Technical Report on Peak Gold Mines, New South Wales, Australia

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TABLE OF CONTENTS

SECTION	PAGE

TABLE **PAGE**

FIGURE **PAGE**

1 SUMMARY

Peak Gold Mines is located near Cobar in New South Wales, Australia and is operated by Peak Gold Mines Pty Ltd (PGM), a subsidiary of New Gold Inc. This report is an internal update of the technical report on Peak Gold Mines compliant with the criteria stipulated in the 43-101 guidelines.

PGM comprises two underground mining districts and a gold / copper processing plant. The deposits accessed for underground mining are New Cobar / Chesney and Peak / New Occidental / Perseverance deposits. The Chesney orebody is in advanced state of development and is due to commence production at the end of quarter one, 2009.

1.1 Licences

PGM holds a 100% interest in four consolidated mining leases, a mining lease, a mining purposes lease, and four exploration licences. In addition, PGM has a 75% farm-in agreement on an exploration lease with Zintoba Pty Ltd. and also a 51% farm-in agreement on an exploration lease with Lydail Pty Ltd.

The PGM leases and licenses including farm-in or purchase agreements cover approximately 855 square kilometres surrounding the Peak Mine.

1.2 Mining Overview

Principal mining activities are conducted underground. The Peak, New Occidental and Perseverance orebodies are accessed via a shaft and surface decline located at the Peak site. The Peak site also hosts the processing facility and administration buildings. The New Cobar and Chesney orebodies are accessed via a decline near the base of the New Cobar open pit which ceased mining in 2004. The Chesney No. 3 shaft provides ventilation and an Emergency Egress system for both the New Cobar and Chesney mines.

The primary mining method used at PGM's operations is bench stoping backfilled with waste from development. Production equipment comprises of modern development jumbos, $8m^3$ scooptrams, 45t and 55t trucks plus utility support vehicles. All ore mined at PGM is treated at the Peak processing facility.

Surface oxide mineralization is known to exist at the Chesney and Peak. Undeveloped sulphide mineralization occurs at the New Cobar and Chesney deposits and remains at the New Occidental, Perseverance and Peak deposits.

1.3 Services and Infrastructure

The Cobar Water Board supplies untreated water to the PGM's mining and processing operations. PGM has well established water recycling systems to maximise use of ground water inflows and water recycled from the underground mines. Potable water is pumped from the Cobar Shire Council's water treatment plant to the site.

Power is provided to the Peak site via a 132 kV transmission line and a substation near the Peak Mine. Emergency power is available from two, 0.8MW diesel and one 0.65MW generating units on site, owned by PGM. This is sufficient power for emergency mine egress and to clear some of the processing lines in a power outage.

Sewage from the mines and surface facilities is treated on-site. Other infrastructure, typical of an operating mine, is present at the Peak Mine.

1.4 History

There has been sporadic gold mining in the Cobar district since the 1870's. The district was better known for its copper deposits and was one of Australia's main sources of copper at the turn of the 20^{th} century. Numerous small gold deposits were discovered in the late 1880's, with the Occidental, New Cobar, Chesney and Peak producing gold in the late 1800's. The greatest period of activity at the Peak Mine was from 1896 to 1911. Most gold mining in the Cobar district had ceased by 1920.

The second phase of sustained mining in the district began in 1935, when New Occidental Gold Mines NL re-opened and operated the Occidental Mine as the New Occidental Mine. The New Cobar (or Fort Bourke) and Chesney mines also re-opened in 1937 and 1943, respectively. Mining again ceased in the district with the closing of the New Occidental Mine in 1952. Between 1935 and 1952, the New Occidental Mine produced around 700,000 ounces of gold.

The Peak deposit was discovered in 1981 and PGM was formed to develop the deposit in 1987. Between 1982 and 1985 a total of 30,840 metres were drilled to delineate the Peak deposit. Production commenced at the Peak Mine in 1992.

Subsequent exploration and investigations led to the development of the New Occidental and Perseverance deposits accessed from the existing Peak underground workings.

Exploration at the Fort Bourke Hill historic workings was conducted from 1989. Following the temporary loss of access to the Peak Mine shaft in mid-1998, PGM developed a trial open pit mine in October 1998 at New Cobar. This mining continued until March 1999, extracting approximately 105,000 tonnes of ore. This positive result led to the mining of the New Cobar open pit which produced in excess of 1,000,000 tonnes of ore prior to its completion in February 2004. In June 2004 a decline was commenced to exploit the underground extensions of New Cobar deposit under the open pit. In 2008, a 700m decline was developed from the underground operations at New Cobar to access the Chesney deposit.

1.5 Geological Setting

The PGM deposits are found in the Cobar mineral field, a mining district stretching over a 60 kilometre section of the north-south trending eastern margin of the Cobar Basin.

The deposits of the Cobar field are characterized by a diversity of metal assemblages, from zinc-lead-silver at the Elura deposit (now the Endeavor Mine) to the north through the copper CSA deposit immediately north of the town of Cobar, to the copper (Great Cobar), copper-gold (New Cobar, Chesney and Perseverance), gold (New Occidental) and gold-copper-zinc-lead-silver deposits (Peak) south of town. This southern, gold-rich section of the Cobar mineral field is known as the Cobar gold field.

The Cobar gold field is defined as the 10 kilometre long belt of historical gold mines that extend northwards from the Peak area to the Tharsis workings, immediately north of the township of Cobar. Cobar gold deposits are attractive from the perspective that they tend to be high-grade discrete bodies, which are ideally suited to underground extraction.

The deposits of the Cobar gold field are structurally controlled vein deposits and hosted by shear zones. They are typically steeply dipping, cleavage-parallel, generally north plunging lodes of short strike length (100 to 300 metres) and narrow width (less than 20 metres), but extensive down dip extensions (in excess of 1,200 metres).

1.6 Exploration

The Cobar gold field is a mature mining district that has been extensively explored for metals near surface. The controls on mineralization are well understood and the location of the two principal controlling structures, the Great Chesney Fault and the Peak Shear are well known.

In 2008 PGM undertook close spaced Aero- Magnetic and Gravity surveys over the Cobar gold field and exploration tenements extending southward. PGM is compiling an extensive 3D model including the geophysical data combined with the model developed by the *Predictive Mineral Discovery – Cooperative Research Centre* developed by Geoscience Australia.

1.7 Mineralization

The mineralization is typically associated with extensive silicification, chlorite alteration and a gangue mineralogy dominated by pyrrhotite, pyrite and to a lesser extent, magnetite. The Cobar gold deposits are steeply dipping pipe-like bodies with short surface dimensions but considerable vertical extent. Mineralization occurs within high strain zones and is localized in zones of dilation that typically form around flexures in the shears caused by lithological competency contrasts. Base metal mineralization is present along most of the shear systems within the Cobar gold field, and in places attains economic significance. The gold mineralization, in contrast, occurs in very discrete high-grade bodies focused in the zones of maximum dilation.

1.8 Drilling

PGM uses NQ sized core to intersect mineralisation for resource delineation and LTK48 sized core for the delineation of underground stope blocks. Diamond core is sampled in 1 metre intervals.

Reverse circulation drilling is conducted with 130 to 140 millimetre face-sampling bits to minimize contamination from material in the drill-hole walls. Reverse circulation drilling samples are collected in a cyclone operated by the crew of the rig. Samples are then logged by the geologist and a representative split sent to the laboratory for assay.

Surface drill data available for the 2008 Mineral Resource estimate at the New Cobar mine were acquired in 21 discrete drill programs completed between 1973 and 2008. Of the holes drilled, 485 were selected for use in the Mineral Resource estimate. Subsequent to commencing underground production, diamond drill core from an additional 205 LTK48 holes were available for use in the December 2008 Mineral Resource estimate.

Surface drilling programs have been completed at the Chesney deposit in 2004 and 2005 with additional holes continuing into 2008. A number of holes were drilled to depths between 350 metres and 800 metres below surface. The oxide mineralization on the property has been drill tested to approximately 100 metres below surface using 100 face-sampling reverse circulation drill holes and 47 percussion drill holes. In addition, several HQ/NQ-sized diamond drill holes tested the deposit at deeper levels below the water table. This drilling was completed in five different drill programs between 1987 and 2001. Of the holes drilled, 147 were selected for use in the Mineral Resource estimate. The unoxidized mineralization at Chesney has been drill tested to approximately 550 metres below surface by NQ/NQ2 diameter diamond drill holes. In addition, 16 AX and EX underground diamond core holes were drilled from the lowest level of the mine (270 metres below surface). In 2008, underground access was gained and a further 73 LTK48 holes were completed. In total, 158 holes have been used to estimate the tonnage and grade of the mineralized system immediately below and in the hanging wall of historic workings.

New Occidental Mineral Resource estimates are based on 795 drill holes from drilling campaigns between 1945 and 2008.

Four drilling campaigns were completed in the Peak Mine area between 1997 and 2000, plus a sporadic series of diamond drill holes between 1948 and 1995. The results of 80 drill holes from such programs were used in the Peak oxide resource estimates. The reverse circulation drill programs were sampled every 2 metres and the diamond drill core was sampled on various intervals of less than 2 metres. Three programs of exploration drilling, utilising NQ/2 core, were conducted on Peak Deeps between 2003 and 2006. The results of 22 holes were used to update the Peak Deep resource model in December 2006. A further 18 holes were drilled during 2007 and were used to further delineate the Peak Uppers above 290L as well as Peak Deep portions of the Peak deposit. The underground mine at Peak was in operation from 1992 to 2002 with mining recommencing at a reduced rate in 2005. It has a very large and extensive database of exploration and delineation drill holes, underground mapping, muck sampling and production reconciliation data from which to estimate and reconcile a resource. Since the last Mineral Resource estimate all new holes have been drilled using underground drills; LTK48 core and whole-core sampling, and NQ core with half-core sampling. These are the same drill rigs as currently being used at New Occidental, New Cobar, Chesney and Perseverance.

Almost all underground drilling at the Perseverance deposit was completed using HQ, NQ/NQ2 and LTK48-sized drilling equipment, except for a few wedged holes which were completed using heavy duty CHD-series drill rods. HQ-sized equipment was used to establish collars and complete up to 300 metres of parent hole to facilitate off-hole wedging and directional drilling. LTK48 core was used to assess the whole of Zone A, Zone B, Upper Zone D and Hulk. Hercules lens is drilled much more sparsely. The results of 599 drill holes were used in the Perseverance Mineral Resource estimates.

1.9 Ore Reserve and Mineral Resource Estimates

Ore Reserves and Mineral Resources are estimated by PGM using the CIM definitions. Table 1-1 shows the Ore Reserves for the Peak Mine as of December 31, 2008.

Table 1-1: Peak Gold Mines Pty Ltd –Mineral Reserve Statement dated 31 December 2008 (1) (2) (3) (4)

Deposit	Tonnes ('000)	Grade Au g/t	Grade Cu %	Contained Metal Au Oz ('000)	Contained Metal Cu Mlb
New Occidental					
Proven	162	6.26	0.13	33	0.5
Probable	165	5.15	0.17	27	0.6
Total	328	5.70	0.15	60	1.1
Perseverance					
Proven	81	2.45	2.17	6	3.9
Probable	1,165	5.70	0.94	213	24.1
Total	1,246	5.49	1.02	220	28.0
Peak					
Proven	359	4.73	0.51	55	4.0
Probable	390	4.77	0.70	60	6.0
Total	749	4.75	0.61	114	10.1
New Cobar					
Proven	172	4.08	0.70	23	2.7
Probable	319	4.20	0.64	43	4.5
Total	491	4.16	0.66	66	7.1
Chesney					
Proven	239	1.58	1.94	12	10.2
Probable	448	2.18	1.83	31	18.1
Total	687	2.00	1.87	44	28.3
Surface Stockpiles					
Proven	34	4.55	0.85	5	0.6
Probable	-	0.00	0.00	-	-
Total	34	4.55	0.85	5	0.6
Reclaim Stockpiles					
Proven	-	0.00	0.00	-	-
Probable	70	2.00	0.65	5	1.0
Total	70	2.00	0.65	5	1.0
Total					
Proven	1,048	3.96	0.95	133	21.9
Probable	2,557	4.60	0.96	380	54.4
Total	3,605	4.40	0.96	514	76.3

1) The Mineral Reserves have been reported with the classification criteria of the CIM Definition Standards.
2) The Mineral Reserve was prepared by Eric Strom, a full-time employee of PGM who is a qualified person as defined under NI 43-101.
3) Tonnes and grade of Mineral Reserves are stated on the basis of delivery to plant.
4) Additions subject to the effects of rounding.

Table 1-2 shows the estimated Mineral Resources for the Peak Mine as of December 31, 2008.

Table 1-2: Measured, Indicated and Inferred Mineral Resources (1) (2) (3) (4) (5) (6) (excluding Proven and Probable Ore Reserves)

Deposit	**Class**	**Tonnes ('000)**	**Au (g/t)**	**Cu %**	**Au Oz ('000)**	**Cu Mlb**
Chesney	Measured	535	1.11	1.31	19	15
	Indicated	493	1.92	1.34	30	15
	M&I	**1,028**	**1.50**	**1.32**	**50**	**30**
	Inferred	625	1.98	1.52	40	21
Chesney Total		**1,653**	**1.68**	**1.40**	**89**	**51**
New Cobar	Measured	249	3.58	0.61	29	3
	Indicated	346	3.65	0.58	41	4
	M&I	**596**	**3.62**	**0.59**	**69**	**8**
	Inferred	613	3.49	0.66	69	9
New Cobar Total		**1,209**	**3.55**	**0.62**	**138**	**17**
New Occidental	Measured	95	6.09	0.14	19	0
	Indicated	139	5.01	0.19	22	1
	M&I	**234**	**5.45**	**0.17**	**41**	**1**
	Inferred	197	5.13	0.18	32	1
New Occidental Total		**431**	**5.30**	**0.17**	**73**	**2**
Peak	Measured	281	3.94	0.51	36	3
	Indicated	219	2.81	0.65	20	3
	M&I	**500**	**3.44**	**0.57**	**55**	**6**
	Inferred	100	5.53	0.48	18	1
Peak Total		**600**	**3.79**	**0.56**	**73**	**7**
Perseverance	Measured	192	4.88	1.50	30	6
	Indicated	652	4.54	1.02	95	15
	M&I	**844**	**4.62**	**1.13**	**125**	**21**
	Inferred	975	6.27	0.66	197	14
Perseverance Total		**1,819**	**5.50**	**0.88**	**322**	**35**
	Class	**Tonnes ('000s)**	**Au (g/t)**	**Cu %**	**Au Oz**	**Cu Mlb**
Total	Measured	1,351	3.04	0.96	132	29
	Indicated	1,85	3.50	0.92	208	37
	M & I	**3,200**	**3.31**	**0.94**	**341**	**66**
	Inferred	2,510	4.40	0.83	355	46
Grand Total		**5,710**	**3.79**	**0.89**	**696**	**112**

(1) The Mineral Resources for the Peak Mine deposits set out in the table above have been estimated by R. Berthelsen who is a qualified person under NI 43-101 and a competent person under the JORC Code. The Mineral Resources are classified as Measured, Indicated and Inferred, and are based on the JORC Code.

(2) The Mineral Resources were estimated using three-dimensional multiple indicator and ordinary kriged block models, constrained by geological and grade domains.

(3) A$95 net smelter return cut-off was applied to in-situ Mineral Resources at Peak, Perseverance and New Occidental while at New Cobar and Chesney A$86 was used, along with appropriate recoveries.

(4) Excluded from the Identified Mineral Resources are mined material and material unlikely to be converted to reserve status for engineering or technical reasons and remnant stope pillars, skins and other material sterilized as a result of mining as well as discontinuous mineralization.

(5) Mineral Resources are not known with the same degree of certainty as Mineral Reserves and do not have demonstrated economic viability.

(6) Numbers may not add up due to rounding.

Table 1-3: Measured, Indicated and Inferred Mineral Resources (1) (2) (3) (4) (5) (6) (including Proven and Probable Ore Reserves)

Deposit	Class	Tonnes ('000)	Au (g/t)	Cu %	Au oz ('000)	Cu Mlb
Chesney	Measured	765	1.52	1.63	37	27
	Indicated	776	1.95	1.64	49	28
	M&I	**1,541**	**1.74**	**1.63**	**86**	**56**
	Inferred	674	1.92	1.67	42	25
Chesney Total		**2,216**	**1.79**	**1.64**	**128**	**80**
New Cobar	Measured	429	4.14	0.69	57	7
	Indicated	585	4.10	0.63	77	8
	M&I	**1,014**	**4.11**	**0.66**	**134**	**15**
	Inferred	649	3.58	0.65	75	9
New Cobar Total		1,663	3.91	0.65	209	24
New Occidental	Measured	205	6.81	0.13	45	1
	Indicated	268	5.63	0.19	49	1
	M&I	**473**	**6.14**	**0.16**	**93**	**2**
	Inferred	215	5.17	0.17	36	1
New Occidental Total		**688**	**5.84**	**0.17**	**129**	**3**
Peak	Measured	653	5.02	0.59	105	8
	Indicated	527	4.10	0.70	69	8
	M&I	**1,180**	**4.61**	**0.64**	**175**	**17**
	Inferred	125	5.60	0.47	23	1
Peak Total		**1,305**	**4.70**	**0.62**	**197**	**18**
Perseverance	Measured	243	4.47	1.76	35	9
	Indicated	1,538	6.39	1.07	316	36
	M&I	**1,781**	**6.12**	**1.16**	**350**	**46**
	Inferred	1,048	6.34	0.67	214	16
Perseverance Total		**2,830**	**6.20**	**0.98**	**564**	**61**
Stockpiles	Measured	110	2.87	0.67	10	2
Stockpiles Total		**110**	**2.87**	**0.67**	**10**	**2**
Total	**Class**	**Tonnes**	**Au (g/t)**	**Cu %**	**Au (oz)**	**Cu Mlb**
	Measured	2,405	3.75	1.02	290	54
	Indicated	3,694	4.71	1.00	559	82
	M&I	**6,100**	**4.33**	**1.01**	**849**	**136**
	Inferred	2,713	4.45	0.87	388	52
	Total	**8,812**	**4.37**	**0.97**	**1,238**	**188**

(1) The Mineral Resources for the Peak Mine deposits set out in the table above have been estimated by R. Berthelsen who is a qualified person under NI 43-101 and a competent person under the JORC Code. The Mineral Resources are classified as Measured, Indicated and Inferred, and are based on the JORC Code.

(2) The Mineral Resources were estimated using three-dimensional multiple indicator and ordinary kriged block models, constrained by geological and grade domains.

(3) A$95 net smelter return cut-off was applied to in-situ Mineral Resources at Peak, Perseverance and New Occidental while at New Cobar and Chesney A$86 was used, along with appropriate recoveries.

(4) Excluded from the Identified Mineral Resources are mined material and material unlikely to be converted to reserve status for engineering or technical reasons and remnant stope pillars, skins and other material sterilized as a result of mining as well as discontinuous mineralization.

(5) Mineral Resources are not known with the same degree of certainty as Mineral Reserves and do not have demonstrated economic viability.

(6) Numbers may not add up due to rounding.

1.10 Mineral Processing and Metallurgical Testing

All the New Cobar open pit ore has been processed save approximately 20,000 tonnes of partially oxidised ore which is batched through the process plant to supplement the underground feed.

Treatment of New Occidental ore, which constituted approximately 60% of the process feed during 2005 is planned to decline and finish in 2011. The ore has been subjected to a comprehensive testing program that commenced in 1996 with initial mineralogical studies. This testing program confirmed the benefit of finer grinding to 80% passing 53 microns and extending the leach time from the current 22 hours to 48 hours. Analysis of performance through the circuit suggests fine grinding to only 75 microns is required if the flotation circuit is operating.

Drill core samples from Perseverance have also been subjected to a comprehensive program of mineralogical examination and metallurgical testing. In addition to testing of Perseverance alone, testing was also conducted on blends with New Occidental ore at the finer grind required for the latter. The testing showed that Perseverance ore is similar in character to the high copper Peak ore, with gold recovery of over 93% using Peak conditions and copper recovery of over 65% to a concentrate. Blending was found to be of benefit to total gold recovery, despite lower gravity recovery with the fine grind required for New Occidental ore, because of a fine-grained gold component in the Perseverance ore and recovery of the New Occidental refractory gold to the copper concentrate.

Testing was performed on Chesney ore by Metcon in 2004. Results indicate low gravity gold recovery with good response to flotation and subsequent cyanidation for an overall recovery of gold above 95%. Copper recovery was acceptable at 72% with reasonable rougher concentrate grades.

1.11 Mining Operations

Mine production operations are located in four distinct underground zones, plus ore stockpiled from the completed open pit operation at New Cobar. Current mining is from zones which are contiguous to, or nearby, earlier mined out areas. Production operations have been underway at New Occidental since December 2001, while development commenced at Perseverance in October 2002 with production commencing in July 2003. The New Cobar open pit mine was completed in February 2004, with the mined ore stockpiled for subsequent treatment. The New Cobar underground development commenced in June 2004 and was commissioned in September 2005. The Chesney ramp development was commenced in 2006 and development of the first mining horizon began in the second quarter 2008.

Access development to the New Occidental and Perseverance deposits allows 45 and 55 tonne truck haulage from the two zones to the Peak Mine crushing and hoisting infrastructure. Ore is hauled to a Jaques jaw crusher on the 630L. Ore from the Peak deposit generally reports to an orepass system which gravitates ore to the crusher access. Crushed ore is loaded into the 10 tonne skip and hoisted to the surface, where it is stockpiled for milling. The Peak shaft is concrete lined, 5.3 metres in diameter and 740 metres deep. The hoisting system is designed to provide capacity in excess of 700,000 tonnes per year.

Ore from New Cobar and Chesney is hauled to the surface using 45 and 55 tonne trucks and stockpiled in the New Cobar ROM pad. The ore is then screened and sized and hauled by surface road trains to the Peak site where it is stockpiled in preparation for milling. All stockpiled ore is fed into the No. 5 feeder with a surface loader.

The mining and milling operations operate on a 4 panel roster with two 12 hour shifts per day, 365 days per year. Maintenance is conducted by PGM personnel and contractors carry out the diamond drilling activities.

Drilling has identified significant down-dip extensions to the Perseverance orebody and also an up dip extension to the Peak orebody. Extensions to the declines are progressing in the New Occidental, Perseverance, New Cobar and Chesney areas to enable further mining. The mining of remnant ore around the original Peak Mine is continuing. The Peak Mine access ramp extension to the surface was completed in December 2007.

At Peak, Perseverance, New Occidental and New Cobar the mining method employed is bench stoping with a 20 metre sublevel interval. At Chesney a combination of bench stoping and open stoping will be employed using a 35m sublevel interval. Mining progresses from bottom up in each panel. Drives are developed along strike in the ore on each level, a slot is developed and ore is blasted into the void. Ore is extracted on retreat to the access crosscut. Waste rock is used to backfill the stoping void.

In wider sections of the orebody (plus 8 metre) and longer strike lengths, footwall or hangingwall drives are also developed. Once the mining void is large enough, waste rock is introduced to the mined out section of the stope in a modified Avoca style mining method in order to increase stability. Minimal pillars are required. An option of Cemented Rock Fill is being examined to assist mining in wide, high value zones.

1.12 Mine Plan

The mine plan based on Proven and Probable Reserves only, is sufficient for five years of mine life. PGM has an active drilling and development program that provides for annual conversion of Resources to Reserves during the 16 year history of the mine. PGM believes that mine life can be extended for a further four years by mining material currently classified as Inferred Mineral Resources supplemented with material from advanced exploration projects.

1.13 Milling Operations

The original flexible process design required by the variability of the Peak ore zones, consisting of SAG milling, gravity, sulphide flotation, cyanide leaching, carbon in leach adsorption, electrowinning and doré production, has facilitated the introduction of new ore sources with different mineralogical and metallurgical characteristics. The original plant capacity of 450,000 tonnes per year was increased incrementally through improvements in removing bottlenecks to 660,000 tonnes per year of hard sulphide ore. Increased leach capacity and the addition of a ball mill were added to treat the finer grinding requirements of the New Occidental ores. In 2008, the mill processed 768,728 tonnes.

Ore from Peak, Perseverance and New Occidental is crushed underground to a nominal top size of 150 millimetres and is delivered to a 6,000 tonne live capacity surface stockpile. Three reciprocating plate feeders deliver ore to the SAG mill feed conveyor from the stockpile. A separate bin equipped with a static grizzly and a reciprocating plate feeder delivers New Cobar and Chesney ore to the SAG mill feed conveyor. The bin is fed by front-end loader from a stockpile of trucked ore.

Gold and silver are recovered in a gravity circuit with Knelson concentrators, then concentrated in the Gekko (intensive leach reactor), electrowon and sludge smelted in a gas fired furnace to produce gold doré bars.

Gold, silver and copper are also recovered as a copper concentrate in a column flotation circuit. The flotation concentrate is thickened, dewatered and stockpiled prior to transporting to the smelter.

The third method of gold and silver recovery is with cyanidation in a tank leach circuit. The flotation tailings are pumped to a series of two leach tanks and seven adsorption tanks. Cyanide and activated carbon are used to recover the gold and silver. A solution of heated caustic cyanide is used in the stripping circuit to recover the gold and silver from the carbon. An electrowinning circuit in the gold room recovers the gold and silver from the strip circuit solution. The resulting sludge is smelted into gold doré bars.

Leach tailings are pumped to a thickener. High-density thickener underflow is pumped to a central discharge tailings storage facility. Water is reclaimed from the thickener overflow and reused within the process.

1.14 Markets and Contracts

Copper concentrate is sold under contract to Glencore International AG. The current contract is due to expire in the first half of 2009. Budgeted production of copper concentrate for 2009 is estimated at 33,800 DMT grading 19.6% copper and 33 grams of gold per tonne. Penalty elements include bismuth, lead and zinc.

Doré bullion is refined under contract by the Perth Mint.

1.15 Environmental Considerations

Enesar Consulting Pty Ltd. (formerly NSR Consultants Pty Ltd.) conducted external environmental audits of the PGM tenements in June 2002, April 2004 and August 2006. While three high ranking environmental issues were identified during the audits, these were addressed as a priority. These three issues classified were related to tailings management.

The report noted a number of significant improvements since the previous audit in 2004, most notably in water recycling, implementation of dust suppression measures and upgrading of the tailings pipelines. No issues classified of very high importance were identified. PGM is due for another external audit in July of 2009. This audit will be a benchmarking against the international standards ISO 14001, the International

Cyanide Code and the World Bank Equator Principles. PGM is using the standard ISO 14001 as a guideline for its environmental management system.

PGM recorded 4 non-compliances during 2008. Three of these were administrative non-compliances that were immediately resolved, while the remaining non-compliance led to an infringement notice and fine following water being discharged off-site following a significant rain event (approximately 1-in-100 yr).

PGM has a responsibility under state law to rehabilitate areas of historic mining as well as current mining activities on its leases to an agreed end land use. The current closure cost estimate is now in the amount of AU$11.4M.

1.16 Taxes

All production from PGM mines is subject to a royalty payable to the State of New South Wales at a net effective rate of 3% of gross revenue.

Both New South Wales state and Australian federal tax are levied on the proceeds from the PGM operations. Federal income tax, after appropriate eligible deductions, is imposed at 30%, while New South Wales state tax effectively is a mining royalty set at approximately 3% of gross revenue, before treatment charges and all other costs. Payroll tax of approximately 6% is incurred on the payroll.

1.17 Production Estimates

The PGM operation is expected to continue to draw the majority of its economic value from the sale of gold in doré bullion. In addition, a concentrate containing copper, gold and silver will continue to be produced for sale.

Most of the planned production is derived from ore mined at the underground operations from the Peak, New Occidental, Perseverance, New Cobar and the Chesney orebodies currently under development.

PGM plans that the total scheduled Mineral Reserves and a portion of the material derived from Resources will be mined and processed. According to the Life of Mine Plan (published third quarter 2008), the planned tonnes, gold and copper output, are approximately 6.8 million tonnes, approximately 825,000 ounces of gold and approximately 147 million pounds of copper respectively over a period of approximately nine years. Mineral Reserves contain 3.6 million tonnes containing 514,000 ounces of gold and 76 million pounds of copper, the balance of material in PGM's Life of Mine Plan is drawn from material currently classified as Inferred Resources and other mineralised material. Metal sales in 2009 are expected to be approximately 92,800 ounces of gold and approximately 14.6 million pounds of copper.

1.18 Conclusions

Geology, Data and Resources
PGM has estimated a total Mineral Resource (exclusive of Mineral Reserves), of 3.2Mt grading 3.31g/t Au and 0.94% Cu in the Measured and Indicated categories and 2.5Mt grading 4.4g/t Au and 0.83% Cu in the Inferred category for the Peak, New Occidental, Perseverance, New Cobar and Chesney deposits.

The Geology Department has detailed plans for future near mine and regional exploration efforts.

Mining
Mining conditions are well understood by PGM. The mining methods and equipment fleets are appropriate for the deposits and forecast rates of production. Infrastructure in place at the mine is appropriate for a mine of its size. Refrigeration has been installed recently to allow all year round development in the deeper sections of Perseverance.

Two new ore sources are planned to enter production in 2009. Chesney is an advanced project. Hosted in good quality rock this orebody has the opportunity for high production rates. Perseverance Zone D is a deeper, down dip and plunge extension of Zone A.

The mine plan based on Proven and Probable reserves only, is sufficient for approximately five years of mine life. PGM is operating to a nine year Life of Mine Plan 2009-2017 that includes an increasing level of Inferred Mineral Resources as the plan matures. Continued focused attention is required on exploration to identify Mineral Resources and delineation for identification of economically mineable material.

Metallurgy
Budgeted metallurgical recoveries are determined from a combination of operating history and, where necessary, metallurgical test work. Recoveries for New Occidental, Perseverance, New Cobar (POX), New

Cobar (Underground) and Peak ores are based on historical operating data. PGM has a system which considers the metallurgy of the individual ores and then produces a predicted recovery based on the feed mixes. The risk of adverse metallurgical responses (such as from any as yet unidentified mineralogical assemblages) can be reduced by performing standard metallurgical tests on composite samples representative of the production plant.

The current age of the processing plant is around 17 years. The planned plant availabilities are in excess of 95%. Maintenance standards are high in the plant and there is a continual program of corrosion repair.

Environmental and Closure Plan

No significant Aboriginal heritage issues have been identified in relation to PGM's operations; however, PGM maintains a strong relationship with the local indigenous community and has provided land for the establishment of a native plant nursery which will supply PGM with rehabilitation products.

PGM updated its Closure Plan and re-estimated closure costs in 2008 at A$11.4M. The Closure Plan is reviewed and updated annually. As with all closure cost estimates, there are uncertainties involved. The greatest uncertainties involve changes and uncertainty over closure requirements and criteria which may add significantly to cost of closure. PGM is currently involved in dialog with regulators and the community to reduce this uncertainty.

1.19 Recommendations

Rehabilitation

PGM has estimated the closure cost to be A$11.4M. This cost can be substantially reduced by progressive rehabilitation. Opportunities exist to rehabilitate areas with the waste products from mining as well as in the mining process by utilising waste products as underground fill.

Mining

The Peak deposit has associated with it zones of lead and zinc that are potentially economic. These zones can be processed with the lead / zinc depressed and treated as penalty elements in the NSR script. Alternately, a lead and zinc circuit can be introduced to the process plant. Lead and zinc could then be recovered and provide an additional income stream. Further work needs to be done to determine the economics of recovering lead & zinc from Peak ores. The A$35,000 that has been budgeted in 2009 for engineering studies should be progressed and possibly increased.

Copper Flotation

PGM is proposing to introduce Flash Flotation in 2009. The Flash Flotation is proposed to increase copper recovery, gain the ability to handle higher grade feeds and to increase the copper grade of the copper concentrate. These are sound reasons and it is recommended that a justification be developed for introducing this process. A$1.2M has been budgeted for this upgrade in 2009.

Regional Exploration

PGM holds arguably the best ground in the Cobar district. A project of 3D alteration modelling may provide additional regional exploration targets which should be ranked along side existing targets and investigated as a priority. Capital should be allocated based on prospectivity.

2 INTRODUCTION AND TERMS OF REFERENCE

Peak Gold Mines Pty Ltd are providing an internal review of the Technical Report compiled initially by Mine and Quarry Engineering Services Incorporated (MQes) in April 2007, of the Mineral Resources, Reserves and operational facilities of Peak Gold Mines Pty Limited (PGM), a subsidiary of New Gold Inc.

PGM is a wholly owned subsidiary of Peak Gold Asia Pacific Pty Ltd, in turn a wholly owned subsidiary of New Gold Inc. The assets, which are located near the town of Cobar, New South Wales, comprise of two underground mining districts, a gold / copper concentrator, administration block and associated land, equipment and housing. Bullion is refined in Australia and sold through bullion banks. Copper concentrates with contained gold and silver are sold locally for subsequent export.

The principal deposits of interest under PGM's control are New Cobar, New Occidental, Peak, Perseverance and Chesney. Underground mining and development is currently occurring at the New Occidental and Perseverance deposits using the Peak shaft and surface decline as access. Mining of an open pit at New Cobar ceased in 2004, however some stockpiles of partially oxidised material continued to be treated. A separate underground mine has been developed at New Cobar and Chesney with decline access from within the completed open pit. Remnant ore and up dip extensions of the Peak orebody are currently being developed. Surface oxide mineralization is known to exist at Chesney and Peak.

The review of the assets of PGM is provided in subsequent sections of this report.

PGM estimates and reports its mineral resources and mineral reserves using the current (2004) version of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code). For this reason, and because of the use of extracts from PGM reports, JORC terminology will be used throughout the report and provide a reconciliation to the CIM Definition Standards – For Mineral Resources and Mineral Reserves (December 11, 2005) in Sections 17 and 19, Mineral Resources and Mineral Reserves.

The Qualified Persons compiling and reviewing this Technical Report are:

Name	**Position in Company**	**Qualifications**	**Membership**	**Sections of Report Responsible for**
Peter Lloyd	General Manager	Bachelor of Engineering (Mining)	Fellow AusIMM	1, 2, 3, 4, 5, 6, 16, 18, 19, 20 & 21
Eric Strom	Mine Technical Services Superintendent	Bachelor of Engineering (Mining)	Licensed Professional Engineer, Ontario	17.8, 17.9
Rex Berthelsen	Principal Geologist	Bachelor of Applied Science (Geology), Post Graduate Diploma of Science (Economic Geology)	Member AusIMM	7, 8, 9, 10, 11, 12, 13, 14, 15, 17.1 – 7, 22

3 RELIANCE ON OTHER EXPERTS

PGM has reviewed and analysed data provided by its consultants and previous operators of the mine properties, and has drawn its own conclusions there from augmented by its direct operational experience. PGM is an established operation with over 17 years of continuous operational history.

The metallurgical, geological, mineralization, exploration and certain procedural descriptions, figures and tables used in this report are taken from reports prepared by PGM and its consultants. PGM is the company which holds the mining leases and exploration licenses and operates the mines in Cobar.

PGM estimates and reports its mineral resources and mineral reserves using the current (2004) version of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The PGM properties are situated in the vicinity of Cobar which is located at 31°34'S 145°53'E (6,515,000N 390,000E in UTM Zone 55J), approximately 600km northwest of Sydney, New South Wales, Australia as shown in Figure 4-1. The township of Cobar has a population of approximately 7,000. The Cobar gold field is defined as the 10-km long belt of operating and historic gold mines that extend northwards from the Perseverance – Peak gold mine area to the Tharsis workings, immediately north of the township of Cobar.

Figure 4-1: Location of Peak Mine

4.2 Tenement Status

The PGM properties include four Consolidated Mining Leases (CML 6, 7, 8 and 9) covering the Tharsis to Peak gold mine area, the Coronation-Beechworth area and Queen Bee area; plus a Mining Lease, Mining Purposes Lease and four Exploration Licences (EL 5933, 6149, 6401 and 6402; (Table 4-1; Figure 4-1 and Figure 4-2).

Table 4-1: Mining Tenements Held or in Farm-in Agreements by PGM

Peak Gold Mines: Location & Expiry Dates of Mining and Exploration Leases as at February 2009								
ML No	**Location**	**Granted**	**Expires**	**Ownership**	**Status**	**Area ha**	**Expenditure Commitment**	**Comment**
CML6	Fort Bourke Hill	29-Apr-96	27-Mar-15	PGM 100%	Granted	1,303.0	$ 300,000	within EL5933
CML7	Coronation/Beechworth	28-Jun-95	13-Feb-12	PGM 100%	Granted	1,185.5	$ 300,000	within EL5933
CML8	Peak to Occidental	16-Sep-99	13-Feb-12	PGM 100%	Granted	1,250.0	$ 300,000	within EL5933
CML9	Queen Bee	26-Sep-95	3-Aug-13	PGM 100%	Granted	527.3	$ 300,000	within EL5933
ML1483	Fort Bourke Hill	30-Apr-01	27-Mar-15	PGM 100%	Granted	47.1		within EL5933
MPL854	Dam	29-Sep-36	13-Feb-12	PGM 100%	Granted	3.9		within EL5933
					TOTAL	4,316.8	$ 1,200,000	
EL No	**Location**	**Granted**	**Expires**	**Ownership**	**Status**	**Area ha**	**Expenditure Commitment**	**Comment**
EL5933	Peak	17-Apr-02	16-Apr-10	PGM 100%	Renewed	26,390.0	$ 125,000	area includes CML's, GL, ML, MPL
EL5982	Norma Vale Farm In	30-Aug-02	29-Aug-10	Zintoba 100%	Renewed	5,250.0	$ 48,000	75% interest in Farm-in with Zintoba Pty Ltd
EL6127	Rookery South	24-Sep-03	23-Sep-09	PGM 51% Lydail 49%	Renewed	27,220.0	$ 69,000	51% interest in Farm-in with Lydail Pty Ltd
EL6149	Mafeesh	17-Nov-03	16-Nov-09	PGM 100%	Renewed	2,332.0	$ 38,000	
EL6401	Rookery East	5-Apr-05	4-Apr-09	PGM 100%	Renewed	15,460.0	$ 56,000	Property reduction as of April 2007
EL6402	Cable Downs	13-Apr-05	12-Apr-09	PGM 100%	Renewed	25,810.0	$ 74,000	Property reduction as of April 2007
* Inclusive of CML, ML, MPL					**TOTAL***	102,462.0	$ 410,000	

Two additional Exploration Licenses (EL 5982 and 6127) are under farm-in agreements as noted in Table 4-1 and described below.

Consolidated Mining Leases ("CML") 6, 7, 8, and 9 are held in the name of Peak Gold Mines 100% and are operated by Peak Gold Mines (Figure 4-3).

- New Cobar and Chesney Mines are contained within the CML 6 tenement.
- No active mines are located within the CML 7 tenement.
- New Occidental, Peak and Perseverance Mines are located within the CML 8 tenement.
- No active mines are located within the CML 9 tenement.

Mining Purposes Lease ("MPL") 854 and Mining Lease ("ML") 1483 are held in the name of Peak Gold Mines and are operated by PGM.

Exploration Licences ("EL") 5933, 6149, 6401 and 6402 are held 100% in the name of Peak Gold Mines and are operated by PGM. A renewal application for EL 5982 has been lodged with the New South Wales Department of Primary Industries – Mineral Resources (DPI-MR) (see Table 4-1 and Figure 4-2).

Figure 4-2: PGM Properties, and Licence and Lease Boundaries

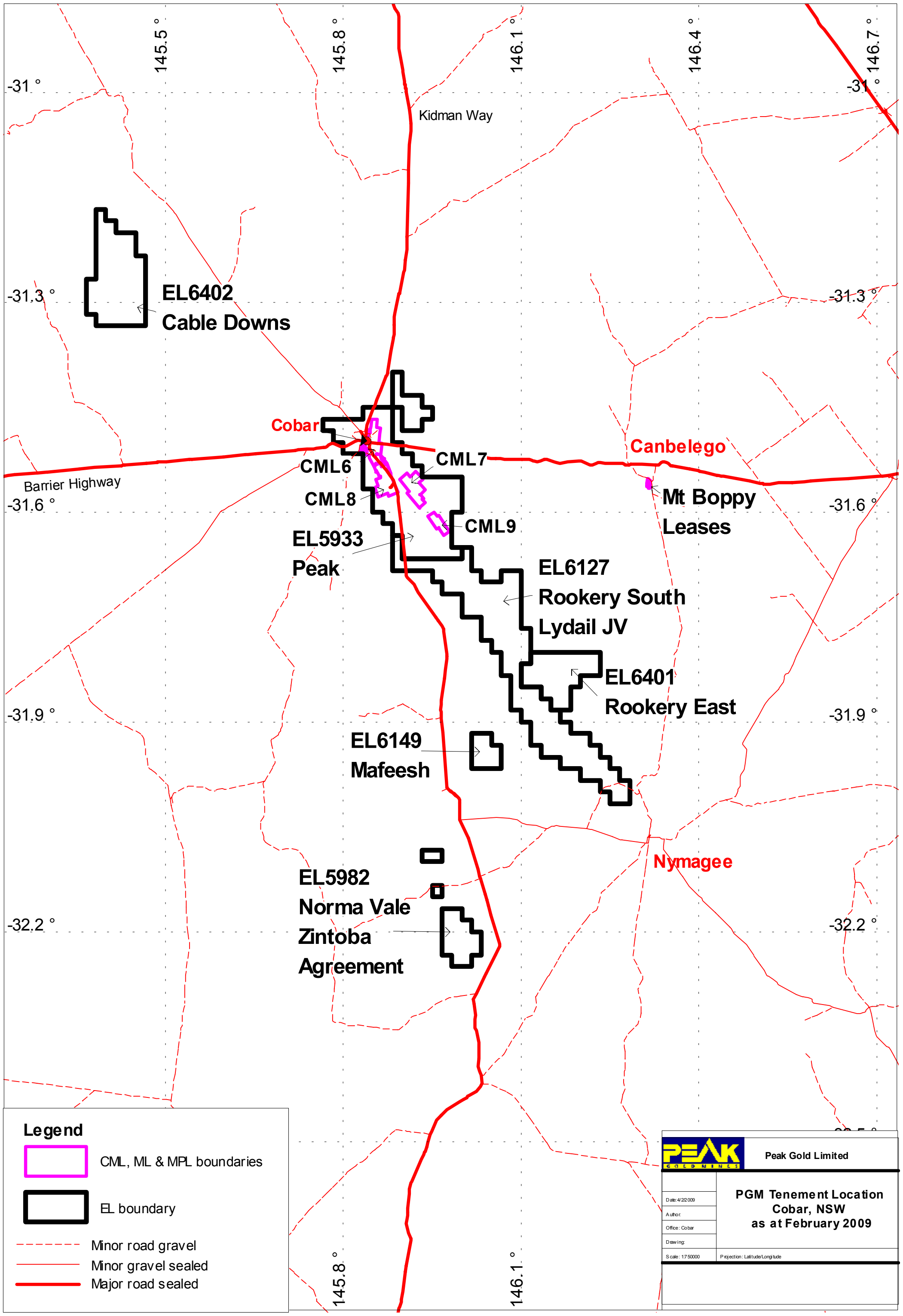
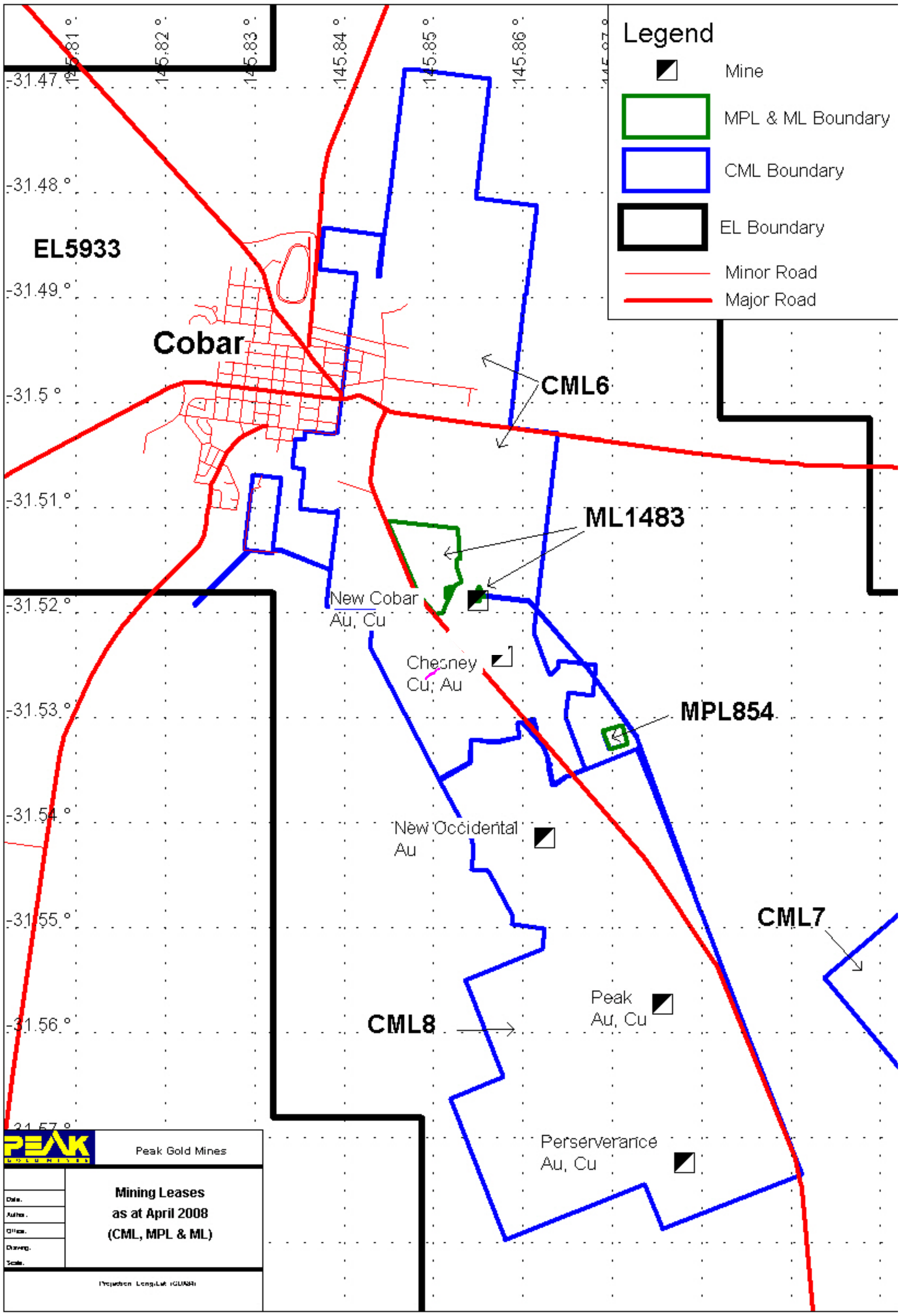


Figure 4-3: Consolidated Mining Lease 6, ML 1483 and MPL 854



EL 5982 is held in the name of Zintoba Pty Ltd and is operated by Peak Gold Mines. A Letter of Agreement with Zintoba was entered into in March 2005, whereby PGM was required to spend $150,000 by 29th August 2006 to earn a 75% interest in the tenement. The expenditure was completed by the due date. Negotiations are continuing with Zintoba to extend the timeframe for evaluation until August 2009.

