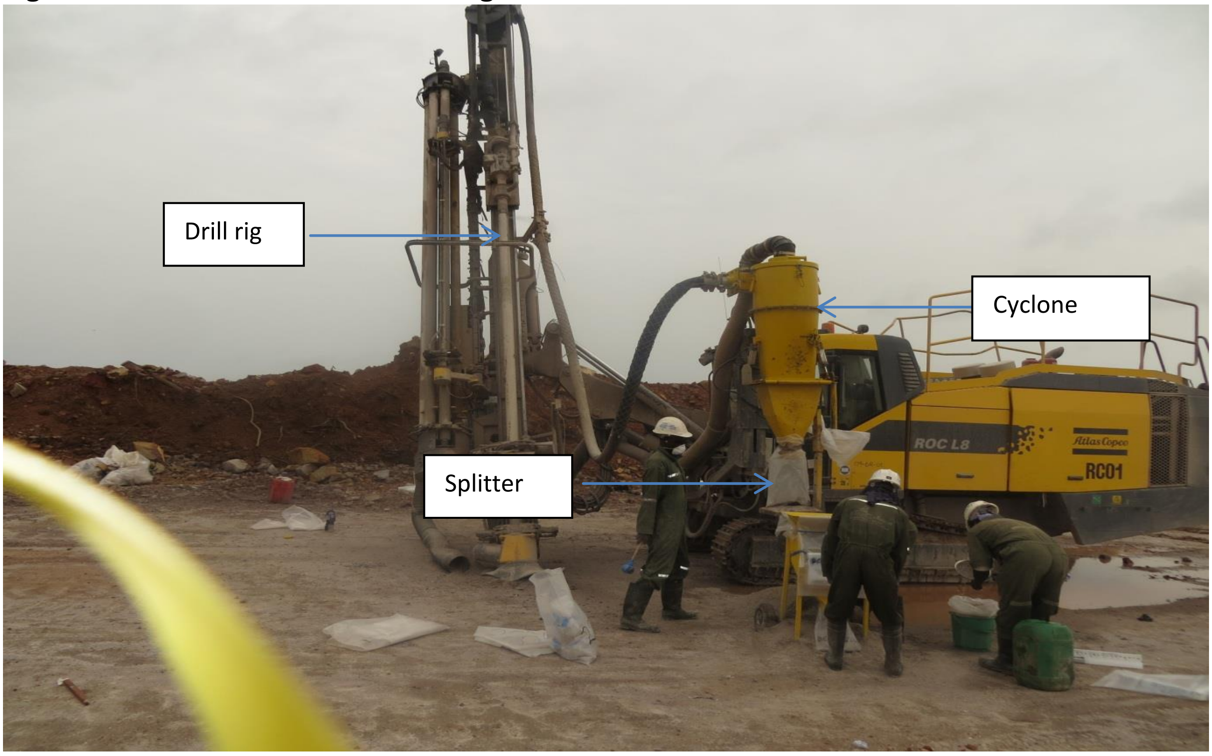
3.7 Conclusion

From the above correlation, it is seen that spot tallies can be a satisfactory indicator and therefore it is appropriate to do it as a check, but it is also proper to question the accuracy of the percentage margins that can be considered accurate for the purpose of metal accounting. The average of the survey over spot tonnes for the period 2008-2012 is 95% at a standard deviation of 9%. At 95% confidence interval the percentage ratio between the survey and spot tonnages should lie between an interval of 92% and 98% and therefore any percentage ratio which is outside this range raises questions for either measurement to be used for the purpose of metal accounting. The circled point (black) in Figure 3.8 has a DTM/spot ratio of 83% which is outside the confidence interval and the circled point (red) has a DTM/spot ratio of 97% which is within the confidence interval of 92% and 98%.

3.8 Grade control sampling

Reverse circulation (RC) method of drilling and chip sampling is used after mining the first two to three meters from the natural surface. The planning of the grade control drill-hole locations is based on geological information such as the Ore shapes of the previous lift, ore reserve block outlines and the current pit profile (Iduapriem, n.d). This information gives an idea of where the ore was on the previous bench and where it is likely to be on the next bench. The holes to be drilled are planned and marked out by the surveyors, indicating the reduced levels on the survey pegs. In order to prevent delimitation errors, the Reduced level (RL) on the pegs are noted. Delimitation error occurs when a dipping ore zone is not drilled at the correct reduced level (RL). Before drilling begins, the sampler ensures that the splitter, cyclone and the drilling rig are clean (Figure 3.6).

Figure 3.6 Reverse circulation drill rig



Source (Picture taken by researcher at Iduapriem Mine)

After drilling every 1m, the sampler removes the sample and places the next plastic bag under the splitter for assay sample collection. A classification system for naming samples is used to give each sample a unique identification which is based on the Pit or block, the bench or reduced level (RL) drilled to, the hole number and the sample interval (Iduapriem, n.d). All samples are then dispatched to the laboratory for assay tests.

3.9 Quality assurance and quality control (QAQC) analysis

This practice involves inserting quality control samples in the batches of grade control samples that are submitted to the laboratory for analysis. Best practices based on the mines QAQC protocol implementation include:

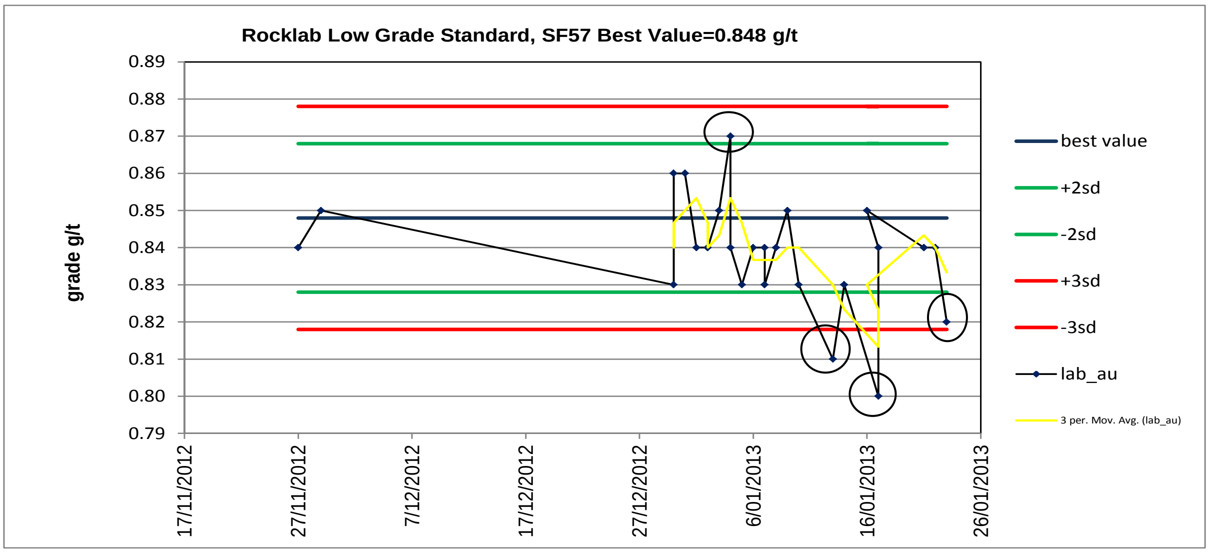
- Duplicates to check quality of sample splitting and the laboratory's precision;
- Standards to check the accuracy of the analytical laboratory;
- Retrieved pulps to check on the precision of the laboratory;

- Blanks to check on how effectively equipment is cleaned at the laboratory and also used to identify mixed sample numbers.

If QAQC protocol of a batch of samples fails, then it is the responsibility of the laboratory to reanalyse the entire batch at their own cost. Pulps of that batch are retrieved and re-submitted to the same laboratory with different sample names (Iduapriem, n.d).

The performance of the laboratory and quality of sampling is assessed for results of low, medium and high grade standards as per plots below.

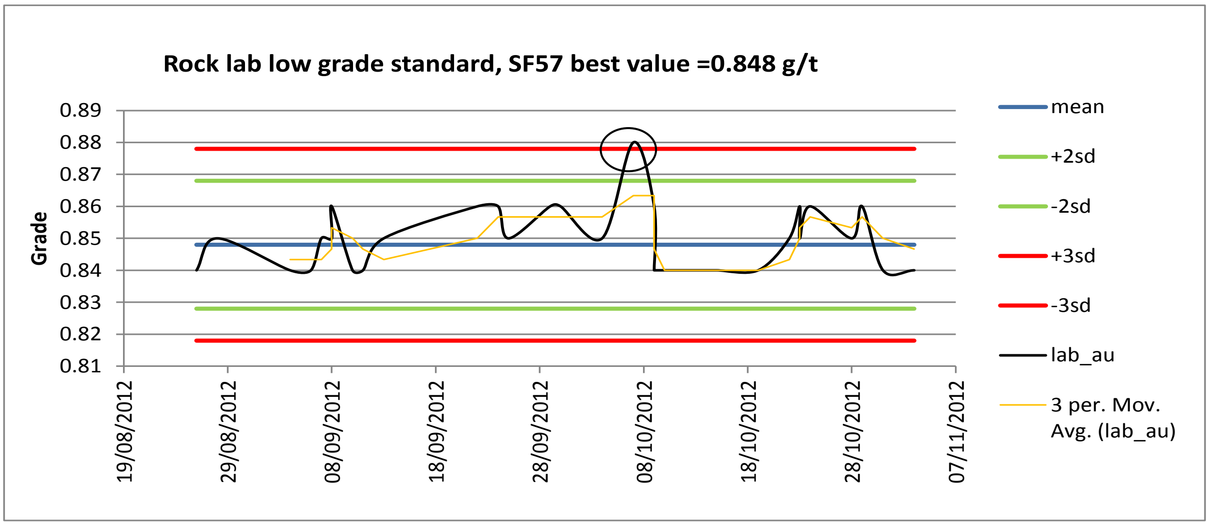
Figure 3.7 Analysis of low grade standards SF57)



Source ((Iduapriem, 2013))

In this result (Figure 3.7), the standard SF57 has an expected grade (best value) of 0.848 g/t and registered assay values of 0.87g/t > +2SD, 0.82g/t < -2SD, 0.81 g/t < -3SD, and 0.80 g/t < -3SD. A 3per moving average line provides trend information of a simple average over a time period which in this graph is consistently below the best value line. A calculated negative bias of (-1.07%) is recorded.

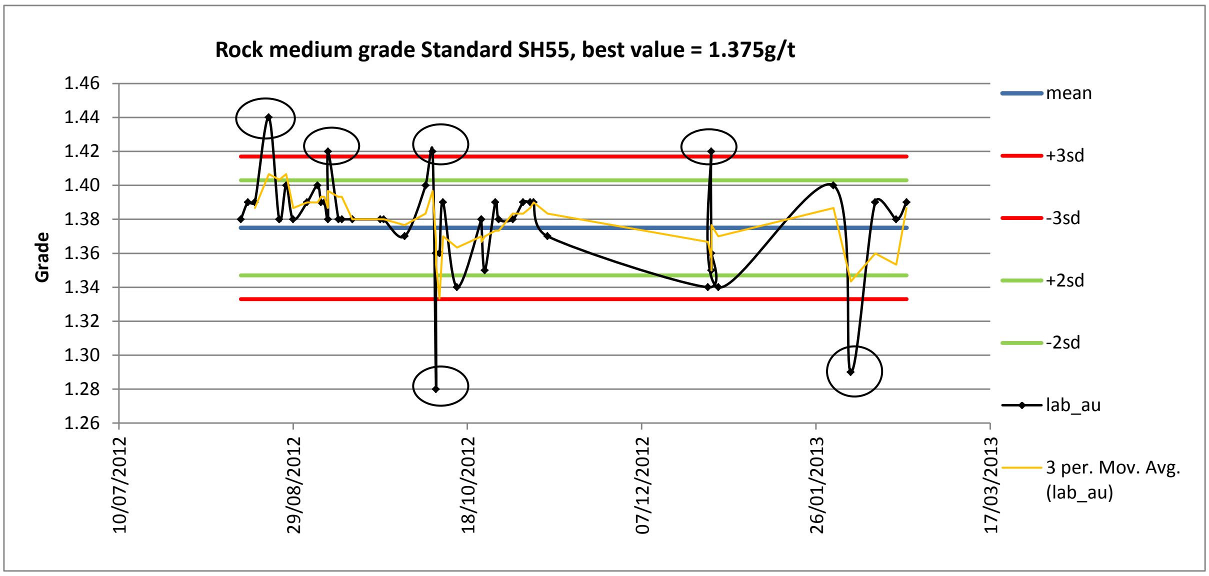
Figure 3.8 Analysis of high standard SF57



Source (Iduapriem , 2012)

Analysis of low grade standard SF57 (0.848 g/t) in figure 3.8 recorded an assay value of 0.88 > +3SD. There is no bias evident in the standard analysis of this batch and therefore the results can be accepted. The calculated bias over the period is 0.

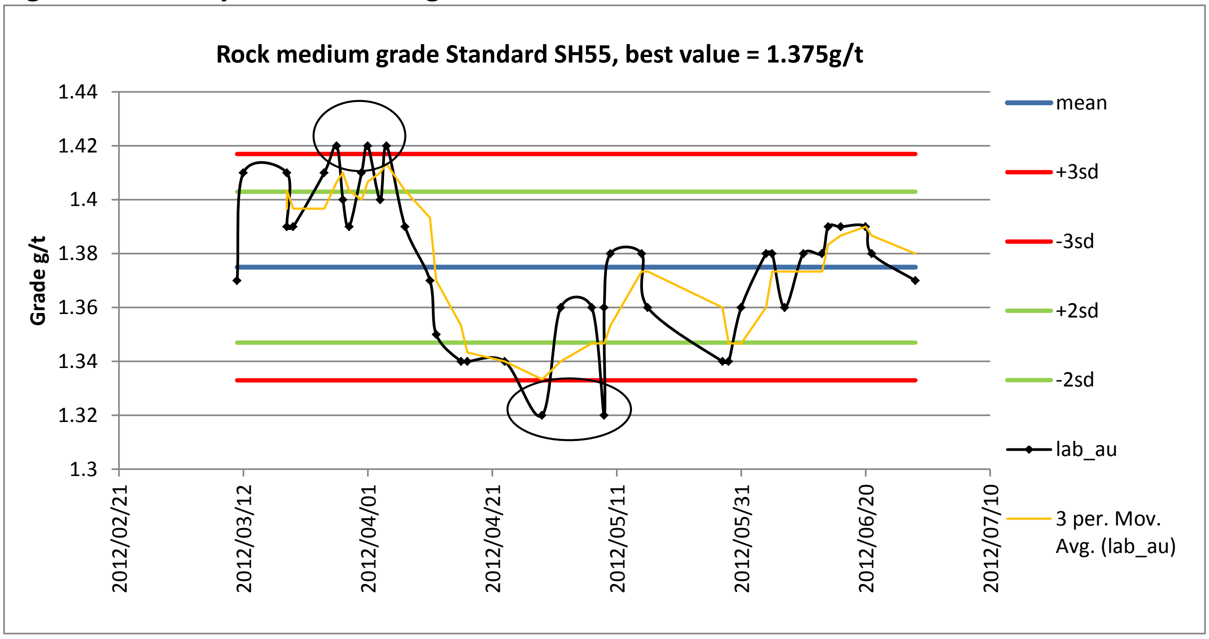
Figure 3.9 Analysis of medium grade standard SH55



Source (Iduapriem, 2013)

The analysis of medium grade standard SH55 (1.375g/t) from August 2012 to February 2013 in Figure 3.9 registered assay values of 1.44g/t ,1.42 g/t > +3SD and 1.28g/t, 1.29g/t <+3SD. The calculated bias is 0%. Since there is no bias evident in the standard of this batch, the results are acceptable for metal accounting.

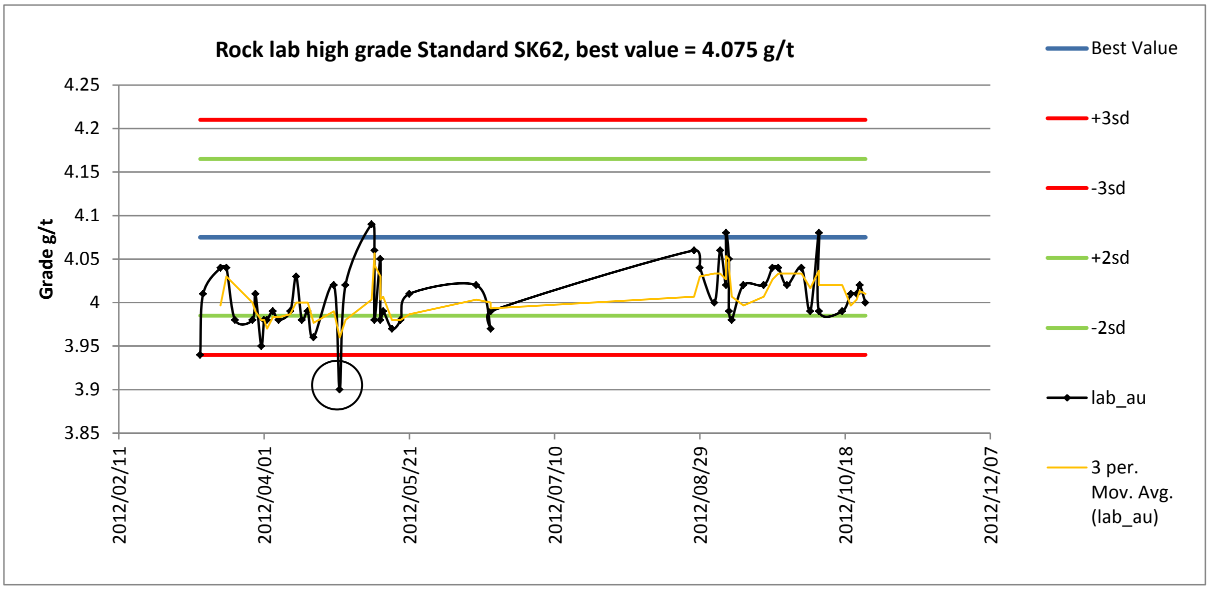
Figure 3.10 Analysis of medium grade standard SH55



Source (Iduapriem , 2012)

Standard SH55 (1.375 g/t) registered values of 1.42 > + 3SD and 1.32 < -3SD in Figure 3.10. The calculated positive bias over this period is 0.09%. Results of the batch of samples with this standard are therefore acceptable for purpose of metal accounting.

Figure 3.11 Analysis of high grade standard SK62



Source (Iduapriem , 2012)

The high grade standard Sk62 (4.075g/t) in figure 3.11 registered assay values of 3.9 g/t < -3SD. An evident and consistent negative bias (-1.66%) is seen and shown in the 3 per moving average which is consistently below the best value line, therefore the pulps of the batch of these samples should be reanalysed.

3.10 Blank analysis

Blank samples are materials which are usually devoid of gold. They are used to monitor contamination of laboratory equipment that is used during sample preparation and analysis. Blanks also show sample number mix ups (For e.g. If a blank has a value but a sample either side shows 0g/t it can be assumed there is a mix-up). Blanks come in two size fractions namely course and pulp (Operational control procedure, 2008).

3.11 Course blanks

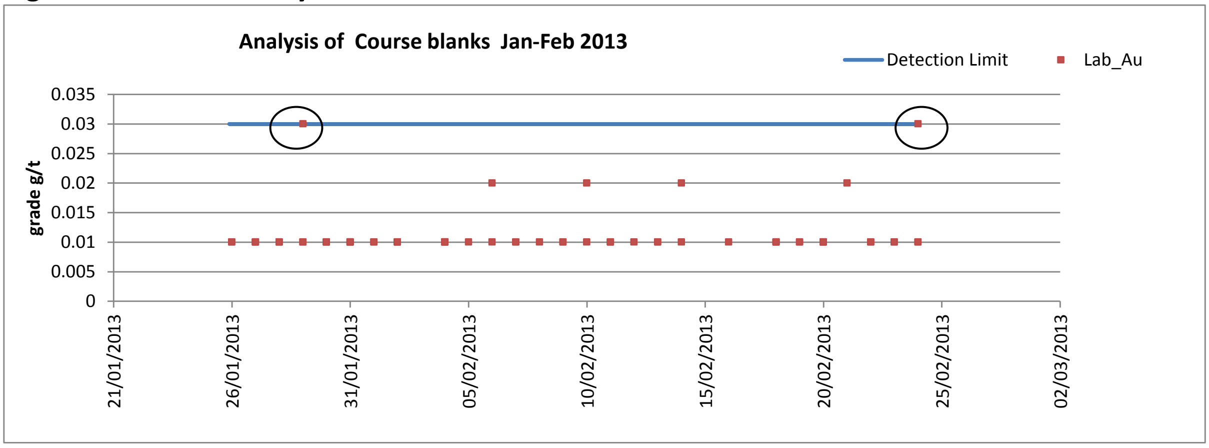
Course blanks is made up of small size barren material. It should be hard and of mesh-size to make it possible for grinding of contaminated material from initial samples which may have adhered to the equipment during previous sample preparation. Course blanks are usually inserted in a batch of samples after an expected ore grade sample.

3.12 Pulp blanks

Pulp blanks consist of barren crushed material (fine material). The purpose of pulp blanks is to test for contamination of laboratory equipment after preparation stage. Pulp blanks are inserted after ore grade sample.

According to Iduapriem's grade control manual, Blanks (course or pulp) grades should be ≤ 0.02g/t for a batch of samples to be accepted. The blank graphs are analysed as per plots below.

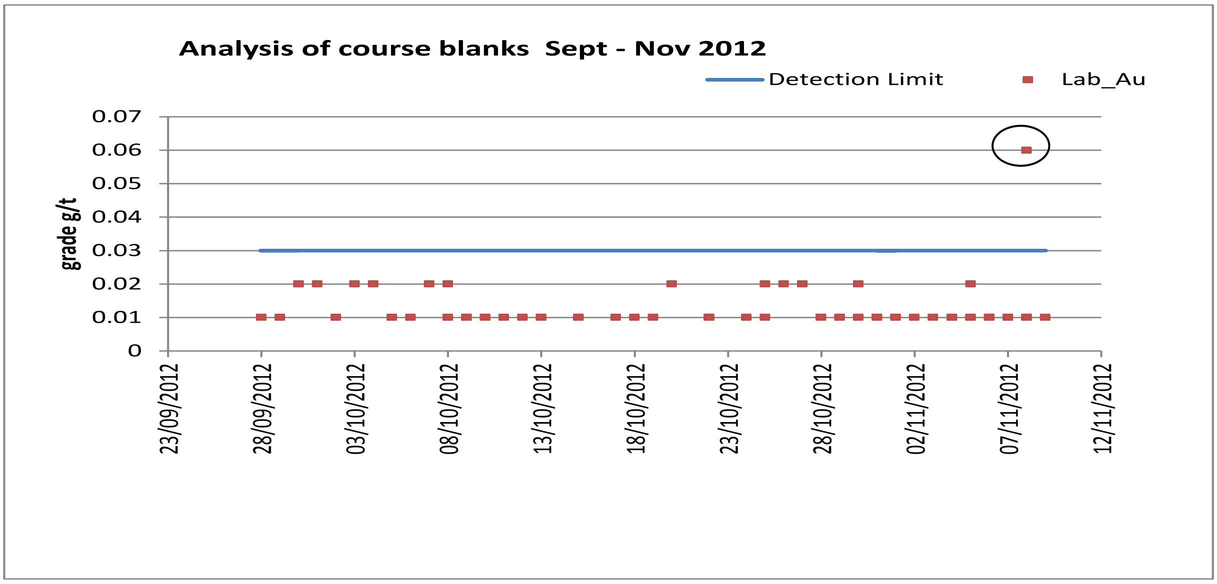
Figure 3.12 Blank analysis Jan- Feb 2013



Source (Iduapriem , 2013)

The course blank analysis in figure 3.12 shows two blank samples that recorded grades 0.03g/t which is > 0.02 g/t. The remaining samples are all below the detection limit of 0.03 g/t. According the mines QA/QC protocol all blank grade should be ≤ 0.02 g/t, but the current practise on the mine, shows that course grade have a detection limit of 0.03 g/t.

Figure 3.13 Analysis of course blanks Sept - Nov 2012



Source (Iduapriem , 2012)

Course blank analysis in Figure 3.13 recorded an assay value of 0.06 g/t which exceeded the detection limit (0.03g/t). This is because of contamination of laboratory equipment from previous sample preparation.

3.13 Pulp re-submits

Another method used to check the reliability of the laboratory's assay results apart from the use of standards and blanks is the use of a check laboratory. This is done by retrieving a sample that has already been checked by an original laboratory, and resubmitting to another laboratory. The results from the two laboratories are then compared (Operational control procedure, 2008).

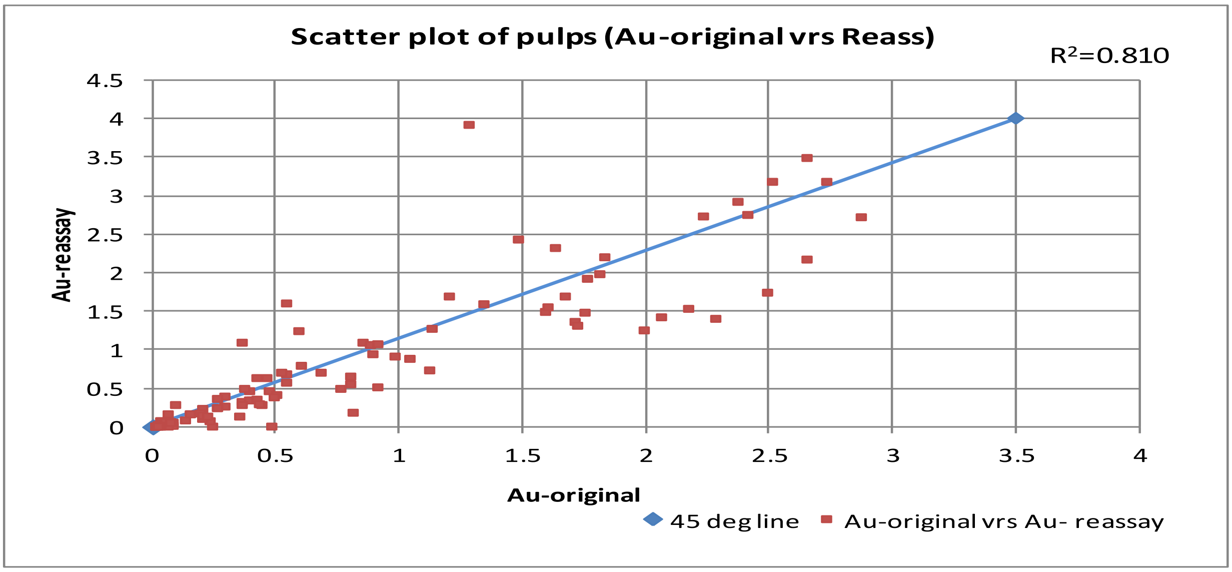
All the quality control measures discussed earlier (insertion of standards, course and pulp blanks) should be submitted with the pulp re-submits as a check on the accuracy on the second laboratory.

3.14 Pulp re-submit analysis

When results from the pulp re-submitted to the secondary laboratory are compared to results of the original laboratory in scatter plots, it should show a 100% correlation line (45^{o} line) starting from the origin, the calculated correlation co-efficient for the data, the average of the original result, the average of the re-submitted pulp result, and the percentage difference between the two averages (Operational control procedure, 2008).

For the data comparison, a correlation coefficient and the percentage difference between the two means should also be calculated. The tolerance for correlation coefficient and the percentage difference between the means will be further discussed and determined in the fourth chapter towards a metal accounting protocol.

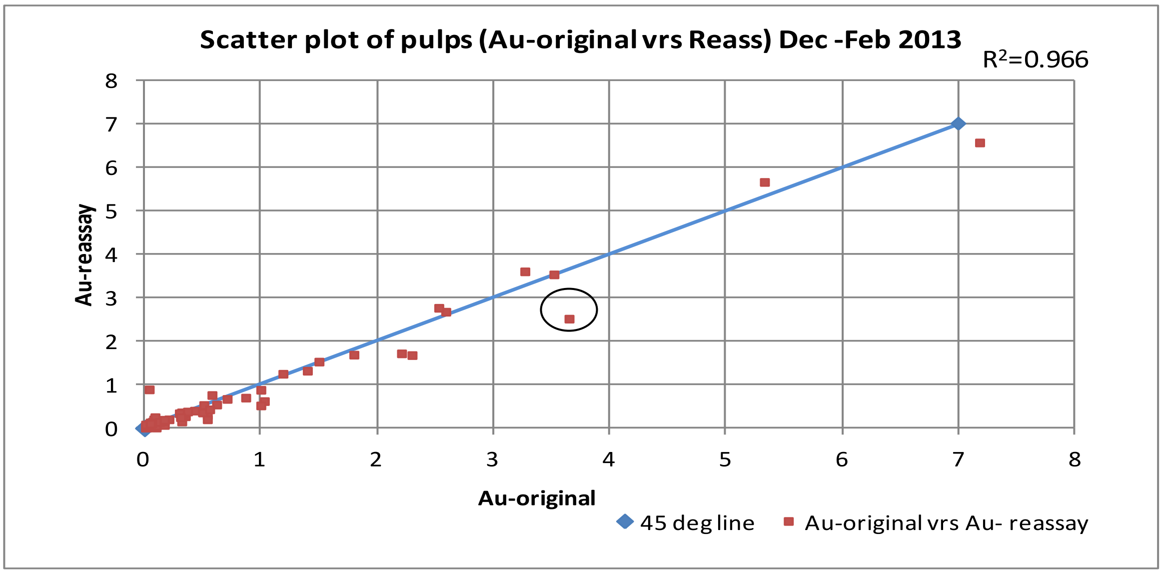
Figure 3.14 Scatter plots pulp (Au original versus Au-re-assay)



Source (Iduapriem , 2013)

The scatter plot of pulp re-analysis in figure 3.14 shows a correlation 0.9 between the Au-original assay results and the pulp re-assay results from the secondary laboratory. The percentage difference between the two means was calculated as (5.09 %).

Figure 3.15 Scatter plot of pulps (Au-original verus re-assay) Dec-Feb 2013



Source (Iduapriem , 2013)

From figure 3.15 above, the scatter plots of pulp (Au-original versus Au re-assay) show a stronger correlation (0.966). The percentage difference between the two results (Au-original versus re-assay) is 5.5%.

At 90 % confidence interval, with an average difference between the two results of 0.04 g/t and a standard deviation of 0.24 g/t, the difference between the two results should be between (0.10g/t and 0.01g/t). The point circled in Figure 3.15 has a difference between the Au-original and re-assay results of 1.13g/t which is outside this confidence interval.

3.15 QAQC conclusion

According to the mine's standard for accepting a batch of assay results, "standards should have a mean grade of ± 2 SD and blanks should be ≤ 0.02g/t". Figures 3.10 and Fig 3.11 showed some measurements exceeded the +3SD. The low grade standard SF57 from Jan – Feb 2013 shows a negative bias. The 3 per moving average trend line is consistently below the best value line. The same standard analysed over the period September to November 2012 does not show any bias (Figure 3.11).

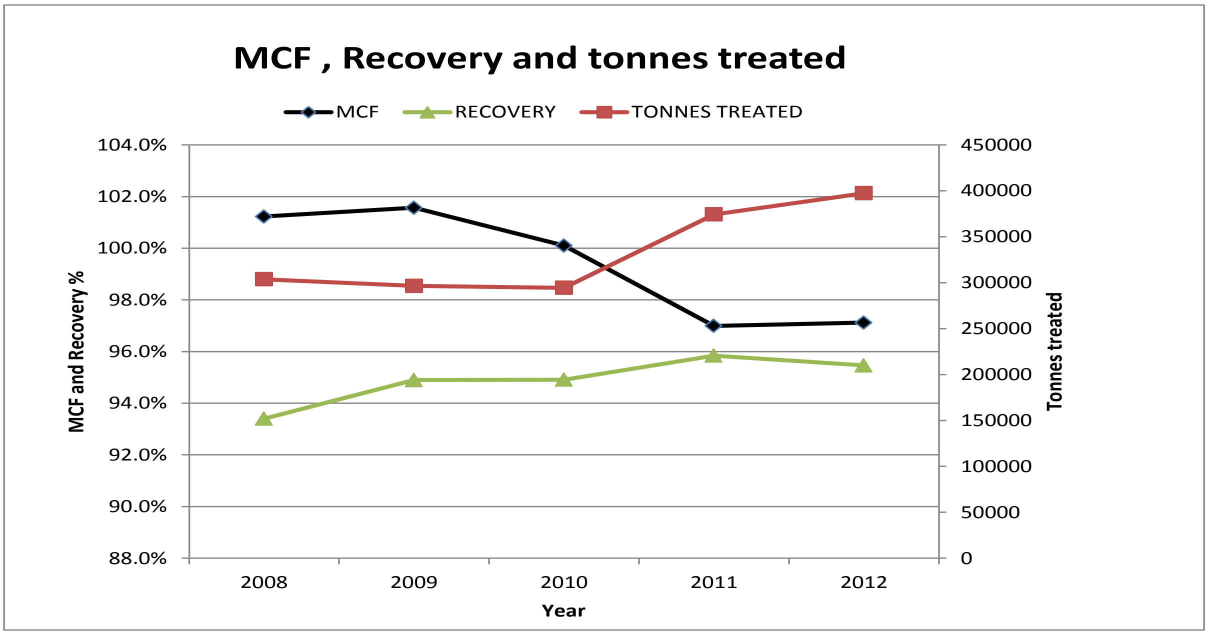
Although the mine's QAQC protocol say all blanks should be ≤ 0.02g/t, the current practice shows that course blanks must not exceed a limit of 0.03 g/t and pulp blanks should not exceed a limit of 0.02g/t. Course blanks analysed in Figure 3.15 meet the requirement, and therefore it can be concluded that there was no contamination of laboratory equipment during sample preparation and analysis however the same cannot be said in the case of Figure 3.16 because one course blank assay registered a value of 0.06 g/t which is > 0.03g/t.

The true value of a measurement is usually not known, but a confidence interval can be established using the standard deviation and the average calculated for a large number of samples to determine what range of values are accurate.

3.16 MCF recovery and tonnes treated

The MCF at Iduapriem Mine ranges from 102% to 97% (Figure 3.16) over the period 2008-2012. Although recovery and tonnes treated in the plant increased in this period, it did not impact on the MCF. The "gold called for component" of the MCF is determined based on the plant feed tonnes and feed grade (grade control) and the "gold accounted" for is based on the gold produced and the plant recovery factor.