EL 6127 is held in the names of Peak Gold Mines Pty Ltd (51%) and Lydail Pty Ltd (49%) and is operated by PGM. PGM entered into a Letter of Agreement with Lydail on the 11th July 2003. PGM was required to spend $900,000 within 3 years to earn 51%. This condition has been met. Under the terms of the agreement, PGM have elected to complete expenditure of a further $1,350,000 before 11 June 2010 to earn 75% equity in EL 6127.

The mining leases and exploration licenses held directly by PGM or under farm-in or purchase agreements cover approximately 850km^2 surrounding the Peak gold mine (Figure 4-2). These form an irregular shaped area straddling the Nurri-Chesney Anticline (see Figure 7-3).

Principal mining activities are conducted at the underground Perseverance, New Occidental and Peak orebodies accessed via the Peak shaft, located 8.5km south-southeast of Cobar on CML 8 and at the underground mine at New Cobar, located 3km southeast of Cobar on CML 6. Stockpiles of partially oxidised ore are also located at New Cobar. A central processing facility is situated at the Peak Mine site. The New Occidental deposit is located 3km north of the Peak Mine and is accessed underground, by a decline from the Peak Mine, as is the Perseverance deposit, which is located 1km south of the Peak. Both utilize the mining and processing infrastructure at the Peak gold mine.

Exploration targets in the area include Great Cobar, Gladstone, Dapville, Young Australia, Mt Pleasant, Queen Bee, Rookery South, Mafeesh and Norma Vale prospects.

4.3 Royalties and Agreements

4.3.1 Royalty

A royalty is paid by PGM to the Department of Primary Industries (formerly the New South Wales Department of Mineral Resources) for the right to extract and use the State's resources. The formula for payment of 4% of metal sales values, less direct treatment charges, realization expenses and a proportion of administration expenses and depreciation. The net effective rate of royalties is 3% of metal sales value and this rate of royalty payment has been used in the economic evaluation of the operations.

4.4 Environmental Liabilities

There has been considerable historical activity on the PGM properties. As holder of CMLs 6 through 9, PGM has a responsibility under the law for addressing the environmental impacts of historic as well as current mining activities on its leases. A Conceptual Closure Plan was prepared by Enesar in 2001 with costings updated yearly to ensure adequate provisions for closure activities. Closure costs were estimated in 2008 to be approximately A$11.4M.

4.5 Permitting Status

Table 4-2 list the environmental and operating permits that are additional to the Mining Lease and Exploration License documents described in Section 4.2.

Table 4-2: Peak Mine Environmental and Operating Permits

Environmental / Operating Permit Title	License / Permit No.	Expiry Date
EPA License*	# 3596	Review date: 15 Dec 2013
EPA Radiation Control License	# 29910	4/6/2011
Peak Development Consent	2004/LDA-00003	N/A
New Occidental Development Consent	27:89 00/01:002	N/A
New Cobar Open Cut Development Consent	99/00:022 98/99:008	N/A
New Cobar Underground Development Consent	2004/CDA-00003	N/A
Peak Surface Decline Development Consent	2007/LDA/00080	N/A

Mining Operations Plan	N/A	Accepted – August 2005 Update due – August 2010
Explosives Licence	N/A	18/4/2011
Dangerous Goods Notification (New Cobar Site)	N/A	Application lodged and is currently under review
Dangerous Goods Notification (Peak Site)	35/026523	18/4/2009
Tourist Permit	N/A	N/A
Mining Leases	*See Section 4.2*	
GL, MPL, EL	*See Section 4.2*	

* PGM is still the official holder of the Environment Protection Licence No. 11583 for the Mt Boppy Mine at Canbelego. PGM is currently reviewing its options with respect to this licence.

Under NSW legislation PGM is also required to submit an annual Type 3 Surveillance Report for the Tailing and Tailing Decant dams. The 2008 report is currently being assessed by the Dam Safety Committee of NSW but as no major issues were found during the surveillance inspection, the Committee is likely to accept the report with few conditions (Section 19.2.1).

PGM is not aware of any current native title claims that have been made to its tenements.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Access

The Peak Mine is accessed by driving 8km south from the township of Cobar on the Cobar-Hillston Road to a 0.5km private sealed road. Regional road access through Cobar is provided by an all weather highway (Mitchell Highway #32) between Sydney and Adelaide (Table 5-1). Road distances from Sydney are provided in Table 5-1. Connection to the NSW rail service is available at Cobar via Nyngan and extends to the port at Newcastle.

Table 5-1: Distances by Road from Sydney to Cobar

From	To	km	Highway #	Highway
Sydney	Bathurst	200	#32	Mitchell
Bathurst	Dubbo	210	#32	Mitchell
Dubbo	Nyngan	170	#32	Mitchell
Nyngan	Cobar	130	#32	Mitchell
Total		**710**		

A regional airport services Cobar. Commercial flights from Cobar to Dubbo were suspended at the end of 2008. Dubbo airport is a 3 hour drive from Cobar where regular commercial flights operate to Sydney. The flight duration is approximately 1 hour.

Ore produced from the New Cobar and Chesney deposits is transported by road 7km to the Peak processing facility. PGM's concentrates are currently transported by road to the point of sale at the Cobar Management Pty Ltd's CSA mine processing facility located 5km north of Cobar under contract to Glencore International AG. Thereafter CSA and PGM concentrates are transported by rail to ports and smelters on the east coast of Australia and overseas.

5.2 Physiography and Climate

The Cobar region has a semi-arid climate and in years of normal rainfall receives on average about 416 mm of rainfall per year. Under long-term average conditions, rainfall distribution is relatively uniform throughout the year, although it may appear as significant storm events. Temperatures range from an average maximum temperature of 16°C in the winter to 33°C in the summer. The mean annual evaporation rate of 2,548 mm causes a pronounced deficit over the summer months. There are no permanent streams or other water bodies on the PGM properties. Weather does not significantly affect the operations and mining is conducted year-round.

The landscape is predominantly flat, composed of sandy plains with minor undulations. A series of minor ridges striking approximately north-northwest to south-southeast, reflect a belt of more resistant, sandstone or sandy tuffs. Cobar is situated 260m above sea level. The Peak is a conical hill, 324.3m above sea level, with the Peak Mine situated at the southern base of the Peak. Vegetation is largely semi-arid low woodland, with minor seasonal creeks and rivers lined by taller eucalypt species.

5.3 Local Infrastructure and Services

The Cobar Water Board supplies untreated water to the Peak Mine via a 130-kilometre dedicated pipeline from the Bogan River west of Nyngan. PGM is entitled to 1,890 million litres per year, although it currently uses on average 300 million litres per year. PGM has agreed to allocate an amount of its entitlement to the CSA Pty Ltd (Cornish, Scottish and Australian) copper mine and, as a result, except in certain circumstances, is not allowed to consume more than 1,000 million litres of its water allocation. Potable water is pumped from the Cobar Shire Council's water treatment plant to the site.

Maximum electricity consumption demand is 9 MW and annual consumption is approximately 79 GWh. Power is provided to the Peak gold mine via a 132 kV transmission line, to a substation at the Peak Mine. Power is converted to 11 kV for use on site, or transformed on site to lower voltages. Emergency power is available from two, 0.8-MW and one 0.65-MW diesel generating units on site, owned by PGM. This is sufficient power for emergency mine egress and to clear some of the processing lines.

6 HISTORY

Portions of the historic summary are excerpted from narratives by Stegman and Pocock (1996) and also quoted in the Micon 2003 technical report.

There has been sporadic gold mining in the Cobar district since the 1870's. The district was historically better known for its copper deposits and was one of Australia's main sources of copper at the turn of the 20th century. Numerous small gold deposits were discovered in the late 1880's, with the Occidental (or United), New Cobar, Chesney and Peak producing gold in the late 1800's. The greatest period of activity at the Peak Mine was from 1896 to 1911 when the Conqueror, Brown and Blue Lodes were worked.

Most gold mining in the Cobar district ceased by 1920 when the copper mines and smelters closed and there was a decrease in gold prices following World War I.

The second phase of sustained mining in the district began in 1935, when New Occidental Gold Mines NL re-opened and operated the Occidental Mine as the New Occidental Mine. The New Cobar (or Fort Bourke) and Chesney Mine also re-opened, in 1937 and 1943, respectively. Mining again ceased in the district with the closing of the New Occidental Mine in 1952. Between 1935 and 1952, the New Occidental Mine was the premier gold mine in New South Wales, and had produced 700,000 oz of gold.

Exploration by various companies was conducted through the late 1940's to 1980 in the district with no significant success. In early 1980, Rio Tinto plc acquired various leases containing the New Occidental, New Cobar and Chesney mines. The Peak gold deposit was discovered in 1981 and PGM was formed to develop the deposit in 1987. Between 1982 and 1985, a total of 30,840m was drilled to delineate the Peak deposit. A pre-feasibility study was prepared in October, 1985. An additional study updated the mining, metallurgical and evaluation information to June, 1986. A proposal for a feasibility study was approved in January, 1987 and the study was completed in 1990. Production commenced at the Peak gold mine in 1992.

Subsequent exploration and investigations led to the development of the New Occidental and Perseverance deposits. In 1995 an exploration program exploring beneath the previously mined area of New Occidental was successful in outlining an inferred resource of 3 million t grading 7.4 g/t gold. In July, 2000, approval was received for development of the New Occidental deposit.

The Perseverance deposit was identified in the 1980's from a coincident gravity and magnetic anomaly centred on historic workings. Deep surface drilling in 1994 yielded a narrow zone of ore-grade gold mineralization. The depth discouraged further exploration until 1996 when a decision was made to proceed with further exploration from an 800m underground exploration drive from the base of the Peak Mine workings. Underground drilling commenced in 1997. Following additional investigations and studies, Perseverance was approved for development in December, 2001.

Exploration at the Fort Bourke Hill historic workings, including shallow and deep diamond drilling, was conducted from 1989. Following the temporary loss of access to the Peak Mine shaft in mid-1998, PGM developed a trial open pit mine, the New Cobar mine, at the site in October, 1998. The trial open pit continued until March, 1999, extracting approximately 105,000 t of ore. The trial mining led to the operation of the New Cobar open pit from 2001 until 2005. In 2005, development of a decline was completed from the New Cobar open pit to access reserves below the base of the pit.

Other exploration targets include the Chesney copper-gold, Gladstone, Dapville and Great Cobar deposits that have been identified through on-going exploration activities in the historic mining district.

Table 6-1 lists the mines associated with the Cobar mineral field and the years of operation. Table 6-2 summarises production from mines on PGM tenements in the Cobar mineral field, including historic mining activities.

Table 6-1: Operational History at the Cobar Mineral Field

MINE	YEARS OF OPERATION
Chesney	1872 to 1919, 1943 to 1952, 1971 to 1975
Coronation / Beechworth	1906 (exploration only)
Gladstone	1908 to 1920
Great Cobar	1871 to 1919
Mount Pleasant	1910 to 1920
New Cobar (Cobar Gold, Fort Bourke)	1889 to 1894, 1910 to 1919, 1937 to 1948, 1998 to 1999, 2001 to present
New Occidental (Occidental, United, Albion)	1871, 1889 to 1918, 1920 to 1921, 1935 to 1952, 1985, 2002 to present
Peak Gold Mine	1992 to present
Queen Bee	1904 to 1909, 1952 to 1973
Tharsis	Late 1960s
Young Australia	1896 to 1901, 1912 to 1916
Various historic operations (Peak-Blue Lode, Brown, Conqueror; Mount Boppy)	Late 19th century
OUTSIDE PGM TENEMENT	
CSA (Cornish, Scottish & Australian) Mine	1905 to 1920, 1965 to 1998, 1999 to present

Table 6-2: Production Statistics from the Cobar Mineral Field to December, 2008

	Gold (ounces)	Silver (kilograms)	Copper (tonnes)	Lead (tonnes)	Zinc (tonnes)
New Occidental	700,500				
New Cobar	242,500	860	5,050		
Chesney	28,500	1,110	6,210		
Historic Peak Workings	20,670	18,220			
Great Cobar(1)	293,500	46,700	114,830		
Gladstone		340	2,160		
Queen Bee			3,960		
Young Australia	1,340	235			
Mount. Pleasant	85	185			
Peak Gold Mine	840,661	16,275	23,110	8,346	6,243
Peak Gold Mine post 1998(2)	1,194,213	18,980	21,353	0	0
Total	**3,321,969**	**102,905**	**176,673**	**8,346**	**6,243**

1) Including significant tonnage of Chesney, New Cobar and the historic Peak workings treated at the Great Cobar smelters from 1910 to 1919.
2) Peak Gold Mine production post-1998 includes New Cobar open pit and underground mining until December 2008.

7 GEOLOGICAL SETTING

The following section is sourced from Bell et al (2000), Dettbarn (2001), and Stegman and Pocock (1996).

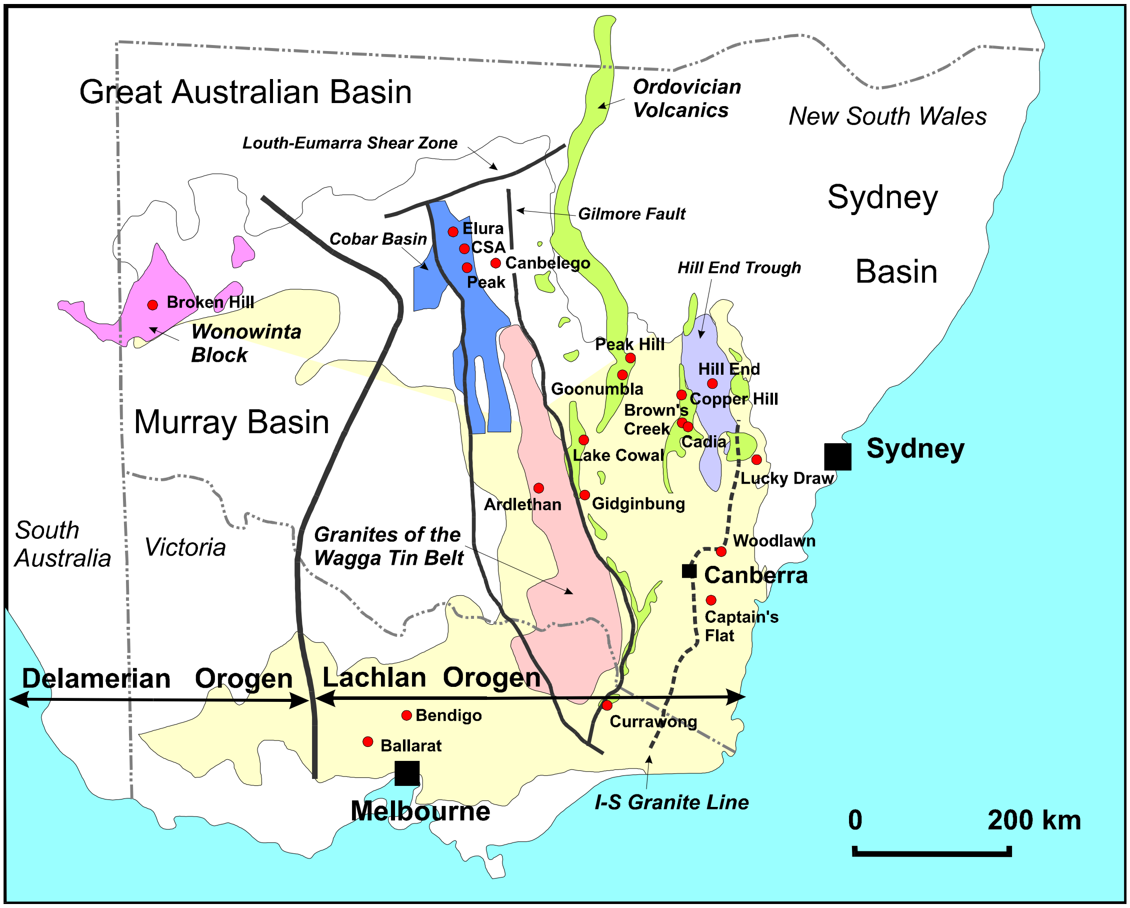
7.1 Regional Setting

The Cobar Gold Field is located on the eastern margin of the Early Devonian Cobar Basin (Glen, 1991). The Cobar Basin lies within the northern part of the Central Belt of the Lachlan Orogen (Figure 7-1). An extensive suite of Silurian "S"-type granitoids of the Wagga Tin Belt is exposed in the southern part of this region. The Eastern Belt of the Lachlan Orogen comprises a mixed Ordovician volcanic and sedimentary sequences characterised by the presence of extensive andesitic volcanic units, which are associated with porphyry-style copper-gold mineralisation including the Goonumbla, Cadia and Browns Creek deposits. Epithermal-style mineralisation including the Lake Cowal, Gidginbung and Peak Hill deposits, are also present within the Eastern Belt. The Western Belt is dominated by Ordovician and Devonian turbidite sequences and contains the extensive, structurally hosted gold deposits of the Victorian Gold Fields.

Regional crustal extension of the Lachlan Orogen in the Late Silurian created a series of north-south trending deepwater troughs and basins in the Cobar region. These include the Cobar Basin and, further to the south, the Raast and Mt Hope Troughs. Onset of regional extension was also marked by the emplacement of Late Silurian S-type and I-type granitoids within the basement on the eastern margin of the Cobar Basin. These include the S-type Thule Granite (dated age of 422 Ma) and Erimeran Granite (dated age of 419 Ma) and the I-type Wild Wave Granodiorite (dated age of 418 Ma) (Glen et al., 1996). The Cobar Basin was flanked by the Kopje Shelf to the east and the Winduck shelf to the west (Glen, 1991; 1995). Formation of the Cobar Basin has an estimated age of approximately 410 Ma (Perkins et al., 1994).

The Cobar Basin comprises predominantly siliciclastic (quartz-rich) turbidites of the Cobar Supergroup. The basin contains two stages of sedimentary fill. The lower part is characterised by coarse grained clastic sediments, while thickly bedded sediments define the upper portion (Glen et al., 1996). Glen (1991) regards these two fill sequences as representing syn-rift and post-rift stages, although Glen et al., (1996) highlight the relatively subtle nature of this sub-division, which is based solely on the relative amounts of sandstone in the respective sequences. The lower unit, the Nurri Group (Figure 7-1), is a fining upwards sequence comprising shallow-water sediments that progress rapidly up into more extensive siliciclastic turbidites. This unit was deposited along the eastern margin of the Cobar Basin from detritus sourced from an eastern landmass. The upper unit, the Amphitheatre Group, comprises more extensive low energy turbidites. The Amphitheatre Group defines an upward coarsening cycle followed by an abrupt change to thinner beds with detritus sourced from an uplifted north-western and western landmass. This unit occupies most of the Cobar Basin (Glen, et al., 1996) and is intercalated with sediments deposited on the western Winduck Shelf. Together, these sediments obscure the western margin of the Cobar Basin.

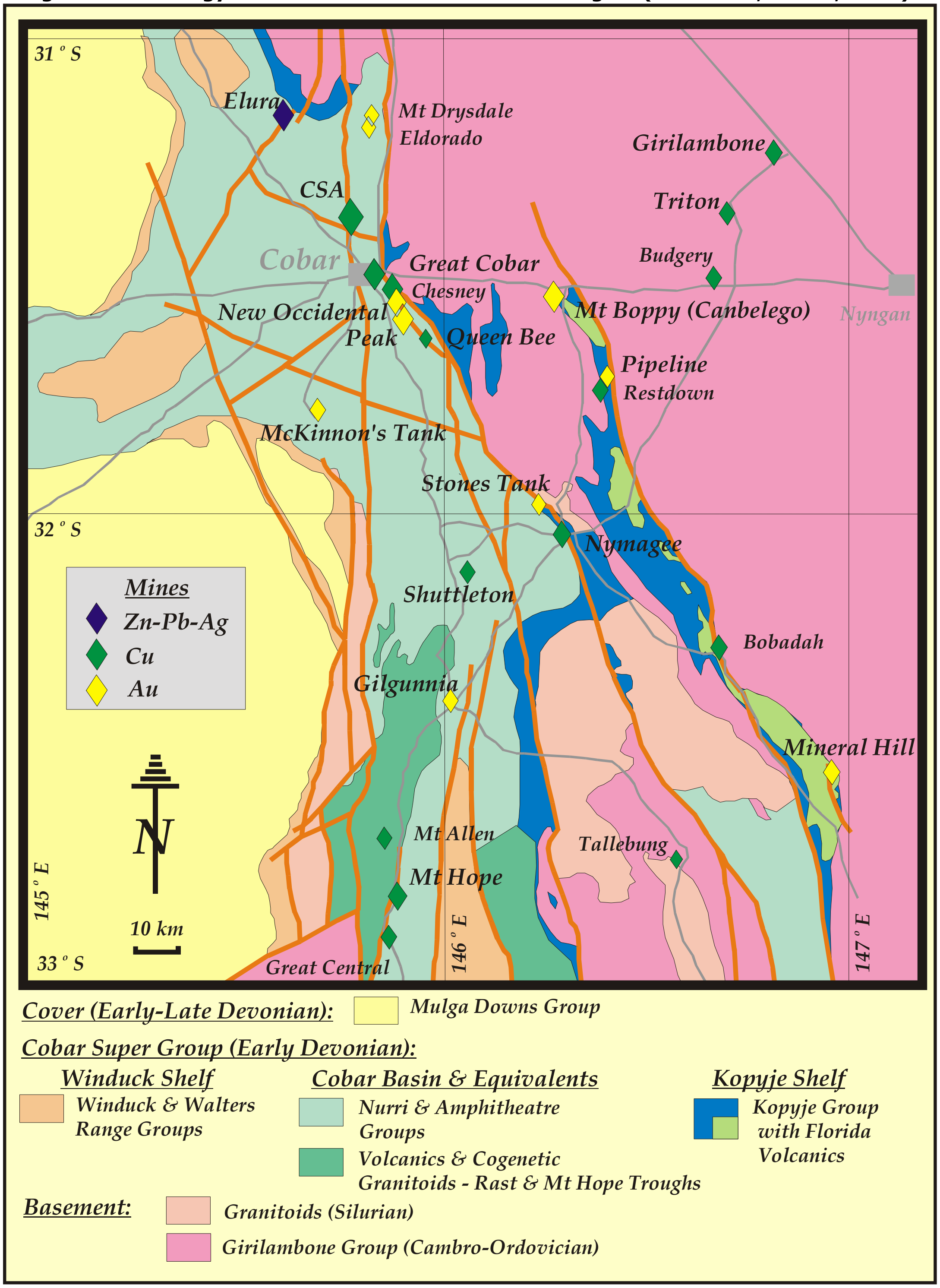
Figure 7-1: Geology and Mineralisation of the Lachlan Orogen (after Walshe et al., 1995).



Glen (1991) indicates that volcanic rocks form only a small proportion of the exposed syn-rift sequence in the Cobar Basin, although volcanic units may be more prevalent at depth in the basin. Small bodies of felsic porphyry outcrop near the Queen Bee mine to the south end of the Cobar Gold Field. Intrusive flow-banded rhyolites are exposed in the underground workings at the Peak Mine (Stegman, 1998).

Portions of the Cobar Basin were structurally deformed and overturned in the late Early Devonian (395-400 Ma) in response to northeast-southwest compression (Glen, 1992). This compression was partitioned into a high strain zone along the eastern margin of the basin, and a low strain zone in the central and western parts of the basin. Reactivation of the early basin-forming extensional faults and formation of new faults controlled the deformation of the basin sediments (Glen, 1991; 1995). Overturning of sediments at the western margin of the basin subsequently occurred in the Carboniferous Period, although with less intense deformation than that experienced along the eastern margin (Glen, 1991). The geology and mineralisation of the Cobar region is shown in Figure 7-2. Cobar Basin sediments are generally of greenschist facies metamorphic grade (Glen, 1995). Glen et al., (1992) constrain the age of metamorphism to between 395-400 Ma, on the basis of 40Ar/39Ar dating of white micas formed during cleavage development in the high strain zone along the eastern margin of the Cobar Basin. Table 7-1 summarises the details of the Cobar Basin stratigraphy.

Figure 7-2: Geology and Mineralisation of the Cobar Region (After Glen, 1987a; 1994).



Mineralisation within the Cobar Gold Field and the broader Cobar Basin demonstrates a strong structural control. There is an intimate association of mineralisation with thrust-type faulting, particularly the thrust faulting focussed along the eastern margin of the basin. Glen (1995) includes the Cobar mineralisation in a distinct class of thrust-associated mineral deposits within the Lachlan Orogen.

Table 7-1: Stratigraphy of the Cobar Basin

Age	Geological Setting	Unit	Composition
Late-Mid Devonian	Cover	**Mulga Downs Group**	Sandstone, siltstone and shale
Early Devonian	Post-rift Shelf	**Winduck Group**	Sandstone, siltstone
Early Devonian	Post-rift Basin	Amphitheatre Group Upper Amphitheatre Gp. Biddaburra Formation Alley Sandstone Member Lower Amphitheatre Gp. CSA Siltstone	 Sandstone, siltstone, mudstone Sandstone, siltstone, mudstone Sandstone Sandstone, siltstone, mudstone minor limestone, volcanics Siltstone, mudstone
Early Devonian	Syn-rift Basin	Nurri Group Great Cobar Slate Unnamed Silicic Volcanics Chesney Formation Bee Conglomerate Member	 Siltstone, mudstone Porphyry, rhyolite Sandstone, siltstone Fan conglomerates, sandstones
Early Devonian	Syn-rift Shelf	Kopyje Group Meryula Formation	Siltstone, sandstone, conglomerate, limestone
Silurian	Basement	Wild Wave Granodiorite	Granodiorite
Cambrian-Ordovician	Basement	Girilambone Group	Sandstones, siltstones, metasediments

7.2 Local Geology

7.2.1 Lithology of the Host Sequences

The Cobar Gold Field is defined as the 10 kilometre long north-trending belt of historical gold mines located east of Cobar which extends from the Peak Mine, south-southeast of Cobar, to the Tharsis workings, north of Cobar. Figure 7-3 shows the geology of the Cobar Gold Field. A generalised stratigraphic column of the Cobar Basin stratigraphy present in the Cobar Gold Field is shown in Figure 7-4.

The Cobar Gold Field is located within the Cobar Basin near its eastern margin and is hosted by sediments of the Nurri Group (Figure 7-3 and Figure 7-4). Detailed mapping of the Cobar Gold Field region has revealed that the Nurri Group consists of a relatively simple upward fining sequence of clastic sediments. The Chesney Formation sandstones, which include basal conglomeratic phases, grade upwards into weakly calcareous silty mudstones and siltstones of the Great Cobar Slate. A distinctive sequence of moderately calcareous siltstones and fine grained sandstones, informally known as the Transitional Unit, marks the change from Chesney Formation to Great Cobar Slate and provides a useful marker horizon. This unit, whilst not described in Glen's (1987a) formal definition of the Chesney Formation, has been recognised in the southern part of the Cobar Gold Field by Enterprise Exploration from mapping of the area in the late 1940's. Hinman (1992) has also recognised this unit in his division of the Upper Chesney Formation in the Peak Mine area. The presence or absence of the Transition Unit between exposures of Chesney Formation and Great Cobar Slate has been used to indicate conformable versus faulted contacts between the units.

Figure 7-3: Geology of the Cobar Gold Field (modified from Stegman and Pocock, 1996).

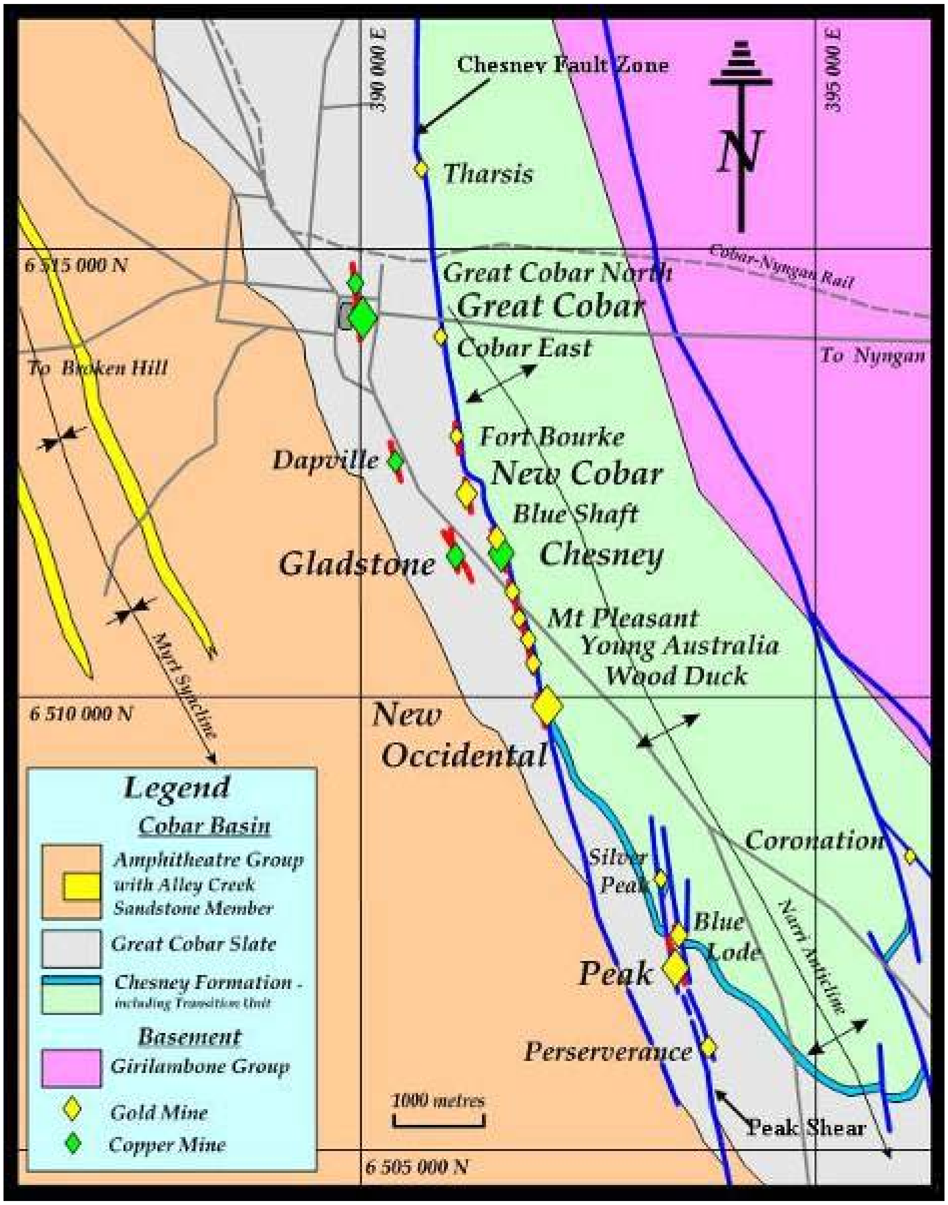
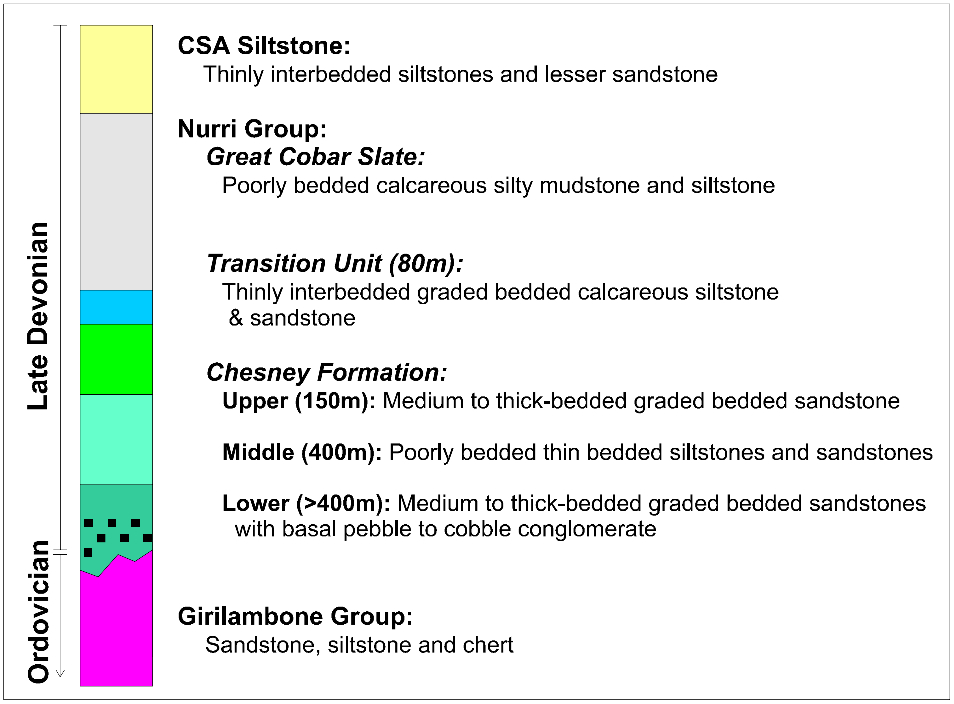


Figure 7-4: Cobar Basin Stratigraphic Column



7.2.2 Structural Framework of the Cobar Gold Field

The eastern margin of the Cobar Basin comprises a north-northwest trending anastomosing fault complex with internal fault-bound plates of strongly folded basin sediments. Glen (1991) interprets the fault complex as a linked thrust system that merges at depth into a single floor thrust. Deep seismic profiling suggests that this fault underlies much of the eastern part of the Cobar Basin. The fault-bounded wedges of sediments are folded into moderate to tight folds with fold axes sub-parallel to the bounding faults. This folding is interpreted to be thrust-related (Glen, 1991).

A strong sub-vertical regional white-mica S_2 foliation and a steep north-plunging mineral and extension lineation L_2 are present throughout the Cobar Gold Field. The S_2 foliation is sub-parallel to the major faults in the area.

The Cobar Gold Field is located on the western margin of the Chesney-Nurri Anticline, a moderate to tight south-plunging anticline (Glen, 1987b). Gold mineralisation is associated with two principal shear zones, the Peak Shear and Great Chesney Fault Zone, both of which juxtapose the Chesney Formation with the Great Cobar Slate (Figure 7-4). Only minor copper mineralisation and no significant gold mineralisation occur along the main basin margin faults, the Rookery and Queen Bee Faults. Similarly, shears immediately to the west of the Peak Shear and Great Chesney Fault Zone host predominantly gold-poor copper mineralisation, for example, the Great Cobar and Gladstone copper deposits.

The setting of the Cobar Gold Field and the relationship of the Great Chesney Fault and Peak Shear to the eastern basin margin fault complex is shown in Figure 7-5. It is interpreted that the Great Chesney Fault is an east-dipping back thrust initiating from the Rookery Fault, while the Peak Shear is a sub vertical to west-dipping imbricate fault associated with the Great Chesney Fault (Glen, 1991). The structural setting of the New Occidental orebody and the relationship of the Great Chesney Fault Zone and Peak Shear to the eastern basin margin fault complex are shown in Figure 7-6.

The Great Chesney Fault (GCF) extends over a distance of approximately 30 kilometres. Gold mineralization is restricted to the southern 8 kilometre long section of the GCF (Glen, 1987b). Within the Cobar Gold Field the GCF strikes north-northwest (350 to 355°) and dips steeply (80 to 85°) towards the east. The fault zone is generally parallel to the regional cleavage, but is slightly oblique to the regional folds including the Chesney-Nurri Anticline. The GCF truncates the western limb of the Nurri Anticline (Glen, 1987a). In detail, the GCF is a composite structure, and component shears of the fault zone are commonly oblique to cleavage. Fault planes are commonly striated, with striations plunging steeply to the north and parallel to a prominent mineral lineation on the cleavage, (Sullivan, 1947; Glen, 1987a).

The Peak Shear zone is approximately three kilometres long and up to 300 metres wide. It comprises a series of anastomosing thrust-type, sub vertical, north-northwest trending shears and faults, locally referred to as the Peak, Blue, Perseverance, Polaris and Lady Greaves shears/faults. The Peak Shear zone is broadly parallel to the eastern margin of the Cobar Basin. These structures parallel the regional cleavage and both the shears and the cleavage dissect a series of south plunging parasitic folds on the Chesney-Nurri Anticline (Hinman and Scott, 1990).

The component shears of the Peak Shear zone have a steep west dip, although they also locally dip steeply towards the east. Displacement of the Great Cobar Slate-Chesney Formation contact across the Peak Shear is west-block up. Both ERA (1987) and Hinman (1992) have identified a left lateral strike slip component of shearing in the Peak area. A prominent topographic high, caused by pervasive silicification of the underlying sediments, is developed along the eastern side of the Peak Shear zone.

Figure 7-5: Schematic Cross-Section through the Cobar Gold Field. Modified from Glen et al. 1994.

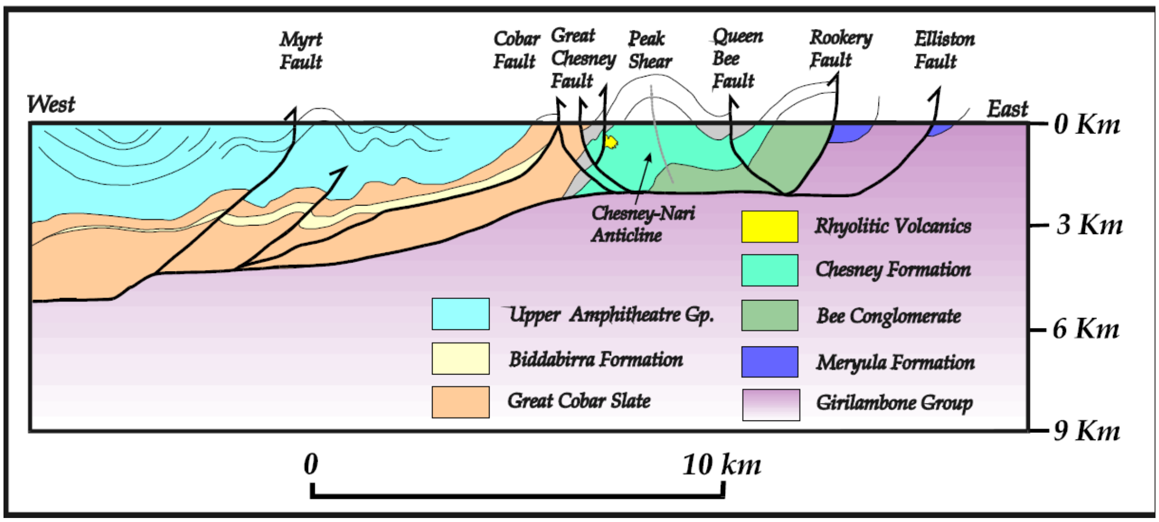
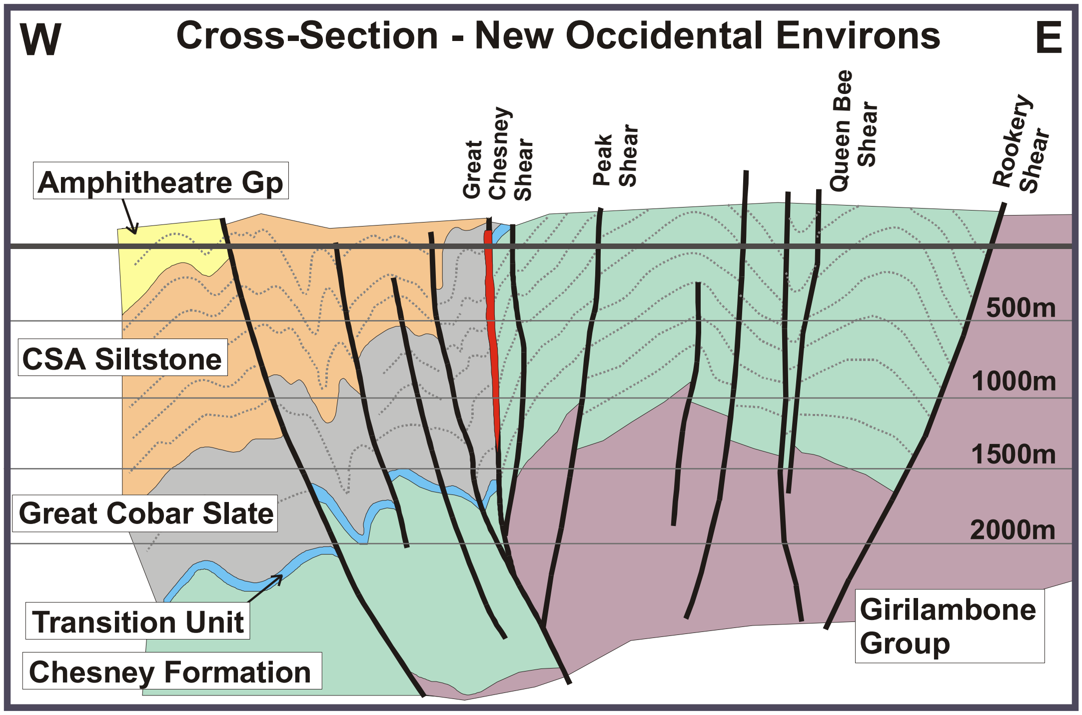


Figure 7-6: Schematic Cross-Section Showing the Structural Setting of the New Occidental Deposit.



The localisation of gold mineralisation in this portion of the Cobar Basin appears be related to the intersection of a major northeast-trending transfer fault, the Sandy Creek Fault, with the eastern basin margin fault complex. The Sandy Creek Fault is inferred from deep seismic reflection transects across the Cobar Basin (Glen et al., 1994) and is interpreted to have been active during basin formation and subsequent deformation.

8 DEPOSIT TYPES

8.1 General Description

Mineralization in the Cobar mineral field has been studied by several authors who have presented well-documented descriptions of the deposits (Stegman, 2001; Munro and Berthelsen, 2004; Glen, 1987, 1995; Berthelsen, 2006; Stegman and Pocock, 1996).

The mineral deposits of the Cobar Gold Field are typically polymetallic, ranging from gold-copper-lead-zinc at the Peak and Perseverance; to gold-copper at New Cobar, New Occidental and Chesney; and copper-minor gold at the Great Cobar, Queen Bee and Gladstone deposits. The gold mineralization occurs as discrete lenses within the broader envelopes of base metal mineralization. Gold mineralization is apparently restricted to two shear zones, the GCF and the Peak Shear system, both of which are localized along the western margin of the Chesney-Nurri Anticline at the contact between sandstones of the Chesney Formation and slates and siltstones of the Great Cobar Slate (Figure 7-4).

All the Cobar deposits are structurally controlled and are interpreted to have been emplaced during deformation and inversion of the Cobar Basin in the Early Devonian (402 to 385 Ma). The deposits occur within high strain zones, typically in intensely cleaved zones, shears and faults. In detail, the mineralization is localised within dilations developed along zones of competency contrast (for example, sandstone-slate contacts, sandstone-rhyolitic intrusive contacts and unaltered sediment-silicified sediment contacts).

The deposits have characteristically steep-dipping pipe-like geometries with uniformly short strike lengths (less than 300m), narrow widths (10 to 30m) but extensive vertical dimensions. The Peak mineralization extends over a vertical extent of 700m and the New Occidental mineralisation has been traced from surface down to 1,200m below surface. At the scale of mining, the Cobar-style mineralisation is characterised by multiple lenses with relatively complex geometries.

8.2 Genetic Model for the Cobar Gold Field Mineralization

Several theories of deposit genesis have been proposed and the following genetic model for the mineralization in the Cobar Gold Field is proposed:

1. Initiation of a series of normal inter-basin extensional faults along the eastern margin of the Cobar Basin during its formation.
2. Intrusion of rhyolitic sub-volcanic magma into unconsolidated Cobar Basin sediments along the precursor of the Peak Shear.
3. Inversion of the Cobar Basin: Major west-directed thrusting on the GCF as part of deformation of the eastern margin of the Cobar Basin.
4. Continued deformation and bulk shortening led to the development of a prominent north-northwest trending steep east-dipping cleavage associated with the GCF and a north-northwest trending steep west-dipping cleavage associated with the Peak Shear.
5. Pervasive silicification of the Chesney Formation along the eastern margins of the GCF and Peak Shear and more focused intense silicification along zones of penetrative cleavage developed along the shear zones themselves and at the western margin of the rhyolite bodies at Peak. The zones of silicification, like the rhyolite bodies, became loci for mineralization because of their competency contrast with the adjacent sediments.
6. Continued deformation of the eastern margin of the Cobar Basin, manifested as left-lateral strike-slip shearing along the GCF and continued reverse shearing along the Peak Shear zones. Shearing created sub vertical pipe-like zones of dilatancy along the GCF and more discrete curvi-linear zones around the rhyolite bodies at Peak. Fluid flow was focused along these zones.
7. Main stage mineralization associated with multiple generations of quartz veining and chlorite alteration occurred during the peak of deformation and cleavage formation. Several broad phases of mineralization are recognised; the gross sequence, from oldest to youngest is:
    - Silver-poor galena, sphalerite, minor chalcopyrite, and gold mineralization at Peak.
    - Magnetite, native bismuth, bismuthinite, and gold mineralization along the GCF.

- Magnetite, chalcopyrite ± gold along the GCF and Gladstone-Great Cobar line of lode and chalcopyrite, pyrrhotite, pyrite, gold, and minor galena mineralization at Peak.

8. Further shearing and dilation within the Peak orebody, causing segmentation of both the rhyolite body and the copper-gold mineralization along the Gecko Shear.
9. Reactivation of early zones of shearing, particularly at Peak where late stage thrusts dismember and displace the rhyolite body and the main stage mineralization. Most displacement is focused along the Polaris/Peak Shear.
10. Late stage mineralization focused in dilations along the reactivated thrust faults. This mineralization typically comprises silver-rich sphalerite, galena, pyrrhotite, and pyrite mineralization associated with black chlorite-talc alteration. Where it overprints main stage mineralization, it inherits a significant gold component.

8.3 Deposits on the PGM Properties

The remainder of this section is principally derived from a paper by Berthelsen (2006) in which a summary of the key deposit types of the PGM properties is presented.

The gold and copper deposits on the PGM properties in the Cobar Gold Field can be characterised based on their own unique mineralisation and alteration signatures. The shear zones which host the mineralisation include the Peak – Perseverance Shear, Blue Shear, Lady Greves Shear and the Great Chesney Fault (Figure 7-2 and Figure 7-4). These northwest trending structures are clearly visible in magnetic data as distinct linear magnetic ridges.

The deposit groupings are as follows.

- Group1 - Peak and Perseverance;
- Group 2 - New Occidental, Chesney and New Cobar;
- Group 3 - Gladstone and Great Cobar.

The parameters for grouping are based on metal ratios and content, host stratigraphy, mineralisation and alteration styles and also northing. Group 1 is southernmost and Group 3 is northernmost (Figure 7-4 and Figure 8-1).

Group 1 - Peak and Perseverance

- Gold and copper deposits with lower grade lead and zinc.
- Orebodies occur within the Peak Shear zone, have a steep west dip, but locally east dip.
- Lithologic host is Great Cobar Slate to Chesney Formation sandstones and siltstones and rhyolite.
- Mineralization also occurs within contact zones (i.e. between sediment and rhyolite).
- Bonanza gold grades occur where rhyolite intersects mineralised shear zones.
- Perseverance has considerably less lead and zinc than that observed at Peak.
- Deposits contain elevated levels of Cu (0.5-2%), Pb (0.1-1%), Zn (0.05 – 1%), Ag (10-12g/t) and Bi (50-100ppm).

Group 2 - New Occidental, Chesney and New Cobar

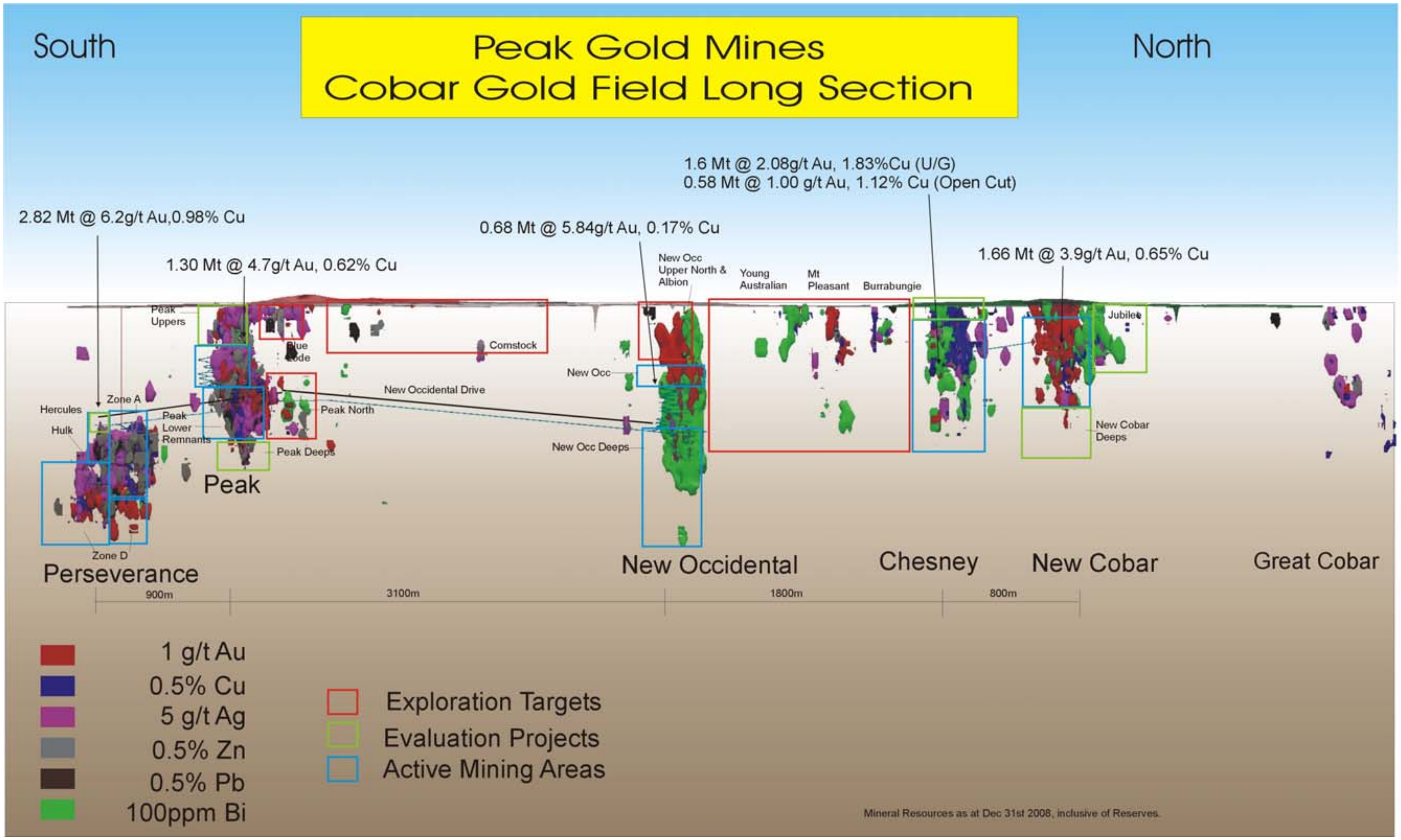
- Gold, gold-copper and copper-gold deposits.
- Lithologic host is Great Cobar Slate.
- Hosted and controlled by the Great Chesney Fault and its related splays.
- Characteristic feature of this group of deposits are the early cryptocrystalline to chalcedonic silica veins that are closely associated with high-grade gold and bismuth mineralization (New Occidental is the best example).

- Chesney and New Cobar have overprinting quartz veining, brecciation, pyrrhotite and chalcopyrite mineralisation.
- Early silicification and pervasive iron chlorite alteration is typical.
- Iron rich stilpnomelane occurs predominantly at New Occidental (Bell et.al, 2000) and to a lesser extent at New Cobar and Chesney.
- Strong Au-Bi correlation and a Cu-Ag relationship (relationship is also seen within the Group 3 deposits).
- Chesney and New Cobar ore zones occur some 20 to 50m respectively west of the GCF and are hosted entirely within Great Cobar Slate.

Group 3 – Gladstone, Queen Bee and Great Cobar

- Copper deposits with minor gold.
- Lithologic host is siltstones and poorly bedded sandstones of the Great Cobar Slate (sandstone units appear to play a controlling part in localising the mineralization).
- Gladstone and Great Cobar occur some 400 to 900m respectively west of the GCF.
- Deposits are characterised by pervasive iron chlorite alteration and dark green magnesium chlorite alteration proximal to the mineralisation (similar to Group 2).
- At Gladstone, the ore consists predominantly of chalcopyrite and minor pyrrhotite. The Gladstone mineralisation is characterised by quartz breccia veins and chalcopyrite. Great Cobar has more complex styles of mineralization.
- At Great Cobar, copper mineralisation occurs within a wide halo, up 100 metres, with extensive vertical extent. At least 4 lenses were identified at Great Cobar: lower grade eastern lens, central lens (historically mined), lead zinc lens occurring to the west, and a lens to the north.
- Great Cobar contains massive sulphide lenses. Porphyroblastic textures are commonly seen in the massive sulphides resulting from recrystallisation.
- Early silicification, like the remainder of the Cobar Gold Field, is accompanied by iron chlorite alteration.

Figure 8-1: Cobar Goldfield Longsection – with Resource Inclusive of Reserves at Dec 2008



9 MINERALIZATION

The descriptions of mineralization in this section of the report were taken from internal PGM documents and a paper published by PGM staff in "The Cobar Gold Field – a 1996 Perspective", published by the Australasian Institute of Mining and Metallurgy. The text is largely the same as the previous technical report (Micon, 2003) but has been reviewed and updated to reflect the current geological concepts and interpretations.

The mineralization of the principal deposits of interest belonging to PGM (Figure 7-4) in the Cobar area are described from north to south on the Great Chesney Fault (Section 9.1) and then on the Peak Shear (Section 9.2). Other deposits that are not located on these two principal faults are described thereafter (Section 9.3).

9.1 The Great Chesney Fault

The New Cobar, Chesney and New Occidental mines are the principal deposits of interest to PGM, based primarily on the presence of identified mineral resources and ore reserves.

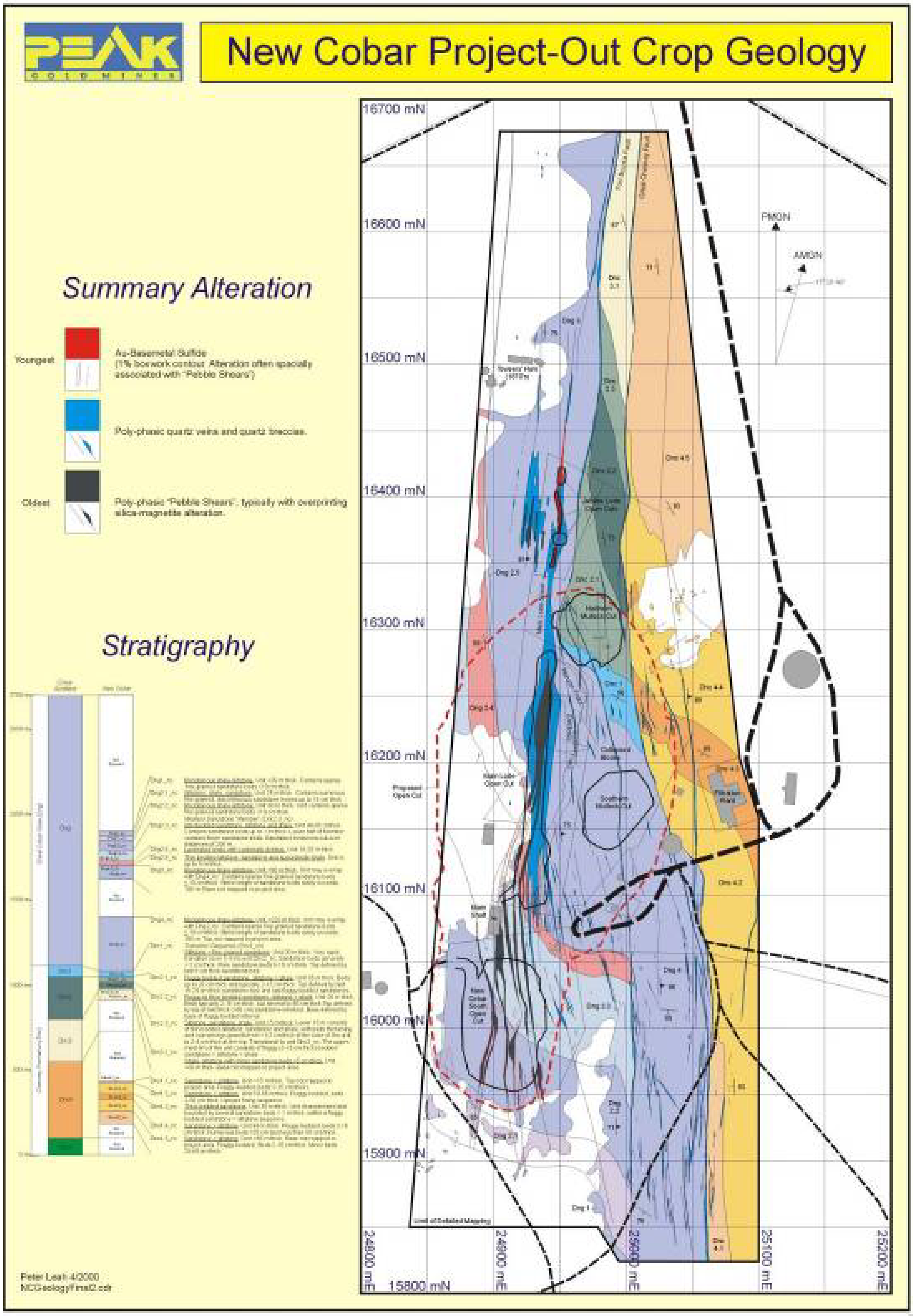
9.1.1 The New Cobar Deposit

Deposit Setting and Morphology

The New Cobar gold deposit is located approximately 2.5km north of the New Occidental Mine on the GCF. It occurs within a late north-northwest trending shear zone developed entirely within siltstones and lesser sandstones of the Great Cobar Slate immediately to the west of the contact thrust. The orebody flanks a pronounced bend in the contact thrust and occurs on the south-western side of a broad silicified ridge (ERA, 1987). At its northern end the New Cobar mineralization is only 20m from the contact thrust but diverges rapidly to the south and is up to 150m from the thrust at its southern extent. The surface geology of the New Cobar deposit is shown in Figure 9-1 (Note: mullock = waste rock).

Mineralization is characterized by a stockwork of pyrrhotite-chalcopyrite-gold veins, which overprint an older quartz-magnetite vein stockwork. Both stockworks are characterized by gradational margins. The mineralization is developed over a strike length of some 500 m, with a central zone 300m long by up to 35m wide; that strikes north-south, dips steeply to the east, and plunges steeply to the north, parallel to regional cleavage. Bedding dips steeply to the west. The contact thrust is apparently not mineralized in the New Cobar area. The New Cobar deposit has been overprinted by a relict lateritic weathering profile.

Figure 9-1: Surface Geology of the New Cobar Deposit



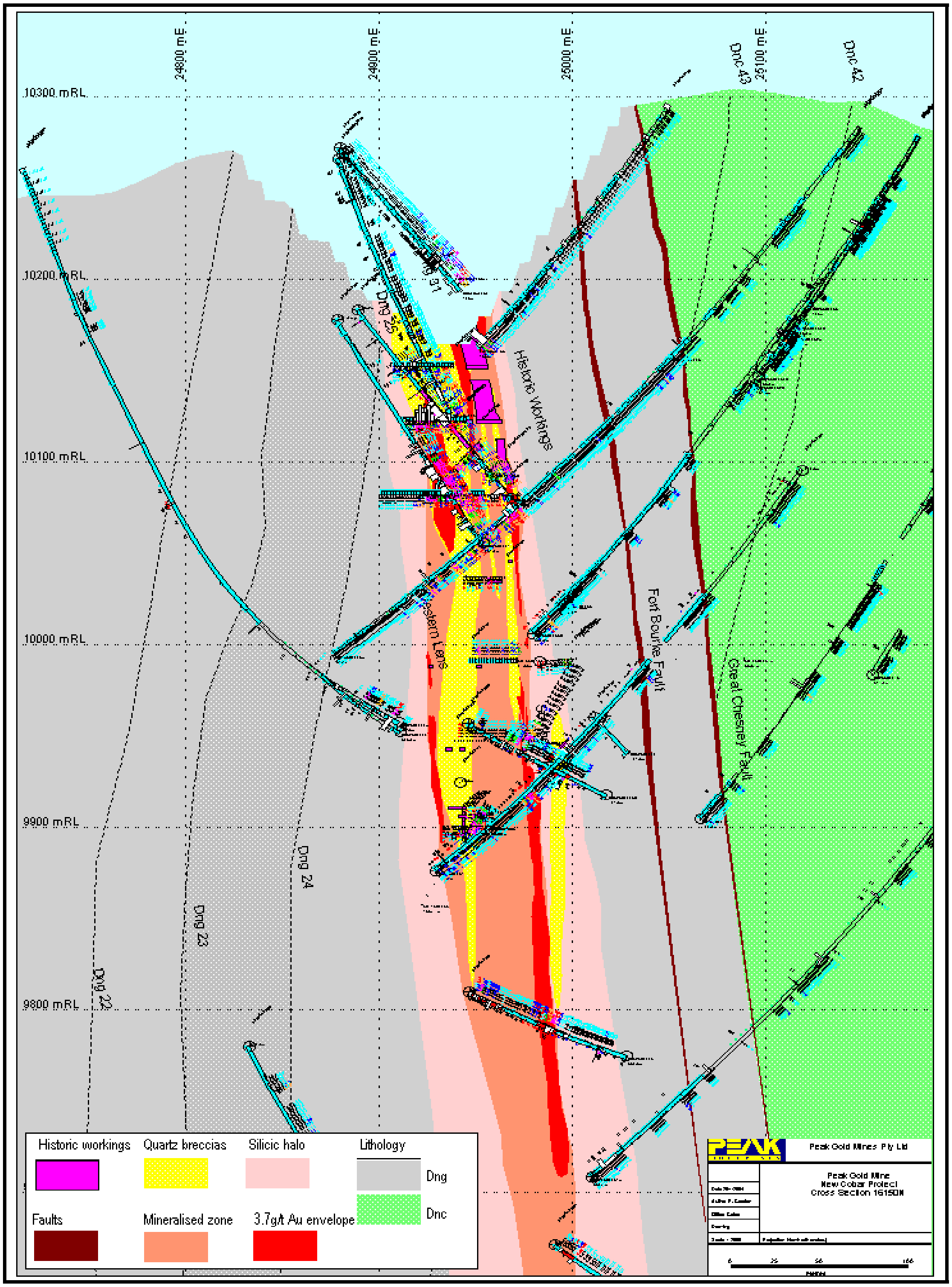
Partially Oxidised (POX) ore was stockpiled during the mining of the New Cobar open pit. The material is processed on a campaign basis and the CIL circuit in the mill is deactivated due to the high proportion of copper oxides in POX material.

Distribution of Mineralization

The New Cobar deposit contains four steep east-dipping and steep north-plunging lenses of gold-copper mineralization. All lodes are associated with curvi-linear sections of the host shears, with concavity to the east (Mulholland and Rayner, 1961). This curvature appears to be related to the intersection of northwest-trending structures with the main north-northwest trending shears. From north to south they are:

1. The Jubilee Lode (located at 16400 N) is a narrow north-northwest trending lode approximately 3 to 6m wide and 50m long (Gilligan and Byrnes, 1995) at the northern extremity of the deposit. The lode is characterized by sharp to rapidly gradational margins, relatively low gold/copper ratios, and accounts for only a small proportion of historical production at the mine. The lode has been traced vertically to a depth of 200m below surface where it appears to bottom out.
2. The Northern Lode occurs immediately along strike from the Jubilee Lode (in the northern half of the historical open cut, approximately 16200N to 16300N) and is separated from the Northern Lode by a zone of barren quartz veining and silicified quartz breccia. The Northern Lode trends north-northwest, is 6 to 12m thick and is 60 to 80m long. A rapid thinning of the lode below 400m depth below surface is apparently associated with a steepening in the dip of the lode (Sullivan, 1947).
3. The Southern Lode occurs immediately along strike from the Northern Lode (in the southern half of the historical open cut, approximately 16100N to 16200N) and is separated from this lode by a short narrower zone of barren quartz veining and silicified quartz breccia. The Southern Lode trends north-northwest, is 6 to 12m thick and is 60 to 80m long. It has been traced to 700m below surface by drilling and is apparently open-ended. The Southern and Northern Lodes were the most productive lodes in the deposit and are collectively referred to as the Main Lode (Sullivan, 1947). The Main Lode is typified by gradational margins, relatively high gold/copper ratios.
4. The Western Lode is located 60m to the west of the Southern Lode (immediately adjacent to the Main shaft). It trends northwest and is approximately 2 to 5m wide and 50m long (Gilligan an dByrnes, 1995). The Western Lode does not crop out and is first discernible at 200m from surface, has been traced with diamond drilling to 500m below surface and is open at depth. Gilligan and Byrne (1995) note that the dip of the Western Lode shallows from 80° to 70 ° and gold grades in the lode improve dramatically.

Figure 9-2: Cross Section through the New Cobar through 16150 N



The various lodes at New Cobar appear to be stacked, becoming progressively deeper to the south. Both Sullivan (1947) and Mulholland and Rayner (1961) believe that the New Cobar Lodes follow the fold in the contact thrust down plunge to the north, even though the stacking of the lodes suggests an overall pitch of the deposit to the south.

Lode Morphology and Associated Alteration and Mineralization

The New Cobar deposit demonstrates a remarkably similar alteration and mineralization paragenesis to that of the New Occidental:

1. Pre-cleavage pervasive silicification, now strongly brecciated.
2. Pre-cleavage white quartz veining, now ptygmatic, brecciated and strongly boudinaged.
3. Cleavage-parallel quartz veining and coeval pervasive green chlorite alteration, associated with a distinctive set of conjugate sub horizontal quartz veins and a set of northwest-trending sub vertical quartz veins. The veins comprise complex quartz breccias, typically with abundant angular cleaved and chlorite-altered sedimentary fragments and colloform quartz-chlorite veins. Gold and base metals are associated with these vein types and include at least three pulses of mineralization (oldest to youngest):

    - colloform banded quartz-magnetite veins.
    - chalcopyrite-pyrrhotite-pyrite splashes and veins.
    - galena-sphalerite-pyrrhotite veins.

    Gold mineralization at New Cobar is also associated with native bismuth and bismuthinite, and together generally occur as extremely fine-grained inclusions in the magnetite veins and with the chalcopyrite-pyrrhotite veining. The New Cobar mineralization is characterized by a much higher copper content (0.8 to 1.2 % Cu) than the New Occidental deposit (0.1 to 0.2 % Cu) and by a broad halo of strong pyrite veining. Veining and mineralization are interpreted to be synchronous with cleavage development (Glen, 1987b).
4. Post-cleavage quartz-carbonate veining.

9.1.2 Chesney

Deposit Setting and Morphology

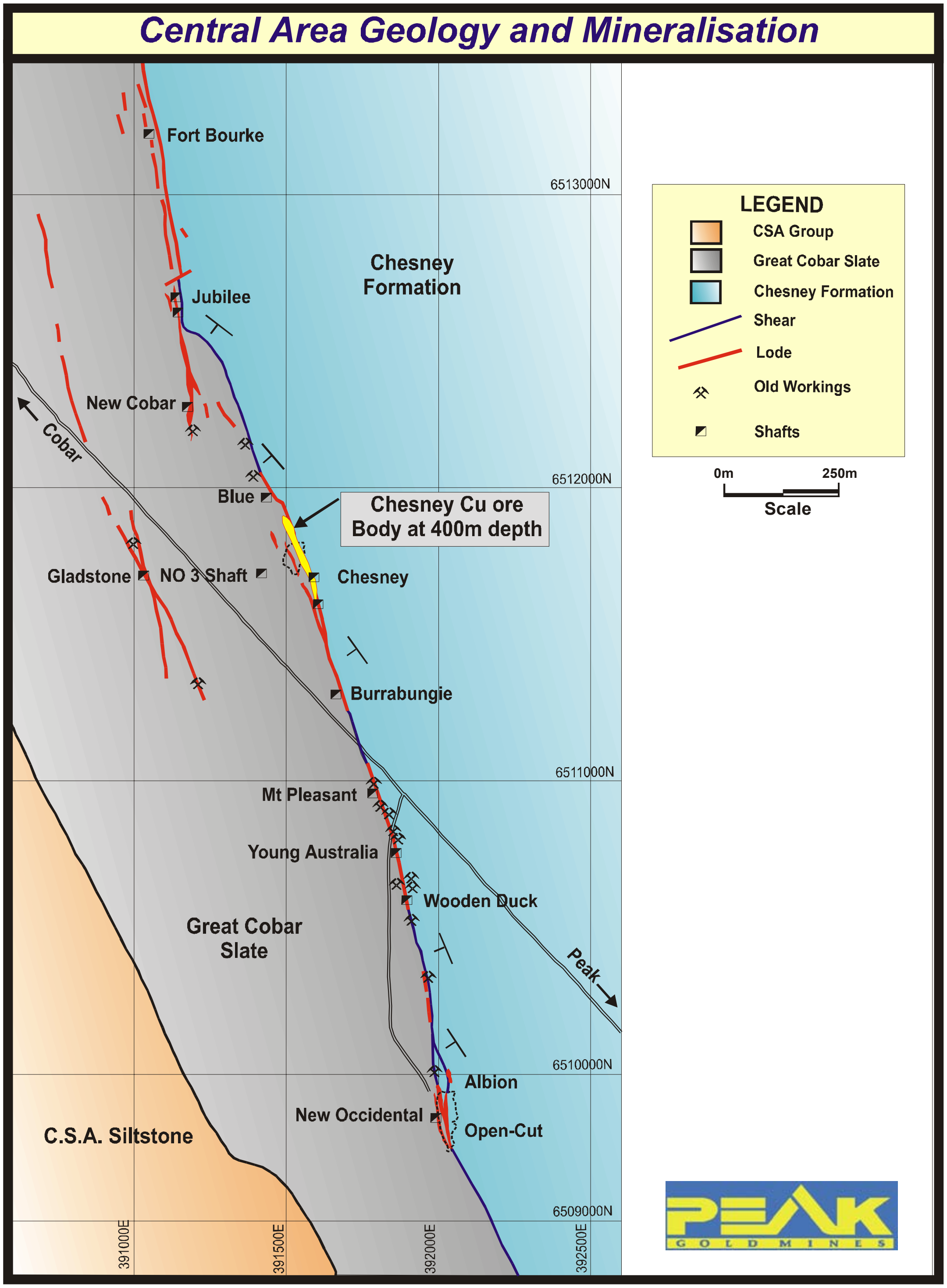
The Chesney copper-gold deposit is located approximately 1.8km north of the New Occidental mine and 600m south of New Cobar on the GCF. The deposit is associated with a late north-northwest trending shear zone located approximately 30m to the west of, and parallel to, the contact thrust. This shear zone is developed entirely within siltstones and lesser sandstones of the Great Cobar Slate and occurs along strike from, and to the south of, a warp in the contact thrust (Sullivan, 1947; Mulholland and Rayner, 1961; and Glen, 1987b). Minor mineralization is also present along the contact thrust (Andrews, 1911). The Chesney deposit is located on the western flank of a broad silicified ridge (ERA, 1987). The deposit is approximately 200m long and up to 25m wide, plunges steeply to the north (80 to 85°) and dips steeply to the east (85°), parallel to the regional cleavage (Mulholland and Rayner, 1961). Near surface oxidized mineralization is known to exist. The surface geology of the Chesney deposit is similar to that seen at the nearby New Cobar deposit.

The oxide mineralization at Chesney has been subdivided into four domains based on oxidation and weathering related metal depletion in Table 9-1.

Table 9-1: Chesney Oxidized Mineralization Domains

Highest	DOX	Completely oxidized, with significant surficial Cu leaching. Typically 30m thick.
	COX	Completely oxidized, with no significant Cu leaching. The thickness varies from 1-30m (depending on the depth of Cu depletion).
	POX	Partially oxidized. Typically 30m thick.
Lowest	NOX	Non oxidized

Figure 9-3: Central Area Geology from New Occidental to New Cobar



The No. 3 shaft was sunk on the Chesney deposit in the 1970's but was never connected to the historical mine workings (Figure 9-3).

Distribution of Mineralization

The Chesney deposit comprises three lodes:

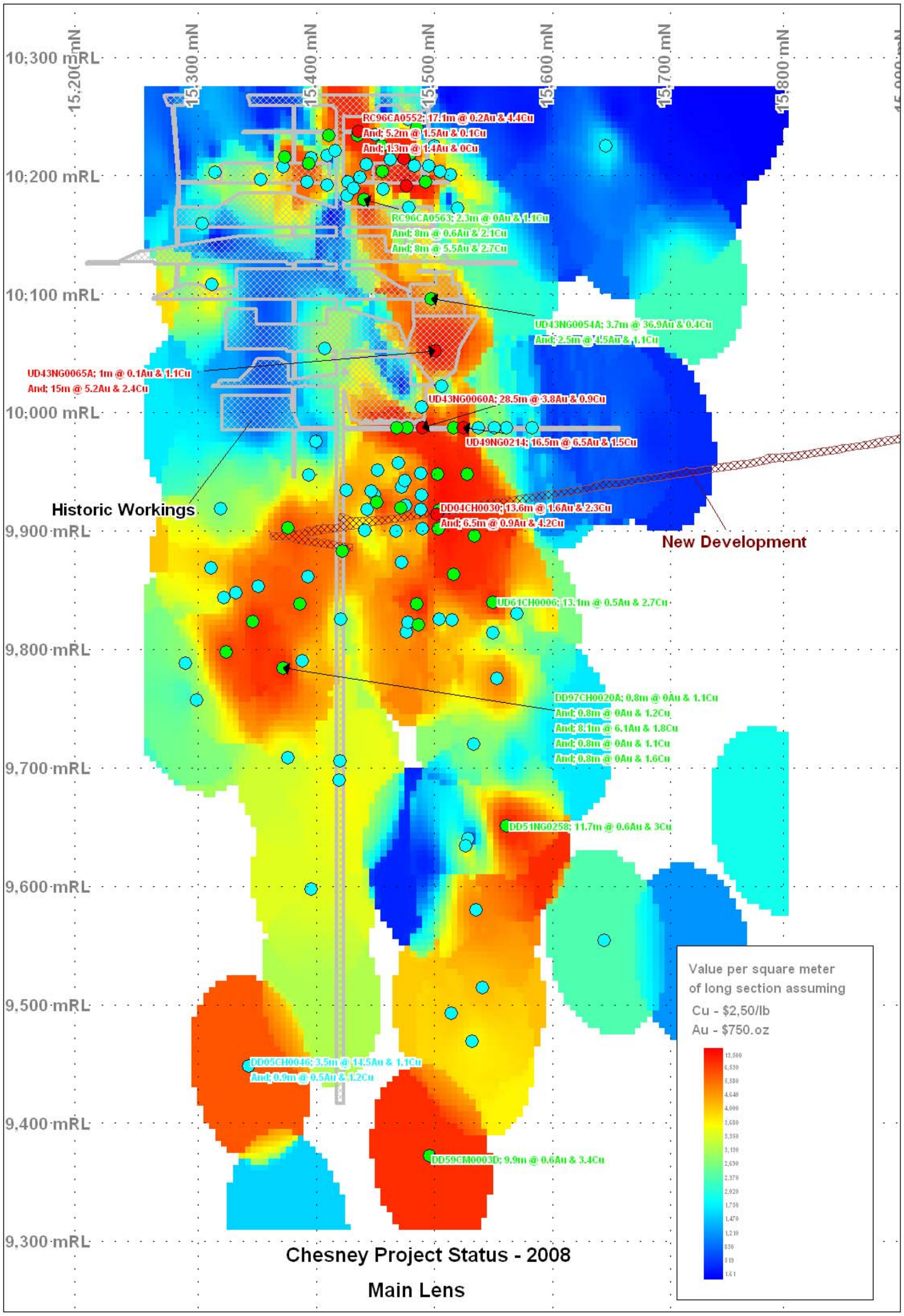
The Main Lode occupies the north-northwest trending shear outboard of the contact thrust and within the Great Cobar Slate. It has an attenuated S-shape (Phillips, 1987) and comprises two short shoots of gold-copper mineralization, the Northern and Southern gold shoots, separated by a central north-northwest trending narrow zone of low-grade copper mineralization (Sullivan, 1947). The gold shoots are pipe-like bodies, each approximately 30m long and 10 to 20m wide. The intervening copper lode is up to 150m long and 10m thick. The gold shoots occur at the intersection of the main north-northwest-trending shear and a series north-west trending cross-shears. The cross-shears serve also to define the northern and southern limits of the Chesney mineralization and are interpreted to extend eastwards across the contact thrust to the East Lode. Sub-horizontal joints and shears are also apparently associated with higher grades of mineralization. Both shoots plunge steeply to the north and dip steeply to the east. The Northern shoot crops out at surface and is associated with a prominent zone of silicification and brecciation. The Northern shoot appears to die out below 250m depth from surface (Sullivan, 1947). The Southern shoot was first encountered in the mine workings at a depth of 200m below surface and is open-ended beneath the deepest workings at 300m from surface (Mulholland and Rayner, 1961).

The East Lode is a poorly defined, narrow, 2 to 3m thick gold lode developed on the contact thrust over a distance of 200m (Gilligan and Byrnes, 1995). Mineralization is apparently developed where the northwest-trending faults intersect the contact thrust. The East Lode was apparently only worked to shallow depths and little is known of its vertical extent (Mulholland and Rayner, 1961).

The Eastern Copper zone is located in the hanging-wall of the GCF. Mineralization is hosted by chalcopyrite bearing quartz veins in the Chesney Formation, but contains insignificant gold.

Secondary mineralization has not been studied in detail. The current understanding is that the DOX domain has been strongly depleted in copper. The POX zone is characterized by native copper, malachite, and lesser azurite, with minor chalcocite at the base of the zone.

Figure 9-4: New Cobar – Chesney Long Section



Lode Morphology and Associated Alteration and Mineralization

The Chesney mineralization comprises zones of intense silicification and quartz vein breccia hosting chalcopyrite-pyrrhotite-magnetite mineralization. Gold demonstrates an intimate association with native bismuth and bismuthinite. Minor galena-sphalerite and pyrite are also present. The Chesney Lodes are associated with a broad green chlorite alteration halo (Andrews, 1911; Gilligan and Byrnes, 1995).

Jiang et al, (1995b) identified a six stage paragenetic sequence for the Chesney deposit which comprises five stages of quartz veining ranging from early cleavage-parallel veins to later fibrous cross-cutting veins overprinted by a sixth stage of vein-type chlorite-chalcopyrite-sphalerite-galena-pyrrhotite-pyrite-lesser native gold and silver mineralization.

Jiang et al, (1995b) argue on the basis of isotopic and geochemical studies that the fluids responsible for the stages of quartz veining demonstrate progressive increases in temperature and salinity, indicating increasing metamorphic fluid activity. The sulphide mineralization is interpreted to have formed from prograde fluids close to the peak of deformation in the Early Devonian.

9.1.3 The New Occidental Deposit

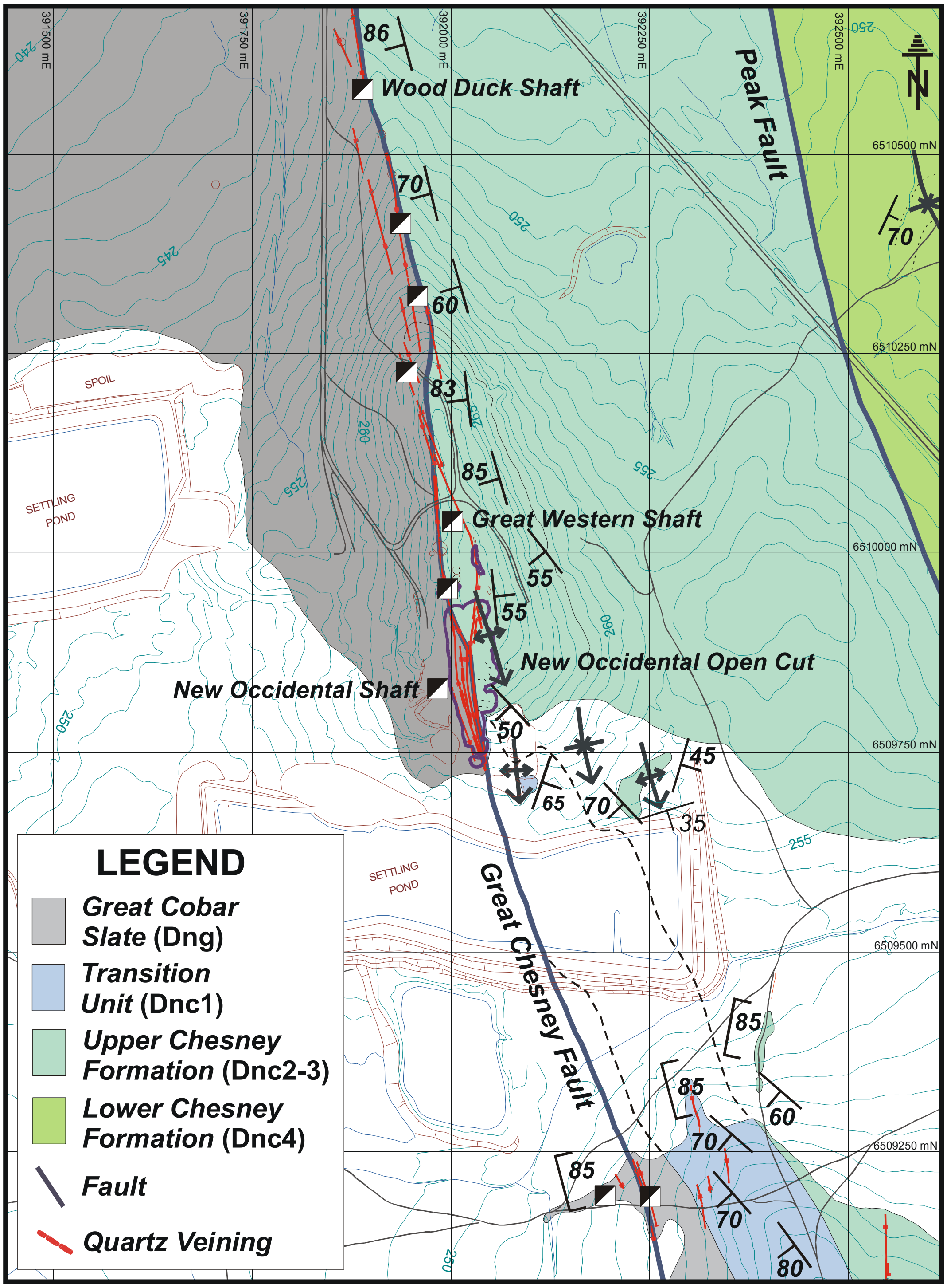
Deposit Setting

Geological mapping by PGM of the New Occidental mine and logging of drill core has identified a minimum thickness of 500m of Upper Chesney Formation along the eastern side of the GCF (Figure 9-5). The Chesney Formation is upward facing and dips generally steeply to the west-southwest. It comprises medium-thick bedded, graded-bedded, coarse-grained lithic sandstone and muddy siltstone.

The top of the Chesney Formation, as defined by the Transition Unit, crops out in the south end of the New Occidental open cut. Interbedded sandstones and siltstones of the lower Transition Unit grade up into a sequence of finely bedded calcareous siltstones and mudstones.

The Great Cobar Slate, which crops out along the western side of the GCF in the New Occidental mine environs, comprises relatively featureless muddy siltstone and mudstone with occasional rare carbonate nodules. Bedding, where present, indicates that the Great Cobar Slate also dips steeply to the west-southwest.

Figure 9-5: Geology of the New Occidental Deposit Environs

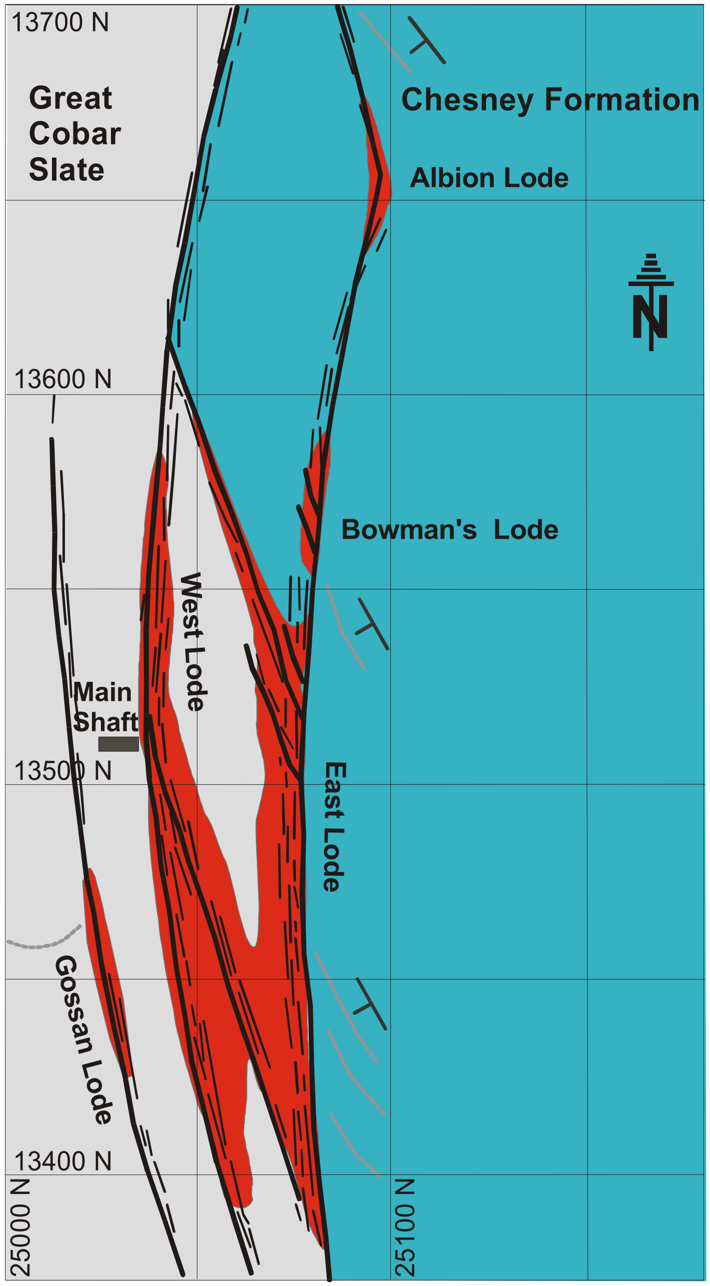


Distribution of Mineralization

Gold mineralization occurs within several discrete quartz-veined lenses that together define a disc-shaped orebody approximately 200m long and up to 25m wide that extends down dip at least 1,200m (Stegman and Pocock, 1996). The orebody is superimposed on the Chesney Formation-Great Cobar Slate contact and parallels the trace of the GCF. The orebody plunges steeply to the north (80 to 85°) parallel to L2 lineation and dips steeply to the east (85°) parallel to S2 (Mulholland and Rayner, 1961; Stegman and Pocock, 1996).

The New Occidental orebody comprises at least six lodes developed along a network of generally north-northwest to northwest-trending curvi-linear shears. The curvature of the shears is concave to the east. Most mineralization occurs within the Eastern and Western Lodes; these two main north-northwest trending lenses define a wishbone-shape coalescing at the southern end of the orebody. Both lodes are 5 to 15m thick and, at the southern end of the orebody are collectively 25m thick. The Eastern Lode is developed within a shear on the Chesney Formation-Great Cobar Slate contact whilst the Western Lode occurs along a shear within the Great Cobar Slate. Mulholland and Rayner (1961) thought that these lodes may converge at depth to form a single but much thicker lode. However, recent drilling by PGM confirms that both lodes remain separated at depth. A third 3- to 5-m thick lens, the Gossan Lode, occupies an additional parallel shear further to the west. A fourth blind lens, the Galena-Sphalerite Lode, has been intersected over narrow intervals (<5m) in workings to the west of the Gossan Lode and is apparently parallel to the other lodes. PGM's deeper drilling failed to find any indication of this lode below the present level of workings (approximately 560m below surface). Several northwest trending cross shears link between these main shears. A small fifth lode, the Bowman's Lode, is developed within the eastern shear, north of the Eastern Lode where the east shear enters the Chesney Formation sandstones. A sixth lode, the Albion Lode, also within the Chesney Formation, occurs at the northern termination of the eastern shear where it intersects another north-west-trending cross-shear. Both the Bowman's and Albion Lodes are 1 to 3m thick and less than 40m long. All lodes at the New Occidental mine demonstrate considerable vertical continuity, with the principal lodes having been individually defined over depth extents in excess of 1,000m (Sullivan, 1947; Mulholland and Rayner, 1961; Stegman and Pocock, 1996).

Figure 9-6: Detailed Surface Geology – New Occidental (Mulholland & Rayner, 1961)



Long-section and cross-section views (looking west or north) of the New Occidental lodes are shown in Figure 9-7 and Figure 9-8 respectively, presenting evaluation drill results as of the May, 2000 mining feasibility study.