Figure 3.16 MCF, Recovery and Tonnes treated

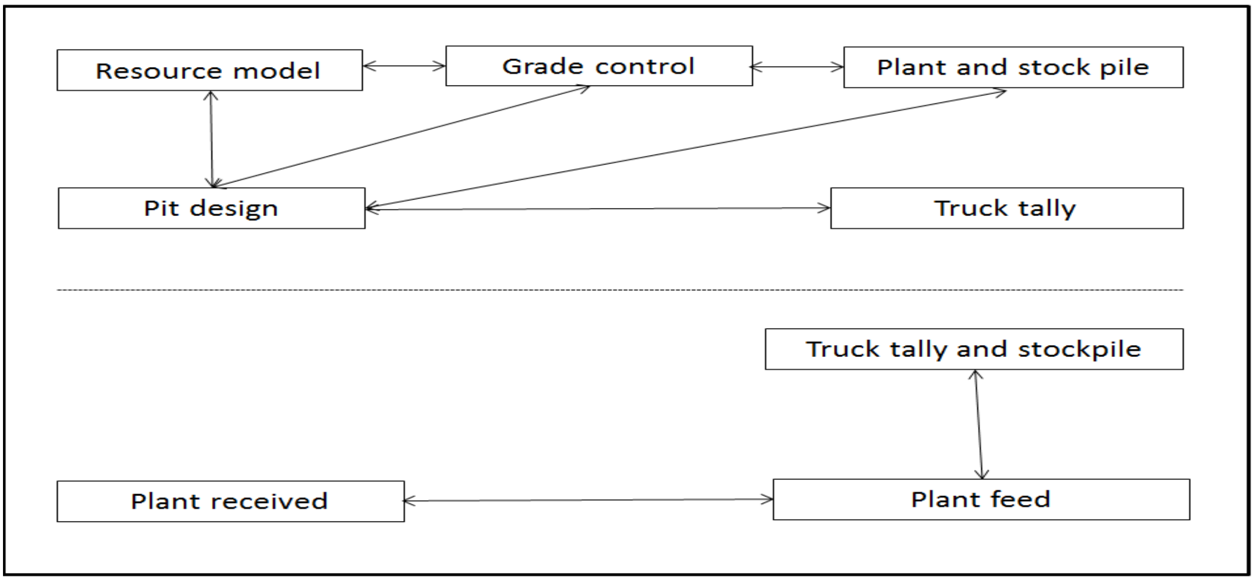


Source (Anglogold Ashanti, 2013)

3.17 Mine to mill reconciliation

Mine to mill reconciliation compares production estimates from various sources (resource model, grade control model, pit design, plant and stockpile, truck tally, stock pile and plant feed, plant feed, and plant received) at the mine over a period (Figure 3.21).

Figure 3.17 Current mine-mill reconciliation (Iduapriem Mine)



Source (Developed by researcher based on current reconciliation practice)

Measurement points are established between two points in Figure 3.17 and a reconciliation factor is calculated. The dashed line indicates a disconnection between the “plant feed and plant received” to the source (resource model) in the current reconciliation strategy. The mine to mill reconciliation data at Iduapriem is statistically analysed and discussed from the period 2009-2010.

The reconciliation factor (expressed as a percentage) calculated in Figure 3.18 below compares the quantity and quality estimates between grade control model and the resource model. The total reconciliation factors calculated in terms of quantity, quality and ounces are 94%, 108% and 101% respectively.

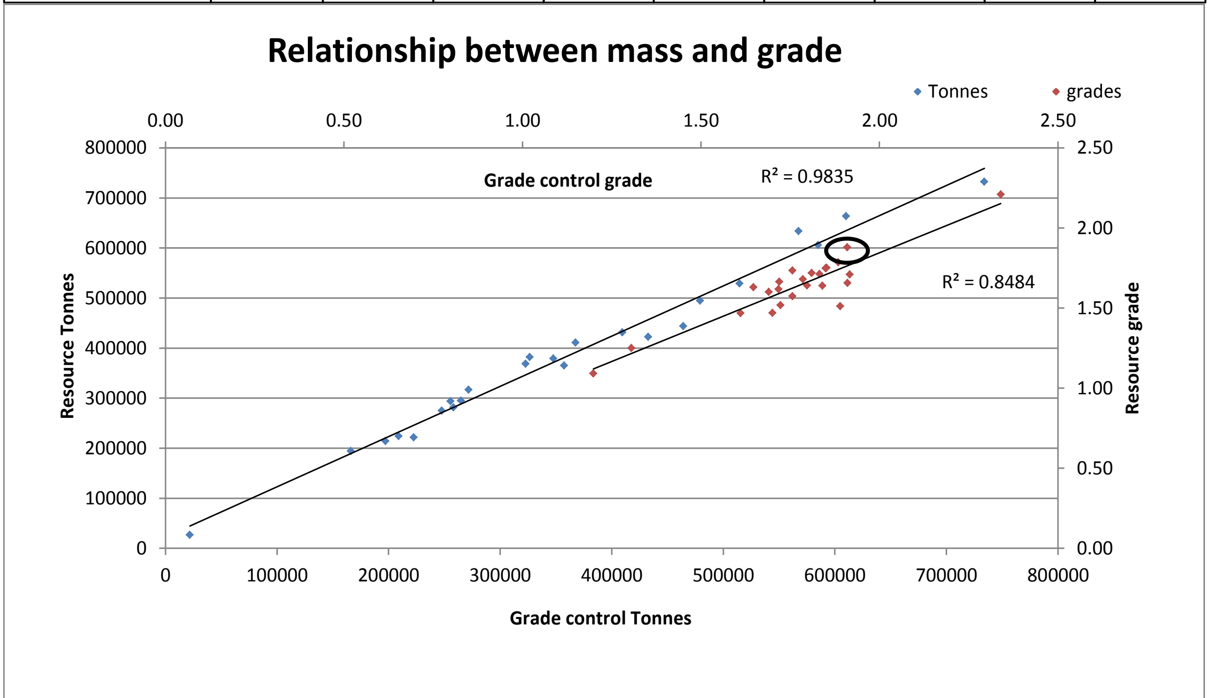
The average reconciliation factor in terms of quantity over this period is 93% at a standard deviation of 6% and average grade (quality) reconciliation factor is 108% at a standard deviation of 6%. At 95 % confidence intervals, the grade reconciliation factor is between 106% and 111%.

The graph in Figure 3.18 shows a stronger correlation for tonnages (0.99) than grades (0.92) between the measuring point “grade control and resource model”. The point circled in the same figure has a grade reconciliation factor of 125% which totally deviates from the straight line, and outside the 95% confidence interval of 106% and 111%. The biggest grade discrepancy between the two models occurred in the month of July 2010.

From the same Figure 3.18 it is seen that there is more confidence in mass measurement compared to grade estimation and therefore grade estimations for especially resource model requires further investigation.

Figure 3.18 Reconciliation between mass and grade (grade control versus resource model)

GRADE CONTROL Versus RESOURCE RECONCILIATION - 2009 & 2010									
MONTH	GRADE CONTROL			RESOURCE			RECONCILIATION FACTORS		
				m7			(%)		
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
JAN 09	166134	1.91	10201.94	194309	1.88	11744.69	85%	102%	87%
FEB 09	264900	1.92	16325.2	294821	1.71	16208.59	90%	112%	101%
MAR 09	479133	1.85	28528.38	494701	1.75	27860.74	97%	106%	102%
APR 09	347613	1.88	21062.24	379302	1.79	21784.98	92%	105%	97%
MAY 09	258141	1.81	15024.76	281328	1.72	15553.04	92%	105%	97%
JUN 09	255475	1.72	14153.42	293871	1.52	14340.19	87%	114%	99%
JUL 09	271645	1.31	11399.99	317031	1.25	12750.75	86%	104%	89%
AUG 09	209082	1.20	8058.298	224290	1.09	7873.425	93%	110%	102%
SEP 09	409498	1.80	23663.14	431686	1.64	22784.54	95%	109%	104%
OCT 09	734061	1.85	43646.55	732762	1.75	41198.98	100%	106%	106%
NOV 09	367505	1.91	22576.68	410884	1.66	21893.89	89%	115%	103%
DEC 09	464245	1.72	25661.71	444023	1.66	23755.67	105%	103%	108%
JAN 10	357378	1.70	19538.97	365382	1.47	17259.93	98%	116%	113%
FEB 10	247638	1.72	13672.17	275195	1.62	14319.88	90%	106%	95%
MAR 10	222457	1.84	13161.7	221644	1.64	11683.95	100%	112%	113%
APR 10	514600	1.69	27959.93	529056	1.60	27221.58	97%	106%	103%
MAY 10	584872	1.76	33020.92	606080	1.57	30664.06	97%	112%	108%
JUN 10	610114	1.61	31593.43	663867	1.47	31330.43	92%	110%	101%
JUL 10	197177	1.89	11983.66	214021	1.51	10400.38	92%	125%	115%
AUG 10	21754	2.34	1636.612	26952	2.21	1915.023	81%	106%	85%
SEP 10	326471	1.79	18741.79	382069	1.68	20641.87	85%	106%	91%
OCT 10	322835	1.76	18227.4	368571	1.74	20560.36	88%	101%	89%
NOV 10	567583	1.83	33438.57	633692	1.71	34906.6	90%	107%	96%
DEC 10	432780	1.65	22918.97	422577	1.63	22152.85	102%	101%	103%
TOTAL/AVERAGE	8633091	1.75	486196.4	9208114	1.62	480806.4	94%	108%	101%



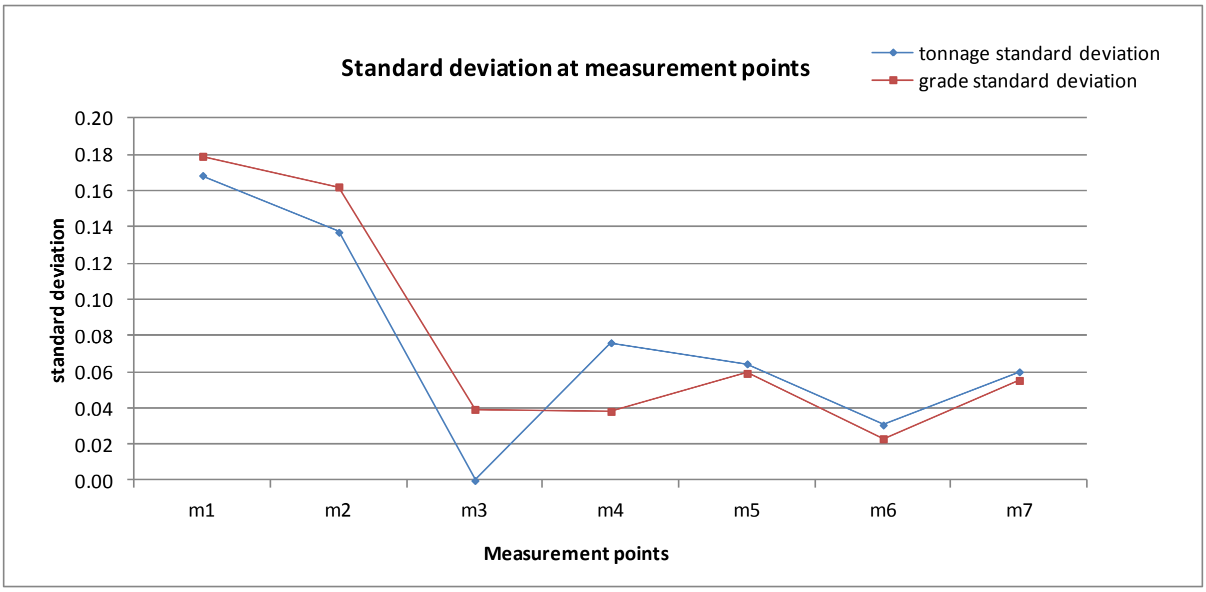
Source (Iduapriem, 2010)

The standard deviations along the measurement points in Table 3.2 for both quality and quantity are shown in Figure 3.19. The highest variance of grade and tonnage occurred between the plant and stockpile versus perimeter design. The lowest grade and tonnage variance occurred between the grade control and perimeter design measurement point.

Table 3.2 Measuring point

Measurement point		
m1	PLANT & STOCKPILE	PERIMETER DESIGN
m2	RESOURCE	PLANT & STOCKPILE
m3	PLANT FEED (Grade Control)	PLANT RECEIVED (Metallurgy)
m4	PERIMETER DESIGN	TRUCK TALLY
m5	RESOURCE	PERIMETER DESIGN
m6	GRADE CONTROL	PERIMETER DESIGN
m7	GRADE CONTROL	RESOURCE

Figure 3.19 Discrepancy at the measuring points



3.18 Conclusion

The biggest grade and tonnage discrepancy exists between the “plant and stockpile” versus perimeter design reconciliation with an average grade standard deviation of 18% and a tonnage standard deviation of 17%.

There is more confidence in mass measurement as compared to grade and the researcher suggests that that grade estimated for resource model as well as “plant and stockpile” grades requires further investigation. A metal accounting protocol is therefore necessary in this regard to be able to establish compliance before any investigation is done.

Currently there is no documented procedure on mine - mill reconciliation at Iduapriem mine and therefore the fourth chapter will work “towards a metal accounting protocol” for the mine. The protocol will highlight on considerable accuracy requirements for tonnage measurements within the mining department.

It is recommended that the current QAQC protocol on the mine should be further expanded to include the various quality control methods and analysis being practiced on the mine.

Chapter 4

4 Towards a metal accounting protocol

4.1 Introduction

The principal objective of mass measurement for metal accounting is to establish the mass of a particular material at a specific time or the flow of material over a defined time period. The determination of mass has to be done to a defined accuracy that is suitable for mass and metal balancing.

In this chapter, “mass” should be read as “mass or volume”, because of practical purposes at the mining department where often volume is measured (e.g. in the case of stockpiles and end of month volume calculation). This then requires the bulk density of “stockpile material” and relative density to be determined by appropriate methods in order to calculate mass of the material of interest. Materials often contain moisture and therefore the proportion of moisture content must be established to obtain dry mass. Significant errors in mass measurements are often incurred from density and moisture measurements and therefore procedures for these measurements can be adopted from the AMIRA protocol.

4.2 Types of errors in mass measurement

There are mainly two types of errors in mass measurements namely random and systematic errors (or bias). An equipment failure or wrong readings will result in gross errors which are also referred as illegitimate errors. These error types are easily identified and removed. In metal accounting, the objective is to reduce the amount of random errors within acceptable limits i.e. (improve the measurement’s precision) and ensure that bias is eliminated. Some basic statistical concepts to establish precision and bias for mass measurements will be useful in defining some error criteria.

4.3 Survey measurement

The scope of the current protocol on survey procedure for end of month measurements highlights the various steps needed to be followed in this procedure. This includes the field work, update of plans, calculation of DTMS, how to treat over spill materials, truck factors, and drilled and blasted volumes. All these issues are relevant in calculating volumes which will later on be converted to tonnages by applying a relative density or bulk density factor. The density determination is also possible source of error. In cases of stockpiles for e.g. measuring the volume accurately is of a higher probability due to modern survey techniques , however it is rather difficult to determine the in-situ bulk density and even more difficult to obtain a representative sample for moisture determination and analysis. This is even worse when there is no homogeneity. These challenges should be considered when doing stockpile surveys.

A very important aspect of survey measurement is the accuracy of instrument used which is based on the type of equipment used. The survey equipment presently used on the mine and their respective accuracies are:

- Trimble S8 with accuracy of 1” angular measurement and precision of 1mm + 1ppm and
- Sokkia set 1X and 2X with accuracy of with 1” and 2” accuracy respectively.

4.4 Truck factor determination

At the end of a month the total number of trucks removed from each bench is obtained from the monthly statistical sheet. The number of trucks loaded from every pit is then entered into a database called "End of Month Pit Volume Calcs.xls" spread sheet to give the average truck factor for each bench (Iduapriem mine survey protocol).

4.5 Static scales

Static scales provide a more accurate and precise method of measuring mass than the in-motion one. It is the preferred choice especially for the purpose of custody transfer as well as primary and secondary metal accounting purpose. These include platform scales, road and rail weigh bridges. Precisions are generally between "± 0.05% to ± 0.2% for platform scales and ± 0.1% to ± 0.5% for weighbridges for gross load measurements" (JKMRC, 2008:82).

The principles of mass measurements are the same for static scale measurements and scales used for the purpose of metal accounting should be certified. The steps for determining mass using weigh bridge are as follows (JKMRC, 2008)

- The gross weight is established by weighing the full truck
- The empty truck is then weighed and the tare is calculated and deducted (or the reverse of the first two steps).
- Representative samples are taken to determine moisture content using recognised procedures
- The dry weight is then calculated

In comparing mass measurements between survey measurements/survey DTM and spot tonnages which are derived by applying the truck factor, it is logical to assume survey DTM's will be of a higher accuracy since they use specialised instruments with high accuracy than the weighbridge scales.

4.6 Sources of errors

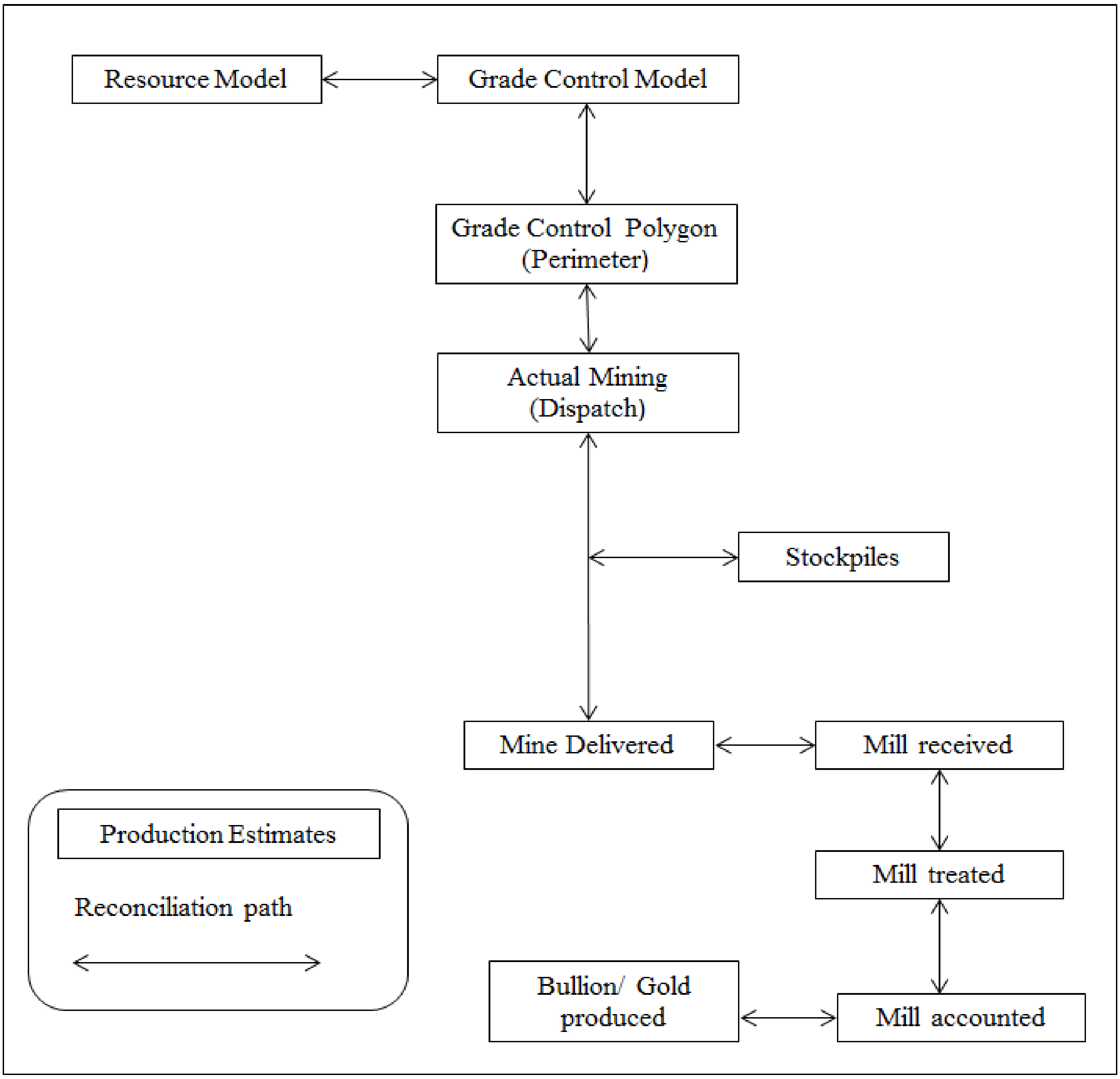
In static mass measurement, the main sources of error are usually spillages on the weigh bridges (JKMRC, 2008) however other sources include:

- Damage caused by impact of overload;
- Distortion of platform due to poor design or misuse;
- Incorrect weighing procedures like not ensuring the container truck is clean and empty for establishing the tare;
- Effect of changes in moisture content during weighing or between weighing and sampling;
- Errors in scale electronics or damage as a result of lightening or voltage fluctuations.

4.7 Mine to mill reconciliation

A useful approach to mine to mill reconciliation is to develop process map (source-to-product maps) indicating a detailed flow of material, measurements (mass and sampling), and measurement errors. A typical mine-to-mill reconciliation should follow the generic model below in figure 4.1.

Figure 4.1 Generic production reconciliation



Source (developed by researcher on Iduapriem Mine)

4.8 QAQC presentation and analysis

The current protocol document indicates that the site practice involves the insertion of quality control samples that are submitted to the analytical laboratory. The QC control methods include:

- Field duplicates (field splits) – to check the quality of sample splitting at the field;

- Standards – check on the accuracy of analytical laboratory;
- Retrieved pulps – to check on the precision of the analytical laboratory;
- Blanks (pulp and course) – to check on equipment cleaning effectiveness at the laboratory;
- Laboratory duplicates – to check on the laboratory's precision.

A batch of results is accepted if blanks and standards within the batch pass the following criteria;

- Standard – mean grade +/- 2SD ;
- Blanks – grades ≤ 0.02g/t.

If the QAQC protocol of a batch fails, the laboratory may be asked to reprocess the entire batch at their own cost or the pulps of that batch may be retrieved and re-submitted to the same laboratory with different sample names.

The performance of laboratory and quality of sampling is assessed as per plots in the sections below. Method of QAQC presentation and analysis is adopted from the Operational procedure manual of Goldfields Ghana Ltd and explained in the subsections below.

4.8.1 Standard reference material-SRM

Standards are materials whose true / best value is known. These should be identified from records as soon as an assay batch returns from the laboratory. The primary information required is batch number, Sample number, reported value, and date batch was submitted.

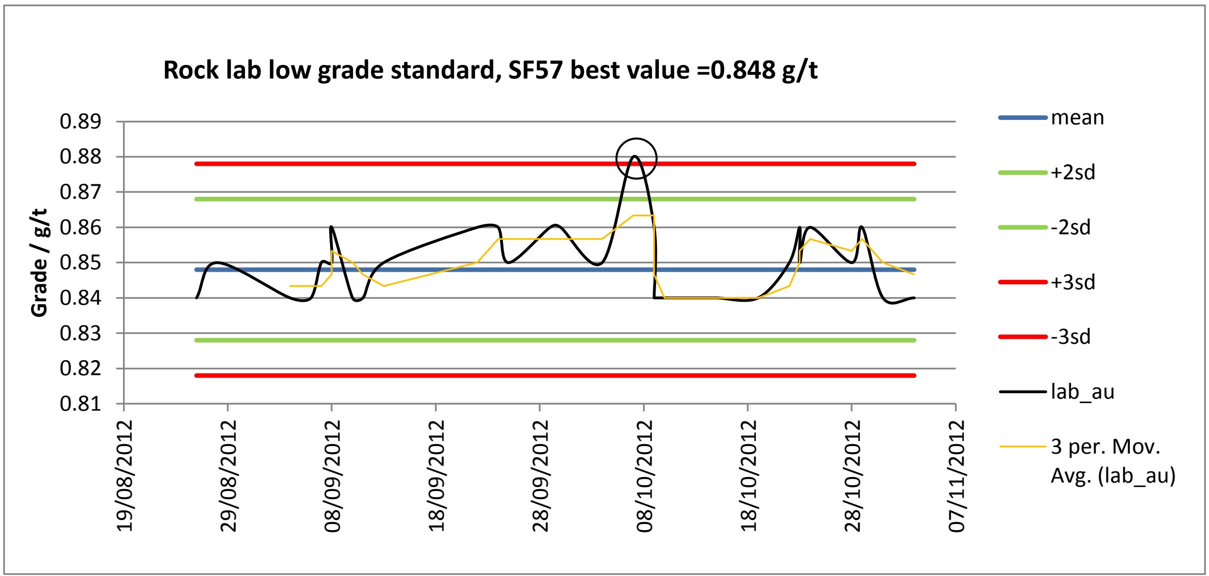
Excel file for that particular standard must contain the expected value for that particular standard and the 95% confidence interval i.e. the ±2 SD and ± 3 SD values. These values can be read from the original container of the standard reference material.

A weekly performance showing the performance of the standard should be prepared. The graph should show the values obtained from the laboratory over time, the expected/best value of the standard and the upper and lower control (±2 SD) and action limit (± 3 SD) . These limits show the reasonable range results from the laboratory are expected to be for that particular standard. In cases where results fall outside the acceptable range, the office geologists in consultation with the chief geologist should do the steps indicated below.

- Scrutinize results of other standards from the batch of the anomalous results if there is any;
- Scrutinize results of blanks for that particular batch;
- Scrutinize results of the laboratory's own standard.

If other standards from the batch do not show any consistent bias, then the result is accepted, however, if they do show a bias (positive or negative), the geologist in charge of database management and QAQC should report to a higher authority (a sectional resource geologist or the chief geologist) as quickly as possible. He or she re-examines the results and decides whether that particular batch needs to be re-assayed or not. The Figure 4.1 below is an example of such graph.

Figure 4.2 E.g. of Standard graph which shows a result that is outside an acceptable limit



Source (Iduapriem, 2012**)**

In Figure 4.2, low grade standard SF57 with expected or best value (0.848 g/t) received an assay value of 0.881 g/t falling above the action limit (+3SD). Other standards examined in this batch showed the following results: medium grade standard SH55 with expected value (1.375 g/t), received 1.401 g/t and high grade standard SK62 with expected value 4.075 g/t, received 3.975 g/t. Since there is no consistent bias evident in the other standards from this batch, the results are accepted.

A very important aspect of determining the reliability of a standard result is calculating the bias. The bias is calculated as: Mean of laboratory _au/ best value – 1 (expressed as a percentage)

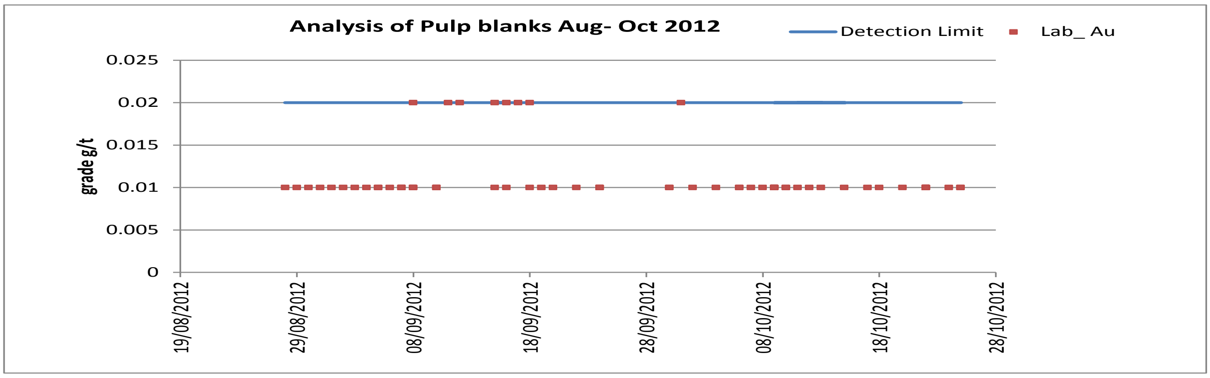
The calculated bias for a standard over a long period should be less than ± 1.5 %. A bias result of more than ± 2% over a long period of time may be indicative of a problem from the analysis procedure used by the laboratory. The bias reported for a particular standard should not exceed ± 3 % for a period exceeding one month.

If the weekly and long term (monthly or more) bias for a standard exceeds ± 1.5 % and ±3 % respectively, the geologist in charge of QAQC should report to the resource geologist and subsequently to the chief geologist and then conduct an investigation to establish the cause of the problem, i.e. If the problem is because of poor analytical technique by the laboratory or the problem is from the standard itself. **Note that these limits are assumed by the researcher based on the mine's data.**

4.8.2 Pulp and course Blanks

Pulp and course blanks should be presented in a simple table format in the reports (weekly or monthly) and with the graph showing blank results versus time (an example is shown in Fig 4.2) should be included in the report. The table format should show the received mean value for the blank and the percentage of received values that are within the acceptable limit. Table 4.1 shows an example of such a table.

Figure 4.3 E.g. of pulp blank sample performance against detection limit



Source (Iduapriem, 2012)

Table 4.1 Blank performance (August - October 2012)

Type	Limit	Lab mean (g/t)	% within limit
Course	0.03	0.013	100
Blank	0.02	0.011	100

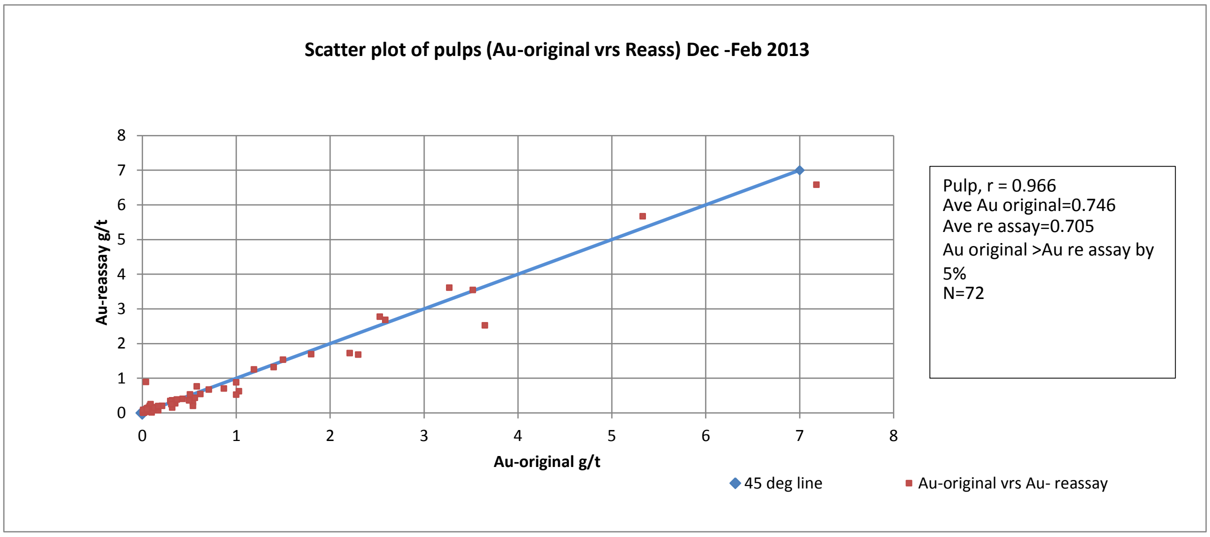
Blanks (pulp and course) results which exceed the detection limits should be brought to the attention of the resource geologist and the chief geologist. This may mean samples were contaminated from poor laboratory sample preparation technique. The history of blank results should then be examined to see if the occurrence was sudden or a trend. If it was a trend then the laboratory should be informed and preparation procedure revised.

4.8.3 Pulp re-submits

When pulp re-submits results return from the secondary laboratory, they should be copied into a specific spread sheet along with the results from the primary laboratory. The two sets of results should then be plotted in a scatter diagram also showing a 100% correlation (45^{o} line starting at the origin), the calculated correlation coefficient, average of the original result of the re-submitted pulp result and the percentage difference between the two averages.

In comparing data, a correlation co-efficient above 0.95 should be expected. The percentage difference between the two means should not exceed +/- 2% if sufficient data are assessed. If these conditions are not met then the QAQC results from secondary laboratory should be examined to determine if bias is existent in the results. Once this is done, the information must be brought forward to the chief geologist for a decision to be taken about the validity of the data comparison. Figure 4.4 is an example of a scatterplot of original assay results versus pulp re- submits assay results.

Figure 4.4 Scatter plot showing original assay results versus pulp re-submit assay result



Source (Iduapriem,2012)

In the example shown in Figure 4.4, it may be necessary to request to check if the splitting method at the rig is accurate or whether there are some deviations from the splitting procedure outlined by the mine.

4.8.4 Field duplicates (field splits)

A field duplicate is an extra sample taken from the opposite side of the splitter (from which normal sample is taken) at the drilling rig (RC rig). Holes that are designated as quality control holes, from which second splits (duplicates) are taken, and submitted to the laboratory as usual under the sample preparation and analysis techniques as carried out on mine.

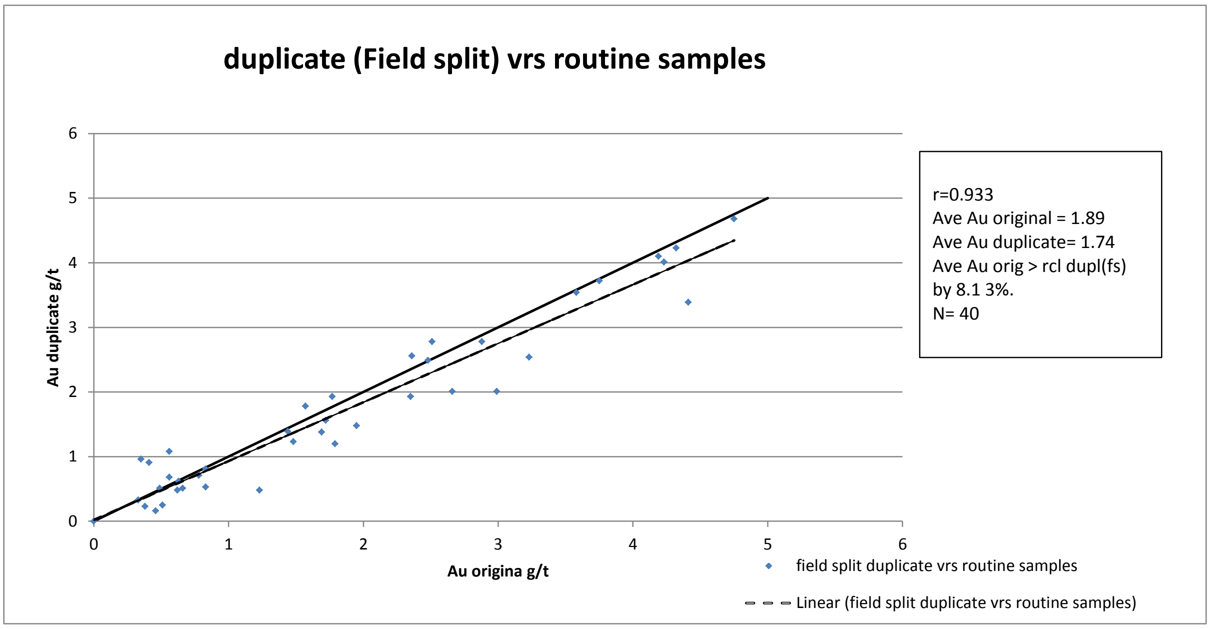
Duplicates from the same reef are submitted to the same primary laboratory. The duplicates play an important role of monitoring any splitting error that may be generated at the splitter in the field. Results from the two splits should indicate no bias towards either split.

Results of field splits from the primary laboratory should be treated similarly as “pulp re-submit results”. The two sets of results should be plotted on a scatter diagram showing the

100 % correlation line (45 % line starting from the origin), calculated correlation co-efficient (r) for the data, averages of the routine and field split results, and the percentage difference (%) between the two averages.

The geologist in charge of QAQC should determine if there is bias existent in the splitting system at the rig. There should be sufficient number of pairs to make a reliable assessment. If the calculated bias exceeds +/- 4 % with a pair of dataset (more than 80 pairs), then it should be investigated for possible causes.