Figure 9-7: Detailed Long Section through New Occidental Deposit

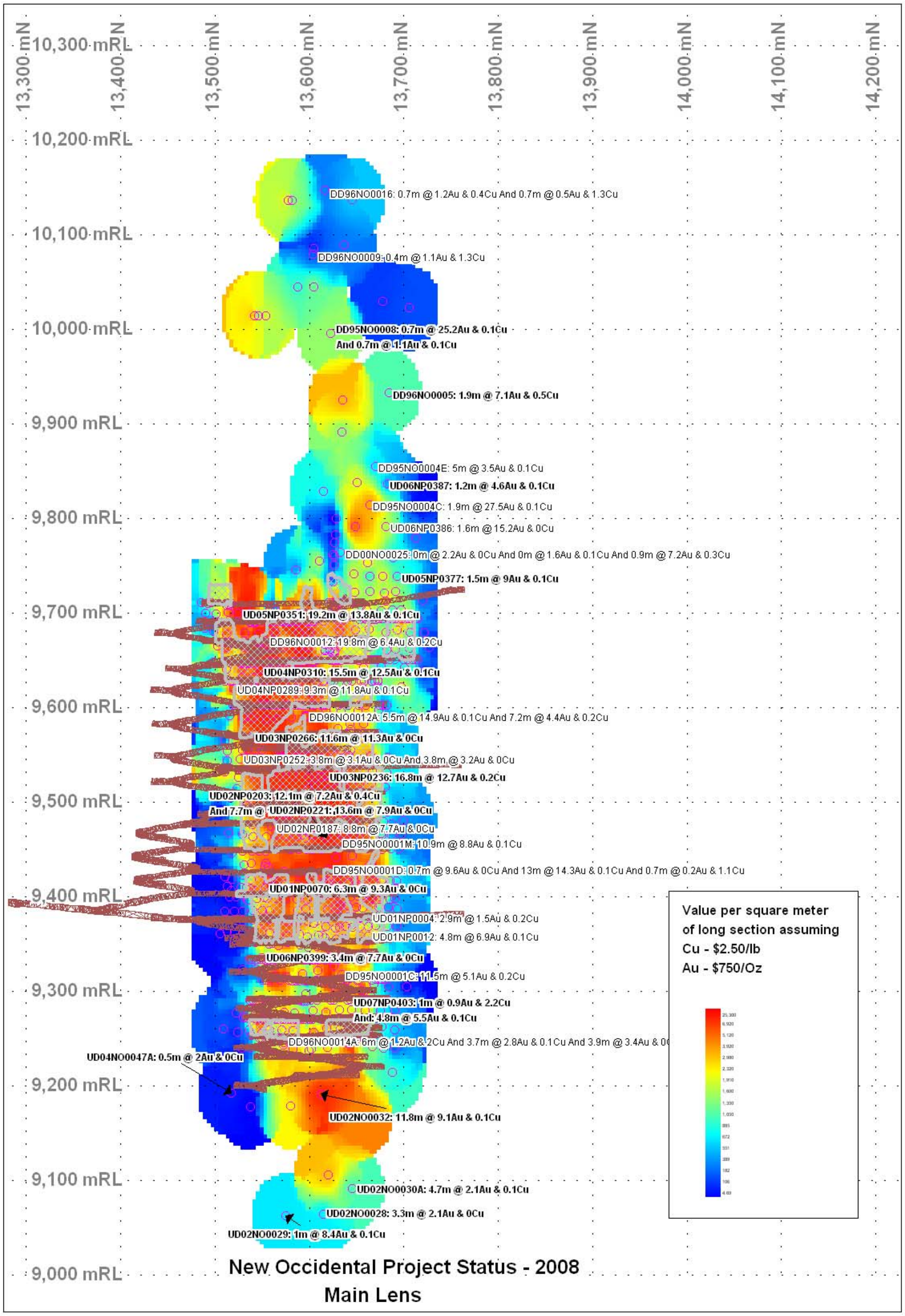
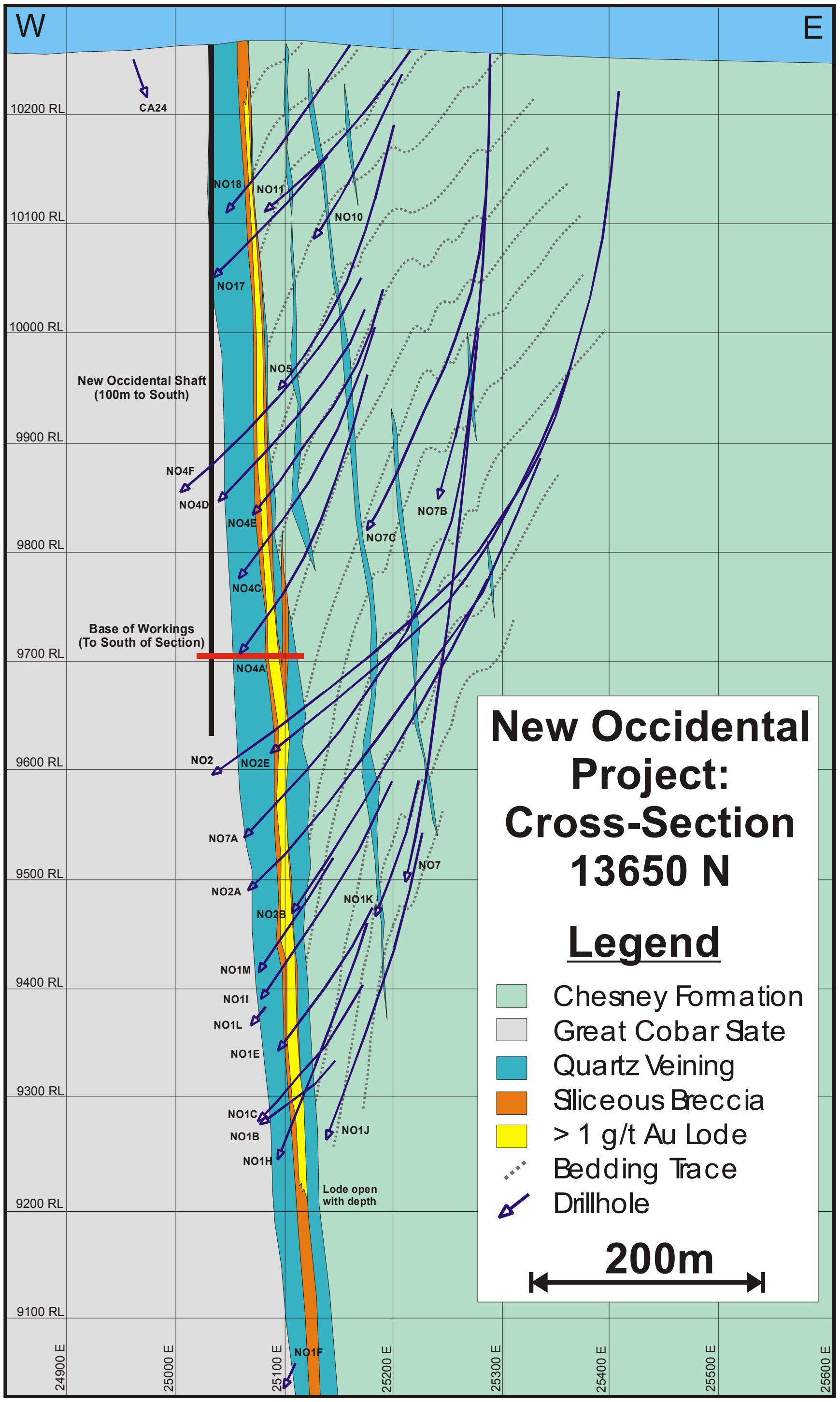


Figure 9-8: Detailed Cross Section through New Occidental Deposit



Quartz Vein Assemblages in the New Occidental Deposit

Detailed logging of drill core indicates three distinctive vein orientations within the New Occidental deposit which chronicle multiple phases of quartz veining formed throughout the main D2 deformation. This vein geometry indicates a prolonged history of repeated fault displacement that is characterized by cyclic hydrothermal precipitation and significant deformation of earlier formed veining. The widespread presence of open-space filling colloform and cockade veining associated with jigsaw breccias in the core of the deposit indicate that early stages of fault movement produced zones of significant dilation, which ultimately focussed fluid flow and mineralization.

Mineralization Sequence within the New Occidental Deposit

A broad paragenetic sequence has been established through core logging while mineral associations have been confirmed by multi-element assaying. A selection of diamond drill core intersections from the New Occidental deposit have been assayed for gold, silver, arsenic, bismuth, copper, iron, manganese, lead, antimony and zinc to support these relationships.

Detailed transmitted and reflected light microscopy, in conjunction with both qualitative and quantitative scanning electron microscopy (SEM), was undertaken on selected drill core samples by Dr. Ivan Reynolds of RTP Research & Technology Development (Reynolds, 1998; 1999a, b). These studies provided considerable additional detail on specific mineral associations within the broad framework identified by geological logging of the drill core. This was considered particularly critical given the extremely fine-grained nature of a significant number of the sulphide phases in the New Occidental deposit. In particular, quantitative SEM was used to provide chemical analyses of a variety of gangue and ore minerals that confirm apparent visual changes in mineral composition through the paragenetic sequence. The SEM studies also identified several new mineral phases.

Infrared spectrometer measurements of drill core pulps using a PIMA portable spectrometer have also been used to identify changes in phyllosilicate alteration mineralogy throughout and peripheral to the New Occidental deposit.

A complex 7-stage paragenetic sequence has been developed for the New Occidental mineralization. This sequence is more detailed than that available for the other deposits in the Cobar area. The sequence, which is summarised in Figure 9-9, comprises:

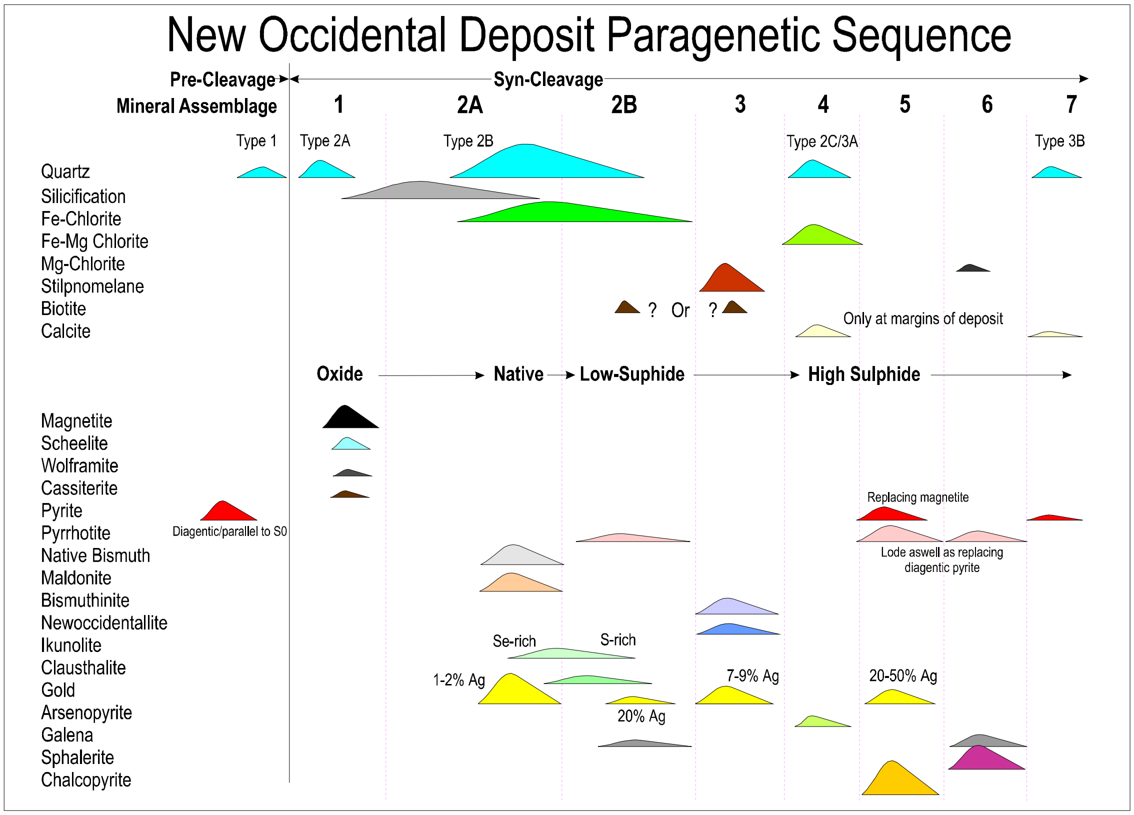
1. An oxide assemblage comprising magnetite-trace scheelite/cassiterite/wolframite as seams along brecciated margins of Type 2A quartz veins. Remnants of this mineralization are present in the hanging wall of the deposit.
2. A gold-bismuth assemblage comprising two parts.
   - A sulphur-poor assemblage of extremely fine-grained and-high fineness (>950 fineness) gold, maldonite (Au_2Bi), native bismuth, selenium-rich ikunolite (Bi_4Se_3-Bi_4S_3 solid solution series), clausthalite (PbSe-PbS solid solution series) and other rare bismuth selenides associated with the earliest fine grained phases of Type 2B veining. It seems likely that this assemblage is also associated with Fe-rich chlorite alteration. Other minor gold-bearing phases like aurostibite ($AuSb_2$) also appear to be associated with this assemblage.
   - Progressive evolution of the Stage 2A sulphur-deficient assemblage to a more sulphur-rich assemblage of sulphur-rich ikunolite, galena and minor pyrrhotite, generally associated with the latter coarse grained quartz within Type 2B veins.
3. An overprinting assemblage of a brittle mica - iron-rich stilpnomelane $(Fe, Mg)_8(Si, Al)_{12}(O, OH)_{12}$ that is associated with minor fine grained gold (850-900 fineness), bismuthinite (Bi_2S_3) and newoccidentallite (Bi_5AuS_4). The latter two minerals extensively replace and rim earlier formed Stage 2A gold, maldonite and native bismuth.
4. An overprinting assemblage of distinctive green more magnesium-rich chlorite that is associated with Type 2C and 3A quartz veins. Coarse-grained arsenopyrite is preferentially developed within "veins" of this chlorite. The arsenopyrite contains minor inclusions of Stage 2A gold and bismuth.
5. Overprinting chalcopyrite-pyrrhotite mineralization that contains minor coarser grained silver-rich gold or electrum (500-750 fineness). The gold grains often contain silver-poor cores that are compositionally similar to the gold formed in Stage 2. The chalcopyrite-pyrrhotite mineralization overprints all quartz vein types and is preferentially developed in the footwall of the deposit. Pyrite and limited chalcopyrite replacement of early-formed magnetite in the hanging wall of the deposit is tentatively correlated with this stage of mineralization.

6. Minor late-stage sphalerite-galena-pyrrhotite veins that are associated with magnesium-rich black chlorite and talc-carbonate alteration.

7. Very minor pyrite-calcite veining that is approximately coeval with Type 3B quartz veining.

This paragenesis highlights a pronounced trend of early oxide mineralization through sulphur-deficient assemblages to final sulphur-rich mineralization. It also indicates that the timing of the gold mineralization is very early relative to the base metal mineralization and that there has potentially been significant remobilization of gold during subsequent stages of mineralization, particularly with the Stage 5 chalcopyrite-pyrrhotite mineralization.

Figure 9-9: Detailed Paragenetic Sequence for New Occidental Orebody



The New Occidental deposit differs from the other gold deposits in the Cobar gold field by being relatively base metal-poor and bismuth-rich (see Table 9-2). For example, the early stages of bismuth-rich mineralization are well developed and the late stage overprinting base metal mineralization is poorly developed at the New Occidental deposit relative to the other deposits in the field.

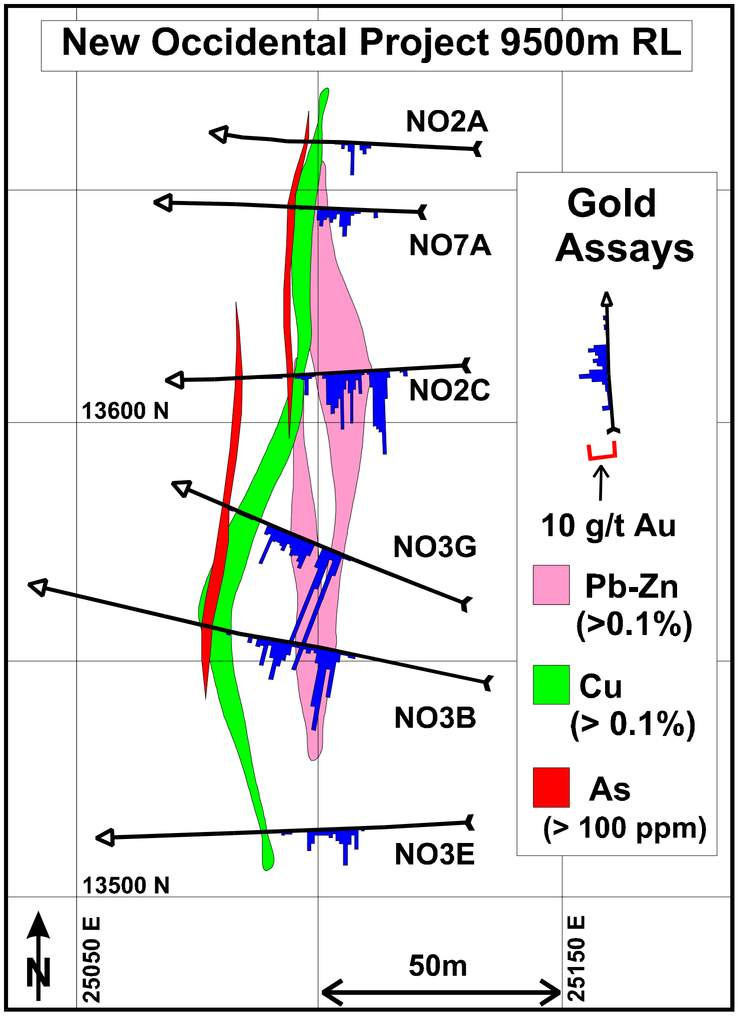
Table 9-2: Comparison of Metal Contents in Cobar Gold Field Deposits

	New Occidental	Peak	New Cobar	Great Cobar
Bismuth	1200 ppm	20-50 ppm	100-200 ppm	20-50 ppm
Copper	0.1 %	0.7 %	0.8 %	2.5 %
Lead	0.1 %	1.0 %	<0.1 %	<0.1 %
Zinc	0.1 %	1.0 %	<0.1 %	<0.1 %
Iron Sulphides	<1 %	>5 %	2-5 %	2-5 %
Magnetite	<0.1 %	Absent	1 %	1-2 %

As would be expected from the mineralization sequence outlined above, bismuth is strongly correlated with gold in the New Occidental deposit. However, the other major base metals in the New Occidental deposit show only a weak correlation with gold. Copper tends to be best developed in the footwall (western side) of the deposit and only overlaps with significant gold mineralization at the northern end. Arsenic is also localized in the footwall of the deposit, but rarely overlaps with the gold mineralization.

Lead and zinc tend to occur in the centre of the deposit and overlap with the main gold lenses. However, the lead and zinc do not extend as far north and south as do the gold lenses. Figure 9-10 shows the distribution of some base metals within the New Occidental deposit on the 9500mRL (relative elevation) based on drill hole intersections.

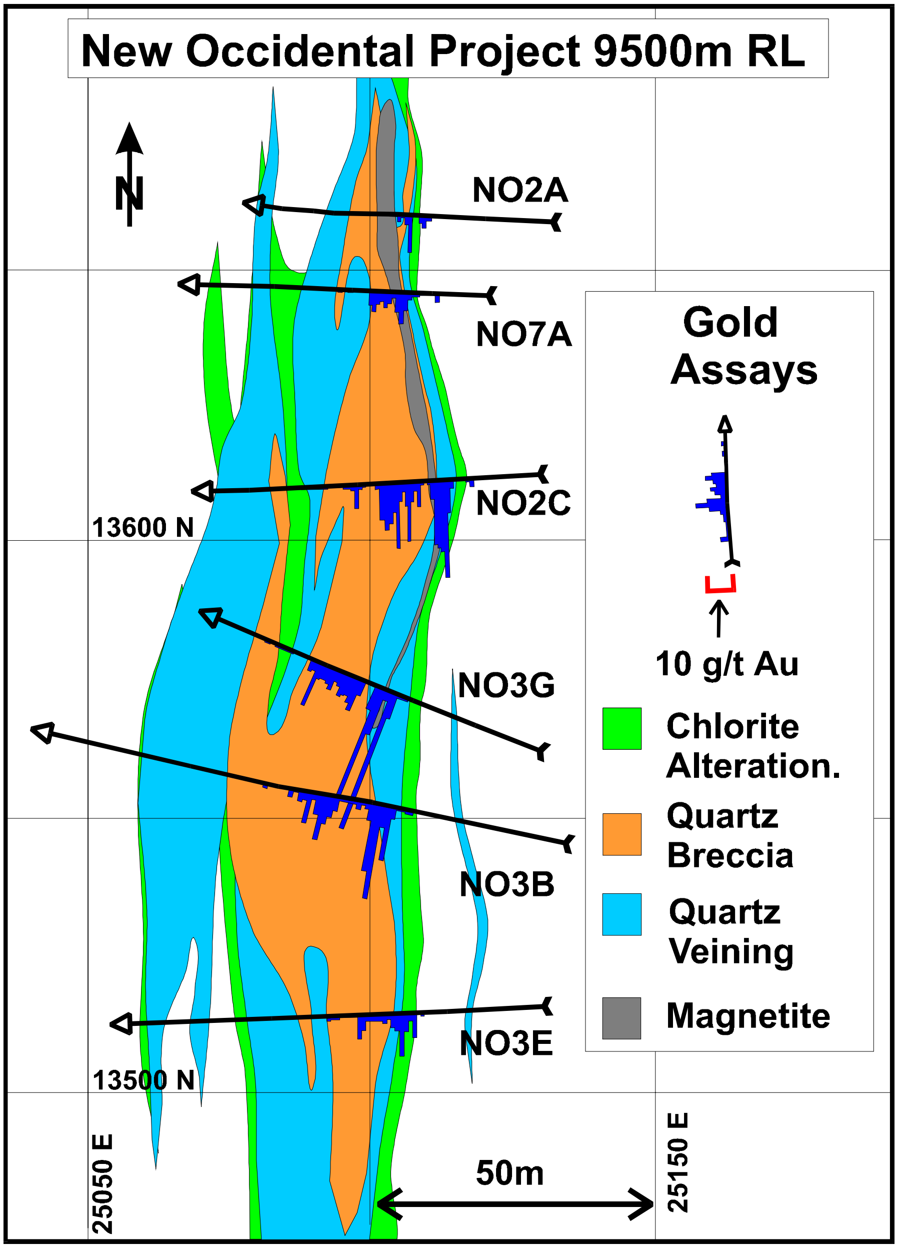
Figure 9-10: Summary of Base Metal Distributions of the 9500mRL



Gold mineralization is localized about the central quartz breccia (Figure 9-11).

The Eastern Lode is localized about the eastern contact of this quartz breccia whilst the Western Lode occurs within the quartz breccia. The Gossan Lode is localized in and around the western limit of the main body of quartz breccia. The deposit is enclosed within a broad quartz vein and Fe-rich chlorite halo that extends well beyond the gold mineralization. Magnetite is largely restricted to the hangingwall (eastern side) of the deposit and overlaps with the gold mineralization in the Eastern Lode position. Stilpnomelane alteration is largely present in the central part of the deposit and is especially well developed in the Eastern and Western gold lodes. In contrast, the magnesian-chlorite overprint is largely restricted to the footwall of the deposit where it overlaps with the Gossan Lode and northern part of the Western Lode.

Figure 9-12: Summary of Magnetite, Quartz Breccia, Quartz Vein and Fe-Chlorite Distributions of the 9500mRL



9.2 The Peak Shear

From north to south along the Peak Shear the principal deposits of interest to PGM, based on the presence of significant infrastructure and identified mineral resources and ore reserves, are the Peak and Perseverance deposits.

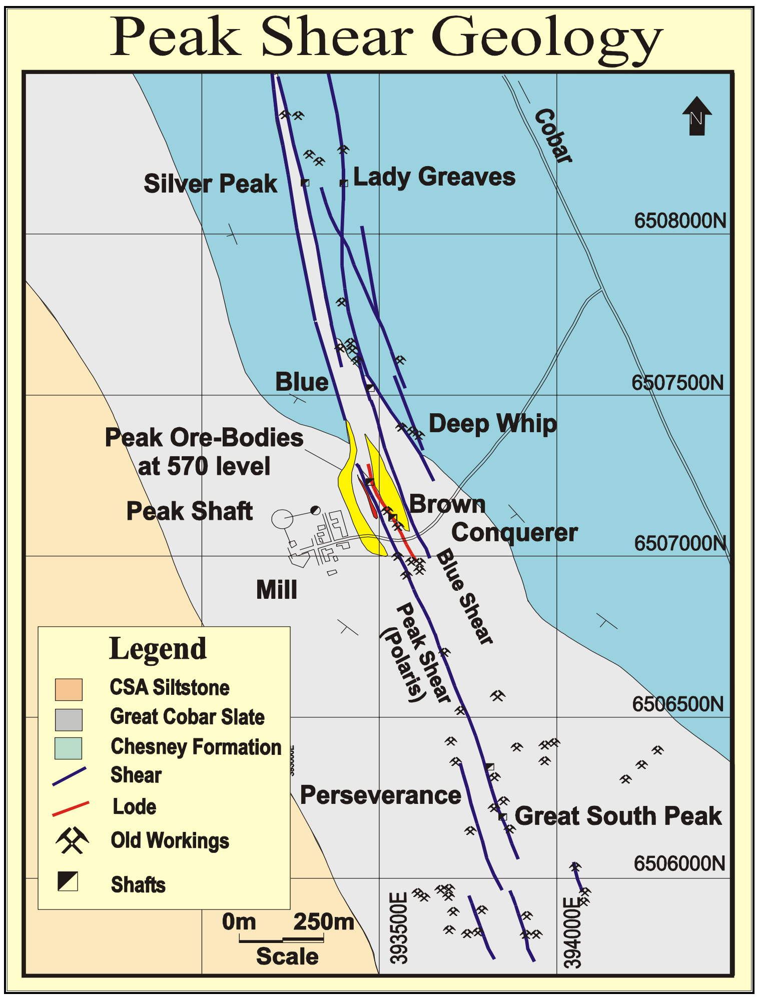
9.2.1 The Peak Deposit

Deposit Setting

The Peak gold-base metal deposit is located 9km south of the town of Cobar on the western flank of a ridge called The Peak. The deposit occurs within and immediately peripheral to the central section of the Peak Shear zone, vertically below the Conqueror, Brown and Big Lodes (Figure 9-13).

The near-surface mineralization occurs at or near the intersection of the Peak Shear and the conformable Great Cobar Slate-Chesney Formation contact. The Peak orebody and the host shears are vertically below the Chesney Formation-Great Cobar Slate contact, within the Chesney Formation itself. The deposit is localized in portions of a series of flow-banded rhyolite and rhyolitic sub volcanic breccia bodies. These rhyolites and rhyolitic breccias do not crop out and are only known from drill core and underground openings. These bodies are shallowest in the centre of the Peak deposit where they are 450m from surface. They are known to extend at least 1,000m south, 500m north and 300m east of the deposit.

Figure 9-13: Peak Mine Area Geology – Plan View



The Peak orebody lies between 300m and 780m from surface and extends over a strike length of 300m along the Peak Shear. In a broad sense, the deposit is vertically elongate parallel to the dominant steep west dipping cleavage and the strike of the Peak Shear zone. The mineralized system is thickest around the volcanics where it attains a maximum thickness of 150 m, and it rapidly tapers upwards into the overlying sediments along the shear zones.

The Peak deposit is hosted in sandstones and siltstones of the Chesney Formation and rhyolitic sub volcanics and volcanic breccias of uncertain affinity. At least three separate fault bounded bodies of rhyolite/rhyolitic breccia are present in the Peak deposit. These bodies comprise a fine-grained flow-banded core within an envelope of often coarse breccia. Contacts between the two rock-types are transitional over distances of several metres.

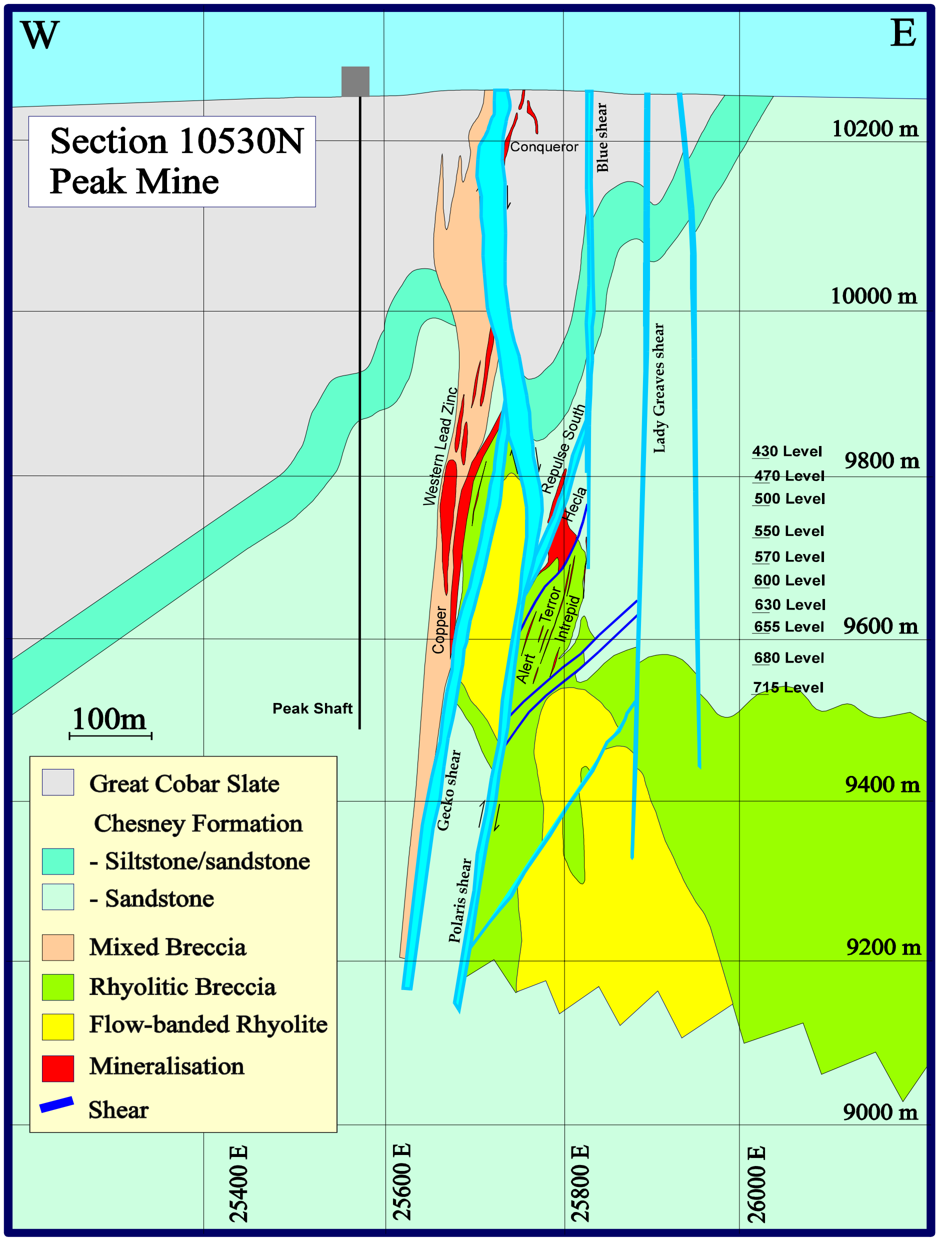
The core of the rhyolite and rhyolitic breccia bodies consists of potassium feldspar, chlorite, quartz and sericite altered devitrified glassy, scoriaceous flow-banded rhyolite. Banding is defined by a weak primary layering of varying proportions of quartz and potassium feldspar, representing flow layering. The unit is vertically elongate and dips steeply west, is up to 50m thick and extends no higher than 400m below surface (Figure 9-13).

To the immediate east and above the rhyolite core, and to a lesser extent to the west, are pervasive quartz, potassium feldspar, sericite and chlorite altered, variably clast- and matrix-supported, poorly sorted, monomictic lithic sub volcanic breccias with fine quartz, potassium feldspar and sericite matrix.

Lesser tectonic breccia, comprising angular clasts of sediment and volcanic material in a quartz-dominated matrix, are also present. However, the tectonic breccias are restricted to narrow shear zones, typically at the contact between the volcanic package and surrounding sediments. These shears and breccias are interpreted to have formed relatively late in the deformation history of the Peak Shear zone, post-dating the main deformation and mineralizing events.

The main breccia complex displays various textures indicating volcanic and sub volcanic emplacement of the felsic bodies approximately coeval with deposition of the Chesney Formation in Early Devonian time. Hyaloclastite textures, including fragments of volcanic material in an altered siltstone matrix, are observed on the northern margin of the eastern subvolcanic package. This is taken to indicate that the rhyolite was intruded into wet sediments with rapid quenching causing brecciation. Pontifex (1993) suggests the brecciation of the subvolcanics is primary, with some samples indicating turbulent flow brecciation within a lava.

Figure 9-14: Peak Mine Geology Section 10530N



Mineralization

The gold and base metal mineralization lies entirely within the broad Peak Shear zone, in a series of host structures that are spatially related to the dome-like bodies of flow-banded rhyolite and volcanic breccia described above.

The Peak orebody consists of seventeen discrete lenses of mineralization. The lenses generally strike and dip parallel to the pervasive regional cleavage, that is, they trend north-south and are sub-vertical to steep west dipping. Another series of northwest-trending, moderately southwest-dipping lenses are localized about the southern contact of the eastern rhyolitic body.

The lenses consist of a series of narrow, stacked en-echelon veins and are elliptical to thin lensoidal shapes in plan. The host structures have strong vertical continuity, persisting at depth below the currently defined ore body. Economic mineralization is less continuous within the host structures. Within each lens, gold grades are highly variable in all directions.

Four styles of gold and associated base metal mineralization occur at Peak.

- *Sediment hosted* - Sheeted veins, to 200 mm in width, consisting of quartz-sphalerite-galena-chalcopyrite-gold occur in zones of intensely silicified sandstones and lesser siltstones of the Chesney Formation. Veins are parallel to the well-developed, steep west dipping cleavage. The Western Lead Zinc (WLZ) lens is the largest example of this style of mineralization at Peak.
- *Contact zone hosted* – Mineralization in altered sediment at the sheared contact with rhyolite and volcanic breccia. These zones are arcuate and surround the western margin (and lesser eastern margin) of the subvolcanic bodies. The zones contain lenses of disseminated to semi-massive pyrrhotite-chalcopyrite-sphalerite-galena-gold mineralization with quartz gangue. Total sulphide content is typically greater than 10%. The Copper lens is the largest of this style of mineralization, representing about 50 % of contained gold in the Peak Mine.
- *Volcanic hosted* - En-echelon quartz veins within riedel shears in the eastern volcanic breccia bodies host pyrrhotite-chalcopyrite-galena±sphalerite-gold mineralization. These lenses lie within the dominant cleavage and vary in thickness from 1 to 20 metres. The Terror lens is the largest lens of this style.
- *Late stage shear hosted* - Small discontinuous pods of banded to semi-massive sphalerite-galena-chalcopyrite-pyrrhotite-pyrite carrying high grade silver and patchy gold mineralization associated with black chlorite-talc-carbonate alteration are developed in late shears including the Peak/Polaris Shear and the Hecla Shear. Mineralization of this style persists to surface in the Peak/Polaris Shear and was exploited in the near-surface Conqueror, Brown and Blue workings. Gold grades are best developed in this mineralization where it is coincident with and directly above the main Peak orebodies, between 300 and 500m from surface. This mineralization is interpreted to be a late stage remobilization phase.

Structure of the Peak Orebody and Host Shears

Inhomogeneous strain in the Peak area during deformation has played a critical role in the alteration and mineralization events that resulted in the Peak orebody (Hinman and Scott, 1990). Deformation causing shearing and bulk shortening has been focused in the sediments within zones of high competency contrast. The greatest competency contrast exists between the rigid rhyolitic subvolcanic units and the surrounding sandstones and siltstones. Lesser competency contrasts exist within the sedimentary package itself, between sandstones and siltstones and between zones of silicified material and lesser or non-silicified material. The typically siliceous subvolcanic rocks show only minor evidence of shearing or bulk shortening.

Of the three shears mapped at surface, only the main Peak Shear is recognised in Peak Mine, where it is referred to as the Polaris Shear. The other component shears of the Peak Shear zone, the Blue and Lady Greaves shears, have only been confidently traced to about 150m below surface.

Several stages of shearing have been documented as follows:

Syn-Main Stage Mineralization Shearing - Zones of strong to intense penetrative cleavage development and tectonic brecciation up to 50m wide are localized within variably altered sediments at the western margins of the rigid rhyolite bodies. The bulk of Peak mineralization is localized within this generation of shears. These shears, even though obviously very intense, do not appear to have appreciably displaced either the Great Cobar Slate-Chesney Formation contact or the rhyolite bodies.

Post-Main Stage Mineralization Shearing - The Gecko Shear is restricted to the deposit environs and is a westerly downward splay off the Peak Shear. The Gecko Shear is interpreted to be a planar dilational zone with little or no displacement. The shear appears to have merely pulled apart the two western rhyolite bodies and dilated the contact mineralization, particularly Copper Lens. The sense of dilation and orientation of the Gecko Shear relative to the Polaris/Peak Shear suggests a west block up sense of shear.

Syn-Late Stage Mineralization Thrusting - The Polaris/Peak and Blue Shears are interpreted to have been reactivated as thrust faults, post deposition of the main stage of mineralization in the Peak orebody. A series of cross-faults, including the Hecla Shear, link these two thrusts. These shears are characterized by black chlorite alteration. Displacement on the Polaris/Peak Shear is estimated to be approximately 250m west block up with a minor (20m) component of left-lateral strike slip. Displacement on the Blue Shear is estimated to be approximately 50m west-block up on the basis of the offset of the Great Cobar Slate-Chesney Formation contact. The Hecla Shear terminates the shears within the eastern volcanic lens and is interpreted to have formed as a short-cut structure along the rhyolite-sediment contact.

A series of north south striking moderately west-dipping shears linking between the Polaris/Peak and Blue Shears at the base of the eastern rhyolite body dismember and displace both the rhyolite and the contained Eastern Lenses. Sense of displacement on these shears is 5 to 30m west-block up.

Mineralization, Paragenesis and Associated Alteration

A broad zone of sodium and lithium depletion coincides with the chloritized alteration envelope surrounding the Peak Shear zone (Robertson, 1983).

Hinman (1992) outlined five stages of mineralization within the Peak deposit, all of which post-date emplacement of the rhyolitic bodies:

1. Disseminated pyrite mineralization within high strain zones;
2. Minor quartz-calcite-pyrite veins;
3. Silver-poor sphalerite-galena mineralization associated with zones of intense cleavage-parallel silicification of the host sediments;
4. Vein-hosted and replacement-style quartz-rich pyrrhotite-chalcopyrite-(sphalerite-galena)-gold-electrum-chlorite-muscovite assemblages; and
5. Silica-absent, black chlorite-(muscovite)-rich, silver-rich sphalerite-galena-pyrrhotite-pyrite-(chalcopyrite) mineralization in late shears.

Stages 3, 4 and 5 are the main ore-forming events. Stage 3 and 4 mineralization is synchronous with cleavage formation and the main period of deformation (Hinman's (1992) D2-3 and Glen's (1994) D1). Stage 5 mineralization cross-cuts and overprints cleavage and is temporally associated with a later period of deformation (Hinman's (1992) D4).

9.2.2 The Perseverance Orebodies

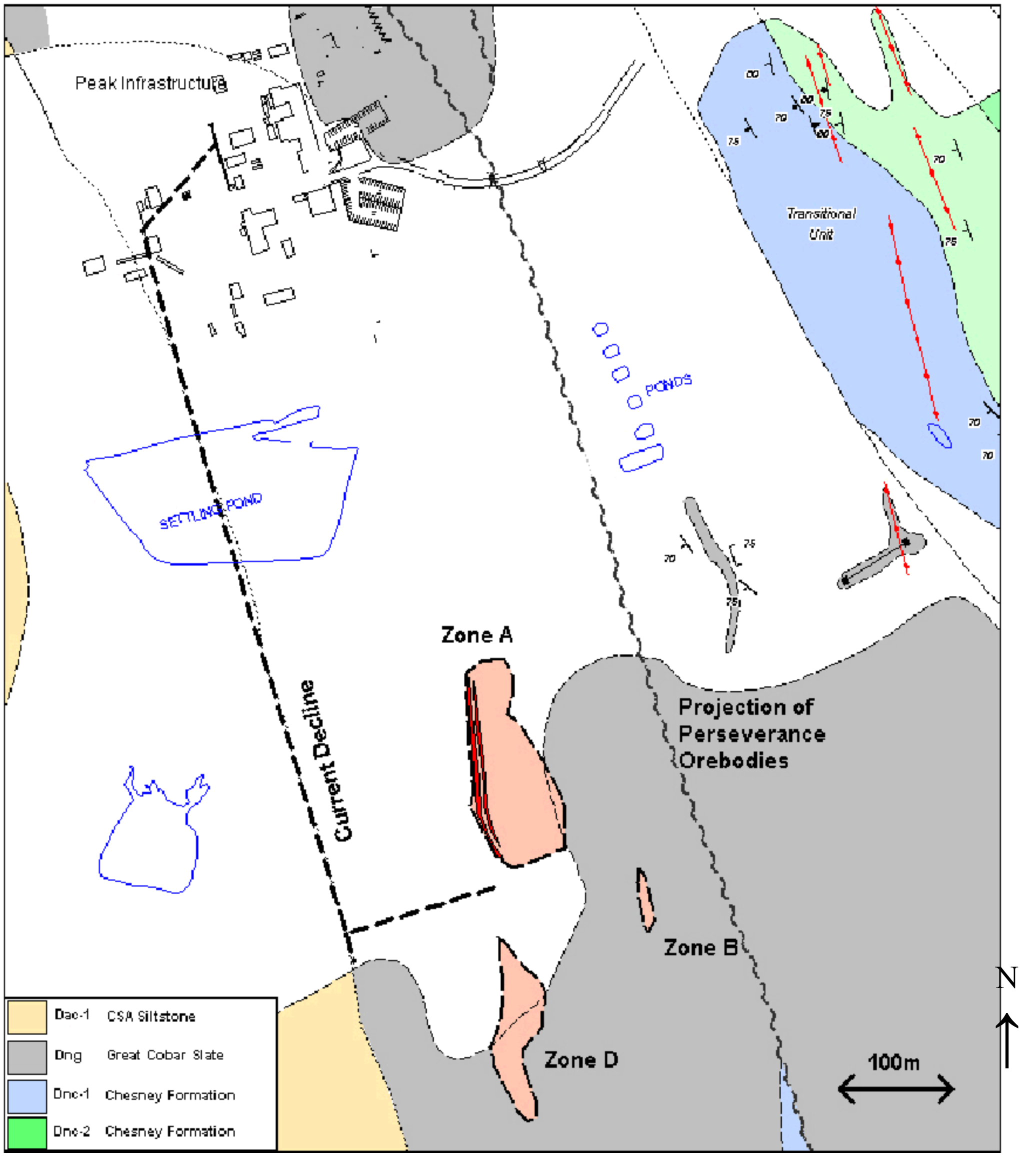
Local Geologic Setting

The Perseverance deposits are part of the gold and base metal deposits of the Peak Shear system. Approximately 200m east of, but parallel to and part of the Peak Shear is the Blue Shear and associated gold and silver mineral deposits, the Blue Lode and the Silver Peak. Two hundred metres further to the east is another parallel shear zone called the Lady Greaves Shear. The Lady Greaves Shear is host to the Lady Greaves gold mine. All the surface deposits are sediment hosted in fault-related veins and breccias. Deposits of the Peak Shear typically have moderate chlorite alteration and variable quantities of pervasive silicification and quartz veining.

Approximately 150m west of, and parallel to, the Peak Shear lies the Perseverance Fault. West of the Perseverance Fault is a stratigraphic sequence comprised predominantly of an upwards fining package of sandstone and slate assigned to the Chesney Formation and Great Cobar Slate. East of the Perseverance Fault are variably chlorite altered, folded and crushed sediments of the lower Chesney Formation and a body of fine-grained, variably perlitic, peperitic, auto-brecciated or flow-banded, sub-intrusive rhyolite (see Figure 9-17).

The dimensions of the rhyolite body are disproportionate such that the overall geometry is that of a long and narrow tabular block oriented roughly perpendicular to sedimentary layering, dipping 30° to 40° to the east and striking roughly parallel to stratigraphy and regional cleavage. In underground exposures the rhyolite margins are irregular and commonly diffuse as a result of brecciation and alteration. The rhyolite is interpreted to be continuous over at least a 2km strike length as it is open to the south of Perseverance and approximately 1km north of Peak, as shown by drilling from the New Occidental access drive. Its horizontal width varies between 100m and 300m and the root zone is at least 700m east from where the rhyolite is faulted off against the Perseverance Fault.