Figure 4.5 Scatter plot showing original sample assay result versus field split



Source (Data created by researcher for demonstration)

4.8.5 Laboratory assay repeats

The laboratory performs an assay reading on the second portion of the assay solution on regular intervals. These are generally reported as repeats. These checks are done by the laboratory and are not usually included as a request on the submission forms. Results of the repeat readings from the AAS (Atomic Absorption spectroscopy) machine are used by the laboratory to determine whether the machine is working properly.

4.8.6 Laboratory second splits

This type of check is done by the laboratory. A second split from pulp duplicate follows an identical process route to the original sample and gets reported as laboratory replicates.

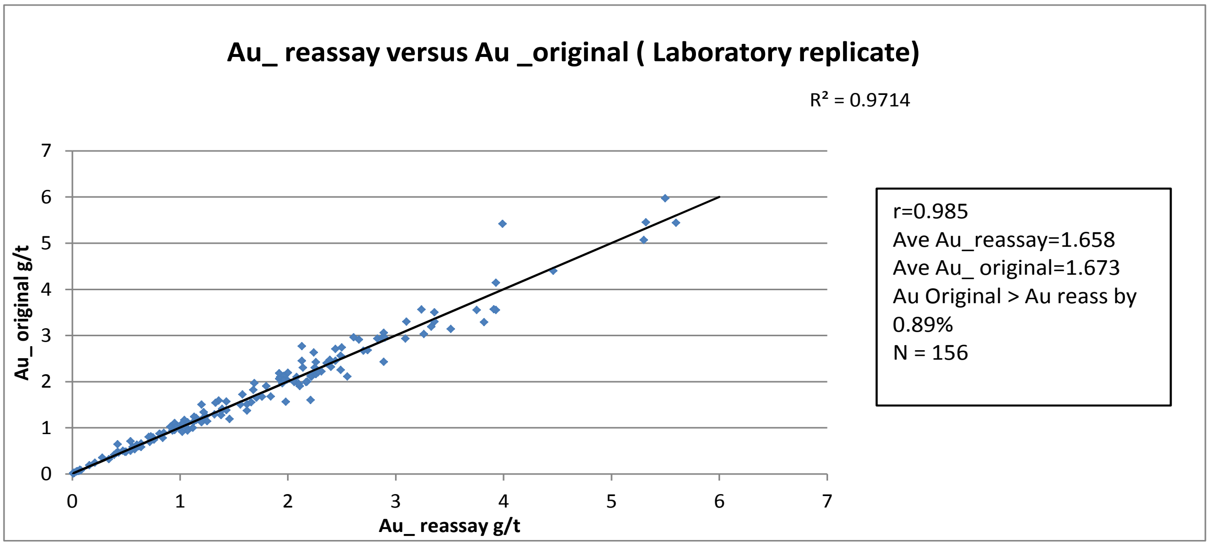
The results of the laboratory replicate sample are used to establish the effectiveness of the homogenization process during sample preparation. These very low grades (say 0.15g/t) will affect the correlation co-efficient (r) even though their comparisons will be of little interest as far as QAQC is concerned, hence they are removed.

The two sets are plotted on a scatter diagram also showing the 100% correlation line (45^{o} line starting from the origin), calculated correlation co–efficient (r), averages of the Au (original) results and those of the replicate ones (Au re-assay), and the percentage (%) difference between the two averages.

In comparing the data, a correlation co-efficient above 0.95 can be expected. The percentage difference between the two averages (Au original and the Au re-assay) should be less than +/- 2% if sufficient data pairs are assessed. If these are not achieved, then it is advisable to contact the laboratory to discuss reasons for the poor performance.

The figure (4.5) shows a graph of comparison between original gold assays and re-assays of a laboratory replicate and the results does not indicate a poor performance for homogenisation process.

Figure 4.6 Laboratory replicate graph



Source (Iduapreim, 2012)

4.9 Conclusion

The fourth chapter discusses the mass measurement techniques that are currently practised on the mine and statistical analysis available for determining the accuracy and bias between any two measurement systems e.g. is between survey DTM and spotter tonnages. The various quality control methods and analysis as well as presentation is recommended based on the operational control manual document of Goldfields Ghana Ltd.

The final chapter five will conclude on the entire research report and some significant findings with regards to the MCF issues at Idupariem mine which includes the tonnage measurements and quality control methods. Some areas of further research and recommendations will also be made.

5 Conclusions and recommendations

The MCF trend at Iduapriem mine from 2006-2012 was averaged at 100%. The difference of gold called for and gold produced is 5890 ounces at average price of 1 098.38 US$/Oz. The percentage of revenue loss is 0.7%. The actual MCF was achieved once in 2010 and has since then been declining to a minimum of 97% in 2012.

The purpose of the study was to establish a relationship between actual measurements (tonnage, volume, relative density, mine to mill reconciliation strategy, truck tonnage determination, sampling and assay standards i.e. QAQC methods and analysis) and reporting against measurement protocols. A significant aim was to identify the various measurement points along the ore flow diagram, study variances over time at those measurement points and investigate the causes of these variances; with a possibility of minimizing over all mine call factor variance.

The methods of tonnage measurement at the mining department included the survey tonnage (survey DTM) and the spot tonnage which is derived based on a truck count and a truck factor applied. It was realised that spot tonnages were consistently higher than survey tonnages over the period 2008 to 2012 but the differences were reducing with time. From the good correlation (0.945) that exists between the spot tonnage and the survey tonnage, it is seen that spot tallies are a satisfactory indicator and is therefore appropriate to do it; however it is appropriate to question the accuracy of the percentage margins to be considered for the purpose of metal accounting.

The researcher recommends that in cases of higher discrepancies between survey and spot tonnages, the survey tonnage is used for metal accounting since the survey tonnage is a controlled measurement which uses specialised instruments with high accuracies

(1mm+2ppm) as compared to spot tonnages based on truck factors determined by a weigh bridge of an accuracy of ±20kg.

The quality assurance and quality control graphs were analysed for low, medium and high grade standards and the rock laboratory high grade standard SK62, of best value 4.075 g/t showed a consistent negative bias of -1.66% from March 2012 to February 2013. It is recommended that the same standard should be reanalysed in a secondary laboratory to establish the authenticity of the best value. The pulp and course blank graphs indicated that the detection limit of pulp blanks was 0.02 g/t and that of the course blanks was 0.03 g/t which was not clearly stated in the mine's current QAQC protocol; however upon analysis of pulp and course blank graphs, the researcher opines that there is minimum or no contamination of laboratory equipment during sampling preparation and analysis. Another recommendation is for the mine to team up with neighbouring mines (say Goldfields Ghana Limited) and work towards a common quality control document with specific bias values as they are mining from similar geology.

The mine to mill reconciliation compared production estimates from various sources (resource model, grade control model, pit design, plant and stockpile, truck tally, stockpile and plant feed, plant feed and plant received) over the period 2009 and 2010. Measurement points between any two of the sources were established and reconciliation factors calculated based on the tonnage (t), grade (g/t) and ounces. The biggest grade and tonnage discrepancy was realised between "plant and stockpile versus perimeter design reconciliation" with a grade standard deviation of 18% and a tonnage standard deviation of 17%. The lowest grade and tonnage discrepancy occurred between the grade control model and perimeter design measurement point. These discrepancies indicate an importance of standardised procedures for measurement particularly for grades because there is more confidence in the mass (tonnage). It is suggested that grade estimation from the resource model requires investigation. The researcher recommends that the mine to mill reconciliation should follow the generic mine to mill model proposed in the metal accounting protocol to avoid the disconnections between measuring points that exist in the current one.

The metal accounting protocol was developed on tonnage measurement, QAQC methods and analysis as well as mine to mill reconciliation with focus on the mining department. Further study to cover up to the plant and processing sections of the mine to unite the mining and processing units into a single system (perhaps called operation) will be a positive step towards dissolving the historic mine to mill boundary.

5.1 Further research

The research was unable to determine reasons why variances occurred at measurement points. A further study is recommended to investigate the measurement (mass and sampling) procedures at the plant and stock pile. The causes of variances at measurement points will bring about opportunities to identify real problems and possible solutions leading to minimising an overall mine call factor variance. A typical MCF investigation from the resource throughout the various processes the ore undergoes until the point of which the metal is produced is recommended.

There are more reliable sources of ore tracking systems available in a number of mining operations. For example, according (JKMRC, 2008), the use of passive radio frequency identification tags (RFID) has proven to be useful in effective ore tracking and they are cheap. This system makes it possible for materials to be tagged with markers at the source representing volume and later detected as it flows through the system, or not detected if sent to waste dumps and later detected if sent to a long term stockpile and eventually processed. Reconciliation can then be conducted from the plant back to the source (resource); which is a reverse of the “historic mine to mill reconciliation”.

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Appendix A Actual Mine Call Factor For Iduapriem Mine (2006-2012)

	ANGLOGOLD ASHANTI LTD - WEST AFRICA DIVISION															
	IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2006							TOTAL
			2006	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2006
	BME 1 - MINED FROM OPEN PITS															
	TOTAL BCM's MINED	bcm	710,754	530,540	655,096	855,861	620,970	669,939	762,310	617,766	759,517	650,681	720,546	910,965	774,862	8,529,053
x	% ORE	%	16.61%	24.60%	19.10%	14.61%	16.78%	17.18%	18.66%	16.80%	16.20%	13.44%	15.19%	12.92%	17.13%	16.61%
=	BCM's ORE MINED	bcm	118,030	130,507	125,148	125,077	104,199	115,063	142,255	103,779	123,031	87,461	109,479	117,653	132,706	1,416,359
x	RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	TONNES ORE MINED	tonnes	312,779	345,844	331,642	331,454	276,127	304,917	376,977	275,015	326,032	231,772	290,119	311,781	351,671	3,753,351
x	IN-SITU GRADE (Grade Control)	g/t	1.79	1.66	1.65	1.75	1.86	1.84	1.82	1.92	1.82	1.74	1.80	1.86	1.82	1.79
=	GOLD MINED	Oz (000)	18.013	18.412	17.629	18.650	16.507	18.049	22.022	16.933	19.025	12.966	16.743	18.645	20.578	216.160
	BME 2 - PLANT FEED															
	PLANT FEED (MET)	tonnes	293,342	340,644	298,998	273,550	327,931	263,394	247,545	289,496	331,440	301,764	310,373	281,268	253,704	3,520,107
x	FEED GRADE (Grade Control)	g/t	1.84	1.79	1.79	1.87	1.85	1.88	1.85	1.91	1.82	1.88	1.78	1.86	1.88	1.84
=	GOLD DELIVERED TO PLANT	Oz (000)	17.387	19.604	17.207	16.446	19.505	15.920	14.724	17.777	19.348	18.194	17.762	16.820	15.335	208.643
+	GOLD IN SCATS	Oz (000)	(0.480)	(1.059)	(0.400)	(0.304)	(0.381)	(0.413)	(0.303)	(0.438)	(0.513)	(0.452)	(0.533)	(0.505)	(0.460)	(5.760)
=	GOLD TREATED	Oz (000)	16.907	18.545	16.808	16.143	19.124	15.508	14.421	17.340	18.835	17.742	17.229	16.315	14.875	202.883
+/-	PLANT INVENTORY CHANGE	Oz (000)	0.048	(0.046)	0.355	0.031	(0.585)	1.243	(0.688)	0.281	(0.152)	0.023	(0.481)	0.079	0.517	0.578
=	GOLD CALLED for	Oz (000)	16.955	18.499	17.163	16.174	18.539	16.751	13.733	17.620	18.683	17.765	16.749	16.394	15.392	203.461
x	MINE CALL FACTOR	%	101.52%	104.42%	103.32%	102.48%	103.44%	101.50%	100.78%	100.65%	100.79%	100.01%	100.66%	98.23%	101.49%	101.52%
x	RECOVERY FACTOR	%	95.14%	94.18%	95.91%	95.92%	95.51%	95.38%	96.28%	94.65%	94.61%	94.84%	94.35%	95.03%	95.46%	95.14%
=	GOLD PRODUCED	Oz (000)	16.377	18.193	17.008	15.899	18.316	16.217	13.324	16.785	17.814	16.850	15.906	15.304	14.911	196.529
	BME 3 -TREATMENT															
	TONNES TREATED (Check in)	tonnes	285,217	322,242	292,053	268,498	321,525	256,566	242,451	282,366	322,649	294,269	301,062	272,830	246,093	3,422,604
x	YIELD	g/t	1.79	1.76	1.81	1.84	1.77	1.97	1.71	1.85	1.72	1.78	1.64	1.74	1.88	1.79
=	GOLD PRODUCED	Oz (000)	16.377	18.193	17.008	15.899	18.316	16.217	13.324	16.785	17.814	16.850	15.906	15.304	14.911	196.529
	INFO : PIT TO CRUSHER RECONCILIATION															
	TONNES (Line 5 - Line 8)	tonnes	19,437	5,200	32,644	57,904	(51,804)	41,523	129,432	(14,481)	(5,408)	(69,992)	(20,254)	30,513	97,967	233,244
x	YIELD	g/t	1.00	(7.13)	0.40	1.18	1.80	1.59	1.75	1.81	1.86	2.32	1.57	1.86	1.66	1.00
=	OUNCES (Line 7 - Line 10)	Oz (000)	0.626	(1.192)	0.422	2.204	(2.998)	2.129	7.299	(0.844)	(0.323)	(5.228)	(1.019)	1.825	5.243	7.517
	TONNES, STRIPPING RATIOS															
	WASTE MINED	bcm	592,725	400,033	529,948	730,784	516,771	554,876	620,054	513,987	636,486	563,220	611,067	793,312	642,156	7,112,694
x	RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	WASTE MINED	tonnes	1,570,720	1,060,087	1,404,362	1,936,578	1,369,443	1,470,421	1,643,143	1,362,066	1,686,688	1,492,533	1,619,329	2,102,277	1,701,713	18,848,640
	MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x	RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
=	MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
=	TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	1,883,499	1,405,931	1,736,004	2,268,032	1,645,570	1,775,338	2,020,120	1,637,081	2,012,720	1,724,305	1,909,448	2,414,058	2,053,384	22,601,991
	SCATS TONNES	tonnes	8,125	18,402	6,945	5,052	6,406	6,827	5,094	7,130	8,790	7,495	9,311	8,438	7,611	97,503
	MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	TOTAL FGO STOCKPILE	tonnes	1,282,021	1,053,977	1,086,621	1,144,525	1,092,721	1,134,244	1,263,676	1,249,195	1,243,787	1,173,795	1,153,541	1,184,054	1,282,021	1,282,021
	TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	SCAT STOCKPILE	tonnes	398,100	318,999	325,945	330,997	337,403	344,230	349,324	356,454	365,245	372,740	382,051	390,489	398,100	398,100
	REHANDLE	tonnes	281,398	340,644	274,015	252,824	314,056	246,253	238,780	277,800	320,657	300,295	293,392	270,375	247,679	3,376,772
=	DIRECT FEED (Line 8 - Line 40)	tonnes	11,945	0	24,983	20,726	13,875	17,140	8,765	11,696	10,782	1,469	16,981	10,893	6,025	143,335
	% REHANDLING	%	95.93%	100.00%	91.64%	92.42%	95.77%	93.49%	96.46%	95.96%	96.75%	99.51%	94.53%	96.13%	97.63%	95.93%
	INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	5.02	3.07	4.23	5.84	4.96	4.82	4.36	4.95	5.17	6.44	5.58	6.74	4.84	5.02
	DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	5.02	3.07	4.23	5.84	4.96	4.82	4.36	4.95	5.17	6.44	5.58	6.74	4.84	5.02
	LOM AVERAGE STRIPPING RATIO	t:t	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90
	DEFERRED STRIPPING TONNES	tonnes	352,325	(287,105)	112,489	645,438	293,822	282,653	174,674	290,777	416,669	589,694	489,204	887,771	331,821	4,227,906
	GRADE, RECOVERIES															
	MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	MILL HEAD GRADE (GRADE CONTROL)	g/t	1.87	1.81	1.85	1.92	1.91	1.90	1.87	1.92	1.83	1.88	1.79	1.83	1.91	1.87
	CALCULATED HEAD GRADE	g/t	1.88	1.86	1.89	1.92	1.86	2.06	1.78	1.95	1.82	1.88	1.74	1.84	1.97	1.88
	RESIDUE GRADE	g/t	0.07	0.11	0.08	0.08	0.09	0.09	0.07	0.10	0.10	0.10	0.10	0.09	0.09	0.07
	FGO STOCKPILE GRADE	g/t	1.18	1.18	1.16	1.16	1.13	1.15	1.21	1.20	1.20	1.13	1.12	1.14	1.18	1.18
	MARGINAL STOCKPILE GRADE	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	SCATS STOCKPILE GRADE	g/t	1.82	1.81	1.81	1.81	1.81	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82

Source (Anglogold Ashanti, 2013)

	IDUAPRIEM GOLD MINE		AVERAGE 2007	SCHEDULE 5 JANUARY	FEBRUARY	MARCH	APRIL	MAY	ACTUAL 2007 JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	TOTAL 2007
	BME 1 - MINED FROM OPEN PITS															
	TOTAL BCM's MINED	bcm	716,204	747,718	748,333	778,202	691,818	717,596	913,474	653,035	763,161	586,919	674,034	685,767	634,388	8,594,444
x	% ORE	%	14.21%	12.56%	16.32%	10.69%	15.67%	15.35%	4.50%	8.87%	18.93%	18.94%	22.45%	14.35%	15.55%	14.21%
=	BCM's ORE MINED	bcm	101,741	93,934	122,133	83,176	108,407	110,124	41,139	57,949	144,446	111,145	151,324	98,439	98,674	1,220,890
x	RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	TONNES ORE MINED	tonnes	269,613	248,925	323,652	220,416	287,279	291,829	109,018	153,564	382,782	294,534	401,008	260,864	261,486	3,235,357
x	IN-SITU GRADE (Grade Control)	g/t	1.84	1.76	1.91	1.86	1.87	1.83	1.94	1.73	1.81	1.90	1.82	1.90	1.81	1.84
=	GOLD MINED	Oz (000)	15.984	14.085	19.875	13.181	17.272	17.170	6.787	8.566	22.276	17.992	23.465	15.925	15.219	191.812
	BME 2 - PLANT FEED															
	PLANT FEED (MET)	tonnes	269,881	209,313	148,515	180,469	308,175	294,422	358,489	335,793	355,307	296,032	219,310	260,375	272,369	3,238,570
x	FEED GRADE (Grade Control)	g/t	1.89	1.85	1.91	1.89	1.90	1.85	1.83	1.72	1.95	1.94	1.99	1.97	1.99	1.89
=	GOLD DELIVERED TO PLANT	Oz (000)	16.435	12.450	9.120	10.966	18.825	17.512	21.092	18.569	22.276	18.464	14.031	16.491	17.426	197.223
+	GOLD IN SCATS	Oz (000)	(0.493)	(0.373)	(0.274)	(0.329)	(0.565)	(0.525)	(0.633)	(0.557)	(0.668)	(0.554)	(0.421)	(0.495)	(0.523)	(5.917)
=	GOLD TREATED	Oz (000)	15.942	12.076	8.846	10.637	18.261	16.987	20.459	18.012	21.607	17.910	13.610	15.997	16.903	191.306
+/-	PLANT INVENTORY CHANGE	Oz (000)	(0.005)	0.578	0.205	(0.392)	0.046	(0.708)	(1.709)	1.789	(0.169)	0.764	0.298	(0.686)	(0.081)	(0.065)
=	GOLD CALLED for	Oz (000)	15.937	12.654	9.051	10.245	18.306	16.278	18.751	19.801	21.439	18.674	13.909	15.311	16.822	191.242
x	MINE CALL FACTOR	%	101.83%	104.80%	105.26%	101.26%	104.73%	97.36%	98.05%	101.70%	97.46%	107.20%	103.64%	99.47%	103.65%	101.83%
x	RECOVERY FACTOR	%	95.08%	95.67%	98.04%	96.99%	96.81%	94.76%	94.44%	95.05%	93.07%	94.70%	94.98%	94.08%	95.01%	95.08%
=	GOLD PRODUCED	Oz (000)	15.430	12.688	9.341	10.062	18.560	15.019	17.362	19.140	19.445	18.957	13.691	14.328	16.565	185.157
	BME 3 -TREATMENT															
	TONNES TREATED (Check in)	tonnes	261,784	203,034	144,060	175,055	298,930	285,590	347,734	325,720	344,648	287,151	212,731	252,564	264,198	3,141,413
x	YIELD	g/t	1.83	1.94	2.02	1.79	1.93	1.64	1.55	1.83	1.75	2.05	2.00	1.76	1.95	1.83
=	GOLD PRODUCED	Oz (000)	15.430	12.688	9.341	10.062	18.560	15.019	17.362	19.140	19.445	18.957	13.691	14.328	16.565	185.157
	INFO : PIT TO CRUSHER RECONCILIATION															
	TONNES (Line 5 - Line 8)	tonnes	(268)	39,612	175,137	39,947	(20,896)	(2,594)	(249,471)	(182,229)	27,475	(1,498)	181,698	490	(10,883)	(3,212)
x	YIELD	g/t	52.39	1.28	1.91	1.72	2.31	4.10	1.78	1.71	0.00	9.80	1.61	(35.97)	6.31	52.39
=	OUNCES (Line 7 - Line 10)	Oz (000)	(0.451)	1.636	10.755	2.215	(1.554)	(0.342)	(14.306)	(10.003)	0.001	(0.472)	9.433	(0.566)	(2.207)	(5.411)
	TONNES, STRIPPING RATIOS															
	WASTE MINED	bcm	614,463	653,784	626,200	695,026	583,411	607,472	872,335	595,086	618,715	475,774	522,710	587,328	535,714	7,373,555
x	RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	WASTE MINED	tonnes	1,628,327	1,732,528	1,659,430	1,841,819	1,546,039	1,609,801	2,311,688	1,576,978	1,639,594	1,260,801	1,385,182	1,556,419	1,419,642	19,539,920
	MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x	RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
=	MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
=	TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	1,897,940	1,981,453	1,983,082	2,062,235	1,833,318	1,901,629	2,420,706	1,730,542	2,022,376	1,555,335	1,786,189	1,817,284	1,681,128	22,775,277
	SCATS TONNES	tonnes	8,096	6,279	4,455	5,414	9,245	8,833	10,755	10,074	10,659	8,881	6,579	7,811	8,171	97,157
	MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	TOTAL FGO STOCKPILE	tonnes	1,278,808	1,321,632	1,496,770	1,536,717	1,515,821	1,513,227	1,263,756	1,081,527	1,109,002	1,107,504	1,289,202	1,289,691	1,278,808	1,278,808
	TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	SCAT STOCKPILE	tonnes	495,257	404,379	408,835	414,249	423,494	432,327	443,082	453,155	463,815	472,696	479,275	487,086	495,257	495,257
	REHANDLE	tonnes	263,696	193,904	143,278	168,218	308,175	292,345	352,979	328,080	342,352	286,032	217,881	258,745	272,369	3,164,357
=	DIRECT FEED (Line 8 - Line 40)	tonnes	6,184	15,410	5,237	12,251	0	2,077	5,510	7,714	12,955	10,000	1,429	1,630	0	74,213
	% REHANDLING	%	97.71%	92.64%	96.47%	93.21%	100.00%	99.29%	98.46%	97.70%	96.35%	96.62%	99.35%	99.37%	100.00%	97.71%
	INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	6.04	6.96	5.13	8.36	5.38	5.52	21.20	10.27	4.28	4.28	3.45	5.97	5.43	6.04
	DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	6.04	6.96	5.13	8.36	5.38	5.52	21.20	10.27	4.28	4.28	3.45	5.97	5.43	6.04
	LOM AVERAGE STRIPPING RATIO	t:t	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97	3.97
	DEFERRED STRIPPING TONNES	tonnes	557,936	744,271	374,498	966,746	405,515	451,213	1,878,874	967,313	119,911	91,472	(206,858)	520,762	381,517	6,695,235
	GRADE, RECOVERIES															
	MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	MILL HEAD GRADE (GRADE CONTROL)	g/t	1.93	1.94	2.01	1.92	1.99	1.81	1.80	1.74	1.90	2.08	2.06	1.97	2.06	1.93
	CALCULATED HEAD GRADE	g/t	1.93	2.03	2.06	1.84	1.99	1.73	1.64	1.92	1.89	2.17	2.11	1.88	2.05	1.93
	RESIDUE GRADE	g/t	0.09	0.08	0.04	0.06	0.06	0.09	0.10	0.09	0.13	0.11	0.10	0.12	0.10	0.09
	FGO STOCKPILE GRADE	g/t	1.02	1.16	1.24	1.26	1.24	1.24	1.13	1.03	1.01	0.99	1.08	1.07	1.02	1.02
	MARGINAL STOCKPILE GRADE	g/t	1.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.00	1.00	1.00
	SCATS STOCKPILE GRADE	g/t	1.82	1.79	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.81	1.81	1.81	1.82	1.82

Source (Anglogold Ashanti, 2013)

	IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2008							TOTAL
			2008	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2008
	BME 1 - MINED FROM OPEN PITS															
	TOTAL BCM's MINED	bcm	547,088	551,452	604,078	543,320	531,929	424,035	405,231	465,794	578,324	575,336	637,245	611,014	637,296	6,565,054
x	% ORE	%	20.60%	17.84%	17.80%	23.09%	25.60%	29.00%	25.23%	27.30%	16.78%	12.06%	21.14%	18.61%	18.41%	20.60%
=	BCM's ORE MINED	bcm	112,677	98,395	107,549	125,426	136,179	122,978	102,256	127,149	97,055	69,361	134,736	113,704	117,334	1,352,122
x	RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	TONNES ORE MINED	tonnes	298,594	260,747	285,005	332,379	360,874	325,892	270,978	336,945	257,196	183,808	357,050	301,316	310,935	3,583,125
x	IN-SITU GRADE (Grade Control)	g/t	1.81	1.77	1.90	1.84	1.72	1.65	1.78	1.79	1.90	2.05	1.80	1.89	1.76	1.81
=	GOLD MINED	Oz (000)	17.372	14.847	17.410	19.712	19.925	17.305	15.496	19.442	15.733	12.098	20.627	18.270	17.594	208.458
	BME 2 - PLANT FEED															
	PLANT FEED (MET)	tonnes	303,714	277,903	250,630	305,002	310,198	320,858	278,571	309,415	290,268	304,587	330,518	328,725	337,899	3,644,573
x	FEED GRADE (Grade Control)	g/t	1.86	1.82	1.96	1.84	1.83	1.84	1.74	1.91	1.88	1.89	1.92	1.86	1.87	1.86
=	GOLD DELIVERED TO PLANT	Oz (000)	18.191	16.261	15.794	18.043	18.251	18.981	15.584	18.988	17.545	18.474	20.403	19.658	20.315	218.296
+	GOLD IN SCATS	Oz (000)	(0.547)	(0.488)	(0.474)	(0.174)	(0.547)	(0.569)	(0.468)	(0.570)	(0.667)	(0.610)	(0.673)	(0.649)	(0.670)	(6.559)
=	GOLD TREATED	Oz (000)	17.645	15.773	15.320	17.869	17.703	18.412	15.116	18.419	16.878	17.864	19.729	19.009	19.645	211.738
+/-	PLANT INVENTORY CHANGE	Oz (000)	(0.016)	0.221	0.109	(0.421)	(1.419)	0.134	0.022	(0.759)	0.390	0.343	0.534	0.406	0.246	(0.195)
=	GOLD CALLED for	Oz (000)	17.629	15.995	15.428	17.448	16.284	18.546	15.138	17.659	17.268	18.207	20.263	19.415	19.891	211.543
x	MINE CALL FACTOR	%	101.23%	108.99%	98.71%	98.98%	96.46%	99.83%	100.45%	99.34%	101.33%	104.45%	103.15%	99.87%	102.53%	101.23%
x	RECOVERY FACTOR	%	93.40%	95.38%	94.75%	94.00%	92.49%	92.84%	92.29%	91.65%	94.48%	93.26%	93.20%	93.30%	93.40%	93.40%
=	GOLD PRODUCED	Oz (000)	16.667	16.627	14.430	16.234	14.528	17.189	14.033	16.078	16.532	17.736	19.481	18.091	19.048	200.008
	BME 3 -TREATMENT															
	TONNES TREATED (Check in)	tonnes	294,601	269,566	243,111	302,055	300,893	311,232	270,214	300,132	279,238	294,536	319,611	317,877	326,748	3,535,212
x	YIELD	g/t	1.76	1.92	1.85	1.67	1.50	1.72	1.62	1.67	1.84	1.87	1.90	1.77	1.81	1.76
=	GOLD PRODUCED	Oz (000)	16.667	16.627	14.430	16.234	14.528	17.189	14.033	16.078	16.532	17.736	19.481	18.091	19.048	200.008
	INFO : PIT TO CRUSHER RECONCILIATION															
	TONNES (Line 5 - Line 8)	tonnes	(5,121)	(17,156)	34,375	27,377	50,676	5,034	(7,593)	27,530	(33,072)	(120,779)	26,532	(27,409)	(26,963)	(61,448)
x	YIELD	g/t	4.98	2.56	1.46	1.90	1.03	(10.35)	0.36	0.51	1.70	1.64	0.26	1.57	3.14	4.98
=	OUNCES (Line 7 - Line 10)	Oz (000)	(0.820)	(1.415)	1.616	1.669	1.674	(1.676)	(0.088)	0.453	(1.812)	(6.376)	0.224	(1.387)	(2.721)	(9.838)
	TONNES, STRIPPING RATIOS												3,961	2,146,466		
	WASTE MINED	bcm	434,411	453,057	496,529	417,894	395,750	301,057	302,975	338,645	481,269	505,974	502,509	497,310	519,962	5,212,931
x	RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	WASTE MINED	tonnes	1,151,189	1,200,601	1,315,802	1,107,419	1,048,738	797,801	802,884	897,409	1,275,363	1,340,832	1,331,649	1,317,872	1,377,899	13,814,268
	MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x	RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
=	MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
=	TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	1,449,783	1,461,348	1,600,807	1,439,798	1,409,612	1,123,693	1,073,862	1,234,354	1,532,559	1,524,640	1,688,699	1,619,187	1,688,834	17,397,393
	SCATS TONNES	tonnes	9,113	8,337	7,519	2,947	9,305	9,626	8,357	9,282	11,030	10,051	10,907	10,848	11,151	109,361
	MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	TOTAL FGO STOCKPILE	tonnes	2,146,466	1,261,652	1,296,027	1,323,404	1,374,080	1,379,114	1,371,522	2,328,157	2,295,085	2,174,306	2,200,838	2,173,429	2,146,466	2,146,466
	TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	SCAT STOCKPILE	tonnes	295,365	503,594	511,113	514,060	523,366	476,794	447,114	455,117	422,985	373,160	323,472	299,326	295,365	295,365
	REHANDLE	tonnes	302,864	277,903	250,630	305,002	310,198	320,858	278,571	309,415	290,268	304,587	323,230	325,809	337,899	3,634,369
=	DIRECT FEED (Line 8 - Line 40)	tonnes	850	0	0	0	0	0	0	0	0	0	7,288	2,916	0	10,204
	% REHANDLING	%	99.72%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	97.80%	99.11%	100.00%	99.72%
	INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	3.86	4.60	4.62	3.33	2.91	2.45	2.96	2.66	4.96	7.29	3.73	4.37	4.43	3.86
	DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	3.86	4.60	4.62	3.33	2.91	2.45	2.96	2.66	4.96	7.29	3.73	4.37	4.43	3.86
	LOM AVERAGE STRIPPING RATIO	t:t	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81	3.81
	DEFERRED STRIPPING TONNES	tonnes	12,491	206,234	228,925	(160,120)	(327,470)	(444,999)	(230,502)	(387,542)	294,537	639,875	(29,976)	168,793	192,137	149,891
	GRADE, RECOVERIES															
	MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	MILL HEAD GRADE (GRADE CONTROL)	g/t	1.89	1.98	1.93	1.82	1.78	1.84	1.75	1.90	1.90	1.97	1.98	1.85	1.92	1.89
	CALCULATED HEAD GRADE	g/t	1.88	2.01	1.95	1.78	1.62	1.85	1.75	1.82	1.95	2.01	2.03	1.90	1.94	1.88
	RESIDUE GRADE	g/t	0.11	0.09	0.10	0.11	0.13	0.13	0.13	0.16	0.11	0.13	0.13	0.12	0.13	0.11
	FGO STOCKPILE GRADE	g/t	1.09	1.12	1.13	1.14	1.14	1.09	1.10	1.09	1.08	1.05	1.04	1.03	1.01	1.09
	MARGINAL STOCKPILE GRADE	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	SCATS STOCKPILE GRADE	g/t	2.11	1.79	1.79	1.76	1.64	1.79	1.88	1.83	1.96	2.21	2.53	2.72	2.74	2.11

Source (Anglogold Ashanti, 2013)

IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2009							TOTAL
		2009	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2009
BME 1 - MINED FROM OPEN PITS															
TOTAL BCM's MINED	bcm	798,783	523,252	647,630	1,092,849	1,010,783	710,622	725,663	853,217	1,079,617	834,148	847,513	808,363	451,741	9,585,397
x % ORE	%	16.15%	17.57%	14.66%	16.91%	13.70%	13.83%	8.75%	10.77%	7.53%	20.96%	28.66%	17.24%	32.31%	16.15%
= BCM's ORE MINED	bcm	129,022	91,924	94,966	184,855	138,524	98,271	63,468	91,904	81,344	174,802	242,863	139,375	145,966	1,548,262
x RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= TONNES ORE MINED	tonnes	341,908	243,599	251,660	489,866	367,089	260,418	168,191	243,545	215,562	463,225	643,587	369,344	386,811	4,102,894
x IN-SITU GRADE (Grade Control)	g/t	1.66	1.75	1.83	1.72	1.72	1.79	1.50	1.14	1.11	1.80	1.74	1.74	1.61	1.66
= GOLD MINED	Oz (000)	18.266	13.699	14.796	27.145	20.323	15.019	8.129	8.964	7.675	26.859	35.948	20.612	20.022	219.190
BME 2 - PLANT FEED															
PLANT FEED (MET)	tonnes	296,450	310,856	179,275	203,413	276,621	275,575	267,621	334,696	380,881	361,506	335,646	294,632	336,679	3,557,403
x FEED GRADE (Grade Control)	g/t	1.78	1.72	1.77	1.79	1.95	1.84	1.80	1.61	1.33	1.84	2.01	1.97	1.87	1.78
= GOLD DELIVERED TO PLANT	Oz (000)	16.984	17.190	10.202	11.706	17.342	16.293	15.488	17.325	16.287	21.386	21.690	18.661	20.242	203.813
+ GOLD IN SCATS	Oz (000)	(0.560)	(0.567)	(0.337)	(0.386)	(0.572)	(0.538)	(0.511)	(0.572)	(0.537)	(0.706)	(0.716)	(0.616)	(0.668)	(6.726)
= GOLD TREATED	Oz (000)	16.424	16.623	9.865	11.320	16.770	15.756	14.977	16.753	15.749	20.680	20.975	18.045	19.574	197.087
+/- PLANT INVENTORY CHANGE	Oz (000)	(0.000)	0.454	0.411	(0.233)	0.612	0.024	(0.172)	(0.266)	0.039	(0.045)	0.236	(0.462)	(0.598)	(0.000)
= GOLD CALLED for	Oz (000)	16.424	17.077	10.276	11.087	17.382	15.780	14.805	16.487	15.788	20.635	21.211	17.583	18.976	197.087
x MINE CALL FACTOR	%	101.57%	100.11%	101.18%	100.93%	100.20%	106.60%	99.29%	109.21%	102.29%	98.91%	101.14%	102.01%	97.95%	101.57%
x RECOVERY FACTOR	%	94.90%	94.19%	96.53%	96.08%	95.97%	96.26%	95.39%	94.72%	94.58%	94.96%	94.61%	93.84%	93.14%	94.90%
= GOLD PRODUCED	Oz (000)	15.830	16.102	10.036	10.751	16.713	16.192	14.021	17.054	15.274	19.383	20.296	16.831	17.310	189.964
BME 3 -TREATMENT															
TONNES TREATED (Check in)	tonnes	286,667	300,598	173,359	196,700	267,493	266,481	258,790	323,651	368,312	349,577	324,570	284,909	325,569	3,440,008
x YIELD	g/t	1.72	1.67	1.80	1.70	1.94	1.89	1.69	1.64	1.29	1.72	1.94	1.84	1.65	1.72
= GOLD PRODUCED	Oz (000)	15.830	16.102	10.036	10.751	16.713	16.192	14.021	17.054	15.274	19.383	20.296	16.831	17.310	189.964
INFO : PIT TO CRUSHER RECONCILIATION															
TONNES (Line 5 - Line 8)	tonnes	45,458	(67,258)	72,385	286,453	90,467	(15,157)	(99,430)	(91,151)	(165,320)	101,719	307,940	74,712	50,131	545,492
x YIELD	g/t	0.88	1.61	1.97	1.68	1.02	2.61	2.30	2.85	1.62	1.67	1.44	0.81	(0.14)	0.88
= OUNCES (Line 7 - Line 10)	Oz (000)	1.281	(3.491)	4.594	15.438	2.980	(1.274)	(7.359)	(8.361)	(8.612)	5.473	14.258	1.951	(0.219)	15.378
TONNES, STRIPPING RATIOS															
WASTE MINED	bcm	669,761	431,328	552,664	907,994	872,259	612,351	662,194	761,314	998,273	659,346	604,650	668,988	305,774	8,037,135
x RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= WASTE MINED	tonnes	1,774,867	1,143,019	1,464,560	2,406,184	2,311,486	1,622,730	1,754,815	2,017,481	2,645,423	1,747,267	1,602,323	1,772,818	810,302	21,298,409
MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
= MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
= TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	2,116,775	1,386,618	1,716,220	2,896,050	2,678,575	1,883,148	1,923,006	2,261,026	2,860,985	2,210,492	2,245,909	2,142,162	1,197,112	25,401,303
SCATS TONNES	tonnes	9,783	10,258	5,916	6,713	9,129	9,094	8,832	11,045	12,569	11,930	11,076	9,723	11,110	117,394
MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL FGO STOCKPILE	tonnes	2,151,593	2,079,208	2,151,593	2,438,046	2,528,513	2,513,356	2,413,926	2,322,774	2,157,455	2,259,173	2,567,114	2,641,826	2,691,957	2,151,593
TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
SCAT STOCKPILE	tonnes	301,602	301,917	301,602	308,315	317,331	315,008	320,794	307,655	296,242	273,126	267,084	264,587	263,477	301,602
REHANDLE	tonnes	273,791	310,856	178,082	199,994	276,621	258,880	261,354	333,901	364,584	278,720	260,254	243,129	319,110	3,285,486
= DIRECT FEED (Line 8 - Line 40)	tonnes	22,660	0	1,193	3,419	0	16,695	6,267	795	16,298	82,786	75,393	51,503	17,570	271,917
% REHANDLING	%	92.36%	100.00%	99.33%	98.32%	100.00%	93.94%	97.66%	99.76%	95.72%	77.10%	77.54%	82.52%	94.78%	92.36%
INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	5.19	4.69	5.82	4.91	6.30	6.23	10.43	8.28	12.27	3.77	2.49	4.80	2.09	5.19
DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	5.19	4.69	5.82	4.91	6.30	6.23	10.43	8.28	12.27	3.77	2.49	4.80	2.09	5.19
LOM AVERAGE STRIPPING RATIO	t:t	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06
DEFERRED STRIPPING TONNES	tonnes	387,055	154,246	443,066	417,807	821,464	565,685	1,072,124	1,028,928	1,770,453	(132,975)	(1,010,012)	273,643	(759,772)	4,644,657
GRADE, RECOVERIES															
MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MILL HEAD GRADE (GRADE CONTROL)	g/t	1.82	1.72	1.79	1.81	2.02	1.96	1.79	1.76	1.36	1.82	2.03	2.01	1.84	1.82
CALCULATED HEAD GRADE	g/t	1.81	1.77	1.87	1.77	2.03	1.96	1.77	1.73	1.36	1.82	2.06	1.96	1.78	1.81
RESIDUE GRADE	g/t	0.10	0.10	0.06	0.07	0.08	0.07	0.08	0.09	0.07	0.09	0.11	0.12	0.13	0.10
FGO STOCKPILE GRADE	g/t	1.01	0.99	1.02	1.10	1.10	1.09	1.04	0.96	0.91	0.95	1.01	1.00	0.98	1.01
MARGINAL STOCKPILE GRADE	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SCATS STOCKPILE GRADE	g/t	1.80	1.78	1.81	1.81	1.82	1.82	1.81	1.81	1.79	1.78	1.79	1.80	1.80	1.80

Source (Anglogold Ashanti, 2013)

IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2010							TOTAL
		2010	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2010
BME 1 - MINED FROM OPEN PITS															
TOTAL BCM's MINED	bcm	733,740	695,252	727,306	331,188	694,708	824,617	745,071	792,289	754,759	811,755	685,588	794,333	948,008	8,804,874
x % ORE	%	18.90%	19.34%	13.00%	25.47%	28.09%	26.78%	30.89%	9.31%	1.69%	15.18%	18.12%	26.01%	17.27%	18.90%
= BCM's ORE MINED	bcm	138,643	134,455	94,514	84,366	195,160	220,847	230,116	73,754	12,730	123,197	124,233	206,600	163,745	1,663,717
x RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= TONNES ORE MINED	tonnes	367,404	356,307	250,462	223,570	517,173	585,245	609,807	195,448	33,735	326,472	329,217	547,490	433,924	4,408,850
x IN-SITU GRADE (Grade Control)	g/t	1.61	1.58	1.72	1.65	1.57	1.62	1.50	1.77	1.81	1.66	1.62	1.66	1.56	1.61
= GOLD MINED	Oz (000)	19.037	18.100	13.850	11.860	26.105	30.482	29.409	11.122	1.963	17.424	17.147	29.220	21.764	228.446
BME 2 - PLANT FEED															
PLANT FEED (MET)	tonnes	294,430	325,178	97,115	0	171,675	368,208	364,384	366,505	371,809	326,668	386,523	354,420	400,678	3,533,164
x FEED GRADE (Grade Control)	g/t	1.79	1.69	1.85	0.00	1.89	1.95	1.84	1.88	1.57	1.78	1.81	1.82	1.75	1.79
= GOLD DELIVERED TO PLANT	Oz (000)	16.988	17.668	5.791	0.000	10.432	23.084	21.544	22.153	18.768	18.695	22.443	20.739	22.544	203.860
+ GOLD IN SCATS	Oz (000)	(0.685)	(0.583)	(0.191)	0.000	(0.344)	(0.797)	(1.029)	(0.929)	(0.786)	(0.809)	(0.946)	(0.862)	(0.941)	(8.219)
= GOLD TREATED	Oz (000)	16.303	17.085	5.600	0.000	10.088	22.287	20.516	21.223	17.982	17.885	21.497	19.876	21.602	195.642
+/- PLANT INVENTORY CHANGE	Oz (000)	(0.033)	(0.314)	0.003	0.000	(0.106)	(0.442)	0.111	0.780	(0.174)	0.006	(0.215)	0.237	(0.287)	(0.402)
= GOLD CALLED for	Oz (000)	16.270	16.772	5.602	0.000	9.982	21.845	20.627	22.003	17.807	17.892	21.282	20.113	21.315	195.240
x MINE CALL FACTOR	%	100.10%	101.10%	102.48%	0.00%	101.36%	97.05%	100.63%	95.53%	112.25%	103.48%	100.61%	94.35%	97.46%	100.10%
x RECOVERY FACTOR	%	94.91%	89.35%	92.83%	0.00%	95.50%	96.10%	95.22%	95.47%	96.01%	94.40%	94.86%	95.97%	96.03%	94.91%
= GOLD PRODUCED	Oz (000)	15.457	15.151	5.329	0.000	9.662	20.374	19.765	20.067	19.191	17.477	20.310	18.212	19.950	185.488
BME 3 -TREATMENT															
TONNES TREATED (Check in)	tonnes	282,550	314,447	93,910	0	166,010	355,491	346,987	351,128	356,238	312,528	370,228	339,684	383,946	3,390,599
x YIELD	g/t	1.70	1.50	1.77	0.00	1.81	1.78	1.77	1.78	1.68	1.74	1.71	1.67	1.62	1.70
= GOLD PRODUCED	Oz (000)	15.457	15.151	5.329	0.000	9.662	20.374	19.765	20.067	19.191	17.477	20.310	18.212	19.950	185.488
INFO : PIT TO CRUSHER RECONCILIATION															
TONNES (Line 5 - Line 8)	tonnes	72,974	31,128	153,347	223,570	345,498	217,036	245,423	(171,057)	(338,074)	(196)	(57,305)	193,070	33,246	875,686
x YIELD	g/t	0.87	0.43	1.63	1.65	1.41	1.06	1.00	2.01	1.55	201.44	2.87	1.37	(0.73)	0.87
= OUNCES (Line 7 - Line 10)	Oz (000)	2.049	0.431	8.060	11.860	15.673	7.398	7.864	(11.030)	(16.805)	(1.271)	(5.296)	8.481	(0.780)	24.585
TONNES, STRIPPING RATIOS															
WASTE MINED	bcm	595,096	560,796	632,792	246,822	499,549	603,770	514,955	718,535	742,029	688,558	561,355	587,733	784,263	7,141,157
x RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= WASTE MINED	tonnes	1,577,006	1,486,110	1,676,899	654,078	1,323,804	1,599,991	1,364,631	1,904,118	1,966,377	1,824,679	1,487,591	1,557,492	2,078,297	18,924,066
MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
= MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
= TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	1,944,410	1,842,417	1,927,361	877,648	1,840,977	2,185,235	1,974,438	2,099,566	2,000,111	2,151,151	1,816,808	2,104,982	2,512,221	23,332,916
SCATS TONNES	tonnes	11,880	10,731	3,205	0	5,665	12,718	17,397	15,377	15,570	14,140	16,294	14,736	16,732	142,565
MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL FGO STOCKPILE	tonnes	2,876,432	2,723,086	2,876,432	3,100,002	3,445,500	3,662,536	3,907,959	3,736,902	3,398,828	3,398,632	3,341,327	3,534,397	3,567,643	2,876,432
TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
SCAT STOCKPILE	tonnes	233,538	249,053	233,538	233,538	235,442	245,501	253,994	267,142	282,712	296,852	311,308	304,362	284,604	233,538
REHANDLE	tonnes	236,839	263,674	94,542	0	93,018	238,721	292,890	339,904	364,291	231,835	330,136	261,301	331,751	2,842,064
= DIRECT FEED (Line 8 - Line 40)	tonnes	57,592	61,504	2,573	0	78,657	129,487	71,494	26,601	7,518	94,833	56,387	93,118	68,927	691,100
% REHANDLING	%	80.44%	81.09%	97.35%	0.00%	54.18%	64.83%	80.38%	92.74%	97.98%	70.97%	85.41%	73.73%	82.80%	80.44%
INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	4.29	4.17	6.70	2.93	2.56	2.73	2.24	9.74	58.29	5.59	4.52	2.84	4.79	4.29
DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	4.29	4.17	6.70	2.93	2.56	2.73	2.24	9.74	58.29	5.59	4.52	2.84	4.79	4.29
LOM AVERAGE STRIPPING RATIO	t:t	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06	4.06
DEFERRED STRIPPING TONNES	tonnes	85,703	39,853	660,267	(253,398)	(775,415)	(775,532)	(1,110,593)	1,110,789	1,829,448	499,520	151,289	(664,783)	316,988	1,028,433
GRADE, RECOVERIES															
MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MILL HEAD GRADE (GRADE CONTROL)	g/t	1.80	1.71	1.90	0.00	1.92	1.90	1.85	1.79	1.76	1.84	1.82	1.72	1.71	1.80
CALCULATED HEAD GRADE	g/t	1.79	1.68	1.90	0.00	1.90	1.85	1.86	1.86	1.75	1.84	1.80	1.74	1.68	1.79
RESIDUE GRADE	g/t	0.08	0.18	0.14	0.00	0.09	0.07	0.09	0.08	0.07	0.10	0.09	0.07	0.07	0.08
FGO STOCKPILE GRADE	g/t	1.02	0.97	1.01	1.06	1.09	1.09	1.08	1.04	0.99	0.98	0.95	0.97	0.95	1.02
MARGINAL STOCKPILE GRADE	g/t	2.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.00	2.00	2.00
SCATS STOCKPILE GRADE	g/t	1.80	1.79	1.79	1.79	1.79	1.80	1.81	1.81	1.80	1.79	1.79	1.79	1.80	1.80

Source (Anglogold Ashanti, 2013)

	IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2011							TOTAL
			2011	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2011
	BME 1 - MINED FROM OPEN PITS															
	TOTAL BCM's MINED	bcm	752,097	796,126	905,289	823,240	704,715	799,452	882,722	611,188	772,918	741,380	430,857	723,723	833,554	9,025,163
x	% ORE	%	25.49%	37.38%	29.69%	31.00%	21.24%	22.29%	18.57%	11.31%	26.26%	37.84%	25.99%	16.66%	24.26%	25.49%
=	BCM's ORE MINED	bcm	191,733	297,591	268,825	255,183	149,669	178,219	163,914	69,113	202,939	280,572	111,985	120,577	202,204	2,300,791
x	RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	TONNES ORE MINED	tonnes	508,026	788,616	712,386	675,456	396,622	472,280	434,372	183,149	537,788	743,515	296,759	319,529	535,841	6,096,314
x	IN-SITU GRADE (Grade Control)	g/t	1.36	1.49	1.52	1.49	1.28	1.29	1.21	1.23	1.22	1.47	1.13	1.39	1.20	1.36
=	GOLD MINED	Oz (000)	22.250	37.778	34.814	32.357	16.322	19.588	16.938	7.227	21.172	35.095	10.811	14.256	20.639	266.998
	BME 2 - PLANT FEED															
	PLANT FEED (MET)	tonnes	374,303	402,921	339,485	404,192	335,049	302,060	342,506	397,295	420,029	380,421	416,756	330,893	420,024	4,491,630
x	FEED GRADE (Grade Control)	g/t	1.54	1.74	1.72	1.68	1.60	1.55	1.42	1.30	1.44	1.69	1.40	1.47	1.54	1.54
=	GOLD DELIVERED TO PLANT	Oz (000)	18.579	22.540	18.773	21.832	17.278	15.053	15.615	16.567	19.487	20.633	18.799	15.596	20.769	222.942
+	GOLD IN SCATS	Oz (000)	(0.752)	(0.902)	(0.754)	(0.895)	(0.708)	(0.617)	(0.640)	(0.676)	(0.757)	(0.839)	(0.756)	(0.633)	(0.850)	(9.028)
=	GOLD TREATED	Oz (000)	17.826	21.638	18.020	20.937	16.570	14.436	14.975	15.891	18.730	19.794	18.043	14.963	19.919	213.914
+/-	PLANT INVENTORY CHANGE	Oz (000)	0.014	(0.399)	0.631	(0.462)	1.455	(0.807)	0.692	(0.800)	(0.267)	0.116	(0.035)	0.552	(0.509)	0.166
=	GOLD CALLED for	Oz (000)	17.840	21.239	18.650	20.474	18.025	13.629	15.667	15.091	18.462	19.910	18.007	15.515	19.411	214.080
x	MINE CALL FACTOR	%	96.99%	97.56%	99.16%	93.43%	102.71%	96.04%	94.45%	94.88%	95.94%	94.28%	101.33%	98.12%	95.95%	96.99%
x	RECOVERY FACTOR	%	95.84%	95.86%	96.77%	96.59%	95.43%	94.70%	95.69%	94.64%	95.58%	95.91%	95.92%	95.92%	96.42%	95.84%
=	GOLD PRODUCED	Oz (000)	16.584	19.863	17.897	18.477	17.668	12.395	14.160	13.551	16.931	18.002	17.502	14.602	17.956	199.005
	BME 3 -TREATMENT															
	TONNES TREATED (Check in)	tonnes	359,146	386,795	325,855	387,620	321,312	289,675	328,463	381,072	403,709	364,946	399,993	317,468	402,837	4,309,747
x	YIELD	g/t	1.44	1.60	1.71	1.48	1.71	1.33	1.34	1.11	1.30	1.53	1.36	1.43	1.39	1.44
=	GOLD PRODUCED	Oz (000)	16.584	19.863	17.897	18.477	17.668	12.395	14.160	13.551	16.931	18.002	17.502	14.602	17.956	199.005
	INFO : PIT TO CRUSHER RECONCILIATION															
	TONNES (Line 5 - Line 8)	tonnes	133,724	385,695	372,901	271,264	61,573	170,221	91,866	(214,146)	117,759	363,094	(119,997)	(11,363)	115,817	1,604,684
x	YIELD	g/t	0.85	1.23	1.34	1.21	(0.48)	0.83	0.45	1.36	0.45	1.24	2.07	3.67	(0.03)	0.85
=	OUNCES (Line 7 - Line 10)	Oz (000)	3.671	15.238	16.040	10.526	(0.956)	4.535	1.324	(9.340)	1.685	14.462	(7.987)	(1.340)	(0.130)	44.055
	TONNES, STRIPPING RATIOS															
	WASTE MINED	bcm	560,389	498,535	636,464	568,351	555,046	621,233	718,808	542,076	569,979	460,808	318,872	603,146	631,350	6,724,667
x	RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
=	WASTE MINED	tonnes	1,485,031	1,321,118	1,686,630	1,506,130	1,470,872	1,646,267	1,904,841	1,436,501	1,510,444	1,221,140	845,010	1,598,338	1,673,077	17,820,368
	MARGINAL MINED	bcm	0	0	0	0	0	0	0	0	0	0	0	0	0	0
x	RD MARGINAL	t/m3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
=	MARGINAL MINED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
=	TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	1,993,057	2,109,734	2,399,016	2,181,586	1,867,494	2,118,548	2,339,213	1,619,651	2,048,233	1,964,655	1,141,770	1,917,867	2,208,917	23,916,683
	SCATS TONNES	tonnes	15,157	16,126	13,630	16,572	13,737	12,384	14,043	16,223	16,320	15,475	16,763	13,424	17,187	181,883
	MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	TOTAL FGO STOCKPILE	tonnes	4,326,239	3,953,338	4,326,239	4,598,282	4,659,856	4,830,077	4,921,943	4,707,797	4,825,557	5,188,653	5,068,656	5,057,293	5,173,109	4,326,239
	TOTAL MARGINAL STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	SCAT STOCKPILE	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	REHANDLE	tonnes	315,267	288,314	234,609	331,616	323,156	255,751	314,935	378,120	317,426	316,360	352,694	290,709	379,517	3,783,207
=	DIRECT FEED (Line 8 - Line 40)	tonnes	59,035	114,607	104,876	72,576	11,893	46,309	27,571	19,175	102,603	64,061	64,062	40,183	40,506	708,423
	% REHANDLING	%	84.23%	71.56%	69.11%	82.04%	96.45%	84.67%	91.95%	95.17%	75.57%	83.16%	84.63%	87.86%	90.36%	84.23%
	INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	2.92	1.68	2.37	2.23	3.71	3.49	4.39	7.84	2.81	1.64	2.85	5.00	3.12	2.92
	DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	2.92	1.68	2.37	2.23	3.71	3.49	4.39	7.84	2.81	1.64	2.85	5.00	3.12	2.92
	LOM AVERAGE STRIPPING RATIO	t:t	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25	4.25
	DEFERRED STRIPPING TONNES	tonnes	(671,974)	(2,027,231)	(1,338,058)	(1,361,756)	(213,126)	(358,966)	60,561	658,876	(772,926)	(1,935,715)	(414,986)	241,664	(602,024)	(8,063,686)
	GRADE, RECOVERIES															
	MARGINAL GRADE MINED	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	MILL HEAD GRADE (GRADE CONTROL)	g/t	1.50	1.70	1.70	1.57	1.64	1.50	1.33	1.24	1.39	1.59	1.42	1.44	1.48	1.50
	CALCULATED HEAD GRADE	g/t	1.50	1.67	1.77	1.53	1.79	1.41	1.40	1.17	1.36	1.60	1.42	1.49	1.44	1.50
	RESIDUE GRADE	g/t	0.06	0.07	0.06	0.05	0.08	0.08	0.06	0.07	0.06	0.07	0.06	0.06	0.05	0.06
	FGO STOCKPILE GRADE	g/t	0.98	0.98	1.01	1.02	1.00	1.00	0.99	0.97	0.96	0.98	0.95	0.95	0.92	0.98
	MARGINAL STOCKPILE GRADE	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	SCATS STOCKPILE GRADE	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	-