Figure 9-15: Peak to Perseverance Surface Geology

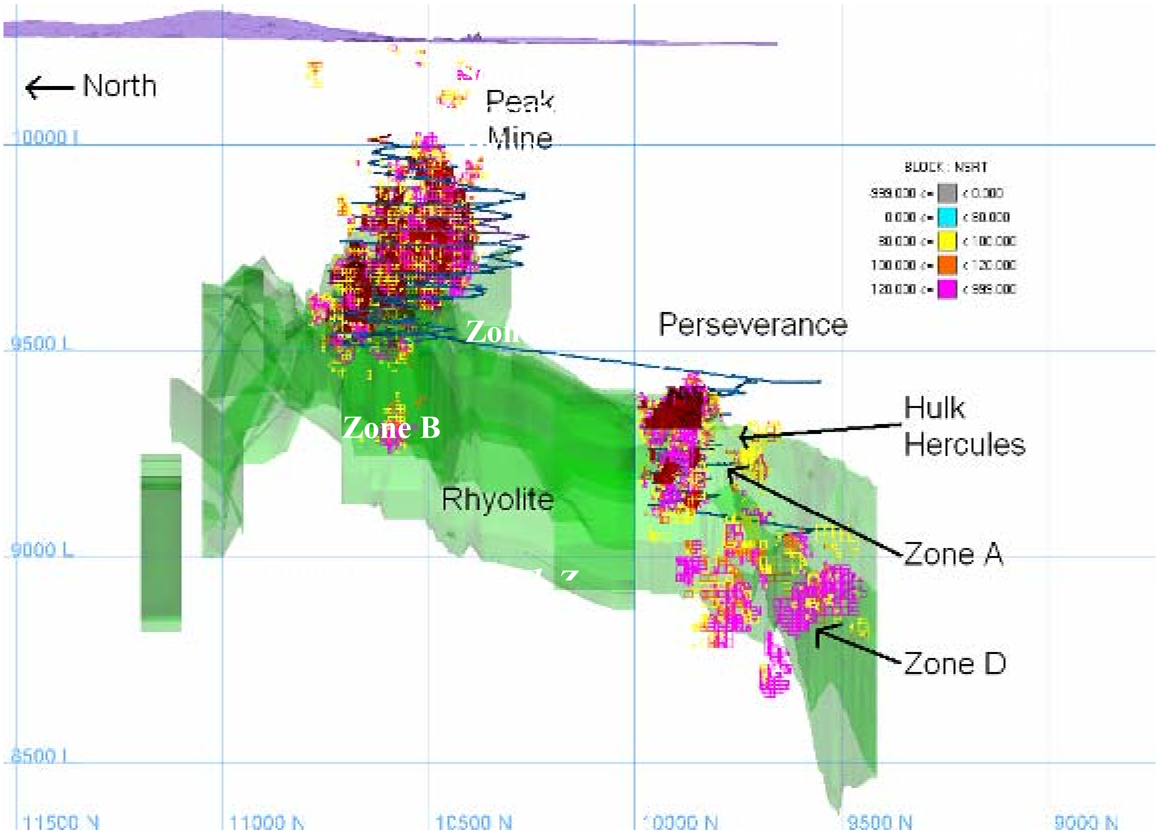


The Peak Shear is shown east of Zone A. Perseverance Fault is not shown.

Deposit Style

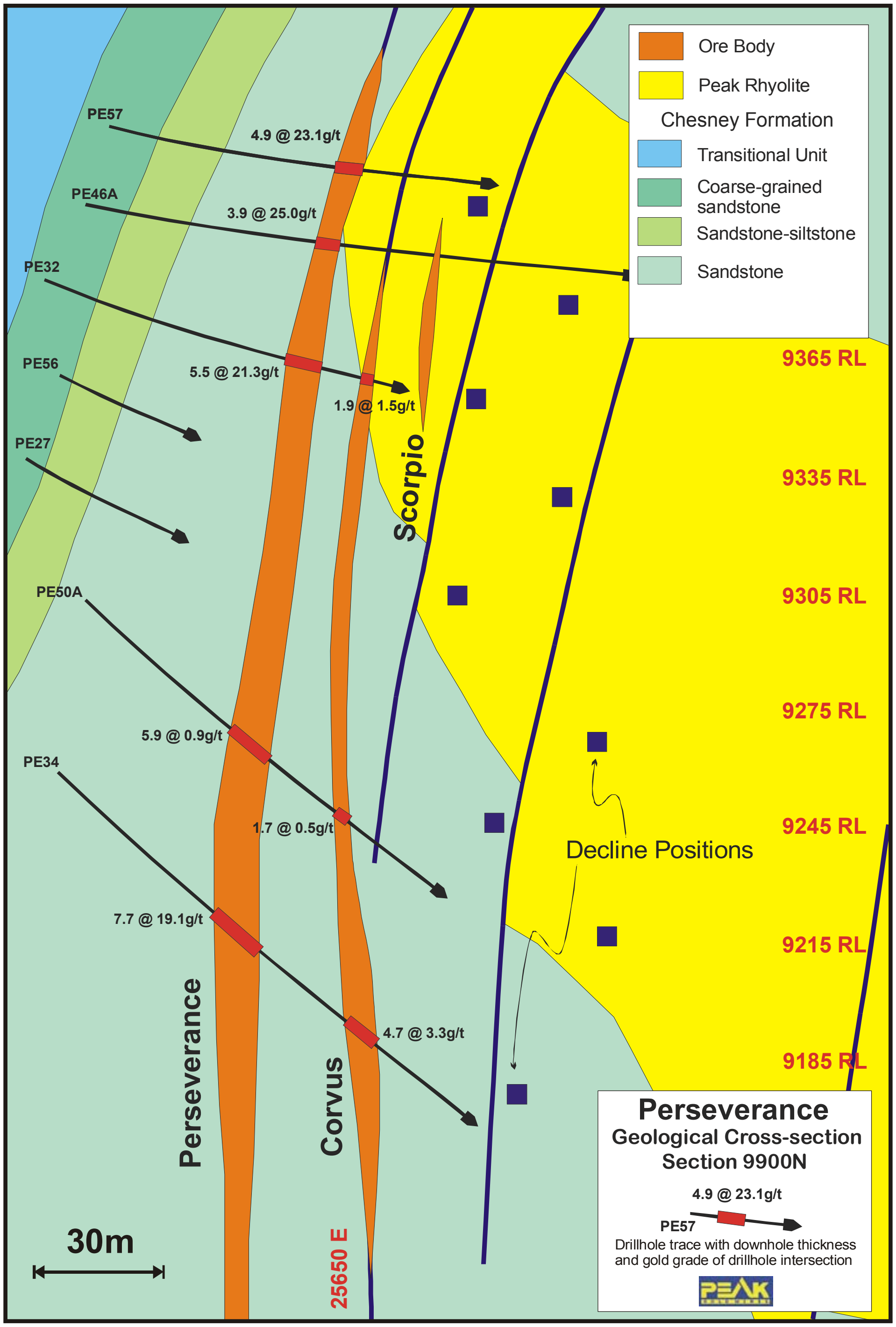
The Perseverance deposits are deformed, fault hosted, sub-vertical gold and copper lenses approximately 800m south of the Peak Mine, which is 9km south of Cobar. The lenses are subdivided into four zones (A to D) based on geometry (see Figure 9-16). The Zone A lens is the closest to existing infrastructure and has the highest density of drilling. Zones B, C and D have been the focus of recent drilling.

Figure 9-16: Oblique View of Perseverance Zones Looking East



The upper-most zone (Zone A) is between 820m and 1,060m deep and located 230m east of the south exploration drive. It is hosted in the steeply east or west dipping, north-south striking, arcuate Perseverance Fault. Coarse native gold is associated with deformed quartz-chlorite veins in variably silica and chlorite altered breccias. The Perseverance Fault is overprinted by chalcopyrite-pyrrhotite-electrum and galena-sulphide mineralization containing remobilized gold. Zone A consists of three sub-parallel lenses side by side and variably between 1m and 25m horizontally apart, striking north-south (340° magnetic) and dipping steeply to the west or vertical. Zone A is approximately 150m long in the north-south direction and 240m vertically. The three sub-lenses of Zone A are called, from west to east, Perseverance, Corvus and Scorpio (see Figure 9-17). The Corvus and Scorpio Shears are believed to be splays off of the Perseverance Fault. Mineralization becomes progressively less continuous in the eastern lenses. The Scorpio Lens mineralization immediately to the East of the Corvus Lens is characterized by erratic high gold with associated silver and molybdenum. Minor copper, if present, appears to be linked with bismuth and antimony. This style is akin to the small high grade ore shoots previously mined as part of the Peak orebody.

Figure 9-17: Perseverance Zone A Geologic Cross Section



Gold and copper mineralization in Zone D is hosted in faults that are interpreted to be equivalent to the Perseverance Fault. Gold mineralization in Zone D is best developed in areas of intense chlorite alteration indicating fluid-wall rock interaction.

Gold mineralization in Zone B is typically very high-grade but erratically distributed in narrow, fractures that are currently unable to be confidently traced over long vertical distances. The drilling density in Zone B is insufficient to accurately characterize the distribution and continuity of the gold bearing fractures.

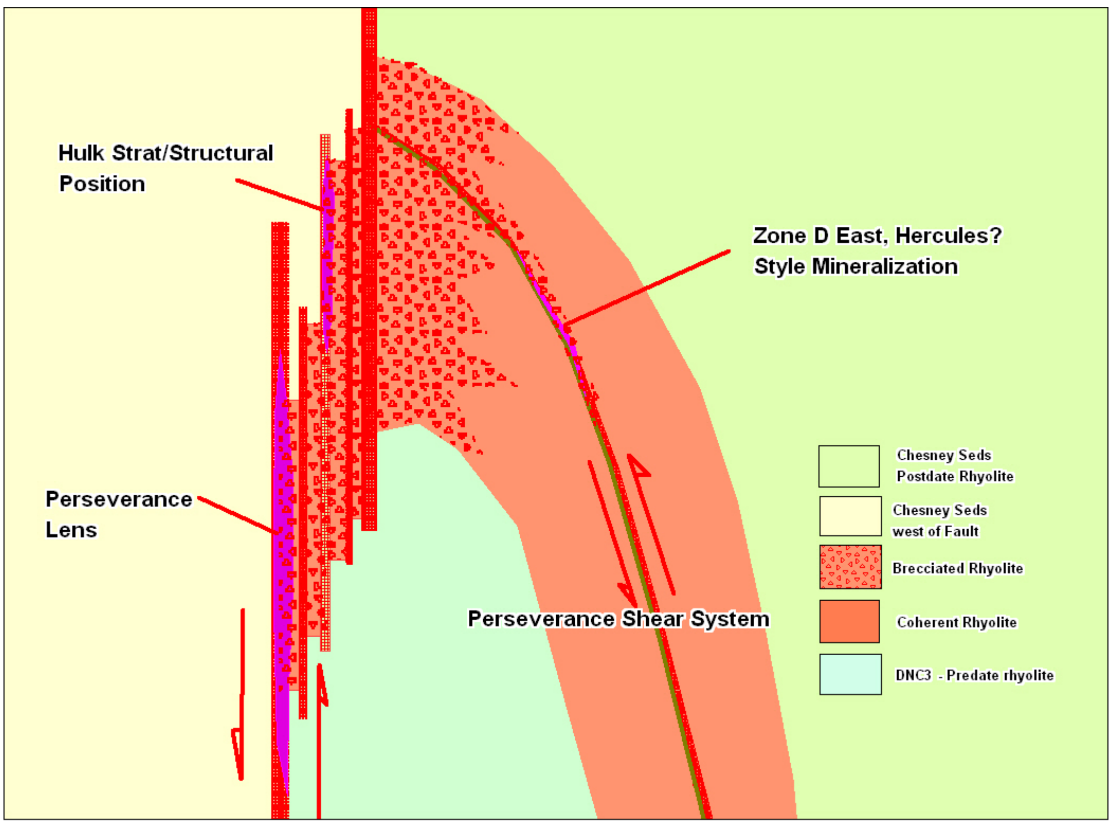
Typically Zone B has a low sulphide content with the most prevalent accessory mineral being pyrrhotite and chlorite.

Several other significant fault zones containing high-grade gold, copper, lead and zinc assays were intersected in the drilling program. Follow-up of these results has not been completed due to their location or immediate priority rating. None of these results are accounted for in the current orebody models.

The Hulk lens (Figure 9-18) appears to be related to a fault splay to the East of the main Peak fault position. The lower levels of Hulk are sediment hosted while the upper sections are hosted by rhyolite breccias where the fault truncates the rhyolite

The Hercules lens is hosted within Rhyolite breccia marginal to the eastern rhyolite contact. The control on the position of the Hercules Lens could arguable be vertical shearing cutting through the rhyolite or alternatively Hercules could be related to shearing wrapping around the eastern contact of the rhyolite with Hercules being the equivalent of the silica flooded rhyolite breccia hosted portion of Zone D.

Figure 9-18: Schematic geological cross section through Perseverance



Alteration

The regional metamorphic grade of the lower stratigraphic section of the Cobar Basin is green-schist facies. Metamorphic grades around the Peak Shear are interpreted to reach upper greenschist facies as indicated by the alteration of chlorite to biotite at the margins of chlorite aggregates. Silicification is also commonly associated with the fault zones as a pervasive addition to the chloritic psammites (sandstones). Table 9-3 lists habits and common associations of transparent gangue minerals.

Table 9-3: Major Alteration Minerals, Habit and Associations

Alteration Mineral	Ideal Stoichiometry	Habit	Common Association
Quartz (silica)	SiO_2	Massive and crystalline	Pervasive in shear zones. Antitaxial and syntaxial quartz veins, dog-tooth crystals.
Chlorite (chamosite)	$(Mg, Fe)_5Al(Si_3Al)O_{10}(OH, O)_8$	Massive	Regionally pervasive. Intense within shear zones.
Sericite (muscovite)	$KAl_2(Si_3Al)O_{10}(OH)_2$	Massive	Sparse regionally, associated with shear zones and rhyolite
Calcite	$CaCO_3$	Crystalline	Intergrown with quartz in veins
Biotite	$K(Mg, Fe^{2+})_3(Al,Fe^{3+})Si_3O_{10}(OH, F)_2$	Alteration rims	Shear zones and quartz veins.

The regional green-schist alteration is interpreted to pre-date the main stage gold and sulphide mineralization. There is also evidence of significant levels of faulting and pressure induced strain in the sediments and the rhyolite prior to mineralization.

Hydrothermal Mineralogy

Minerals identified to be associated with hydrothermal fluid activity are listed in Tables 9-4, 9-5 and 9-6. The predominant assemblages are chalcopyrite-pyrrhotite-(cubanite), sphalerite-galena, sphalerite-pyrrhotite, coarse-grained euhedral galena and arsenopyrite. Other minerals less frequently seen, or only seen in thin section and SEM, include minor precious metal tellurides, antimony and bismuth minerals and amalgams of gold, silver and bismuth. Mineralogical relationships are elaborated in Reynolds (2000) and processing implications are discussed by Ratcliff (2001).

Table 9-4: Early Stage Hydrothermal Mineralogy – Pre-Gold

Mineral	**Ideal Stoichiometry**	**Mineralogical Relationship to Gold Potential for Refractory Behaviour**
Scheelite	$CaWO_4$	Not associated.
Rutile	TiO_2	
Ilmenite	$FeTiO_3$	
Sphene	$CaTiSiO_5$	
Molybdenite	MoS_2	Not related. May be associated with the rhyolite.
Arsenopyrite	FeAsS	Not related. Relationship with gold not known.

Table 9-5: Main Stage (Gold) Hydrothermal – Deformation Mineralogy

Mineral	**Ideal Stoichiometry**	**Mineralogical Relationship to Gold Potential for Refractory Behaviour**
Galena	PbS	Early phases not related to Au. Late phases associated with Au by accessory minerals. Not refractory but could report to a Pb-Zn blend.
Sphalerite	(Zn, Fe)S	
Bismuthinite	BiS_3	Bismuth minerals apart from native bismuth are rare. Understood to be an alteration product of native Bi. Associated with gold. Quantity assures that it will not have a significant refractory affect.
Maldonite	$AuBi_2$	
Chalcopyrite	$CuFeS_2$	Common. Intimately associated with pyrrhotite, cubanite and electrum. Not considered refractory.
Cubanite	$CuFe_2S_3$	Occurs as ex-solution lamellae within chalcopyrite. Associated with electrum. Not considered refractory.
Pyrrhotite	$Fe_{1-x}S$	Common. There are up to 3 phases of pyrrhotite. Stage 1 is rare and associated with high-fineness Au. This phase may have refractory qualities dependent on grind size. The

		dominant pyrrhotite phase, stage 2 is associated with chalcopyrite, cubanite and electrum. Late stage pyrrhotite is associated with late galena, troilite and the antimony minerals. This phase is only minor and the relationship to gold mineralogy is not understood.
Troilite	FeS	Occurs as ex-solution intergrowths with late stage pyrrhotite. Refractory qualities not significant.
Pyrite	FeS_2	No association or only minor association between pyrite and gold. Relationship not significant.

Table 9-6: Late Stage Hydrothermal Mineralogy Antimony-Lead-Silver Association

Mineral	**Ideal Stoichiometry**	**Mineralogical Relationship to Gold Potential for Refractory Behaviour**
Costibite	CoSbS	Minor phase. Relationship to gold not significant.
Allargentum	$Ag_{1-X}Sb_{1-X}$	Minor phases. Intimately associated with gold. Solid solution series exists ranging from auriferous metallic silver to allargentum including dyscrasite. Au and Sb substitute in the allargentum crystal lattice
Dyscrasite	Ag_3Sb	
Aurostibite	$AuSb_2$	Understood to be an alteration product of maldonite, this minor mineralogical phase is likely to be insignificant as a refractory phase.
Breithauptite	NiSb	Associated with the chalcopyrite-pyrrhotite-cubanite-electrum assemblage, this minor phase is not considered to have a significant refractory influence.
Gudmundite	FeSbS	Not significant as a refractory phase.
Hessite	Ag_2Te	The tellurides are rare, associated with the late stage galena minerals. They are associated with gold but are not considered to have significant refractory qualities.
Sylvanite	$(Au,Ag)_3Te_4$	
Calaverite	$AuTe_2$	
Volynskite	$AgBiTe_2$	

Vein Types

Cleavage parallel quartz-chlorite veins, quartz breccias, deformed quartz veins and late quartz-calcite veins are common at the Perseverance deposits. The most significant vein morphology for gold mineralization is the tight to isoclinally folded, sub-horizontal, recrystallized dog-tooth quartz-chlorite veins of noted abundance in the Perseverance Lens, Zone A. This vein type has a noted association with high-grade gold assays. These sub-horizontal vein sets are believed to open vertically when hydrothermal fluid pressures becomes greater than lithostatic pressure. These veins are known to crosscut, and be cut by, sub-vertical cleavage parallel quartz-chlorite veins. To date visible gold has only been recorded in sub-horizontal or highly deformed veins.

9.3 Other Shears and Copper Deposits

9.3.1 Gladstone Deposit

The Gladstone deposit is located approximately 500m west of the Chesney deposit (Figure 7-4). The deposit occurs at the intersection of two steep east-dipping shear zones, a north-northwest trending shear that is parallel to the main shears in the nearby GCF, and a more northwest trending shear zone which parallels the cross-shears in the New Cobar, Chesney and Occidental deposits (Mulholland and Rayner, 1961). The slates surrounding the deposit are strongly deformed and Thompson (1950) notes that folds in the slates are typically steep north plunging adjacent to the lodes but south dipping elsewhere. Glen (1987b) suggests that the shears hosting the Gladstone deposit transects a broad south plunging fold.

The deposit comprises a single lode approximately 5 to 10m thick and in excess of 450m long (Phillips. 1987). Drilling data is widely spaced and therefore the deposit may be considerably more complicated. Mineralization has been traced down plunge to 450m from surface and open with depth (Phillips, 1987) and Mulholland and Rayner (1961) believe the deposit has a steep northerly plunge like all of the other deposits in the gold field.

The lode comprises a central quartz breccia containing abundant shale fragments and massive veins of chalcopyrite, within a halo of both cleavage parallel and transgressive quartz veining and strong pervasive green chlorite alteration containing patchy chalcopyrite-pyrrhotite-sphalerite mineralization. Pyrrhotite disseminations are ubiquitously developed along cleavage planes outboard of the deposit (Gilligan and Byrnes, 1995).

9.3.2 Dapville Deposit

The Dapville deposit is located approximately 1.3km south-southwest of the Gladstone deposit and 800m northwest of New Cobar mineralization (Figure 7-4). The deposit occurs at the southern termination of the Great Cobar Shear system (Andrews, 1911) and was discovered by drill-testing of a 250m long, north-northwest trending zone of ferruginous and siliceous sheared siltstones (Thompson, 1950). Thompson (1950) interprets the Dapville mineralization as a single lode position developed along the steep north-plunging intersection of a north-trending, steep west-dipping shear and a broad north-northwest trending steep east-dipping main shear (the latter shear being the Great Cobar shear).

The lode at Dapville is interpreted from sparse drilling to be 1 to 5m wide and approximately 120m long (Gilligan and Byrnes, 1995). The deposit consists of weakly quartz veined strongly brecciated and chlorite altered slates. Mineralization comprises splashy chalcopyrite-magnetite-pyrrhotite-minor pyrite mineralization. The best copper grades are associated with magnetite-rich ore. Extensive zones of cleavage-parallel quartz veining are present to the east of the deposit (Thompson, 1950).

9.3.3 Great Cobar Deposit

The Great Cobar deposit, at which the original discovery of copper in Cobar was made, is located approximately 1.5km north-northwest of the Dapville mineralization (Figure 7-4). The deposit occurs within a north-northwest trending shear zone that extends south to the Dapville deposit and north at least 300m beyond the deposit (Thompson, 1950). The shear zone appears to cut through the hinge zone of a north-plunging anticline, although the eastern limb area is poorly exposed (Andrews, 1913; Glen, 1987b). Bedding-cleavage intersections plunge north around the shear zone and to the south, further west of the shear (Thompson, 1950).

The Great Cobar deposit comprises three steeply (75 to 85°) north plunging lodes developed over a 400m strike length of the shear zone, separated by zones of weakly mineralized silicified and quartz veined slates. However, surface gossans extend over a 730m strike length of the shear (Gilligan and Byrnes, 1995). The lodes have an elliptical shape and generally dip steeply to the east (88°) but locally dip steeply to the west. Individual dimensions of the lodes are:

Northern Lode: 110m long and 30m wide.

Central Lode: 130m long and 25m wide.

Southern Lode: 45m long and 15m wide.

Mineralization intersected at depth in workings at the Great Cobar North deposit is interpreted to be the northern extension of the Northern Lode (Andrews, 1911). The Great Cobar Lodes were mined to a depth of 500m below surface and deep drilling has determined that they extend to 1,000m below surface (Thompson, 1969).

The lodes consisted of chalcopyrite ore developed in talcose black chlorite, a central zone of brecciated silicified and chlorite-altered slates hosting magnetite-pyrrhotite-chalcopyrite mineralization (basic ore) and an eastern hangingwall zone of quartz vein breccias and strong quartz veining hosting chalcopyrite-pyrrhotite-lesser magnetite mineralization (siliceous ore) (Andrews, 1911). The proportion of the basic mineralization appears to diminish with depth at the expense of the siliceous mineralization, although copper grades do not greatly change (Thompson, 1950).

Free gold was recognised in the oxide zone at Great Cobar (Andrews, 1911) and small tonnages of gold-bearing oxide ore were extracted from remnants of the Central Lode in the 1940’s (Mulholland and Rayner, 1961). However, little indication of the gold distribution in the primary mineralization is provided by previous mining data.

10 EXPLORATION

The Cobar Gold Field (CGF) is defined as the 10 kilometre long north-trending belt of historical gold mines located east of Cobar which extend from the Peak Mine, south-southeast of Cobar, to the Tharsis workings, north of Cobar. The CGF occurs on the eastern margin of the Devonian Cobar Basin and has produced in excess of 2.75 million ounces of gold and 200,000 tonnes of copper since mining commenced 134 years ago (Stegman, 1998). The currently producing mines within the Cobar Gold Field operated by Peak Gold Mines Pty Ltd include the New Occidental, New Cobar, Peak and Perseverance mines. Mining and exploration has occurred in the CGF area since the late 1800's. It is outside of the scope of this document to describe historic exploration efforts. This section will concentrate on exploration since 2000 by PGM.

10.1 Recent Exploration

10.1.1 Near Mine Exploration

New Cobar

During the year 2000, thirteen RC holes totalling 578 metres were completed around the New Cobar open cut to test for extensions to the south. No additional mineralisation was discovered. In 2003, 27 holes totalling 10,559 metres were drilled during the feasibility study for the underground resource. A further 36 holes were drilled in 2007 for 3496.5m. Of these 20 were included in the 2007 resource estimate of the 30 to 40 levels of the mine.

Chesney

In 2004 and 2005, a drilling program comprising 52 diamond drill holes totalling 17,252 metres was completed to test both the Main Lode from below the old workings to 800 metres below surface and the Eastern Gold Lens below the oxide zone. A down-hole electromagnetic survey (DHEM) was completed on all drill holes and generated additional drill targets. In 2007, 3 diamond holes were drilled from surface to test the southern shoot of the Main lens at depth. These holes were drilled as daughters with the parent hole being 1200m in depth.

New Occidental

In 2002, 2003 and 2004, a total of 16 diamond drill holes were completed to explore the New Occidental Deeps area below the 92 level to further define the resource. The result was an increase of 4,906 ounces of gold in the resource base. In 2007 a program of 5 NQ exploration holes were started from the 101 Diamond drill cuddy to further augment the delineation drilling of the 102 – 108 levels. Two of these holes completed late in the period and were not added to the resource estimate. A further 56 holes were drilled delineating the area between the 98 and 108 Levels.

Peak

Since 2003, 46 diamond and RC drill holes totalling 13,249 metres have been completed, testing the Peak Uppers, Peak Deeps, Peak Oxide and Peak North areas. The Peak Uppers program comprised 16 holes totalling 3,962 metres and resulted in the delineation of a Measured, Indicated and Inferred Resource. During 2003, 2004 and 2006, 18 underground diamond drill holes totalling 5,538 metres were drilled to further evaluate the Peak Deeps Resource, resulting in increased confidence in the resources. During 2002 and 2004, 12 diamond drill holes totalling 3,808 metres were completed north of the Peak Mine to test geophysical targets; no significant intercepts were identified. A further 14 holes were drilled in 2007 for mine delineation in the Peak Uppers and the Peak North Areas. No holes were drilled in 2008 in the Peak Deposit.

Perseverance

The evaluation of the Perseverance Zone D commenced in 2002 and continued intermittently through 2003 and 2005 to 2008. A total of 120 underground diamond drill holes have been completed totalling 38,440 metres which were used for the Zone D resource estimate. During this time, the Hulk, Hercules and Zone D East mineralisation lenses were discovered in the footwall to Zone D.

10.1.2 Other Near-Mine Prospects

Exploration in other near-mine prospects (Figure 10-1) includes the following:

Great Cobar

In 2004, nine diamond holes totalling 5,622m were completed to test below the old mine workings and to follow up down-hole EM anomalies. The results included a high grade copper intercept in hole GC2B (17.5m at 4.45% Cu) and two narrow high grade gold intercepts in hole GC2 (1m at 9.98g/t Au and 1m at 5.77g/t Au).

Dapville

A single 447m deep diamond hole was drilled in 2004 to follow up previously identified copper mineralisation, magnetic and down-hole EM targets. The hole intersected 3m of 0.83% Cu, similar in style to Great Cobar style mineralisation. DHEM surveying shows a strong off-hole conductor, which remains untested by drilling.

Gladstone

In 2004, four diamond holes totalling 1,085.9m were completed to verify previous high-grade intercepts (DD42GS0002, 4.7m true width at 5.2% Cu) and increase confidence in the continuity of high-grade Cu mineralisation at depth. The drill holes confirmed continuity of a steep north-plunging, higher grade lens in the southern strike length of the deposit up to 10m in width. The results indicated that Gladstone consists of Cu-Ag mineralisation only. No significant assays were returned for gold, lead or zinc.

Jubilee

In 2008 two series of diamond wedge holes were drilled at the Jubilee prospect, 100m north of New Cobar, with the aim of infilling a 100m vertical gap in the information between the lowermost most historical stoping block and the deeper pattern of previous exploration holes. Results were encouraging, with a best intersection in DD08NC0097C with 6m of 2.09 g/t Au and 3.68% Cu. Down-hole EM surveying detected two moderate to good conductor targets not directly attributable to the known New Cobar or Jubilee mineralisation. The deeper conductor does not appear to have been tested by previous drilling, and is therefore a high-priority target.

Burrabungie

In 2008 two diamond drill holes totalling 627m were completed at Burrabungie, located between Chesney and Young Australia, to test the continuity of mineralisation at depth. Results were promising, with a best intersection of 6m of 3.1% Cu. Down-hole EM surveying of these holes detected conductors suggesting further continuation of the known mineralised structure at depth.

Young Australia / Mount Pleasant

Nine diamond drill holes were completed during 2007 and 2008 to test below sub-economic drill intersections returned in earlier drilling. A total of 2861.3m were drilled to test for the continuation of down-plunge mineralisation interpreted to occur in three sub-vertical mineralised zones. Areas of broad low level anomalous copper were intersected as well as several narrower high-grade copper and gold zones (best 3m of 9.4 g/t Au and 0.8% Cu). Down-hole EM surveying of selected holes did not detect any significant conductors.

Comstock

Located south of New Occidental, Comstock was tested with five diamond drill holes totalling 2,714m in 2003, 2004 and 2006, following up down-hole EM targets generated from previous drilling. The holes successfully tested the targets but only minor mineralisation was intersected.

Coronation / Beechworth

A programme of RC drilling (17 holes for 1,681m) were completed in the vicinity of the Coronation and Beechworth workings in 2000 / 2001, however only scattered anomalous copper and lead, and no significant gold intersections were returned. In 2005 a deep diamond drill hole (DD05CB0064, 1,110.3m) was completed at Coronation to test a conceptual target, the interpreted position of the Chesney Formation – Great Cobar Slate contact approximately 600m below surface. The target contact was intersected at 555m and comprised an unmineralised quartz vein. Unmineralised quartz veining and alteration were also intersected at the Coronation structure at 925m. No significant assays were returned and down-hole EM detected only a weak conductor at the Coronation structure intersection.

Queen Bee / Porphyry North

An IP survey was conducted over the Queen Bee workings in 2008. Chargeable anomalies detected were then tested by diamond drilling, which failed to intersect significant mineralisation or explain the IP anomalies, but did intersect previously unknown feldspar porphyry. Down-hole EM surveying conducted on two of these drill holes failed to detect any in-hole or off-hole conductors.

At the Porphyry North prospect, 4 diamond holes were completed in 2001 to follow up anomalous gold in soils and rock chips and test geophysical anomalies. No mineralisation was intersected.

10.1.3 Regional Exploration

Regional exploration on the PGM prospects included the following:

EL5933 Peak

This tenement surrounds the PGM mines at Cobar. Exploration of EL5933 has comprised IP, gravity, and down-hole EM geophysical surveys; and RAB, RC, and diamond drilling targeting the Langtons, Illawong, Central Shaft, Cobar Lucknow and Fortitude prospects.

At the Langtons prospect, RC drilling followed up near-surface highly anomalous gold detected in previous RAB drilling; however the deeper drilling did not intersect any significant mineralisation. At the Illawong prospect, strong coincident IP and CSAMT geophysical anomalies were tested by RAB and RC drilling, however no significant mineralisation was intersected, the anomalies being due to formational (stratigraphic) sources. Similar strong IP anomalies at the Central Shaft were tested by RC drilling, but were also found to be sourced from graphitic siltstones with no significant mineralisation intersected. Previously detected gold anomalies at Cobar Lucknow, located northeast of Cobar, were tested by IP surveying which detected strong responses associated with silicification. These anomalies were subsequently tested with RAB drilling which returned scattered anomalous gold assays. Diamond drilling to test CSAMT anomalies at Fortitude, south of the Peak, intersected only weak mineralisation and quartz veining.

In 2008 an airborne magnetic and radiometric survey was completed over EL5933, and gravity surveying was extended over the full tenement. Inversion modeling of the magnetic and gravity data was completed, and merged with a 3D geology model. Follow-up exploration of the anomalies detected is planned in 2009.

EL6127 Rookery South

This tenement covers 48km of prospective Rookery Fault environs southeast of Cobar, including the Stones Tank Resource located at the south end of the tenement, approximately 70km southeast of Cobar. An RC and diamond drilling program completed in 2004 at Stones Tank to confirm and test for extensions to the known mineralisation increased confidence in the resource, but did not significantly change its size.

Elsewhere in EL6127, exploration has focussed on the Nurri to Rookery area, with magnetics and IP surveys and a substantial soil sampling programme completed, followed up with RC drilling at the Rookery prospect and diamond drilling at Copper Burr. RC drilling at Rookery tested several geological and geophysical targets; however the holes did not intersect any significant mineralisation. At the Copper Burr prospect, diamond drilling tested a very strong IP response and anomalous Au rock chip results. The holes intersected quartz veining, silicification, chlorite alteration and minor mineralisation. A number of geophysical and geochemical anomalies remain untested at Nurri to Rookery.

In 2008 an airborne magnetic and radiometric survey and ground-based gravity surveying was completed over all of EL6127. Inversion modeling of the magnetic and gravity data was completed, and merged with a 3D geology model. Follow-up exploration of the anomalies detected is planned in 2009.

EL6401 Rookery East

The tenement is located 50km southeast of Cobar. In 2008 airborne magnetic and radiometric surveying and ground-based gravity surveying were conducted over all of EL6401. Inversion modelling of the magnetic and gravity data was completed, and merged with a 3D geology model. Follow-up exploration of the anomalies detected is planned in 2009.

EL6149 Mafeesh

This tenement, located 50km south of Cobar, was taken up to investigate geochemical anomalies detected by previous explorers. Exploration to date has included RAB drilling, IP surveying, RC drilling and diamond drilling over the main geochemical anomaly within EL6149. RAB drilling defined a 400m long multi-element geochemical anomaly, and IP surveying detected three chargeable anomalies. Subsequent RC drilling confirmed the geochemical anomaly but failed to intersect significant mineralisation. Deeper diamond drilling returned disappointing results with only minor quartz veining intersected in a sequence of interbedded sandstone and shale/mudstone, with minor highly altered mafic volcanics identified by petrology.

Further geochemical sampling to follow-up the remaining geochemical anomalies within EL6149 was conducted in January 2009. Assays are awaited.

EL5982 Norma Vale

This project is located 80km south of Cobar and contains prospective magnetic features and soil geochemistry. Exploration to date has included soil sampling, RAB drilling, IP surveying, diamond drilling and down-hole EM surveying.

RAB drilling in 2005 over magnetic features detected two geochemically anomalous areas requiring further work. IP surveying over these areas detected chargeability anomalies associated with the coincident geochemical and magnetic anomalies. This was followed up in late 2007 and early 2008 by four diamond drill holes for a total of 1,123.5m. These holes intersected generally low-grade mineralisation within a sequence of intensely altered sediments and volcanics cut by multi-phase altered intrusives. Down-hole EM surveying in the northernmost hole detected an off-hole conductor warranting further drill testing.

Geochemical sampling over magnetic anomalies associated with acid volcanics in two isolated northern blocks of EL5982 was completed in late 2008 and early 2009. Assays are awaited.

EL6402 Cable Downs

This tenement is located 40km northwest of Cobar. Exploration to date has comprised RC drilling following up gold anomalies detected in previous RAB drilling at Mountain Tank. No significant mineralisation was intersected.

Geochemical sampling to follow-up anomalies generated by previous explorers in the northern half of EL6402 was completed in late 2008. This programme detected several subtle gold anomalies, and infill and extension sampling is planned.

Figure 10-2 shows the location of the Cobar Goldfields targets projects and regional prospect respectively. Figure 7-4 and Figure 8-1 shows the location of current near-mine exploration targets.

10.2 Exploration Data Collection

The site geological database contains the details of all available historic drill holes on the project tenements. Historic drill-hole data that has the potential to impact on resource estimates is flagged in the database based upon perceived reliability and excluded from estimates if deemed appropriate.

The Rio Tinto plc (RTP) group of companies has been exploring in the Cobar gold field since the early 1940’s. PGM has access to much of this information, including underground survey data, production records and drill hole data The PGM Geology department hosts a large library of technical documents, both historic and current. The details of these documents are stored in an electronic database system for ease of use and identification.

Figure 10-1: Cobar Goldfield Targets (EL5933)

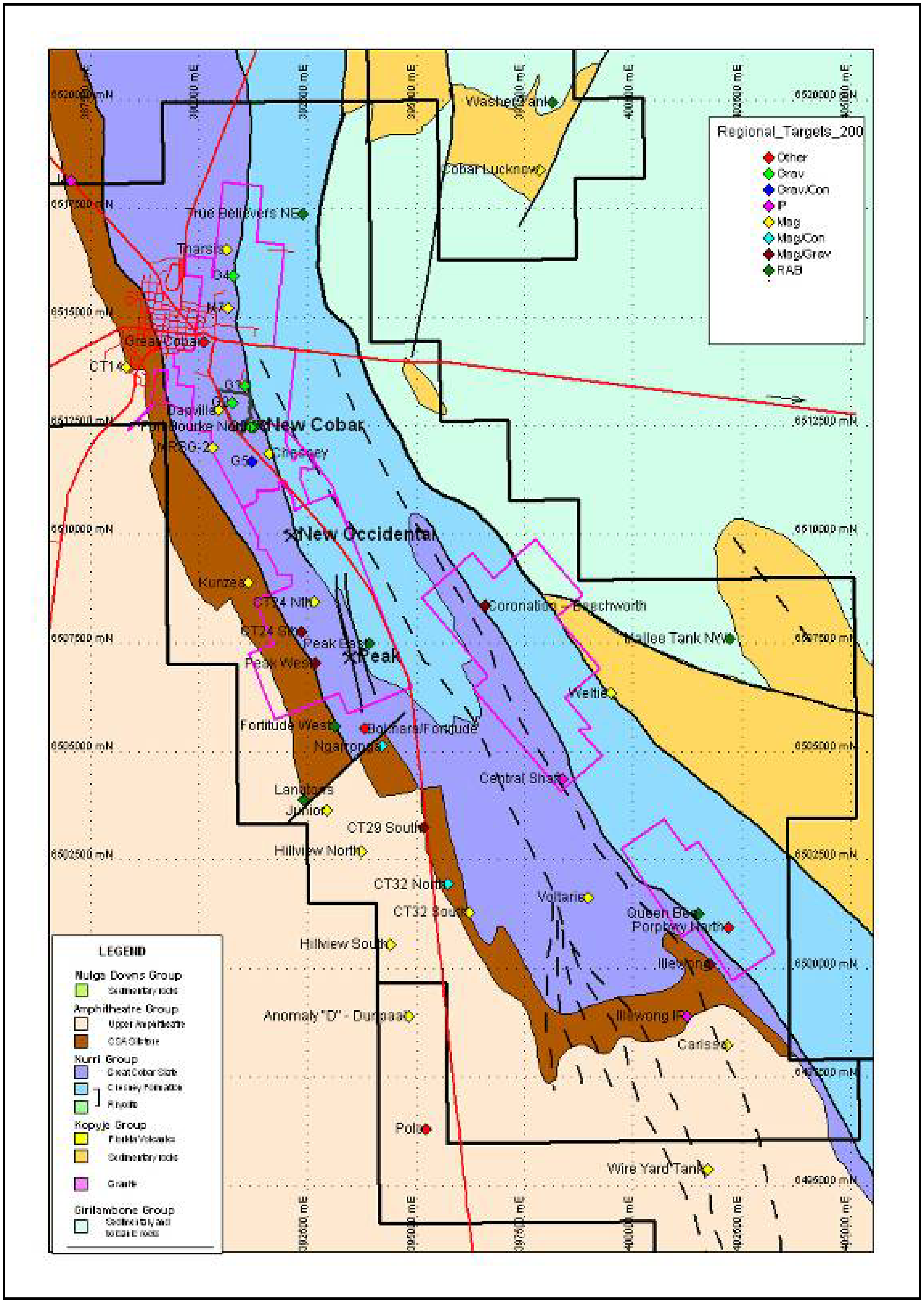
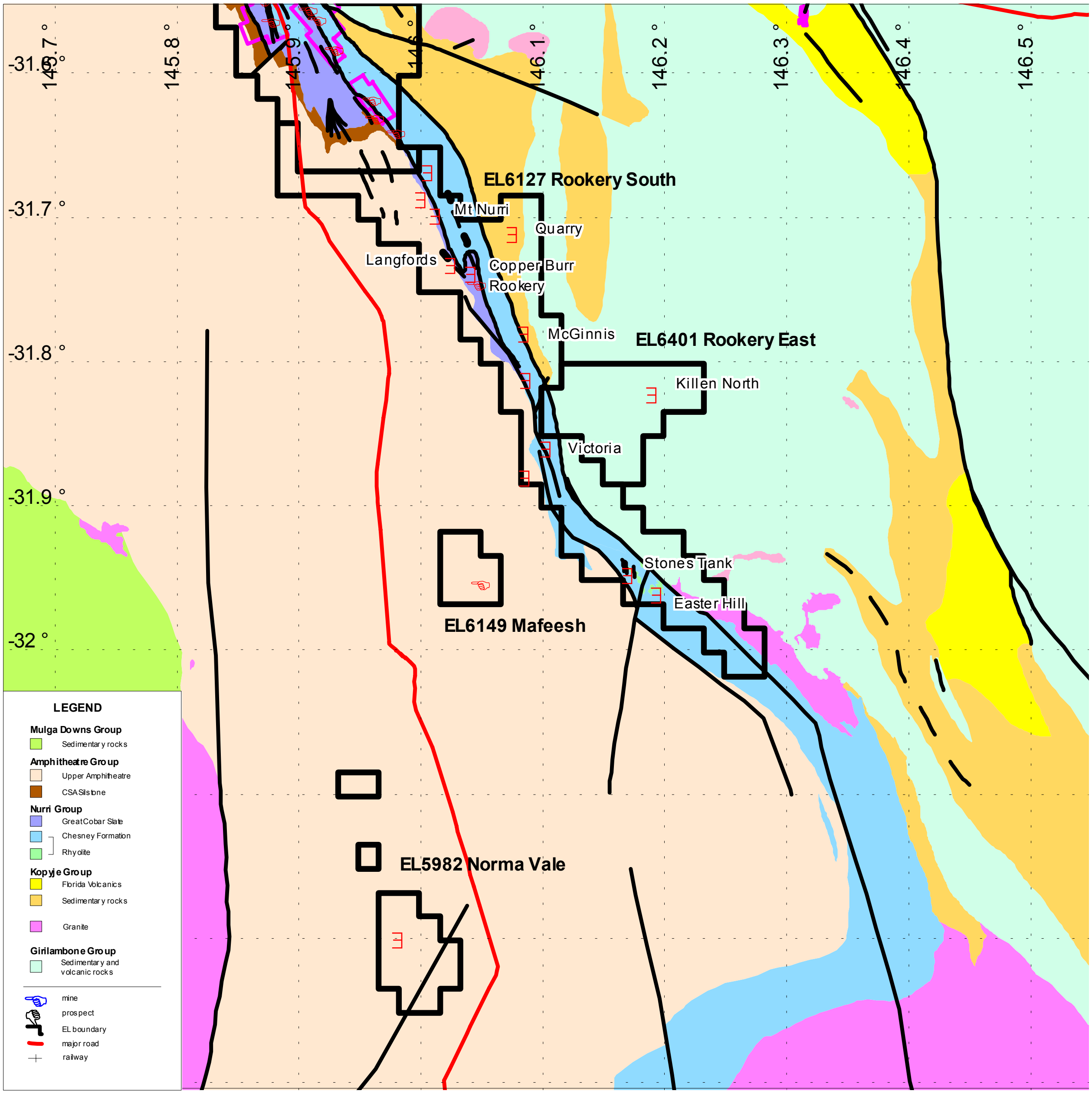


Figure 10-2: Regional Exploration Targets



11 DRILLING

Drilling data is available for the various operations dating back to 1931. For the purpose of this report, this section will concentrate on the type and extent of drilling since 1997, which is most relevant to the current Mineral Resources.

All relevant drill holes in the project area have been recorded in the site database using the Peak Mine grid. Grid conversions exist for converting the Peak Mine grid to the national Australian Mine Grid (AMG (AGD66)) and older grid systems.

11.1 Current Site Practice

11.1.1 Diamond Core

It is current PGM practice to use HQ diameter core for exploration diamond drill holes, stepping down to NQ/NQ2 diameter core at about 300 metres depth or when drilling problems are encountered. Occasionally, holes are collared using PQ diameter core. For this reason, most exploration intercepts of mineralized zones tend to be from NQ diameter core. Drilling of stope blocks for grade control, mining, and Measured Resource estimates utilises LTK48 diameter core. The drill core is not routinely oriented; instead the relatively consistent regional S_2 foliation is used as a reference feature for determination of core cutting lines and reference lines for structural readings.

All drill core is logged by qualified geologists in a core yard and storage facility at the Peak Gold Mine. The exploration geologists log the exploration holes, while mine geologists log the grade control/stope delineation drilling. All logging utilises portable computers connected to a wireless network. Basic tasks related to routine processing of core is completed by contract staff at the PGM core yard under the direct supervision of a PGM geologist, and includes tasks such as measurement of core recovery, metreage mark ups, tray photography, cutting, sampling, density measurement, dispatch and storage of core. Diamond core is currently sampled on 1m intervals.

Until December 1998 drill holes were logged using MS-DOS-based computer-logging software called Datcol™. Logging is now performed with Windows-based core logging software called CoreView™ which includes a database system. All drill hole information, including geological logging data, logistical information regarding drilling, collar data, down hole survey data and assay data, is stored using the Drillview™ database, which is a Microsoft Access-based logging and database system that was purpose built and is administered by senior staff.