Source (Anglogold Ashanti, 2013)

IDUAPRIEM GOLD MINE		AVERAGE	SCHEDULE 5					ACTUAL 2012							TOTAL
		2012	JANUARY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER	2012
BME 1 - MINED FROM OPEN PITS															
TOTAL BCM's MINED	bcm	822,301	948,874	809,898	829,163	994,898	966,623	764,225	542,809	476,352	758,641	787,967	944,843	1,043,313	9,867,606
x % ORE	%	19.31%	18.05%	15.88%	23.86%	15.73%	17.94%	16.82%	57.88%	27.25%	18.80%	13.35%	16.67%	9.62%	19.31%
= BCM's ORE MINED	bcm	158,814	171,258	128,624	197,802	156,462	173,391	128,563	314,169	129,797	142,643	105,179	157,466	100,409	1,905,763
x RD ORE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= TONNES ORE MINED	tonnes	420,856	453,835	340,854	524,175	414,624	459,486	340,692	832,547	343,961	378,004	278,724	417,285	266,084	5,050,271
x IN-SITU GRADE (Grade Control)	g/t	1.16	1.26	1.23	1.04	1.10	1.11	1.06	0.98	1.13	1.18	1.26	1.39	1.55	1.16
= GOLD MINED	Oz (000)	15.680	18.407	13.482	17.447	14.706	16.451	11.564	26.232	12.445	14.288	11.281	18.597	13.263	188.163
BME 2 - PLANT FEED															
PLANT FEED (MET)	tonnes	397,425	411,488	394,465	392,811	424,105	418,697	381,977	384,760	436,680	404,214	390,461	345,313	384,130	4,769,101
x FEED GRADE (Grade Control)	g/t	1.31	1.35	1.30	1.28	1.29	1.26	1.41	1.31	1.37	1.25	1.32	1.43	1.21	1.31
= GOLD DELIVERED TO PLANT	Oz (000)	16.776	17.820	16.485	16.127	17.584	16.930	17.338	16.205	19.254	16.245	16.558	15.854	14.907	201.307
+ GOLD IN SCATS	Oz (000)	(0.648)	(0.715)	(0.663)	(0.661)	(0.707)	(0.681)	(0.696)	(0.652)	(0.775)	(0.656)	(0.666)	(0.588)	(0.313)	(7.774)
= GOLD TREATED	Oz (000)	16.128	17.105	15.821	15.466	16.877	16.249	16.643	15.553	18.479	15.589	15.892	15.266	14.593	193.533
+/- PLANT INVENTORY CHANGE	Oz (000)	0.072	(0.206)	0.095	1.174	(0.462)	0.695	0.010	(0.687)	0.074	0.779	0.041	0.017	(0.661)	0.870
= GOLD CALLED for	Oz (000)	16.200	16.899	15.916	16.640	16.414	16.944	16.652	14.866	18.553	16.368	15.933	15.283	13.932	194.403
x MINE CALL FACTOR	%	97.12%	94.28%	95.12%	97.26%	96.24%	99.88%	97.34%	85.59%	93.05%	103.54%	105.57%	99.66%	97.53%	97.12%
x RECOVERY FACTOR	%	95.46%	95.04%	94.06%	95.25%	95.72%	96.21%	95.87%	94.41%	95.40%	95.78%	96.05%	95.96%	95.75%	95.46%
= GOLD PRODUCED	Oz (000)	15.020	15.142	14.241	15.414	15.121	16.283	15.539	12.013	16.470	16.231	16.157	14.617	13.010	180.238
BME 3 -TREATMENT															
TONNES TREATED (Check in)	tonnes	382,118	394,979	378,594	376,706	407,041	401,845	366,653	369,275	419,107	387,901	374,751	332,511	376,058	4,585,421
x YIELD	g/t	1.22	1.19	1.17	1.27	1.16	1.26	1.32	1.01	1.22	1.30	1.34	1.37	1.08	1.22
= GOLD PRODUCED	Oz (000)	15.020	15.142	14.241	15.414	15.121	16.283	15.539	12.013	16.470	16.231	16.157	14.617	13.010	180.238
INFO : PIT TO CRUSHER RECONCILIATION															
TONNES (Line 5 - Line 8)	tonnes	23,431	42,347	(53,611)	131,364	(9,481)	40,789	(41,285)	447,787	(92,719)	(26,211)	(111,737)	71,972	(118,046)	281,170
x YIELD	g/t	(1.45)	0.43	1.74	0.31	9.44	(0.37)	4.35	0.70	2.28	2.32	1.47	1.19	0.43	(1.45)
= OUNCES (Line 7 - Line 10)	Oz (000)	(1.095)	0.587	(3.002)	1.320	(2.878)	(0.479)	(5.774)	10.027	(6.809)	(1.957)	(5.277)	2.743	(1.644)	(13.144)
TONNES, STRIPPING RATIOS															
WASTE MINED	bcm	663,487	693,206	592,926	631,361	838,436	793,232	635,662	401,399	346,556	615,998	682,788	787,378	942,904	7,961,845
x RD WASTE	t/m3	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
= WASTE MINED	tonnes	1,758,241	1,836,996	1,571,254	1,673,107	2,221,855	2,102,065	1,684,503	1,063,707	918,373	1,632,394	1,809,388	2,086,551	2,498,696	21,098,888
MARGINAL MINED	bcm	0	84,410	88,348	0	0	0	0	(172,758)	0	0	0	0	0	0
x RD MARGINAL	t/m3	0.00	2.65	2.65	0.00	0.00	0.00	0.00	2.65	0.00	0.00	0.00	0.00	0.00	0.00
= MARGINAL MINED	tonnes	0	223,686	234,122	0	0	0	0	(457,808)	0	0	0	0	0	0
= TOTAL MINED (Line 5 + Line 26 + Line 29)	tonnes	2,179,097	2,514,516	2,146,230	2,197,282	2,636,479	2,561,551	2,025,196	1,438,447	1,262,333	2,010,398	2,088,113	2,503,836	2,764,780	26,149,159
SCATS TONNES	tonnes	(21,562)	(51,712)	(38,851)	(46,446)	(46,781)	(33,455)	(33,251)	(9,515)	(33,892)	12,430	9,669	4,991	8,072	(258,741)
MARGINAL ORE TREATED	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL FGO STOCKPILE	tonnes	5,161,845	5,215,456	5,161,845	5,751,017	5,741,537	5,782,326	5,741,041	5,731,019	5,638,300	5,612,090	5,500,353	5,572,325	5,454,279	5,161,845
TOTAL MARGINAL STOCKPILE	tonnes	457,808	223,686	457,808	0	0	0	0	(457,808)	0	0	0	0	0	457,808
SCAT STOCKPILE	tonnes	259,437	298,288	259,437	212,991	166,210	132,755	99,504	89,989	0	0	0	0	0	259,437
REHANDLE	tonnes	365,023	357,946	363,762	322,571	379,182	403,830	369,804	342,423	399,621	375,127	369,463	315,251	381,297	4,380,276
= DIRECT FEED (Line 8 - Line 40)	tonnes	32,402	53,542	30,703	70,240	44,923	14,867	12,173	42,338	37,059	29,087	20,998	30,062	2,833	388,825
% REHANDLING	%	91.85%	86.99%	92.22%	82.12%	89.41%	96.45%	96.81%	89.00%	91.51%	92.80%	94.62%	91.29%	99.26%	91.85%
INDIRECT STRIPPING RATIO (Waste + Min Waste / (Ore + Marginal))	t:t	4.18	2.71	2.73	3.19	5.36	4.57	4.94	2.84	2.67	4.32	6.49	5.00	9.39	4.18
DIRECT STRIPPING RATIO (Waste + Min Waste + Marginal) / Ore	t:t	4.18	4.54	5.30	3.19	5.36	4.57	4.94	0.73	2.67	4.32	6.49	5.00	9.39	4.18
LOM AVERAGE STRIPPING RATIO	t:t	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97	4.97
DEFERRED STRIPPING TONNES	tonnes	(333,413)	(194,879)	111,334	(932,045)	161,175	(181,581)	(8,737)	(3,531,859)	(791,112)	(246,286)	424,128	12,645	1,176,258	(4,000,958)
GRADE, RECOVERIES															
MARGINAL GRADE MINED	g/t	0.00	0.77	0.73	0.00	0.00	0.00	0.00	(0.75)	0.00	0.00	0.00	0.00	0.00	0.00
MILL HEAD GRADE (GRADE CONTROL)	g/t	1.27	1.27	1.24	1.23	1.24	1.25	1.37	1.13	1.28	1.29	1.39	1.42	1.18	1.27
CALCULATED HEAD GRADE	g/t	1.28	1.25	1.24	1.34	1.21	1.31	1.38	1.07	1.28	1.36	1.40	1.42	1.12	1.28
RESIDUE GRADE	g/t	0.07	0.06	0.07	0.06	0.05	0.05	0.06	0.06	0.06	0.05	0.05	0.06	0.05	0.07
FGO STOCKPILE GRADE	g/t	0.88	0.96	0.95	0.92	0.90	0.90	0.87	0.87	0.84	0.84	0.82	0.83	0.84	0.88
MARGINAL STOCKPILE GRADE	g/t	0.00	0.77	0.75	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SCATS STOCKPILE GRADE	g/t	1.43	1.72	1.65	1.55	1.43	1.14	0.94	0.85	0.00	0.00	0.00	0.00	0.00	1.43

Source (Anglogold Ashanti, 2013)

Appendix B Spangenbergs check list for checking the status of sampling practice

Potential Influence of Sampling Errors for : ___1 ___

			Rating of Potential Influence of Sampling Errors (1 = low ; 3 = medium ; 5 = high									Average Potential Influence			
	Sampling area & element of sampling		INE	FSE	GSE	IDE	IEE	IPE	PIE	IWE	AE	marks	out of	%	rating
1	EXPLORATION														
	Diamond drilling												5.0	0.0	
	Other														
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!							0.0	
	Sub-sampling														
	diamond saw												5.0	0.0	
	guillotine														
	other														
	Average for sub-section in %			#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!						0.0	
	Average for section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!						**0.0**	
	Info : Reef wide homogenous [X] or narrow composite [ ]														
	QAQC														
	CRM	X													
	blanks	X													
	duplicates	X													
	referee	X													
2	MINING														
2.1	Open-pit grade control sampling														
	Diamond drilling														
	Blast hole sampling														
	RC drilling														
	Other														
	Average for sub-section in %														
	Sub-sampling														
	diamond saw														
	guillotine														
	radial collectors														
	riffler														
	rotary cone - stationary collectors														
	rotary cone - rotating collectors														
	rotary cone - slots														
	stationary cone - stationary collectors														
	stationary cone - rotating collectors														
	stationary cone - slots														
	other														
	Average for sub-section in %														
	Average for section in %														
2.2	Underground grade control sampling														
	Grab														
	Chip											#DIV/0!	5.0	#DIV/0!	
	Coffin														
	Drill: core														
	fines														
	Other														
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!							#DIV/0!	
	Sub-sampling														
	diamond saw														
	guillotine														
	riffler														
	cascade rotary splitter														
	other														
	Average for sub-section in %														
	Average for section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!							**#DIV/0!**	
	Info : QAQC														
	CRM	X													
	blanks	X													
	duplicates	X													
	referee	X													
2.3	Broken ore plant feed sampling														
	Stop-belt														
	Go-belt											#DIV/0!	5.0	#DIV/0!	
	Cross-stream														
	Other														
	Average for section in %			#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!				**#DIV/0!**	
	Info : Individual	X													
	Composite														
	Mass flow: weightometer	6-idler													
	primary														

Source (Spangenberg, 2012)

Potential Influence of Sampling Errors for : ___1 ____

	Sampling area & element of sampling	INE	FSE	GSE	IDE	IEE	IPE	PIE	IWE	AE	marks	out of	%	rating
		Rating of Potential Influence of Sampling Errors (1 = low ; 3 = medium ; 5 = high)									**Average Potential Influence**			
3	METALLURGICAL PLANT													
3.1	Head grade sampling													
	Grab													
	Poppit													
	Cross-stream : launder										#DIV/0!	5.0	#DIV/0!	
	2-in-1													
	Other													
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!		#DIV/0!	#DIV/0!				#DIV/0!	
	Sub-sampling													
	vezin-type										#DIV/0!	5.0	#DIV/0!	
	cascade rotary splitter													
	riffler													
	filter cake													
	other													
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!				#DIV/0!	
	Average for section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!				**#DIV/0!**	
	Info : Mass flow: flowmeter & densitometer — X													
	primary — X													
3.2	Residue grade sampling													
	Grab													
	Poppit													
	Cross-stream : launder										#DIV/0!	5.0	#DIV/0!	
	2-in-1													
	Other													
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!		#DIV/0!	#DIV/0!				#DIV/0!	
	Sub-sampling													
	vezin-type										#DIV/0!	5.0	#DIV/0!	
	cascade rotary splitter													
	riffler													
	filter cake													
	other													
	Average for sub-section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!				#DIV/0!	
	Average for section in %		#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!				**#DIV/0!**	
3.3	Bullion													
	Dip													
	Drill										#DIV/0!	5.0	#DIV/0!	
	Other													
	Average for section in %			#DIV/0!	#DIV/0!	#DIV/0!							**#DIV/0!**	
4	LABORATORY													
	Aliquot selection									2	2.0	5.0	40.0	
	Average for section in %				0.0	0.0	0.0			40.0			**40.0**	
								Total rating for all sections:					**#DIV/0!**	

Potential influence of sampling errors:

Potential influence of management on sampling practice:

Rating of Potential Influence on Sampling Practice (low = 1 ; 3 = medium ; 5 = high)

MANAGEMENT	marks	out of	
COP - Sampling Standard documented (TOS)			
SOP's			
PTO's			
availability of finance			
internal audits			
external audits			
in-house training (operators)			
formal training (management)			
Sampling champion : on mine			
in company			
consultant			
supplier			
Total :	0	0	**#DIV/0!** %

Scale	
Interval in %	Rating
0.0 - 33.3	Low
33.4 - 66.6	Moderate
66.7 - 100.0	High

Source (Spangenberg, 2012)