The following information is routinely recorded during core logging:

- Rock types, brecciation and fault zones.
- Alteration and vein types.
- Mineralization and mineralization assemblages.
- Identified coarse gold for assay purposes.
- Rock mass quality data (RQD) for mine planning purposes.
- Intervals of core-loss.

11.1.2 Reverse Circulation Drilling

Reverse Circulation (RC) drilling is typically conducted using 130 to 140 millimetre face-sampling hammers. Cuttings are collected by a cyclone and reduced by a Jones riffle splitter to nominal 2 to 2.5 kilogram samples. Representative chips from each sample interval are stored in plastic chip logging trays for geological logging and future reference. All drilling is supervised by a PGM technical employee.

11.1.3 Collar and Down-Hole Surveys

All near-mine drill-hole collars are surveyed by the mine surveyors. Underground drill holes are routinely down-hole surveyed every 30 metres using an Eastman camera as drilling progresses. At the completion of drilling, an electronic multi-shot device (e.g. Reflex model) may be used to take additional survey measurements at 3 to 6 metre intervals. All Eastman single shot discs and original records of the electronic camera surveys are stored in the site Geology Offices in lever-arch files.

11.2 New Cobar Mine

The New Cobar Mine was discovered in 1889, with historical mining during the periods of 1880 to 1922 and 1934 to 1948. During the later period, underground grade control diamond drilling was performed.

From 1973 to 1997, sixteen drilling programs were undertaken to target the near-surface oxide potential.

During 2000 a total of 13 RC holes were drilled to test for continuation of the open pit resource. Surface drilling by PGM in 2003 delineated an underground resource amenable to mining, and a portal was started from the base of the pit in 2004. Subsequent surface and underground diamond drilling programs since 2005 have further delineated the underground resource. A single surface exploration program since 2005 was drilled in 2008 into the Jubilee lens north of the New Cobar lens but on the same structural trend. Only delineation drilling using LTK48 sized core has been drilled in the New Cobar deposit since 2005 between the 16 and 46 mining levels.

The oxide portions of the New Cobar deposit have a nominal drill hole spacing of 20 metres by 20 metres to approximately 100 metres below surface and 25 metres by 25 metres (northing and elevation) to a vertical depth of approximately 120 metres. Drilling of the deeper portions of the deposit since 2003 has subsequently resulted in a nominal drill hole spacing of 15 metres by 20 metres between a vertical depth of 105 and 340 metres, and a spacing of 50 metres by 40 metres to a vertical depth of 430 metres. Between 430 and 680 metres vertical depth, there is a nominal drill hole spacing of 70 metres by 40 metres.

Table 11-1 summarises the various drilling programs undertaken at New Cobar.

Table 11-1: Drilling Programs at New Cobar

Drilling Program		Year	Number of Drill Holes	Hole Diameter (mm)/ Core Size	Hole Size at Lode Intersection (mm)
1	First exploration PD	1987	37	125, 115	115
2	First exploration RC	1988	70	150, 140	140
3	Second exploration RC	1988	18	125, 115	115
4	Third exploration RC	1989	70	150,140,125,115	140, 125, 115
5	Fourth exploration RC	1995	62	150, 140, 125	140, 125
6	Fifth exploration RC	1996	77	150, 140	140
7	Sixth Exploration RC	2000	13	140	140
8	First exploration DD	1996	7	60 (BX roller bit), BQ	BQ
9	Second exploration DD	1991	7	PQ, HQ, NQ	All NQ
10	Third exploration DD	1987	8	140, PQ, HQ, NQ	HQ, NQ
11	Fourth exploration DD	1989	27	178, 150, 140, PQ, HQ, NQ	NQ
12	Fifth exploration DD	1996	7	HQ	HQ
13	Sixth exploration DD	1996	89	PQ, HQ, NQ	NQ
14	Seventh exploration DD	1997	17	PQ, HQ, NQ	NQ
15	First metallurgical DD	1988	1	PQ	PQ
16	Second metallurgical DD	1997	12	PQ, HQ	PQ, HQ
17	First geotechnical DD	1997	7	HQ	HQ
18	UG Resource - Surface	2003	32	HQ/NQ	NQ
19	UG Drilling	2005/6	97	LTK48/BQ	LTK48/BQ
20	UG Drilling	2007	36	LTK48	LTK48
21	UG Drilling	2008	73	LTK48	LTK48

11.2.1 Surveying Practices

PGM near mine surface drill holes and underground drill holes are surveyed by site surveyors. Exploration drill holes are surveyed by external contractors using the Digital Global Positioning System (DGPS).

Prior to 1991, the drill holes were not down hole surveyed during drilling, but surveys were completed later on a nominal 30 metre down-hole interval using a down-hole camera recording magnetic azimuth and dip. From 1991 to 2003, drill holes were down-hole surveyed during drilling at nominal 30 metre down-hole intervals.

Since 2003 all drill holes have been surveyed at nominal 30 metre intervals using either a Reflex multi-shot camera or an Eastman camera. It is current practice to regularly check the down hole cameras against surveyed ‘totem stations’ on surface to ensure the accuracy of the instruments.

11.2.2 Sample Recovery

Sample recovery data is not available for the pre-1995 RC data. Later diamond programs have good mean recoveries in the mineralised portions of the deposit with most intervals exhibiting 100% recovery.

11.3 Chesney Oxide and Underground Mine

The Chesney deposit was discovered circa 1872, with its last period of operation from 1943 to 1952. From 1971 to 1975, a new shaft was sunk, but was not connected to the old workings. During the last period of historical production, diamond drill holes were used for grade control.

The oxide mineralization at Chesney has been drill tested to approximately 100 metres vertically at a spacing of 20 metres by 40 metres or 20 metres by 20 metres using 100 face-sampling RC drill holes and 47 percussion drill holes (PD) were completed. In addition, several HQ/NQ diameter diamond drill holes tested the deposit at deeper levels. This drilling was completed in five different drill programs between 1987 and 1996. Table 11-2 summarises the three drilling programs undertaken for the oxide portion of the Chesney deposit.

Table 11-2: Drilling Programs for Chesney Oxide

Drilling Program		Year	Number of Drill Holes	Hole Diameter (mm)/ Core Size	Hole Size at Lode Intersection (mm)
1	First exploration PD	1987	47	125	115
2	Exploration RC	1988, 89, 96	100	150, 140	140
5	First exploration DD	1992 - 93	15	PQ, HQ, NQ	HQ, NQ

Since 1992, the sulphide portion of the Chesney deposit has been tested by four drilling programs, down to approximately 800 metres below surface. Details of the programs are summarised in Table 11-3.

The shallow mineralisation down to approximately 120 metres below surface is drilled to nominal 20 metre by 20 metre spacing. Drill hole spacing is wider around the old workings and on the extremities of the deposit to the north and south.

Back and face sample data from the historic levels were compiled from original linen plans. These samples are spaced approximately 30 metres vertically apart on sub-level intervals. A number of underground diamond holes drilled from 1943 to 1950 intersect the original orebody within the old workings.

Drilling below the base of historic workings has been completed to a nominal 40 metre by 40 metre spacing by holes over several programs down to 520m below surface (the 9,750mRL). The majority of holes which are used in the resource estimation for this area are from programs completed in 1997, 2004 and 2005 as well as . Below the 9,750mRL, drill hole spacing increases significantly.

In 2008 the Chesney crossover drive from New Cobar was completed. The first of the diamond drill cuddies was cut below Stockpile 5 and drilling commenced from it in June 2008. This programme covered the northern strike length of the ore body with a 20 x 20 drill hole pattern which extended from just below the 9900 level to the base of the old workings. A second cuddy was cut as a stockpile below the 9900 access takeoff on the first downward turn of the decline to drill out the southern strike length. Both programs were completed in 2008 for a total of 8756m.

Table 11-3: Drilling Programs for Chesney Sulphide

Period	Details	Number of Drill Holes	Hole Diameter (mm)/ Core Size
1943	**Stage 1 UD drilling program: UD43 series holes**	13	EX
1948-1949	**Stage 2 UD drilling program: UD49 series holes** Underground drilling from the north drive on 8 level. Some UD48* series of holes lack assay data.	20	AX, BX
1992-1993	**Stage 1 DD drilling program: DD92/93 series holes** DD drilling program testing eastern gold lode and main lode position beneath historic workings to approximately 550 metres below surface.	9	NQ, HQ, PQ
1997	**Stage 2 DD drilling program: DD97 series holes** DD drilling program testing the eastern gold lode and main lode position beneath historic workings at approximately 450 metres below surface.	33	NQ, HQ
2001	**Stage 3 RC and DD drilling program: RC01 and DD01/02 series holes.** Infill drilling programme to raise the resource status of the Chesney Oxide to Measured.	8	140, NQ
2004	**Stage 4 DD drilling program: DD04 series holes** Tested the higher grade gold lodes below the historic workings to 9700mRL, tested the northern shoot of the Main Lode to 9400mRL	32	NQ, NQ2, HQ
2005	**Stage 5 DD drilling program: DD05 series holes** Infilled the area beneath the historic workings to 9700mRL	14	NQ,HQ
2007	**Stage 5B DD drilling program: DD07 series holes** Re-entry of DD05CH0046 with three daughter holes	3	HQ, NQ
2008	**Underground Delineation Program**	73	LTK48

11.3.1 Surveying Practices

PGM near mine surface drill holes and underground drill holes are surveyed by site surveyors. Exploration drill holes are surveyed by external contractors using DGPS. It is not recorded how the holes drilled in 1943 and 1949 were down-hole surveyed.

Up to 1989, the drill holes were not down-hole surveyed during drilling, but surveys were completed later on a nominal 30 metre interval using a down-hole camera. Due to collapse of some drill holes, it was not possible to survey all drill holes. From 1992 onwards, the drill holes were down-hole surveyed during drilling at nominal 30 metre intervals.

The recent delineation holes are surveyed with a multishot camera firstly at 15m to check the setup and then at the end of the hole at 30m intervals.

11.3.2 Sample Recovery

Sample recovery data is not available for the pre-1992 RC data. Diamond drilling programs since 1992 in the mineralised portions of the deposit have good mean recoveries with most samples exhibiting 100% recovery.

11.4 New Occidental Mine

The New Occidental deposit was first worked as the United Mine and is believed to have been discovered in 1871. The last period of historic mining was from 1932 to 1935. It was during this latter period that records of diamond drilling commenced. The programs of exploration and evaluation (delineation) drilling that are known to have been undertaken at the New Occidental deposit include:

- Underground grade control drilling by New Occidental Gold Mines (NOGM), 1940-1952.
- Underground exploration drilling by NOGM, 1950-1952. These holes have been excluded from the database due to collar/sample location issues.
- Two campaigns of surface exploration diamond drilling by Cobar Mines Pty Limited, 1975 and 1986. The diamond drill holes completed in 1975 occur well to the north of the deposit and do not contain any significant mineralization. The latter program involved limited reconnaissance deep diamond drilling targeting the down-dip extensions of the deposit.
- Surface RC, percussion and limited diamond drilling undertaken by CRA Exploration Pty Ltd (CRAE) as part of the evaluation of the remnant oxide gold potential, 1987-1998.
- Evaluation surface diamond drilling by PGM, 1995-1996.
- Exploration surface diamond drilling by PGM to the south of the deposit at the New Occidental South prospect, 1998-1999.
- Underground delineation drilling by PGM in 2001-2008.

The drilling programs are summarised in Table 11-4.

Table 11-4: New Occidental Deposit Drilling Programs

Company	Year	Number of Drill Holes	Type of Drilling	Diameter	Sample Type	Down-hole Surveys
New Occidental Gold Mines	1945-1952	104	Diamond	AX/BX	Full Core	-
New Occidental Gold Mines	1948-1952	13	Diamond	AX	Full Core	-
Cobar Mines Pty Ltd	1980	5	Diamond	BQ	½ Core	Single shot camera
Cobar Mines Pty Ltd	1986	3	Diamond	PQ/HQ with NQ tails	½ Core	Single shot camera
CRA Exploration Pty Ltd	1987	11	Percussion	115mm	Chip	
CRA Exploration Pty Ltd	1987	1	Diamond	HQ with NQ tails	½ Core	Single shot camera
CRA Exploration Pty Ltd	1988	4	Reverse Circulation	140mm	Chip	
PGM	1995-1996	55	Diamond	PQ/HQ with NQ tails	½ Core	Single shot camera
PGM	2001-2002	257	Diamond	LTK48	Full Core	Single shot camera
PGM	2003-2007	284	Diamond	LTK48, HQ, NQ	Full Core, ½ Core	Single shot camera
PGM	2008	52 6	Diamond	LTK48 NQ	Full Core ½ Core	Multishot

Data distribution varies throughout the deposit. From the 92 Level (914m below surface) up to the base of historical workings the drill-hole data has a nominal spacing of 15 metres by 20 metres (northing and elevation). Below the 92 Level, the data has a nominal spacing of 30 metres by 50 metres for the first 150 metres of depth then increases to an approximately 50 metres by 75 metres for the next 100 metres.

11.4.1 Surveying Practices

PGM near mine or underground drill holes are surveyed by site surveyors. Exploration drill holes are surveyed by external contractors using DGPS.

For the CRA and Cobar Mines drilling programs, down-hole surveys were taken every 6 to 30 metres. Historic reverse circulation drill holes were not down hole surveyed.

All PGM drill holes were surveyed using an Eastman single shot camera or by a digital multi-shot camera. Surveys were typically collected at 30 metre intervals. The frequency was increased to 15 metre intervals in controlled drilling (navi-drilling) or wedging sections. Additional surveys were also taken at 6m intervals in the rods immediately behind the navi-bit and barrel during a navi-cycle to confirm dip measurements.

11.4.2 Sample Recovery

Sample recovery for the PGM series of holes is high in the mineralised portions of the deposit with most samples recording recoveries of 100%.

11.5 Peak Mine

The Peak gold deposit was discovered in 1887 and was mined intermittently through the late 1800's from underground. From 1992 through to 2002, underground mining of the Peak orebody continued from 300 metres below surface (300 Level) down to 760 metres below surface (740 Level). Although previously considered to be exhausted, there are significant resources that remain both within and proximal to the deposit. Exploratory surface diamond drilling during 2005, up-dip from the current workings, identified a gold/base metal resource immediately below the base of oxidation.

Four drilling programs were completed in the area between 1997 and 2000 to define an oxide resource. Prior to this, several sporadic and minor programs were completed in the area from 1948 to 1995. These programs are summarised in Table 11-5.

Table 11-5: Drilling Programs in the Area of the Peak Oxide Resource

Program	Year	Hole Type	Operator	Number of Drill Holes	Total Metres Drilled	Hole Size
1	1987	RC	CRA Exploration	21^	2,335	
2	1988	PD	CRA Exploration	7^	669	
3	1948 - 1995	DD	CRA Exploration	12	3,160	
4	1996	RC	PGM	9	712	133
5	2000	RC	PGM	59	3,677	133
^ Not used in the resource estimations						

Since 2000, approximately 319 drill holes have been drilled to define the underground resources at Peak. Underground resource definition holes are typically drilled either as NQ2 or LTK48 diameter core. Drilling from surface is typically collared using HQ diameter core followed by an NQ diameter tail. Similarly, some underground drilling was collared using HQ diameter core followed by an NQ2 diameter tail. Navigational drilling for the HQ diameter holes was used in 2001. The underground drilling utilises the same drill rigs as used at the New Occidental and Perseverance mines.

The drilling programs used to define the underground resource are summarised in Table 11-6.

Table 11-6: Peak Drilling Programs Since 2000

Company	Year	Number of Drill holes	Type of Drilling	Hole Diameter (mm)/Size	Sampling	Down-hole Surveys
PGM	2000	135	Diamond	BQ, NQ@, LTK48	Full Core, 1/2 Core	Single Shot Camera
PGM	2001	80	Diamond	LTK48, NQ2, HQ	Full Core, 1/2 Core	Single Shot, Multi Shot camera
PGM	2002	17	Percussion	LTK48, NQ, NQ2, HQ	Full Core, 1/2 Core	Single Shot, Multi Shot camera
PGM	2003	6	Diamond	HQ, NQ, NQ2	Full Core, 1/2 Core	Single Shot, Multi Shot camera
PGM	2004	10	Diamond	BQ, HQ, NQ, NQ2	Full Core, 1/2 Core	Single Shot, Multi Shot camera
PGM	2005	21	Diamond	HQ, NQ, LTK48	Full Core, 1/2 Core	Single Shot, Multi Shot camera
PGM	2006	29	Diamond	HQ, NQ, NQ2, LTK48	Full Core, 1/2 Core	Single shot camera
PGM	2007	21	Diamond	LTK48	Full Core	Single shot camera

11.5.1 Surveying Practices

Accurate collar surveys are available for the 1987 and 1988 series of RC and PD drill holes, but no down hole surveying was performed (these holes are not used in the resource estimations). Since 1996, all RC and diamond holes have been down hole surveyed at nominal 30 metre intervals using either Eastman single shot cameras or electronic multi-shot cameras (Reflex). All recent drilling (post 2000) has used standard PGM down-hole survey and collar survey procedures.

11.5.2 Sample Recovery

Sample recovery for the PGM series of diamond drill holes in the mineralised portions of the deposit is high with most samples recording recoveries of 100%. RC drilling during 2000 for the Peak oxide material averaged 72% recovery, with 67% of the samples having a recorded recovery of 70% or greater.

11.6 Perseverance Mine

The Perseverance deposit was discovered in 1994 by PGM. All drilling within the project has been completed by PGM using its standard procedures. Drilling for the underground resource has been completed using HQ, NQ, NQ2 and LTK48 diameter drilling equipment. HQ diameter holes are used to establish a parent hole to facilitate later directional drilling and wedging using either NQ or NQ2 diameter core. LTK48 diameter core is used for infill drilling to delineate mineralisation from underground drives and cuddies. In 1998, 5 PQ diameter holes were used as a parent hole to subsequent HQ and NQ diameter tails. Table 11-7 summarises the drilling programs at Perseverance since 1997.

Table 11-7: Perseverance Drilling Programs Since 1997

Company	Year	Number of Drill Holes	Type of Drilling	Hole Diameter (mm)/Size	Sampling	Down-hole Surveys
PGM	1997	1	Diamond	NQ2, LTK48	Full Core, 1/2 Core	Single shot camera
PGM	1998	27	Diamond	BQ, CHD76, PQ, HQ, NQ, NQ2	1/2 Core	Single shot camera
PGM	1999	32	RC And Diamond	HQ, NQ2, 140	1/2 Core	Single shot camera
PGM	2000	50	Diamond	HQ, NQ2	1/2 Core	Single shot camera
PGM	2001	20	Diamond	HQ, NQ2, LTK48	Full Core, 1/2 Core	Single shot camera
PGM	2002	27	Diamond	BQ, LTK48, HQ, NQ, NQ2	Full Core, 1/2 Core	Single shot, Multi shot camera
PGM	2003	173	Diamond	BQ, HQ, LTK48, NQ, NQ2	Full Core, 1/2 Core	Single shot, Multi shot camera
PGM	2004	56	Diamond	NQ2, LTK48	Full Core, 1/2 Core	Single shot camera
PGM	2005	124	Diamond	LTK48	Full Core	Single shot, Multi shot camera
PGM	2006	144	Diamond	NQ, NQ2, LTK48	Full Core, 1/2 Core	Single shot, Multi shot camera
PGM	2007	91	Diamond	NQ, NQ2, LTK48	Full Core, 1/2 Core	Single shot, Multi shot camera
PGM	2008	62 14 9	Diamond	LTK48 NQ Delineation NQ Evaluation	Full Core ½ Core ½ Core	Single shot, Multi shot camera

The spacing of drill-hole intersections with mineralisation varies throughout the deposit. Zone A, which contains most of the stope delineation LTK48 diameter drilling, has a nominal drill hole spacing of 20 metres by 20 metres (northing and elevation), with some areas near drill cuddies having a drill hole spacing of 10 metres by 10 metres. Zone B has a nominal drill hole spacing of 20 metres by 20 metres. The drill hole spacing in the Percy Deeps, Hulk and Hercules zones varies from 20 metres by 10-20 metres to 40 metres by 20 metres. In the peripheral areas of the deposit, drill hole spacing increases to 60 metres by 80 metres.

Delineation drilling of Zone D commenced in 2008 with a nominal spacing of 15 x 12.5 over the high grade zones and 15 x 25 over periphery zones.

11.6.1 Surveying Practices

All of the diamond and RC holes at Perseverance were collar and down hole surveyed using the standard PGM site procedures as described in previous sections. Down hole surveys are taken at nominal 30 metre spaced intervals. In certain cases such as wedging operations, areas of bad ground conditions, or the start of a drill hole, the frequency of down-hole survey readings has been decreased to 9 metre intervals to enable adequate spatial location of the holes. Electronic multi-shot devices (Reflex and Ranger models) were used for some programs since 2002.

A comparison of gyroscopic surveys to magnetic down-hole surveys has indicated that the presence of magnetic pyrrhotite near the mineralised zones has not significantly affected the magnetic survey readings.

11.6.2 Sample Recovery

As for the other deposits, the recorded sample recovery for the diamond core in the mineralised portions of the deposit core is excellent, with most samples recording recoveries of 100%.

12 SAMPLING METHOD AND APPROACH

Records of the sampling methods and approach for the drilling campaigns prior to the 1980's are not readily available. Drill logs are available for most of the historic underground holes from the New Occidental, New Cobar and Chesney mines. Most of the older holes were sampled on 5 foot intervals (approximately 1.52 metres). It is thought that the historic (1930's and 1940's) underground drill holes were full core sampled as no split core remains and this was standard practice for delineation drilling at local mines during this period. Samples were generally assayed at the onsite laboratory.

Certain older diamond drilling programs are recorded to have used a "groove" sampling technique whereby a small continuous sample was cut from the side of the core using an angle grinder for subsequent assay. The holes on which this technique was used are known, and the corresponding assay data has not been used for resource estimation.

RC chip samples collected by CRA were reduced to 2 to 3 kilogram samples using Jones riffle splitters or rotary splitters fed directly from the cyclone on the drill rig. The RC samples were generally collected as 2 metre composite samples. Some of the exploration percussion drilling samples were collected as 6 metre composite samples.

Based on the due diligence and competency of the author, there are no drilling, sampling or recovery factors that could material impact the accuracy and reliability of the results. There are no known factors that would result in a sample bias at any of the Cobar Goldfield deposits discussed in this report.

12.1 Reverse Circulation Samples

The current RC sampling practice is discussed briefly in Section 11.1.2. PGM's current standard practice is to use face-sampling hammers to minimize sample contamination from drill hole walls, and riffle splitters to reduce samples in the field to a size small enough (2 – 2.5 kilograms) suitable for pulverisation using an LM5 mill.

Sampling strategies are devised for individual projects depending on requirements and project status. The initial sampled intervals utilise 1, 2 or 4 metre composite samples depending on the resolution required. One metre samples are retained to allow more detailed analyses where required.

A typical example of an RC sampling strategy from the Peak Oxide deposit is supplied below:

- Drill cuttings are split directly upon exiting the cyclone into 50% and two 25% portions compositing 2 metre intervals, beginning from surface.
- One of the 25% portion composite samples is collected and stored for later use (metallurgical or other).
- The other 25% portion composite sample is weighed (then recalculated to 100% as a recovery check) and then further reduced using a 25/75 Jones riffle splitter.
- Two nominal 2.5 kilogram samples are obtained from the splitter:
- A sample to be submitted for assay.
- A field duplicate sample to be stored and assayed at a later stage as required in accordance with sampling quality assurance and quality control (QAQC) protocols.

The sampling methods and approach are reasonable for this style of mineralization. The samples are considered representative and there appear to be no sample biases introduced during sampling.

12.2 Core Sampling

During logging of the core, the geologist selects intervals to be sampled and supplies the site geotechnician with details via a Core Sampling Request Form. This form details which intervals are to be sampled, which samples are to be density tested (usually all samples), the sample method (1/2 core or whole core), and which standards and blanks are to be used and at what frequency. The sampling details are then entered into the Drillview database which outputs a sample ledger form. The sample ledger contains the sequential list of which intervals are to be sampled, the corresponding sample numbers, and the placement of specified blank and standard samples. These details are incorporated as part of the main drill-hole database.

Selective sampling of core is based on the presence of significant quartz veining, alteration mineralogy (usually silicification) and/or sulphide minerals. PGM currently samples core using 1 metre intervals. Unmineralised material from the hangingwall or footwall is not necessarily sampled.

HQ, NQ or NQ2 diameter core is cut in half for sampling, with the sampling line based upon the regional S2 cleavage. Because of the similar volumes of LTK48 core and NQ half core, the LTK48 core is sampled whole. Unsampled mine production core from delineation drilling is discarded.

The samples are placed into numbered calico bags (Figure 12-1) which are then bundled into poly-weave bags and secured using cable ties. The poly-weave bags are labelled using hole number and sample intervals.

The samples are held within a secure shed until collection by a representative of the SGS Cobar assaying laboratory (previously Analabs Cobar).

The sampling methods and approach are reasonable for this style of mineralization. The samples are considered representative and there appear to be no sample biases introduced during sampling.

Figure 12-1: Samples Awaiting Dispatch at the Peak Coreyard

12.3 Density Determination

It is current site practice to determine bulk density values on all core that is sampled. This is done using the water immersion method, with results entered directly into a computerised system which includes calibration values for the weight of the measuring apparatus. The accuracy of the density readings is tested and calibrated against pre-determined standard samples at a frequency of one in 30 samples and at the start of new sampling campaigns. Figure 12-2 shows the setup of the bulk density measuring stations, along with a poster outlining the correct procedure to be used for bulk density testing.

Figure 12-2: Peak Mine Bulk Density Measuring Station

There is a comprehensive chapter in the site Coreyard Procedures Manual which details the correct procedure for conducting bulk density measurements.

13 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Numerous drilling programs have been conducted on the PGM projects over the last 60 years. This section will concentrate on the sample preparation, analysis and security protocols currently used by PGM. PGM personnel do not conduct any aspect of the sample preparation.

13.1 Sample Preparation for Core

No records were available for the sample preparation protocols of the historic underground drilling. The sample preparation procedures since 1995 are described below.

The current sample preparation procedure is as follows:

- All delineation drill core is submitted to SGS Cobar (previously Analabs Cobar) for preparation and analysis.
- Core (approximately 2.5kg) is dried for 12 hours and crushed to –25mm using a Jaques jaw crusher.
- Total preparation of all of the crushed material by LM5 puck and bowl pulveriser (Figure 13-1) to a nominal 90% passing 75μm (usually 6 to 8 minutes). The average mass per metre allows the full 1m sample to be pulverised in the LM5 in a single run.
- Sizing tests are performed on one in every 5 samples. If a sample fails the test the entire sample is reground, plus all samples since the last sample tested. Sizing data is reported to PGM along with analytical results.
- Barren wash (blue metal or feldspar) is used between all samples in the LM5.
- The first split is by taking a scoop of material out of the LM5 bowl. A 250g sub-sample is taken from the LM5 bowl and put into a kraft (brown paper) packet. A second 350g sample may be collected and placed in a coloured (usually black) kraft bag if duplicate samples are to be analysed. Bulk residues are placed in the original calico bag and stored for at least 3 months before being discarded.
- A 50g charge is scooped from the 250g kraft bag for gold determination by fire assay. A second 50g charge may be collected for repeat analysis.
- A 0.5g charge is taken from the primary sample packet for base metal analysis.
- All remaining pulp in the kraft packet is returned to PGM for storage. Until November 1998 all pulps were sent to the Rio Tinto Exploration pulp storage facility in Bundoora, Victoria. From December 1998 onwards all pulps are stored at PGM's core shed.

Figure 13-1: Sample Pulverisation at SGS Cobar

13.2 Sample Preparation for RC Samples

Sample preparation protocols for RC samples are similar to those for drill core except that drying times may be longer and, typically, only roll crushing (or no crushing) is required before pulverizing.

13.3 Screen Fire Assaying

Screen fire assays (SFA) are not routinely performed by PGM, but were used on core samples in the Perseverance evaluation as a secondary assay method. SFA sample preparation is the same as for regular core until completion of pulverization.

- A 500g assay charge is scooped from the calico bag storing bulk residue;
- Wet screen 500g over a 75µm screen;
- Fire assay all screen overflow (approximately 10% of sample or 50g);
- Dry 450g (approximately 90%) underflow and sub-sample 50g for fire assay. Discard remainder;
- Weigh average coarse and fine gold grades and report weights and grades of both size fractions and total;

13.4 Sample Analysis

No records were available for the assaying protocols for the historic underground drilling at New Cobar, Chesney and New Occidental. It is believed that the samples were analysed at the mine's onsite laboratory by fire assay for gold and wet chemical methods for base metals.

Most of the early exploration sample analyses for New Cobar Mines, CRAE and PGM used the Analabs Cobar (formerly Australian Assay Laboratories Cobar, now SGS Cobar) laboratory for assaying. ALS

Chemex (2001- 2006) was occasionally used for check assaying. CRA RC samples were assayed for gold by fire assay and for silver, copper, lead, zinc, lithium and sodium by AAS at Analabs Adelaide. CRA drill core samples were usually analysed for gold by fire assay and for base metals and silver by AAS methods. Some surface diamond core drilled during the 1970's and 1980's was assayed for gold using acid digestion with an AAS finish, but many of the mineralised intersections were later re-assayed by fire assay.

Other laboratories including Aminya (1989 – 1996), Amdel (1987 – 1999) and CMPL (1973 – 1975) have been used for early metallurgical test work or for assaying of some exploration samples.

PGM currently uses external laboratories for all core and RC chip analyses. All drill core is analysed for gold, copper, lead, zinc, silver and bismuth. Standard contract analytical work is currently carried out by SGS Cobar. Prior to June 2001, ALS Chemex in Orange was used for analysis of some Perseverance drilling. Since 1997, over 80% of all sample submissions and over 88% of all sample analyses have been completed at either Analabs (Cobar or Orange) or SGS Cobar.

13.4.1 Gold Analysis

Prior to 2004 the Analabs Cobar laboratory was mainly used for gold assaying. Special requirements, such as screen fire assay, were performed at the Analabs Orange laboratory. All samples include the S091 method, which is a silica wash in the pulveriser between samples. Some samples have been analysed at the ALS Chemex laboratories in Brisbane and Orange. These include some Perseverance drilling samples (2000, 2001) and some check assays (2005).

On December 31st 2001, Analabs Cobar became SGS Cobar. In 2004 the Analabs Cobar Laboratory became referred to as SGS Cobar and the analysis methods were changed. A barren silica wash (method WSH78) is still included between each sample.

The gold analysis methods used by PGM are shown below in Table 13-1.

Table 13-1: Gold Analysis Method Used by PGM

Element	Laboratory Code	Digestion	Finish	Units	Detection Limit	Laboratory
Pre 2004						
Au	F650	50g FA, Pb collection	AAS	ppm	0.01	Analabs Cobar, Orange
Au	F642(SFA)	FA, Pb collection	AAS	ppm	0.01	Analabs Cobar, Orange
Post 2004						
Au	FAA505	50g FA, Pb collection	AAS	ppm	0.01	SGS Cobar
Miscellaneous						
Au	Au-AA2	50g FA, solvent extraction	AAS	ppm	0.001	ALS Orange
Au	PM209	50g FA, Pb collection	AAS	ppm	0.01	ALS Orange
Au	Au-AA26	50g FA, solvent extraction	AAS	ppm	0.001	ALS Orange
Au	PM212 (SFA)	50g FA, Pb collection n	AAS	ppm	0.01	ALS Orange
FA = Fire Assay AAS = Atomic Absorption Spectroscopy ppm = Parts per million, equivalent to grams per tonne (g/t) SFA = Screen Fire Assay						

13.4.2 Base Metal Analysis

Prior to mid-2004, Analabs performed the base metal analyses at their laboratory in Townsville using the Inductively Coupled Plasma – Optical Emission Spectroscopy method (ICP-OES). Over-range (ore grade) base metal assays (typically above 1%) were repeated using a triple acid digestion with an AAS finish.

ALS laboratories in Orange were also used as discussed in the previous section. Subsequent to mid-2004 the bulk of all base metal analyses were performed at the SGS Cobar laboratory using a triple acid digestion on a 0.5g pulp sample with an AAS finish.

Table 13-2 and Table 13-3 summarise the analytical techniques used by the various laboratories.

Table 13-2: Base Metal Analysis Methods Used for PGM

Element	Laboratory Code	Digestion	Finish	Units	Detection Limit	Laboratory
Prior to Mid-2004						
Cu	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
Cu (over range)	I112/(A103)	Triple acid	AAS	%	0.01	Analabs Townsville
Pb	C112 / I112	Four Acids	ICP-OES	ppm	3	Analabs Townsville
Pb (over range)	I112/(A103)	Triple acid	AAS	%	0.01	Analabs Townsville
Zn	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
Zn (over range)	I112/(A103)	Triple acid	AAS	%	0.01	Analabs Townsville
Ag	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
Sb	C112 / I112	Four Acids	ICP-OES	ppm	5	Analabs Townsville
Bi	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
As	C112 / I112	Four Acids	ICP-OES	ppm	5	Analabs Townsville
Co	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
Mo	C112 / I112	Four Acids	ICP-OES	ppm	1	Analabs Townsville
Fe	C112 / I112	Four Acids	ICP-OES	%	0.01	Analabs Townsville
Mn	C112 / I112	Four Acids	ICP-OES	ppm	5	Analabs Townsville
Post Mid-2004						
Cu	AAS21R	Triple acid	AAS	%	0.0002	SGS Cobar
Cu (ore grade)	AAS22S	Triple acid	AAS	%	0.01	SGS Cobar
Pb	AAS21R	Triple acid	AAS	%	0.0003	SGS Cobar
Pb (over range)	AAS22S	Triple acid	AAS	%	0.01	Analabs Townsville
Zn	AAS21R	Triple acid	AAS	%	0.0002	SGS Cobar
Zn (over range)	AAS22S	Triple acid	AAS	%	0.01	Analabs Townsville
Ag	AAS21R	Triple acid	AAS	ppm	1 ppm	SGS Cobar
Bi	AAS21R	Triple acid	AAS	ppm	10ppm	SGS Cobar
AAS = Atomic Absorption Spectroscopy						
ppm = Parts per million, equivalent to grams per tonne (g/t)						
Four acid digest: includes perchloric, nitric, hydrochloric and hydrofluoric acids, added sequentially						
Triple acid digest includes nitric, hydrochloric and perchloric acids added sequentially						
ICP-OES = Inductively Coupled Plasma - Optical Emission Spectroscopy						

Table 13-3: Summary of ALS Chemex Analytical Techniques

Element	Laboratory Code	Digestion	Finish	Units	Detection Limit	Laboratory
Prior to 2005						
Cu	IC587	Four Acids	ICP-OES	ppm	1	ALS Orange
Pb	IC587	Four Acids	ICP-OES	ppm	3	ALS Orange
Zn	IC587	Four Acids	ICP-OES	ppm	1	ALS Orange
Ag	IC587	Four Acids	ICP-OES	ppm	1	ALS Orange
Sb	IC587	Four Acids	ICP-OES	ppm	5	ALS Orange
Bi	IC587	Four Acids	ICP-OES	ppm	10	ALS Orange
As	IC587	Four Acids	ICP-OES	ppm	5	ALS Orange
Co	IC587	Four Acids	ICP-OES	ppm	1	ALS Orange
Mo	IC587	Four Acids	ICP-OES	ppm	1	ALS Orange
Fe	IC587	Four Acids	ICP-OES	%	0.01	ALS Orange
Mn	IC587	Four Acids	ICP-OES	ppm	5	ALS Orange
Post 2005						
Cu	ME-ICP61	Aqua Regia	ICPAES	ppm	1	ALS Orange
Pb	ME-ICP61	Aqua Regia	ICPAES	ppm	2	ALS Orange
Zn	ME-ICP61	Aqua Regia	ICPAES	ppm	2	ALS Orange
Ag	ME-ICP61	Aqua Regia	ICPAES	ppm	0.5	ALS Orange
Sb	ME-ICP61	Aqua Regia	ICPAES	ppm	5	ALS Orange
Bi	ME-ICP61	Aqua Regia	ICPAES	ppm	3	ALS Orange
As	ME-ICP61	Aqua Regia	ICPAES	ppm	5	ALS Orange
Co	ME-ICP61	Aqua Regia	ICPAES	ppm	1	ALS Orange
Mo	ME-ICP61	Aqua Regia	ICPAES	ppm	1	ALS Orange
Fe	ME-ICP61	Aqua Regia	ICPAES	%	0.01	ALS Orange
Mn	ME-ICP61	Aqua Regia	ICPAES	ppm	5	ALS Orange
Cu	ME-ICP41	Multi Acid	ICPAES	ppm	1	ALS Orange

Table 13-3: Summary of ALS Chemex Analytical Techniques

Element	Laboratory Code	Digestion	Finish	Units	Detection Limit	Laboratory
Pb	ME-ICP41	Multi Acid	ICPAES	ppm	2	ALS Orange
Zn	ME-ICP41	Multi Acid	ICPAES	ppm	2	ALS Orange
Ag	ME-ICP41	Multi Acid	ICPAES	ppm	0.2	ALS Orange
Sb	ME-ICP41	Multi Acid	ICPAES	ppm	5	ALS Orange
Bi	ME-ICP41	Multi Acid	ICPAES	ppm	2	ALS Orange
As	ME-ICP41	Multi Acid	ICPAES	ppm	2	ALS Orange
Mo	ME-ICP41	Multi Acid	ICPAES	ppm	1	ALS Orange
Fe	ME-ICP41	Multi Acid	ICPAES	ppm	0.01	ALS Orange
Mn	ME-ICP41	Multi Acid	ICPAES	%	5	ALS Orange
ICP-AES = Inductively Coupled Plasma - Atomic Emission Spectroscopy						

13.5 Sample Security

Core samples that are to be collected at site are stored within a secure shed in a fenced and locked yard behind the main gate at the Peak Mine offices. All visitors who enter the Peak Mine site must be tagged through an electronic gate, which is manned by security personnel 24 hours a day. It is current practice for the samples to be collected daily by an employee of the SGS Cobar laboratory.

Sample pulps and field splits of RC samples are also stored in a secure facility either at the mine or at the CRAE/RTE office in Bundoora, near Melbourne, Australia. Prior to November, 1998 pulps were sent to the pulp storage facility in Bundoora. Since December, 1998 all pulps have been stored at the Peak Mine core shed.

Each sample dispatch is given a unique data processing order (DPO) number, which is used to track the sampling details (laboratory, assaying method, and sample preparation code) in the site Drillview database. Original DPO sheets are on file at CRAE/RTE in Bundoora, or at the Peak Mine offices.

Post-1975 assay reports issued by analytical laboratories detail the sample preparation and analytical codes used to perform the analyses. Original laboratory reports are filed by DPO at CRAE Bundoora and PGM. Sample preparation codes for all DPO's are detailed in the DPO and analytical code tables in Drillview.

13.6 Quality Control Procedures

There is little information available regarding historic quality assurance and quality control (QAQC) procedures. During the 1970's and 1980's QAQC was reportedly usually limited to duplicate assaying of one in every fifteen samples. This data is available in the site database.

Drilling programs during the 1990's included the use of a barren quartz wash between each sample during the pulverisation stage. Duplicate samples were taken at a nominal frequency of one in ten samples, and certified assay standards were routinely used. Umpire check-assaying of 10% of the mineralised samples was also undertaken. This data is available in the site database. Checks on the pulverisation process were not routinely undertaken prior to 1996.

The current site QAQC practices include the following checks:

- Routine insertion of certified standards and site-sourced blanks into the sample stream. The sample insertion point is based upon instructions by the supervising geologist and sample numbers are automatically generated by the Drillview database system. Two standards are routinely inserted at a frequency of one in 20 samples, resulting in an overall 1:10 ratio. Blank samples are routinely inserted at the start of a sampling run and after identified mineralised intervals.
- Routine replicate samples. Sample replicates are defined as a second 50g sample taken from the original 250g pulp packet. These are taken at a nominal rate of 1 in every 10 samples.
- Routine duplicate samples. Duplicate samples are defined as gold analysis of a 50g charge taken from a second 250g pulp packet. This is performed at the nominal rate off 2 in every 42 samples at

the laboratory. The insertion of laboratory duplicate is controlled by the computerised laboratory information management system (LIMS).

- Insertion of standards and blanks at pre-defined interval by the laboratory during the assaying process. Blank standards are shaken in the same 'cocktail' mixer that is used to mix the pulp and flux prior to firing.
- Routine calibration of the density recording equipment on site, using two pre-determined density standards
- Umpire samples. Re-assaying of selected residual pulp by a separate laboratory. This is done on an occasional basis, with the latest lot of umpire assays analysed in December 2005.
- Check assaying using the Screen Fire Assay (SFA) technique. Selected samples have been analysed using the SFA technique.
- Field duplicates consisting of analysis of either quarter core or second nominal 2.5kg RC split.

During the assay loading procedure, the Drillview database system automatically generates reports based on the loaded QAQC data and flags standards results which are outside the accepted limits for subsequent action. Representatives from the Geology Department meet at least once a month with the SGS Cobar site manager to discuss the previous month's QAQC performance and any issues that have been identified.

Blank samples that return a grade above ten times the detection limit are requested to be re-assayed to confirm the contamination. If contamination is confirmed, then the three samples on either side of the blank sample are requested to be re-assayed, with the new sample taken from the bulk pulp residue.

The routine standard/blank, replicate and duplicate assaying QAQC procedures are followed for gold and base metals assaying. Currently the site database does not capture the copper replicate or duplicate data as part of the automated data capture system, although it is understood that this will commence shortly.

The Qualified Persons for this report can verify that the above procedures have been and are being followed. The verification has been established by personal supervision of the procedures over a period of seven years, revision at regular (up to six monthly) intervals of the drilling database, which encompasses validation checks of the QAQC results, geological logging and other supporting data documentation.

14 DATA VERIFICATION

14.1 Quality Control Data

14.1.1 Assay Accuracy

PGM measures the accuracy of the assaying process by the use of a series of certified standards. Material for the PGPL series of standards was derived from the 500 Level of the Copper Lens at the Peak mine and prepared by Gannet Holdings Pty Ltd (Gannet). The certified standards have been prepared for a range of gold and base metal grades. Table 14-1 summarises the most common standards used on site since 2000.

An additional 3 standards were prepared by ORE Pty Ltd in 2008 from stillages generated from core samples at Perseverance, Chesney, Peak and New Occidental. The standards are certified for Gold and Copper and were prepared with projected gold values of 2.5, 5.0 and 10g/t. These three new standards were sourced to replace some of the existing standards which were diminishing in supply. No sample dispatches containing the new standards were received prior to the end of 2008.

The Au and Cu standards results for the top 6 standards (by volume) analysed from 2000 to present are presented in Appendix A. These cover a range of gold values from 1.51g/t Au to 29.74g/t Au, and a range of Cu values from 0.27% Cu to 1.52% Cu. The performance statistics for these standards are summarised in Table 14-1 and Table 14-2.

Analabs (2000- 2004)

The Au standards assayed by Analabs generally exhibit good accuracy with the bulk of the standards having more than 92% of the assays within +/-10% of the certified expected value (EV). The exception is standard PGPL2700, where only 86% of the returned values were within the +/-10% limits. Review of the control plots indicates that some of the more extreme spikes may be due to mislabelling of inserted standards prior to submission. Most of the returned standards results exhibit a slight negative bias in Au grades, ranging from -0.9% to -3.8%. Only standards ST42_7192 (with an EV of 1.51g/t Au) and ST06_8208 (with an EV of 1.01g/t Au) have neutral or slightly positive biases for Au grades (0.04% and 1.14% respectively).

The Cu standards sent to Analabs generally exhibit moderate accuracy with the standards having only 67% to 83% of the returned assays within +/-10% of the certified expected value. The standard control plots indicate potential calibration issues with the AAS machine around December 2002. As with the gold standards, some of the more extreme outliers illustrated in the control plots may be due to mislabelling of inserted standards prior to submission. Four of the six standards batches reviewed showed a positive bias in the range of 2.2% to 5.8%.

Table 14-1: Main Standards Used by PGM since Jan 2000 to Dec 2007

Standard	Number of Samples	Description	Supplier	Matrix Type	Certified Value			
					Au Mean (g/t)	Au SD	Cu Mean (ppm)	Cu SD
PGPL3600	848	Au-base metal standard (Peak ore)	Gannet	Sulphide	3.73	0.24	4651	210
PGPL2700	835	Au-base metal standard (Peak ore)	Gannet	Sulphide	2.8	0.19	3509	175
PGPL5000	788	Au-base metal standard (Peak ore)	Gannet	Sulphide	5.21	0.37	6337	385
PGPL2100	639	Au-base metal standard (Peak ore)	Gannet	Sulphide	2.15	0.15	2731	133
PGPL10000	551	Au-base metal standard (Peak ore)	Gannet	Sulphide	10.32	0.41	13169	440
ST42_7192	388	Au standard	Gannet	Sulphide	1.51	0.09		
ST147	139	Au standard	Gannet	Sulphide	2.66	0.155		
ST43_7194	104	Au standard	Gannet	Sulphide	3.64	0.24		
PGPL30000	96	Au-base metal standard (Peak ore + WA Mine Ore)	Gannet	Sulphide	29.74	1.32	15153	550
ST49_8242	95	Au standard (Paired STANDARD)	Gannet	Sulphide	2.06	0.1		
ST06_8208	84	Au standard	Gannet	Oxide	1.01	0.06		
ST148	83	Au standard	Gannet	Sulphide	8.67	0.59		
ST153	83	Au standard	Gannet	Sulphide	13.8	0.93		
PGPLORIG	78	Au-base metal standard (Peak ore)	Gannet	Sulphide	20.15	1	25756	810
ST242	60	Gold standard	Gannet	Sulphide	2.06	0.1		
BM23	47	Base metal standard	Gannet	Sulphide			36000	1600
ST07_9258	43	Au standard	Gannet	Oxide	0.22	0.02		
ST42_1292	39	Au standard	Gannet	Sulphide	1.48	0.06		
GLG304-4	38	Low level gold standard from Geostats	Geostats	Fresh	0.121	0.013		
ST05_9280	29	Au standard	Gannet	Oxide	2.52	0.12		
ST43	25	Gold standard	Gannet	Sulphide	3.58	0.24		
GLG304-2	23	Low level gold standard from Geostats	Geostats	Oxide	0.065	0.01		

Table 14-2: Summary Statistics for Gold and Copper Standards Results since 2000

ToLab	Data	Sampel Fraction						
		PGPL10000	PGPL2100	PGPL2700	PGPL30000	PGPL3600	PGPL5000	PGPLORIG
ALS	Expected Au ppm	10.32	2.15	2.80	29.74	3.73	n/a	20.15
	Mean Au ppm	9.81	2.15	2.74	29.83	3.76	n/a	18.93
	Std Dev Au Cert	0.41	0.15	0.19	1.32	0.24	n/a	1.00
	Count of AuPpm	58.00	63.00	60.00	10.00	70.00	n/a	4.00
	Average	9.81	2.15	2.74	29.83	3.76	n/a	18.93
	Min Acceptable Au ppm	9.50	1.85	2.42	27.10	3.25	n/a	18.15
	Max Acceptable Au ppm	11.14	2.45	3.18	32.38	4.21	n/a	22.15
	Min of Au Value ppm	10.32	2.15	2.80	29.74	3.73	n/a	20.15
	Max of Au Value ppm	13.50	2.88	3.65	31.10	11.10	n/a	19.60
	Expected Cu ppm	13169.00	2731.00	3509.00	15153.00	4651.00	n/a	25756.00
	Mean Cu ppm	13313.38	2716.17	3545.50	15590.00	4797.57	n/a	26427.50
	Count Cu ppm	57.00	63.00	60.00	10.00	70.00	n/a	4.00
	Cu Std Dev Cert	440.00	133.00	175.00	550.00	210.00	n/a	810.00
	Count of Min Acceptable Cu	58.00	63.00	60.00	10.00	70.00	n/a	4.00
	Count of Max Acceptable Cu	58.00	63.00	60.00	10.00	70.00	n/a	4.00
	Minimum Cu ppm	2.50	49.00	2790.00	15200.00	3910.00	n/a	25500.00
	Maximum Cu ppm	14300.00	3070.00	3950.00	16100.00	5200.00	n/a	27500.00
Analabs	Expected Au ppm	10.32	2.15	2.80	29.74	3.73	5.21	20.15
	Mean Au ppm	10.16	2.12	2.72	29.00	3.65	5.17	20.23
	Std Dev Au Cert	0.41	0.15	0.19	1.32	0.24	0.37	1.00
	Count of AuPpm	278.00	440.00	236.00	52.00	528.00	320.00	12.00
	Average	10.16	2.12	2.72	29.00	3.65	5.17	20.23
	Min Acceptable Au ppm	9.50	1.85	2.42	27.10	3.25	4.47	18.15
	Max Acceptable Au ppm	11.14	2.45	3.18	32.38	4.21	5.95	22.15
	Min of Au Value ppm	10.32	2.15	2.80	29.74	3.73	5.21	20.15
	Max of Au Value ppm	11.50	3.45	3.50	31.30	5.30	6.25	21.20
	Expected Cu ppm	13169.00	2731.00	3509.00	15153.00	4651.00	6337.00	25756.00
	Mean Cu ppm	12968.49	2847.01	3588.24	15090.38	4927.41	6701.63	25300.00
	Count Cu ppm	278.00	440.00	236.00	52.00	528.00	320.00	12.00
	Cu Std Dev Cert	440.00	133.00	175.00	550.00	210.00	385.00	810.00
	Count of Min Acceptable Cu	278.00	440.00	236.00	52.00	528.00	320.00	12.00
	Count of Max Acceptable Cu	278.00	440.00	236.00	52.00	528.00	320.00	12.00
	Minimum Cu ppm	3270.00	340.00	25.00	12800.00	148.00	1110.00	23500.00
	Maximum Cu ppm	18000.00	6150.00	8230.00	24000.00	51500.00	8890.00	26500.00
SGS	Expected Au ppm	10.32	2.15	2.80	29.74	3.73	5.21	20.15
	Mean Au ppm	10.20	2.12	2.68	29.51	3.60	4.90	19.59
	Std Dev Au Cert	0.41	0.15	0.19	1.32	0.24	0.37	1.00
	Count of AuPpm	215.00	135.00	539.00	34.00	249.00	468.00	62.00
	Average	10.20	2.12	2.68	29.51	3.60	4.90	19.59
	Min Acceptable Au ppm	9.50	1.85	2.42	27.10	3.25	4.47	18.15
	Max Acceptable Au ppm	11.14	2.45	3.18	32.38	4.21	5.95	22.15
	Min of Au Value ppm	10.32	2.15	2.80	29.74	3.73	5.21	20.15
	Max of Au Value ppm	11.80	2.72	3.64	34.00	4.16	5.83	23.50
	Expected Cu ppm	13169.00	2731.00	3509.00	15153.00	4651.00	6337.00	25756.00
	Mean Cu ppm	13271.24	2845.87	3656.01	15506.97	4702.02	6763.29	25280.33
	Count Cu ppm	215.00	135.00	539.00	34.00	248.00	460.00	61.00
	Cu Std Dev Cert	440.00	133.00	175.00	550.00	210.00	385.00	810.00
	Count of Min Acceptable Cu	215.00	135.00	539.00	34.00	249.00	468.00	62.00
	Count of Max Acceptable Cu	215.00	135.00	539.00	34.00	249.00	468.00	62.00
	Minimum Cu ppm	49.50	1910.00	131.00	12200.00	1835.00	1090.00	15200.00
	Maximum Cu ppm	21000.00	3360.00	34000.00	22000.00	6080.00	8790.00	33500.00

ALS Laboratories (2000 – 2001)

The Au standards sent to ALS Laboratories generally exhibit moderate accuracy with seven of the eight standards having between 80% and 91% of the assays within +/-10% of the certified expected value. The exception is standard PGPL30000 (n=10, EV=29.74g/t Au), where 100% of the returned values were within +/-10% of the certified standard value. A review of the control plots (e.g. PGPL 3600) indicates that some of the more extreme spikes may be due to mixing of standards prior to submission. The standards exhibit a slight negative bias ranging from -7.35% to -1.56% for five of the eight reviewed standards.

The Cu standards sent to ALS Laboratories exhibit good accuracy with all of the standards having greater than 90% of the returned assays within +/-10% of the certified expected values. Some of the more extreme spikes illustrated in the control plots may be due to the mixing of standards prior to submission. A slight positive bias of between 1% and 3.2% is indicated for four of the five reviewed standards.

SGS (2004 – 2008)

The Au standards sent to SGS Cobar generally exhibit good accuracy with all of the standards having over 90% of the assays within +/-10% of the certified expected values. A review of the control plots indicates the rare occurrence of mixing of standards prior to submission. The standards exhibit a slight negative for all of the reviewed standards.

The Cu standards sent to SGS Laboratories exhibit moderate to poor accuracy with all of the standards having between 50% and 80% of the returned assays within +/-10% of the certified expected values. These standards are however certified for Au and not for Cu. A slight positive bias is indicated for all of the reviewed standards.

14.1.2 Assay Precision

PGM measures the precision of the sample preparation and assaying process by the use of repeat and duplicate assays. Field duplicate samples (from quarter core samples) are not usually collected for diamond core, although field duplicate samples are sometimes collected for the RC samples.

Micon (2003) discussed the assay precision data for the projects, covering the period from 1997 to 2001. In general Micon's comments on assay precision were:

- For Perseverance, Analabs gold pulp duplicate sample data has low mean percent difference (MPD) values that compare well with the Analabs repeat data.
- For Chesney, duplicate sample data indicate that acceptable precision has been achieved.
- For New Occidental, moderate levels of precision were achieved for repeat and duplicate assay data.
- For Peak, moderate to fair levels of precision were achieved with more recent assaying (after 1999) displaying better precision that previous campaigns.
- For Perseverance, the gold assay data was considered reasonable.

The Au repeats and duplicate statistics since 2000 for the various labs for Au grades above 0.1 g/t.

Table 14-3: PGM Gold Replicate and Duplicate Sample Data since 2000 to 2008

Lab	Project	Total	STD	Blanks	Rep's	Dup's
Analabs	Chesney	1388	61	15	221	59
	Great Cobar-Gladstone	342	15	8	32	14
	New Cobar	2616	120	49	417	112
	New Occidental	16272	767	350	4034	688
	Peak	10365	391	114	1559	417
	Perseverance	16141	1034	359	3337	987
	Queen Bee	183	12	10	12	8
	Regional	319	13	16	13	13
	Stones Tank	193	9	6	19	9
	Total	47819	2422	927	9644	2307
ALS	Chesney	74	4	6	2	0
	New Occidental	763	26	4	0	0
	Peak	7479	262	123	750	331
	Perseverance	2199	104	20	196	101
	Queen Bee	130	25	13	0	0
	Regional	491	24	7	18	3
	Total	11136	445	173	966	435
SGS	Chesney	5793	333	135	440	284
	Great Cobar-Gladstone	999	20	10	74	42
	Mt Boppy Mine	125	6	3	22	7
	New Cobar	14433	589	235	2402	742
	New Occidental	6649	294	167	1236	346
	Peak	6300	207	65	727	276
	Perseverance	24781	1051	361	2897	1188
	Queen Bee	68	6	10	10	3
	Regional	318	14	6	28	17
	Total	59466	2520	992	7836	2905

Summary statistics for the replicate and duplicate data are shown in Table 14-4.

Table 14-4: Summary Statistics for PGM Gold Replicates and Duplicates Results since 2000 to 2006

Laboratory	Number of Samples	Original Au mean	Repeat Au mean	Pearson C.C.	Mean HRD	Median HRD	Mean HARD	Median HARD	% within 10% HARD
Replicates									
ALS	421	13.27	13.64	1	0.99	0.15	0	6.38	87
Analabs	8,034	9.44	9.56	1	1	-0.38	-0.32	4.07	92
SGS	3,555	11.02	10.72	0.95	1	-0.19	0	3.58	95
Duplicates									
ALS	163	4.98	5.09	1	0.99	-0.9	0	5.71	88
Analabs	875	24	23.42	1	1	-0.31	0	4.12	91
SGS	606	14.44	13.35	0.97	1	-0.71	-0.45	4.03	93

The plot of assay replicates in Figure 14-1 covers the drilling campaigns from the project areas since 2000 to 2008. The plots and summary statistics indicate that there is a good level of precision for gold analyses during this period with Analabs and SGS reporting 92% and 95% of the data above a 10% HARD limit. The correlation plot in

Assay duplicate results plotted against original assays on a per project per lab basis are presented below.

Figure 14-6 indicates that there is an acceptable level of correlation between the two datasets with high Pearson correlation coefficients.

The Rank HARD plot of Au assay duplicates (Figure 14-5) indicates slightly lower levels of precision than for the replicates (as expected), with Analabs and SGS reporting 91% and 93% of the data above a 10% HARD limit. The associated correlation plot in

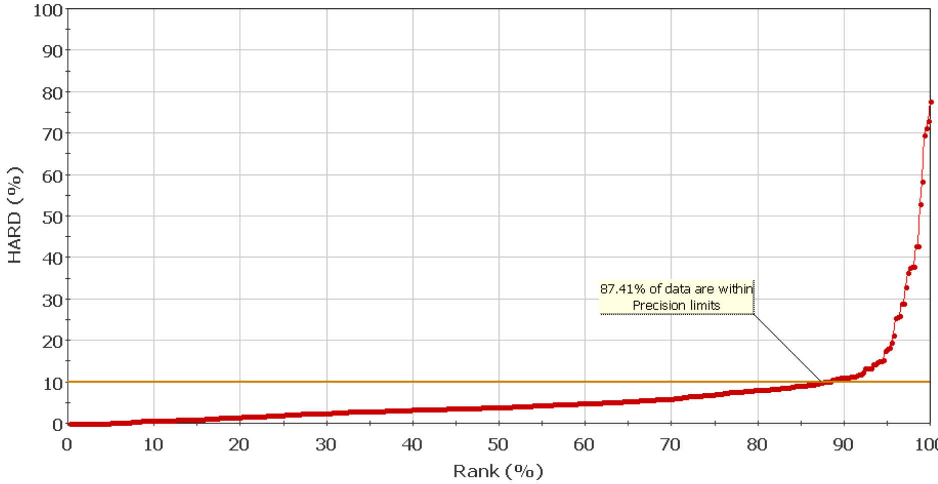
Assay duplicate results plotted against original assays on a per project per lab basis are presented below.

Figure 14-6 shows duplicate and original data displays acceptable correlation levels between the two datasets with high Pearson correlation coefficients. These plots were not recreated to include the 2007 as it is considered that the addition of the 2007 data in separate graphs was sufficient to identify any deficiencies.

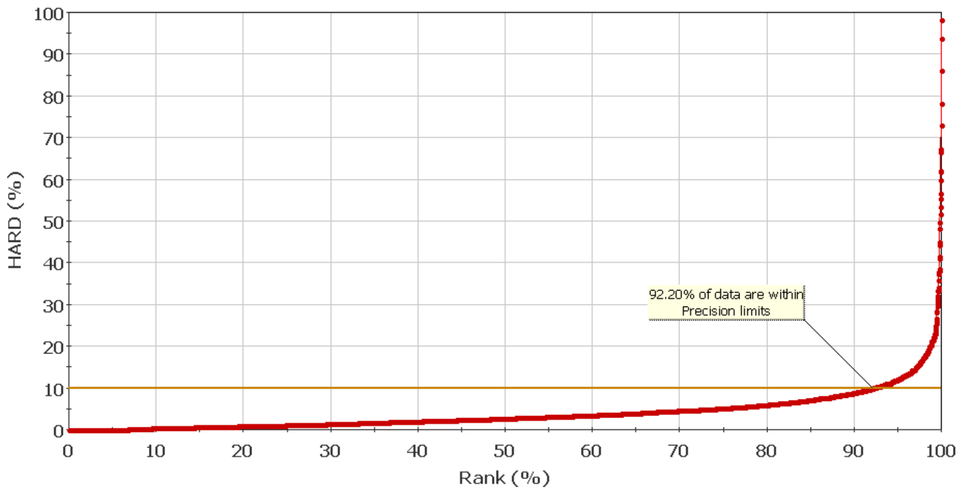
The site database importing routine was modified during early 2007 to capture the replicate and duplicate data for the copper. Historical repeat and duplicate data for copper was loaded into the site database in April 2007.

Figure 14-1: Rank HARD Plot of Gold Assay Replicates since 2000 to 2006

ALS



Analabs



SGS

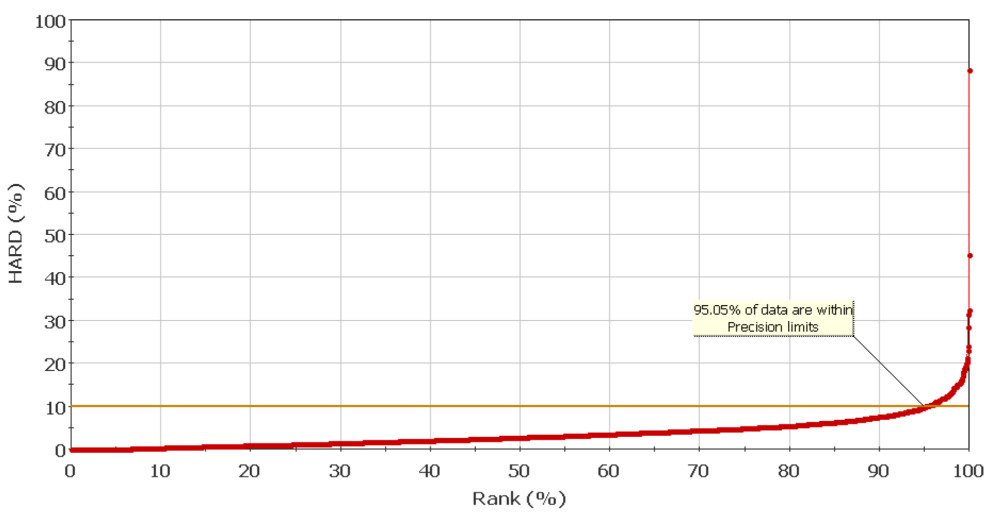
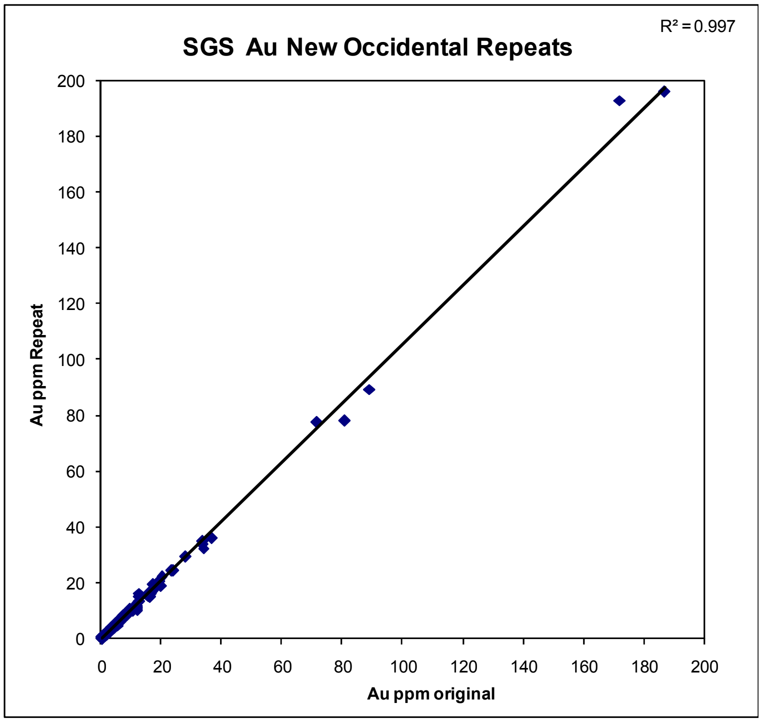
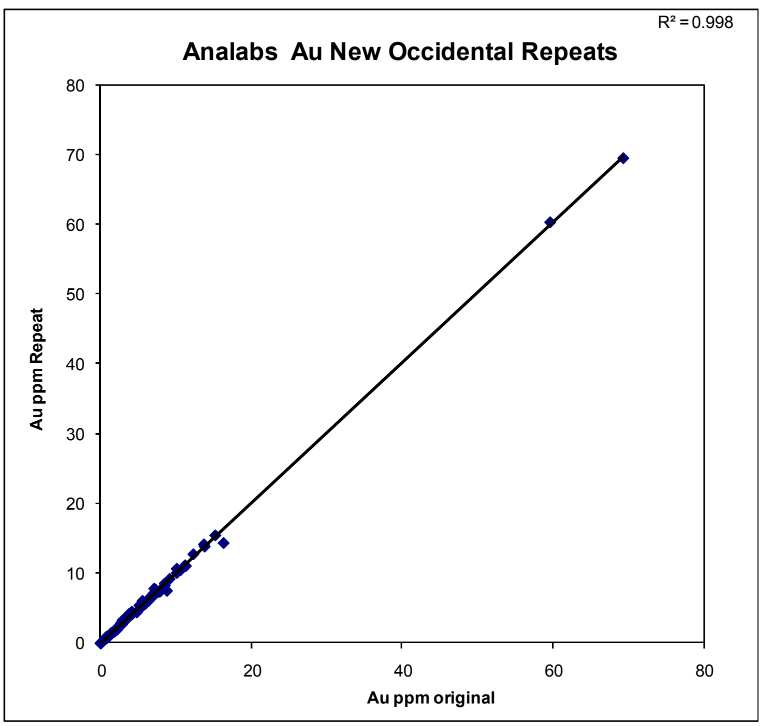
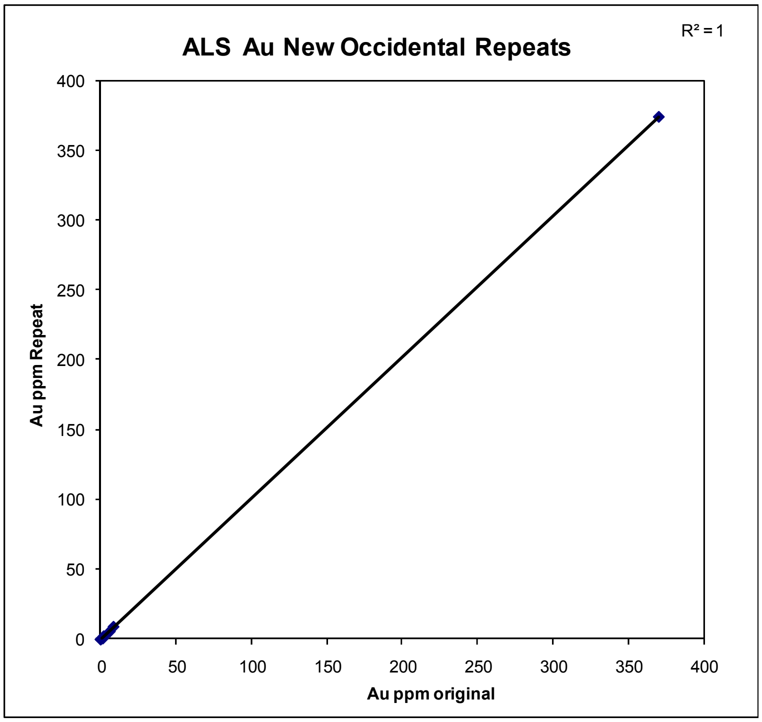


Figure 14-2: Correlation plots of Gold Assay Replicates since 2000 to 2008 by project and lab

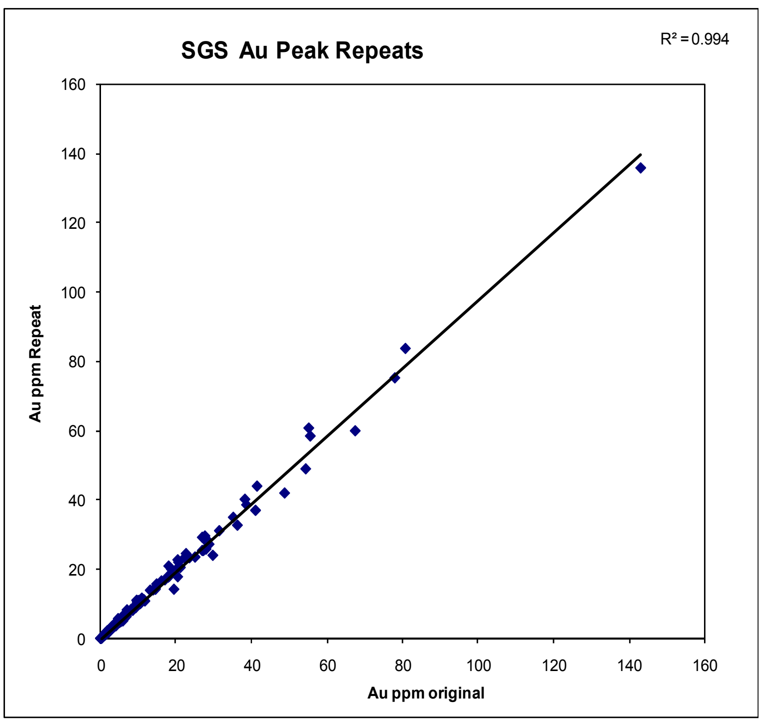
New Occidental

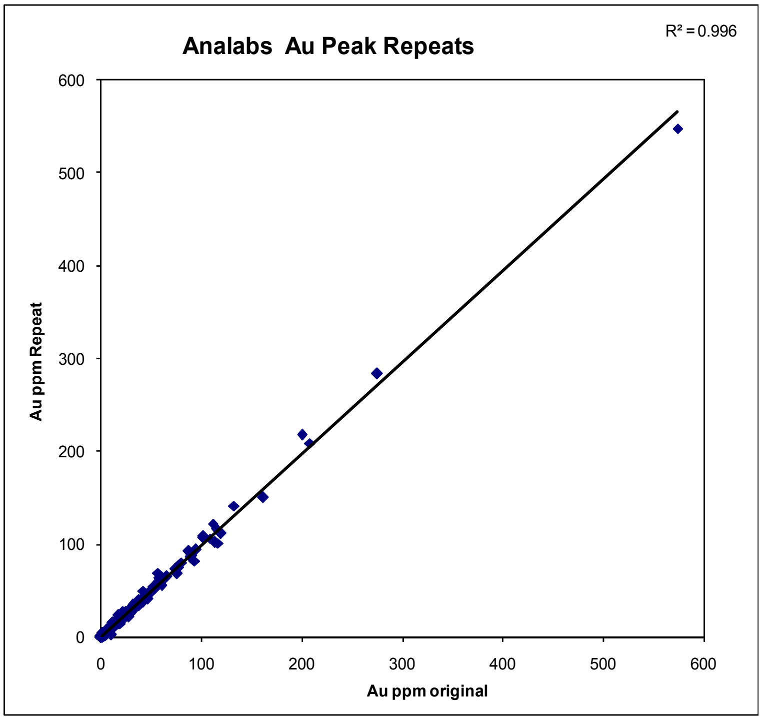


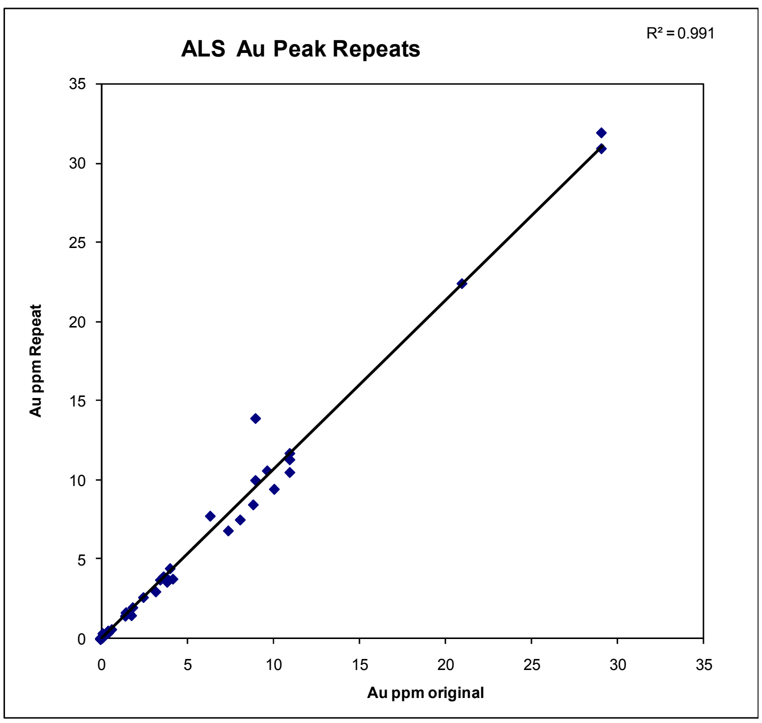



ALS Au New Occidental Repeats
R² = 1
Au ppm Repeat
Au ppm original
0
50
100
150
200
250
300
350
400

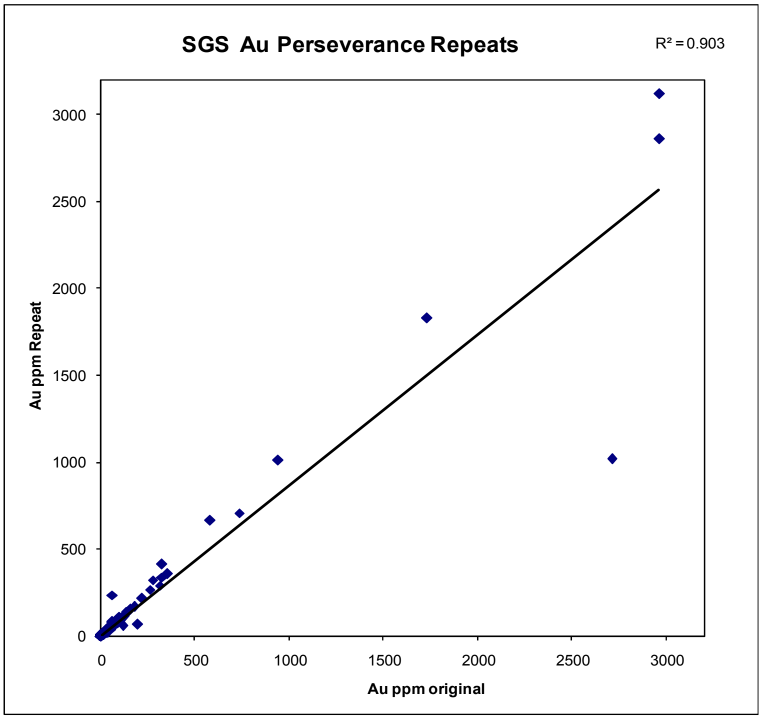
Peak

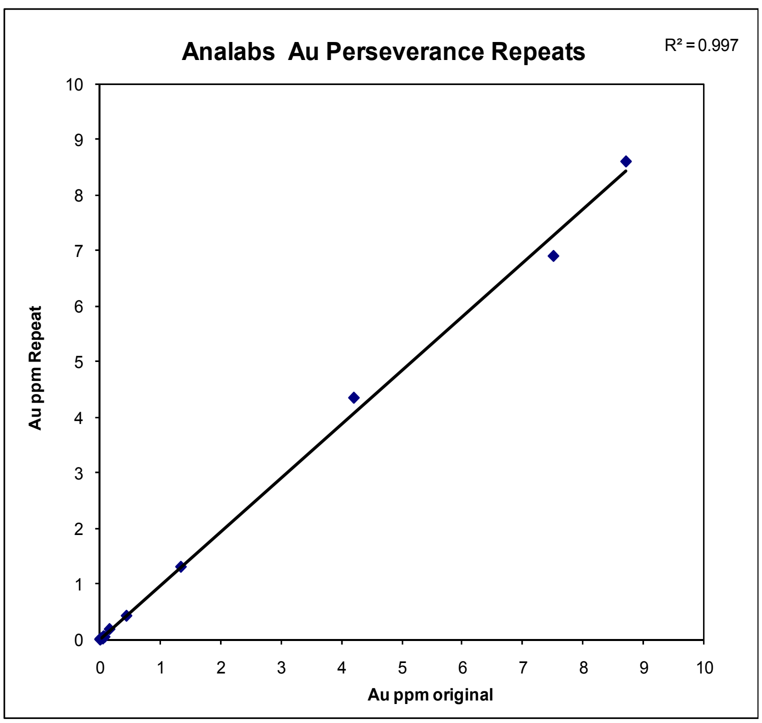

SGS Au Peak Repeats
R² = 0.994
Au ppm Repeat
Au ppm original
0
20
40
60
80
100
120
140
160

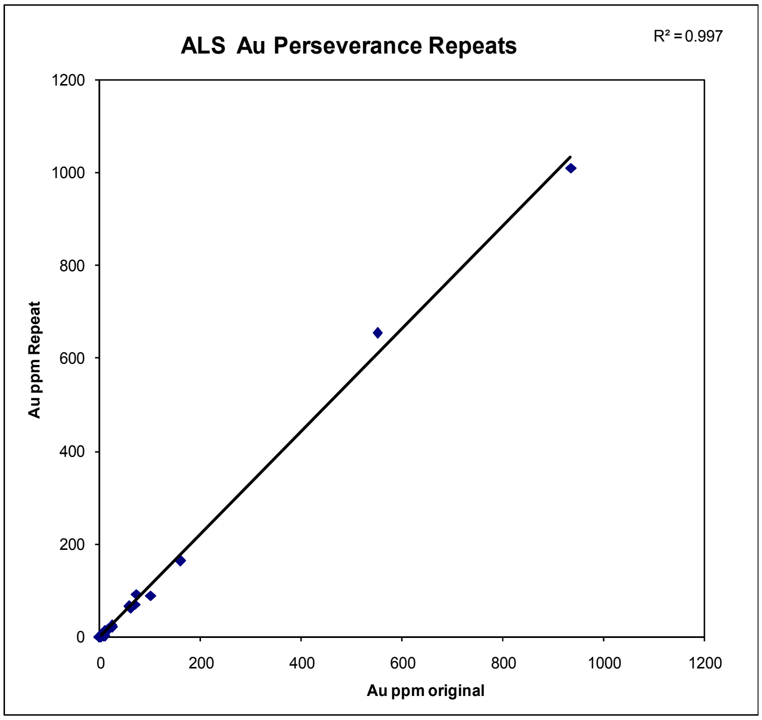

Analabs Au Peak Repeats
R² = 0.996
Au ppm Repeat
Au ppm original
0
100
200
300
400
500
600


ALS Au Peak Repeats
R² = 0.991
Au ppm Repeat
Au ppm original
0
5
10
15
20
25
30
35

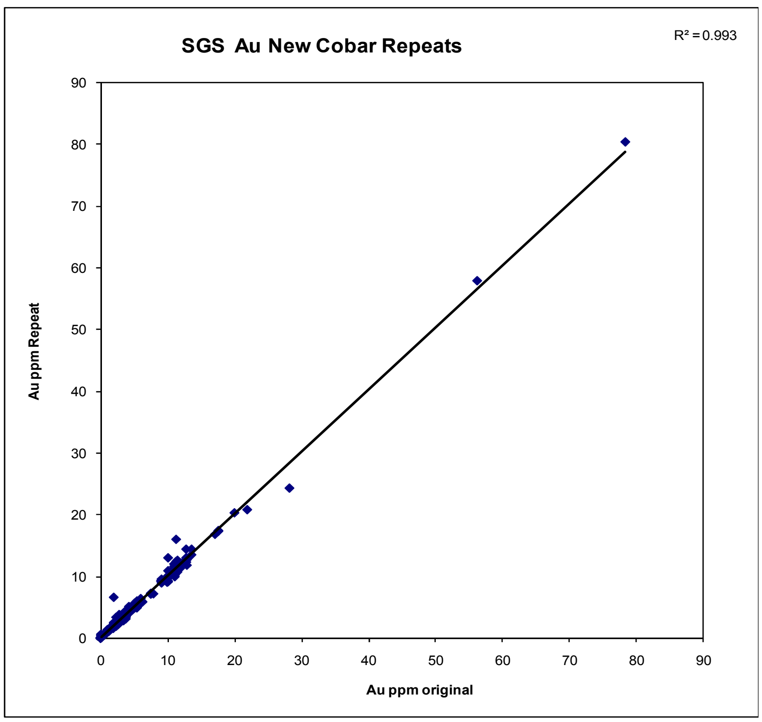
Perseverance

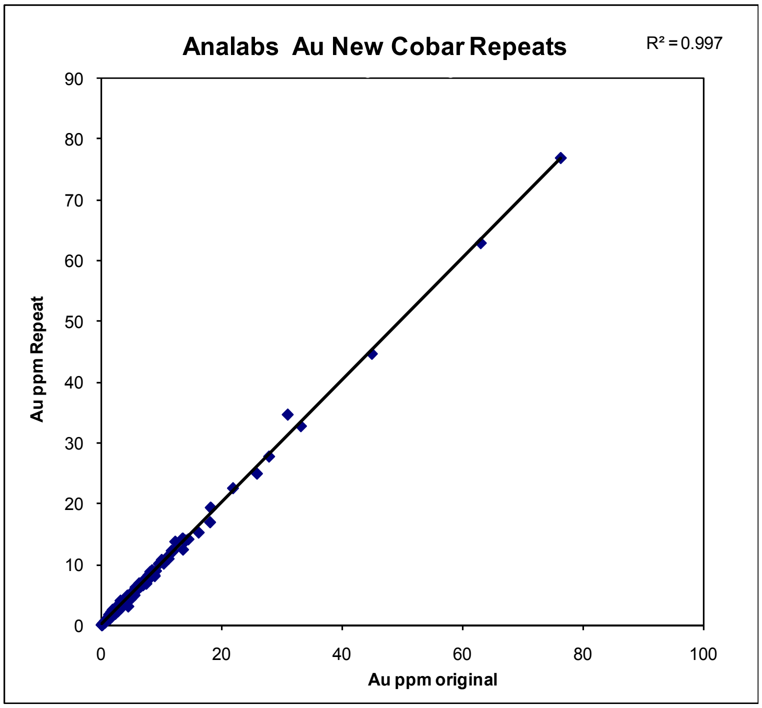




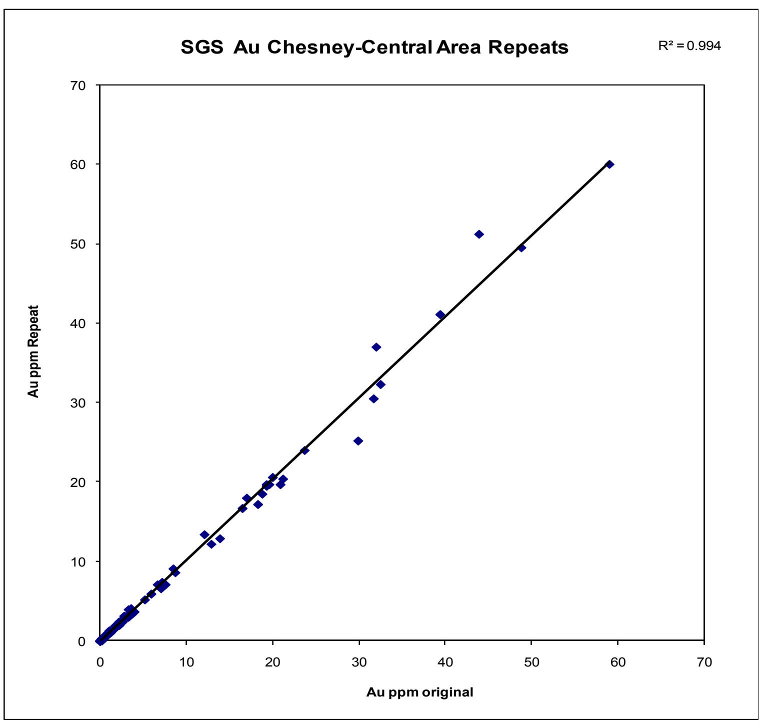

ALS Au Perseverance Repeats
R² = 0.997
1200
1000
800
600
400
200
0
Au ppm Repeat
0
200
400
600
800
1000
1200
Au ppm original

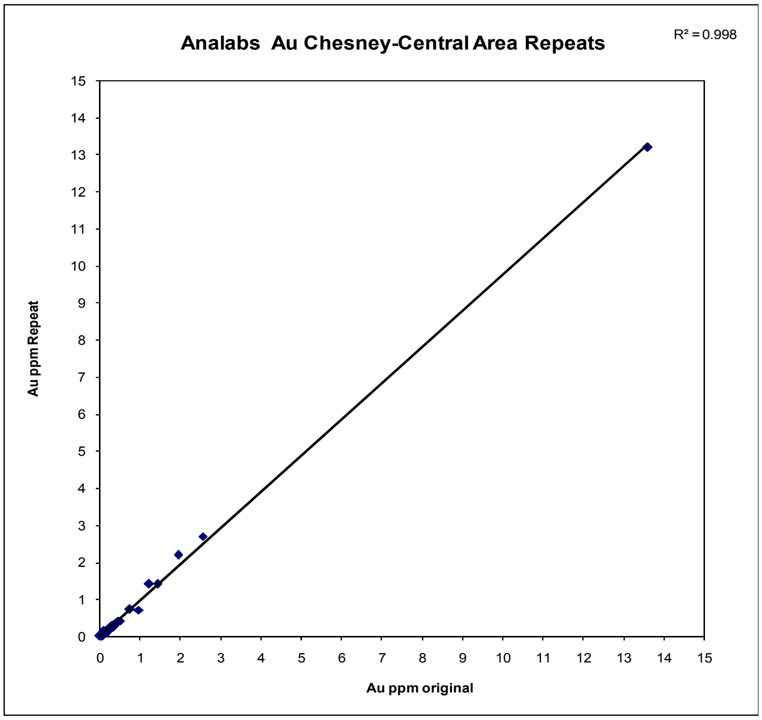
New Cobar


SGS Au New Cobar Repeats
R² = 0.993
90
80
70
60
50
40
30
20
10
0
Au ppm Repeat
0
10
20
30
40
50
60
70
80
90
Au ppm original



Chesney Central Area

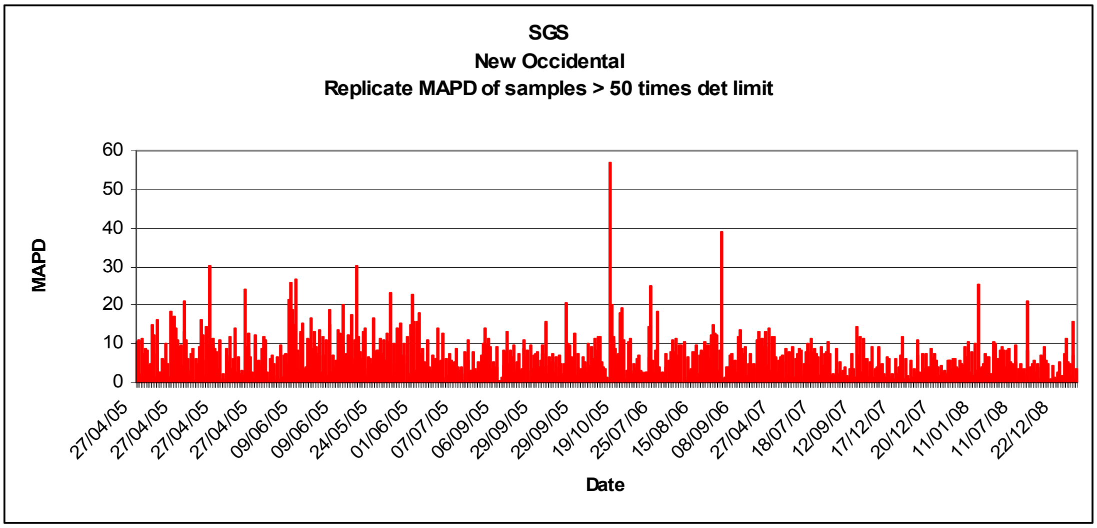


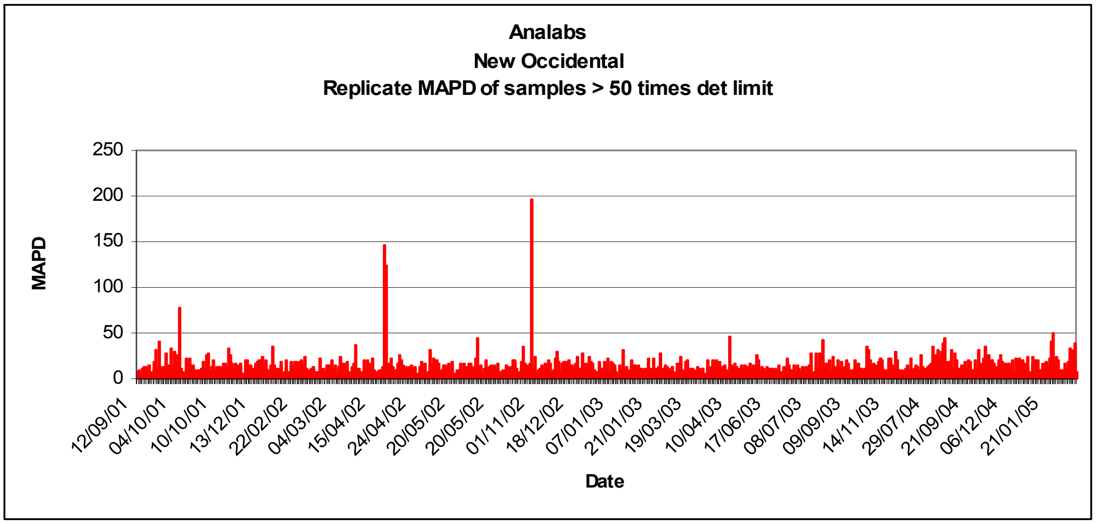


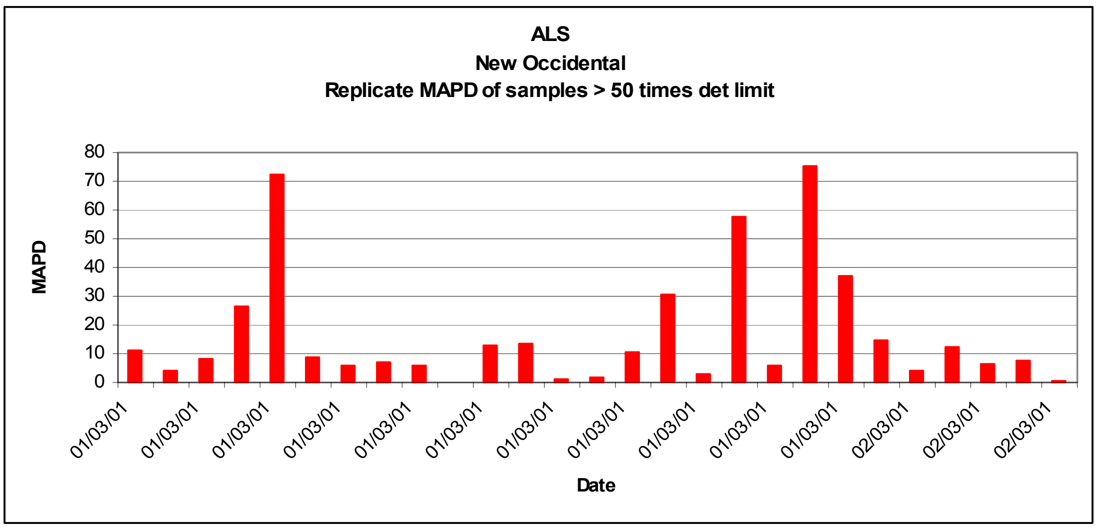
Replicate MAPD plots covering the period from 2000 to 2008 on a per project per lab basis are presented below

Figure 14-3: Au Replicate MAPD Plots on a per lab per Project basis

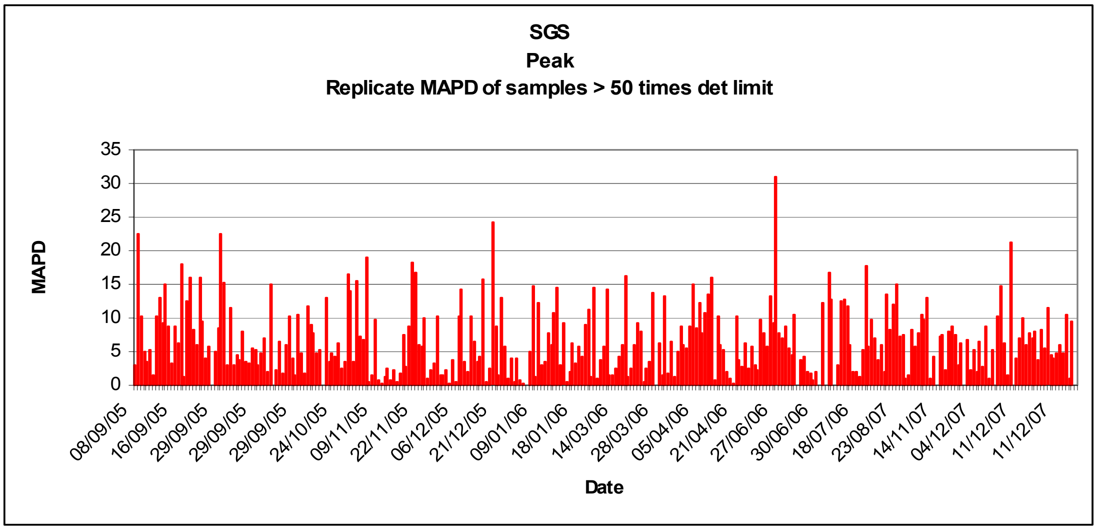
New Occidental

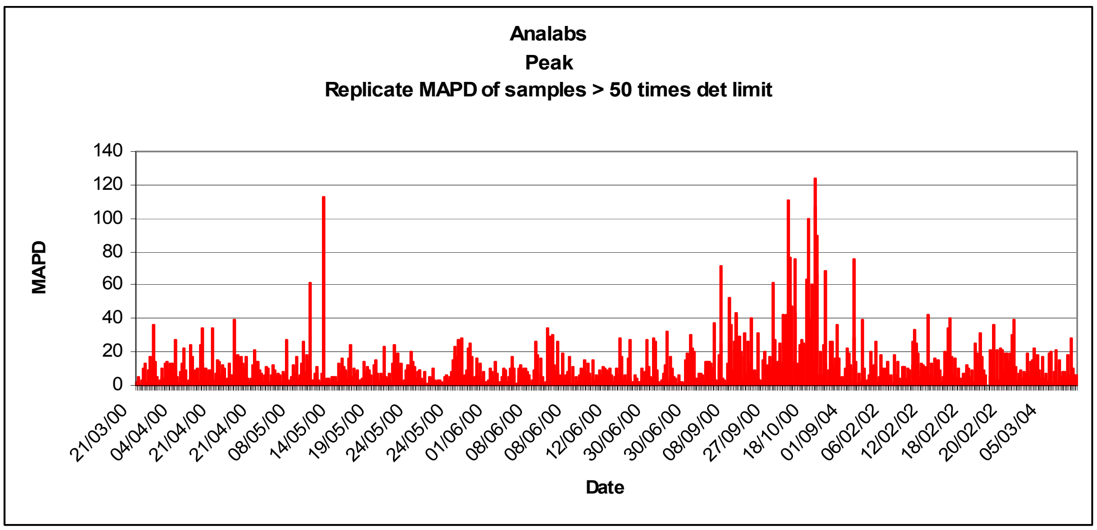


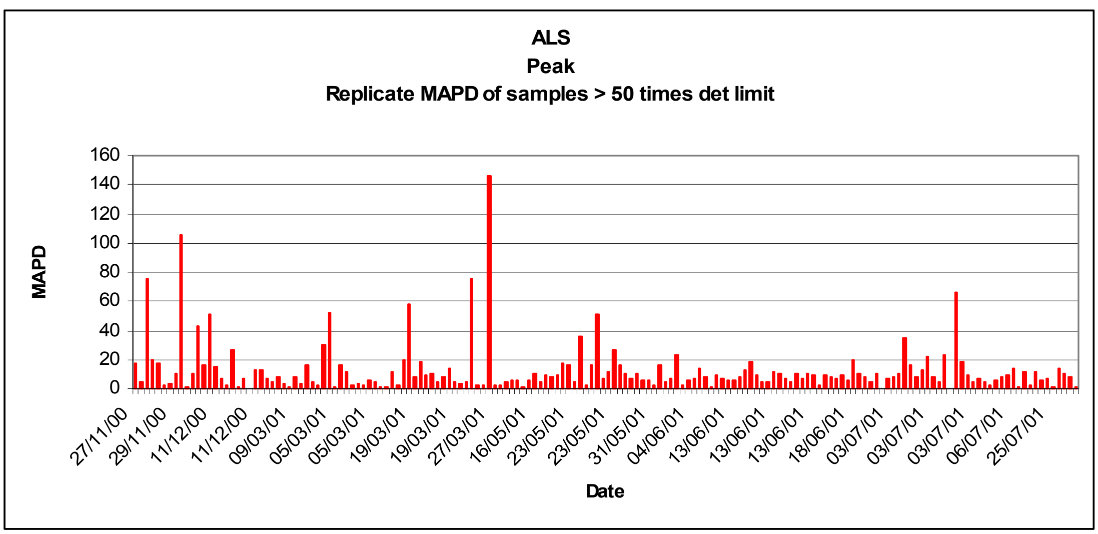

Analabs
New Occidental
Replicate MAPD of samples > 50 times det limit
MAPD
250
200
150
100
50
0
12/09/01
04/10/01
10/10/01
13/12/01
22/02/02
04/03/02
15/04/02
24/04/02
20/05/02
20/05/02
01/11/02
18/12/02
07/01/03
21/01/03
19/03/03
10/04/03
17/06/03
08/07/03
09/09/03
14/11/03
29/07/04
21/09/04
06/12/04
21/01/05
Date


ALS
New Occidental
Replicate MAPD of samples > 50 times det limit
MAPD
80
70
60
50
40
30
20
10
0
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
01/03/01
02/03/01
02/03/01
02/03/01
Date

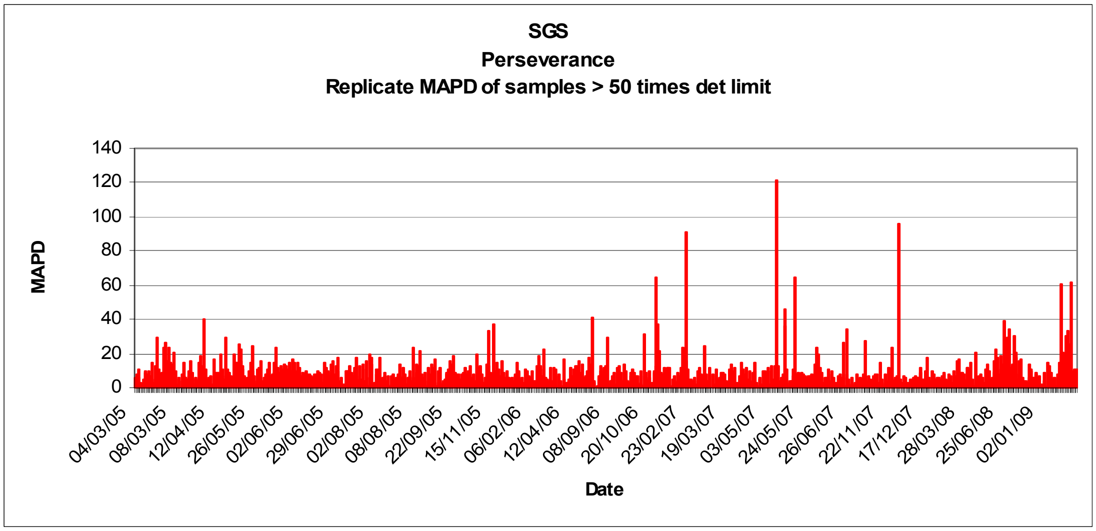
Peak

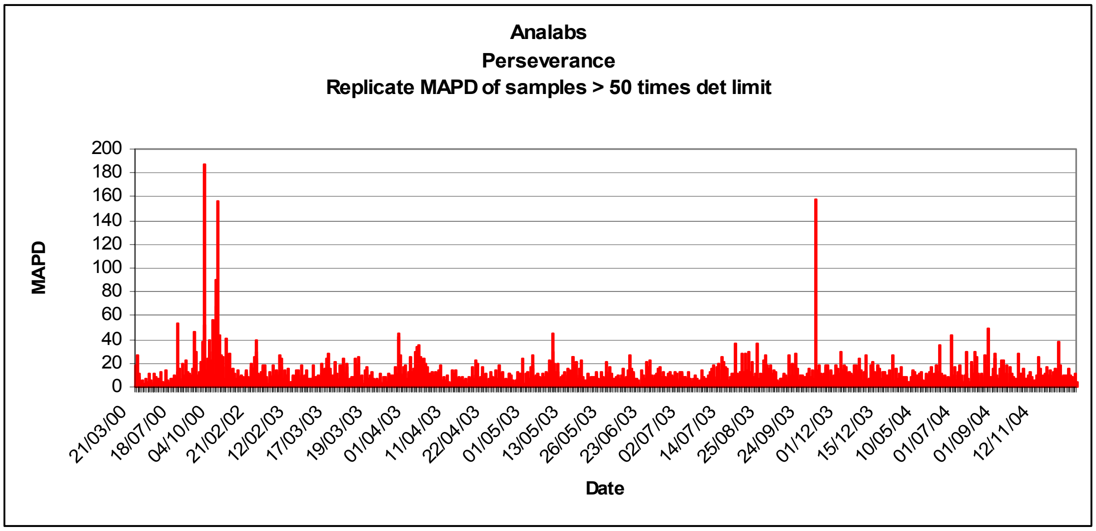

SGS
Peak
Replicate MAPD of samples > 50 times det limit
MAPD
Date


Analabs
Peak
Replicate MAPD of samples > 50 times det limit
MAPD
Date

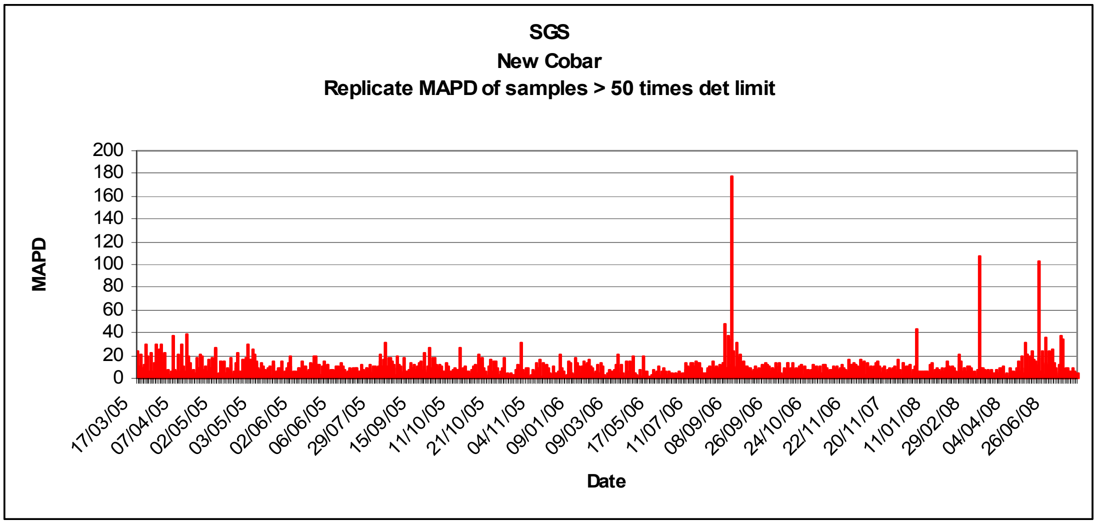

ALS
Peak
Replicate MAPD of samples > 50 times det limit
MAPD
Date

Perseverance

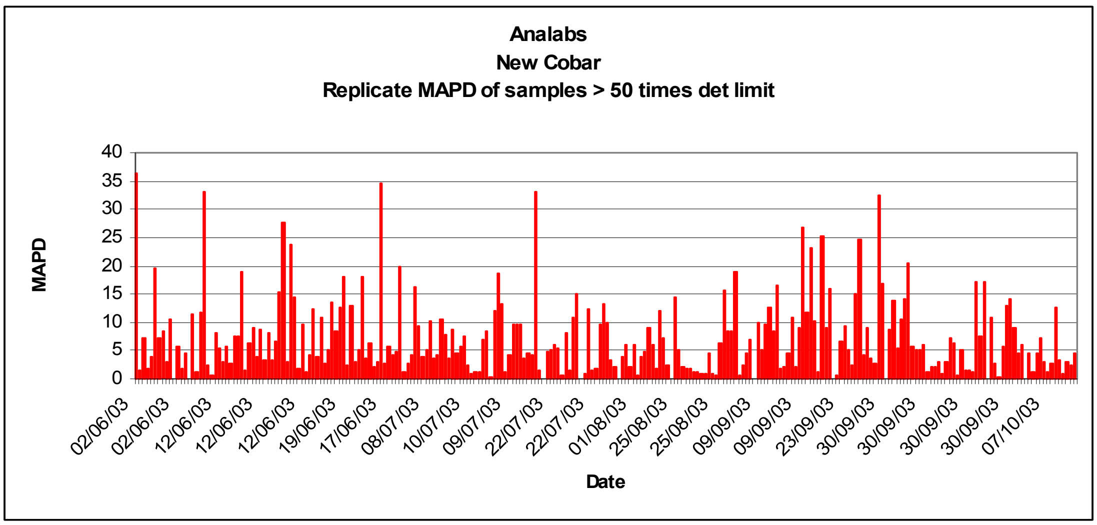




New Cobar

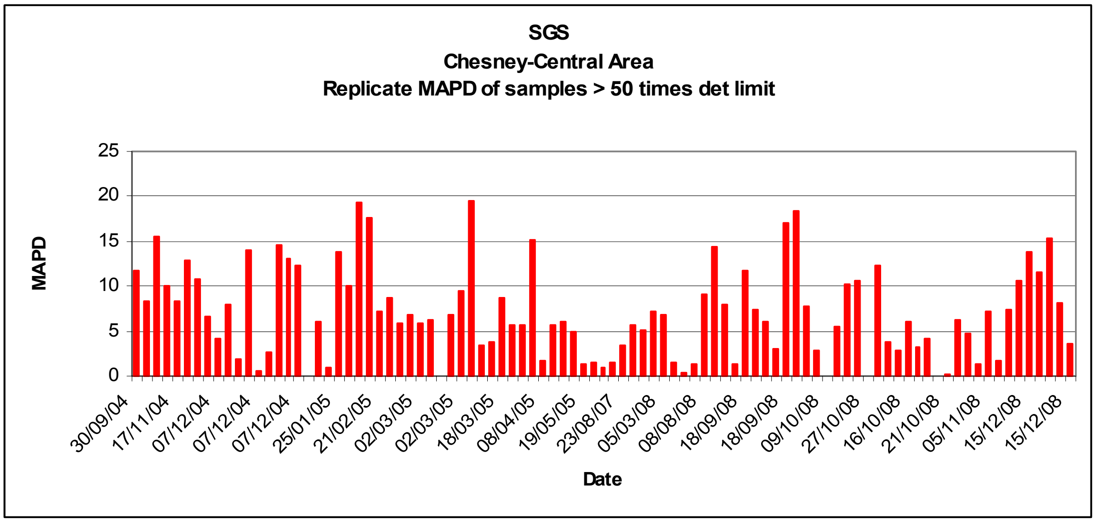


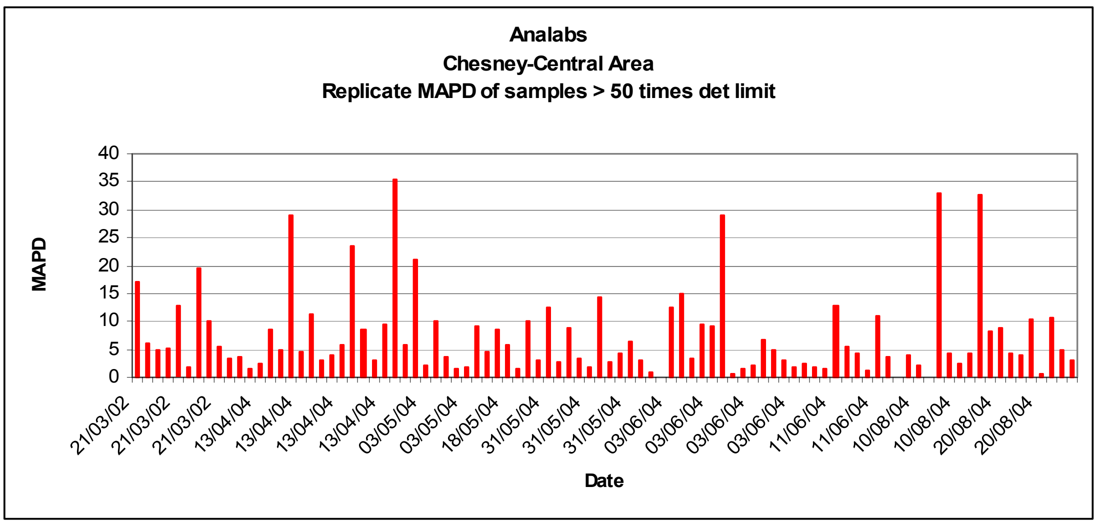
SGS
New Cobar
Replicate MAPD of samples > 50 times det limit
MAPD
200
180
160
140
120
100
80
60
40
20
0
17/03/05
07/04/05
02/05/05
03/05/05
02/06/05
06/06/05
29/07/05
15/09/05
11/10/05
21/10/05
04/11/05
09/01/06
09/03/06
17/05/06
11/07/06
08/09/06
26/09/06
24/10/06
22/11/06
20/11/07
11/01/08
29/02/08
04/04/08
26/06/08
Date



Analabs
New Cobar
Replicate MAPD of samples > 50 times det limit
MAPD
40
35
30
25
20
15
10
5
0
02/06/03
02/06/03
12/06/03
12/06/03
12/06/03
19/06/03
17/06/03
08/07/03
10/07/03
09/07/03
22/07/03
22/07/03
01/08/03
25/08/03
25/08/03
09/09/03
09/09/03
23/09/03
30/09/03
30/09/03
30/09/03
30/09/03
07/10/03
Date

Chesney Central Area



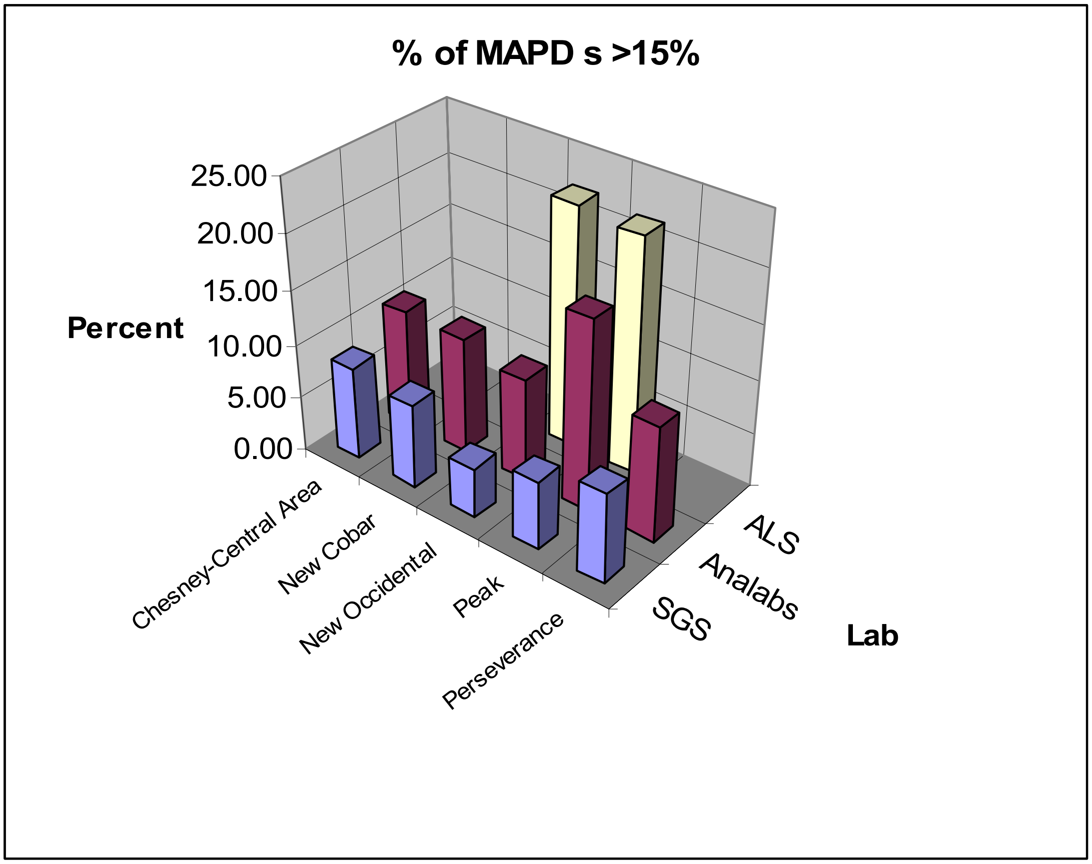


On a per project per lab basis the percentage of replicate assays of samples greater than 50 times detection limit with a MAPD of greater than 15% are given in the following table and graph.

Table 14-5: Percentage or Replicate samples with MAPD > 15%

Percentage of Replicate samples with MAPD > 15%			
	SGS	Analabs	ALS
Chesney-Central Are	8.60	10.87	
New Cobar	7.79	10.80	
New Occidental	4.63	9.55	22.22
Peak	6.35	17.48	21.69
Perseverance	8.27	10.64	

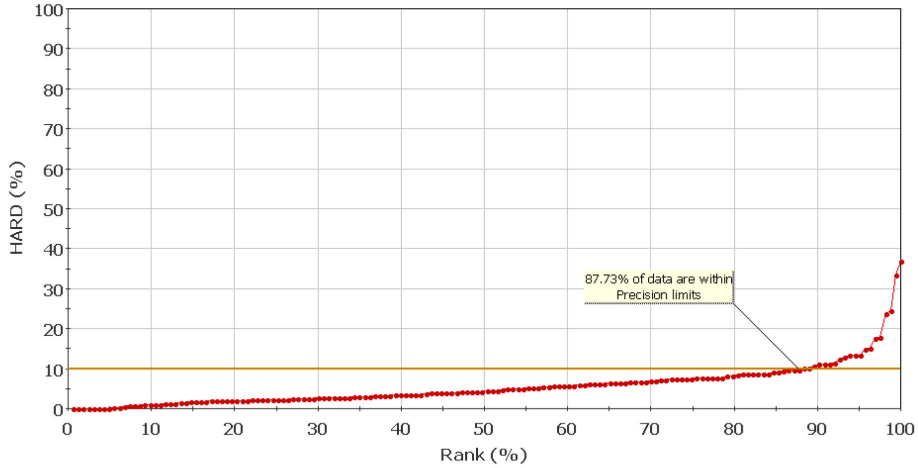
Figure 14-4: Replicate greater than 15% rate per project per lab



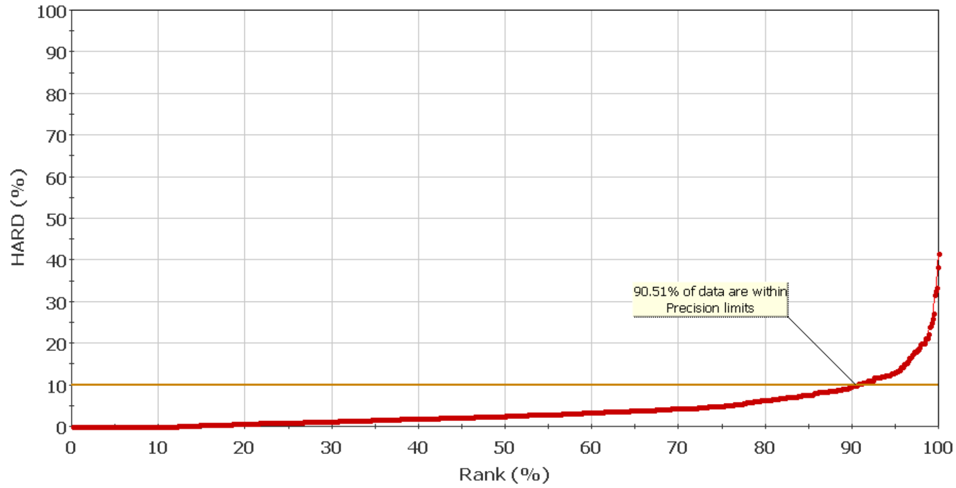
On a statistical basis ALS have the poorest performance record with greater than 20% of replicate assays falling outside a 15% MAPD range. SGS and Analabs both performed to an acceptable standard on all project areas.

Figure 14-5: Rank HARD Plot for Gold Assay Duplicates 2000 - 2006

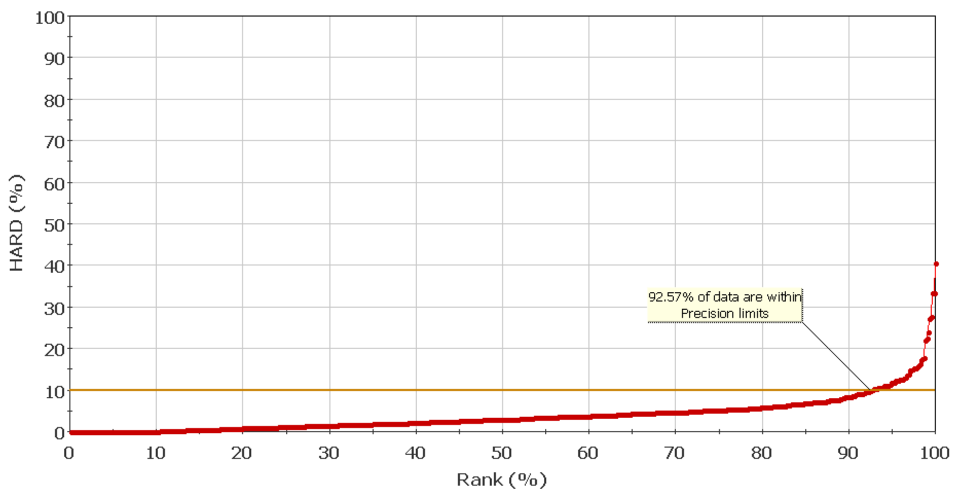
ALS



Analabs



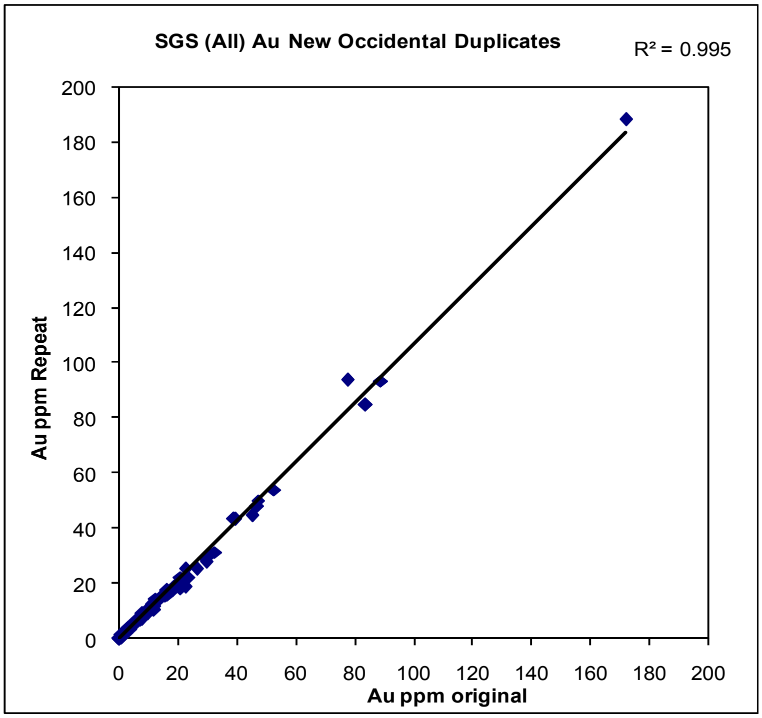
SGS

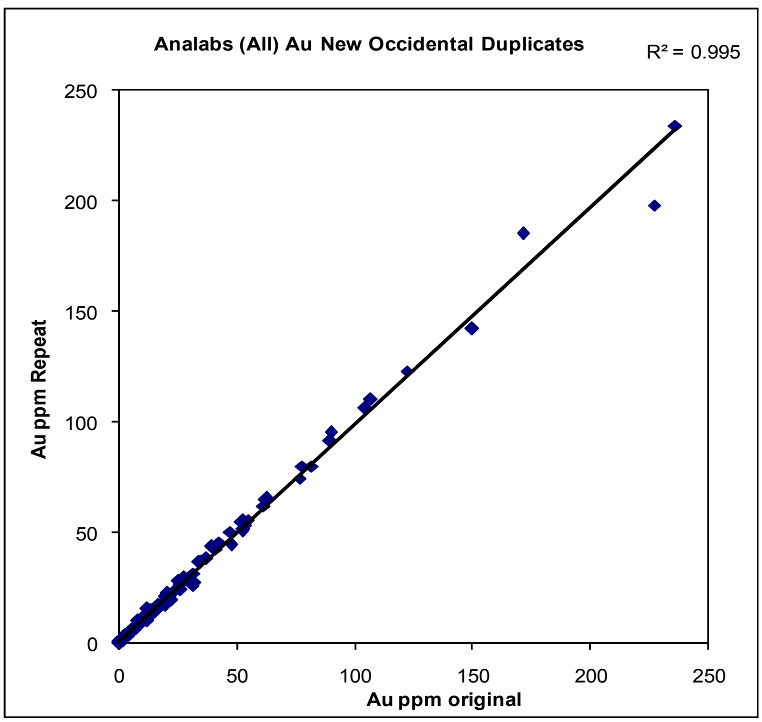


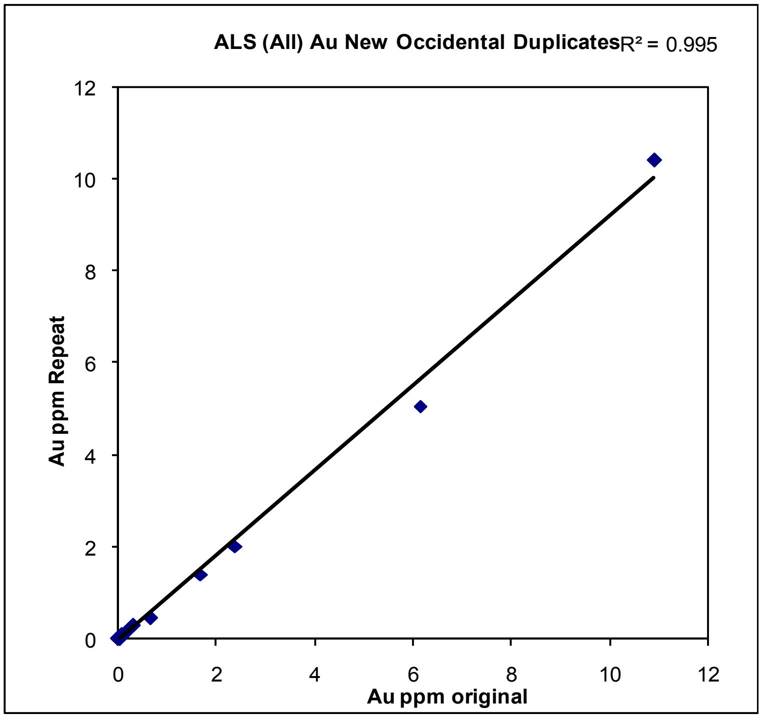
Assay duplicate results plotted against original assays on a per project per lab basis are presented below.

Figure 14-6: Correlation Plot for Au Assay Duplicates since 2000

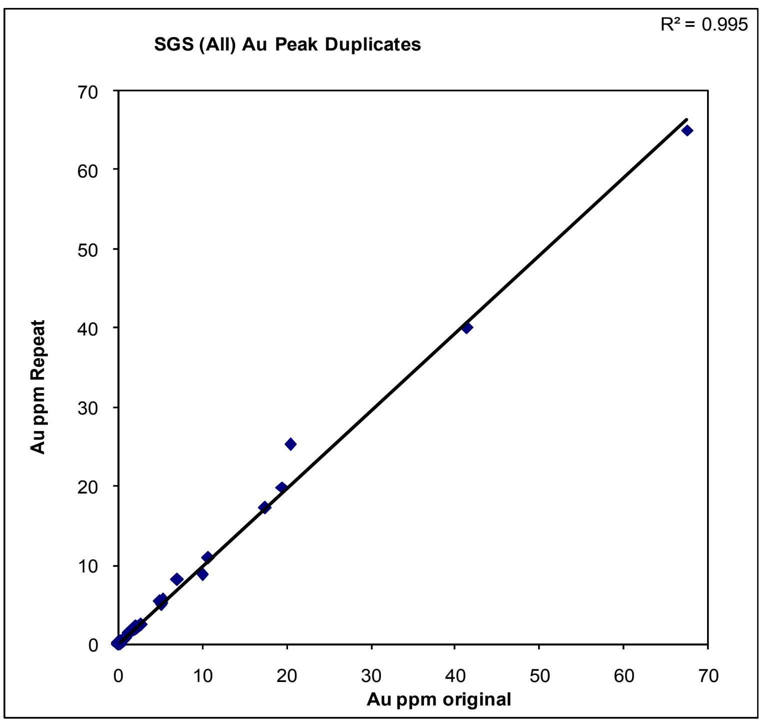
New Occidental

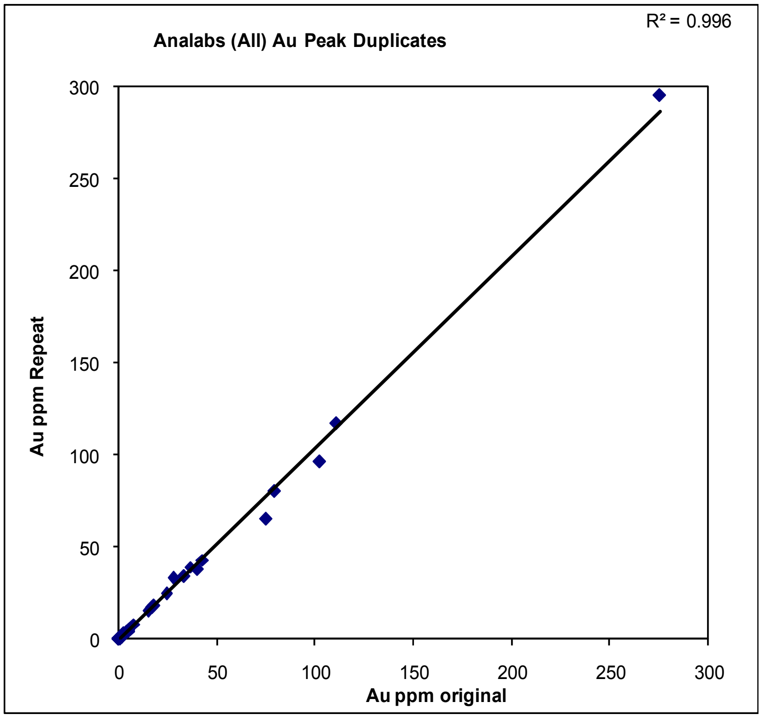




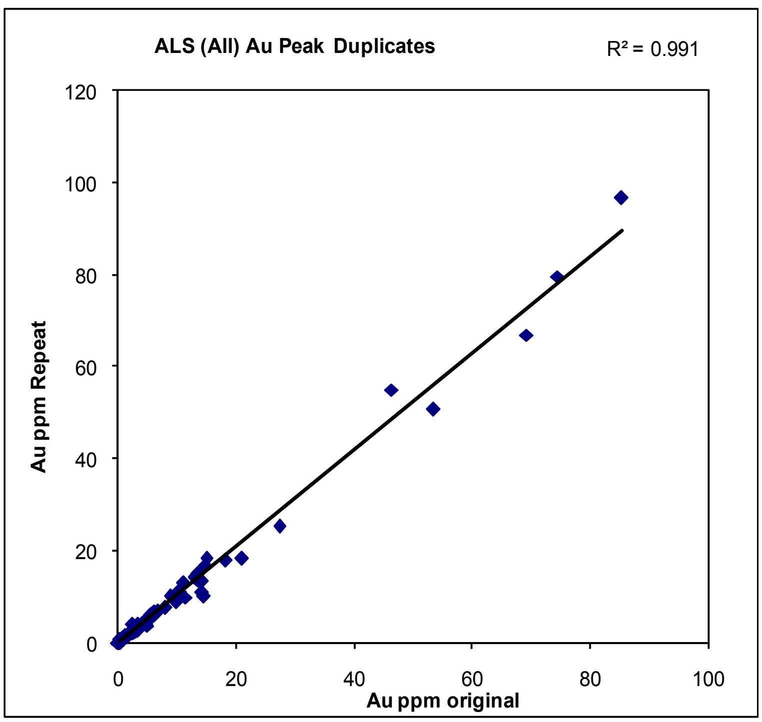

ALS (All) Au New Occidental Duplicates
R² = 0.995
Au ppm Repeat
Au ppm original
0
2
4
6
8
10
12

Peak


R² = 0.995
SGS (All) Au Peak Duplicates
Au ppm Repeat
Au ppm original
0
10
20
30
40
50
60
70

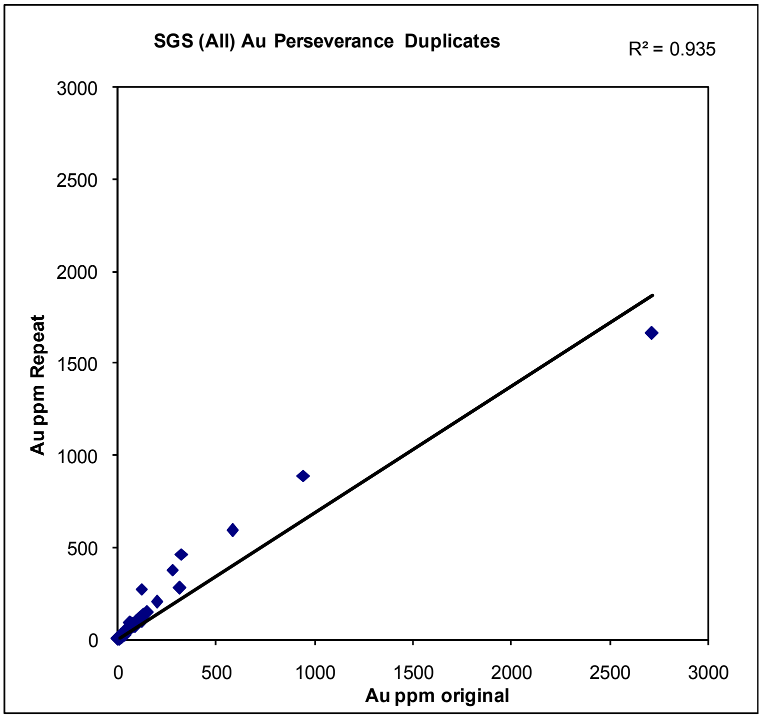


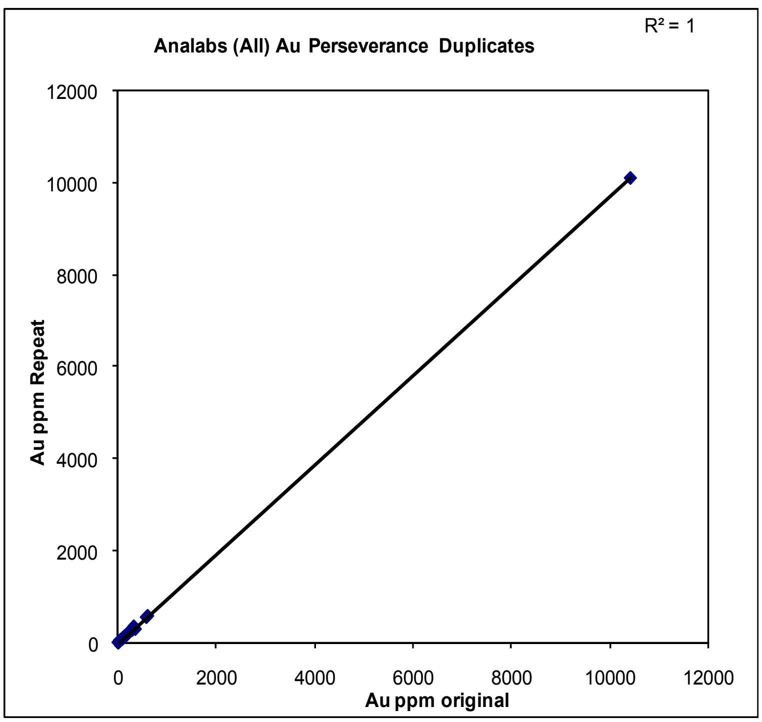
R² = 0.996
Analabs (All) Au Peak Duplicates
300
250
200
150
100
50
0
Au ppm Repeat
0
50
100
150
200
250
300
Au ppm original

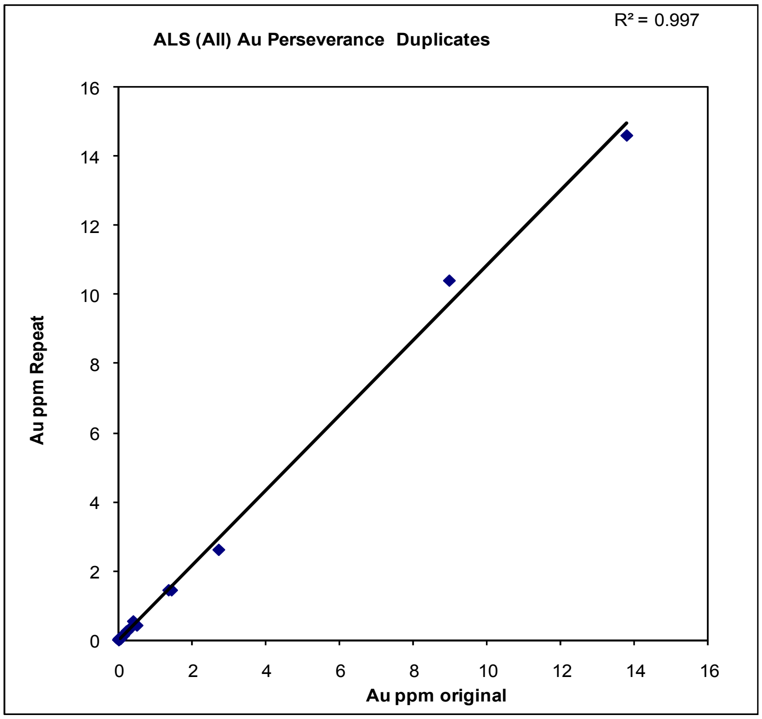


ALS (All) Au Peak Duplicates
R² = 0.991
120
100
80
60
40
20
0
Au ppm Repeat
0
20
40
60
80
100
Au ppm original

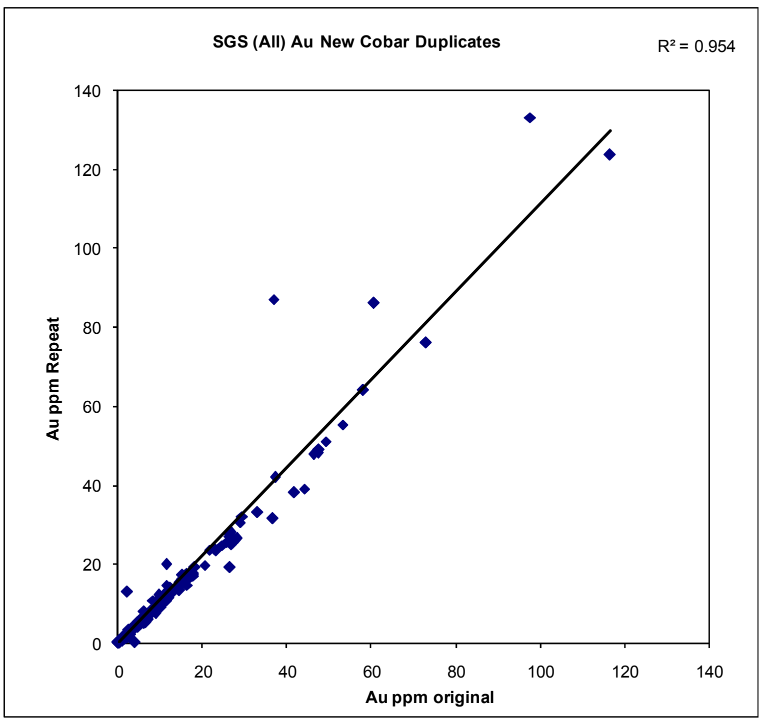
Perseverance

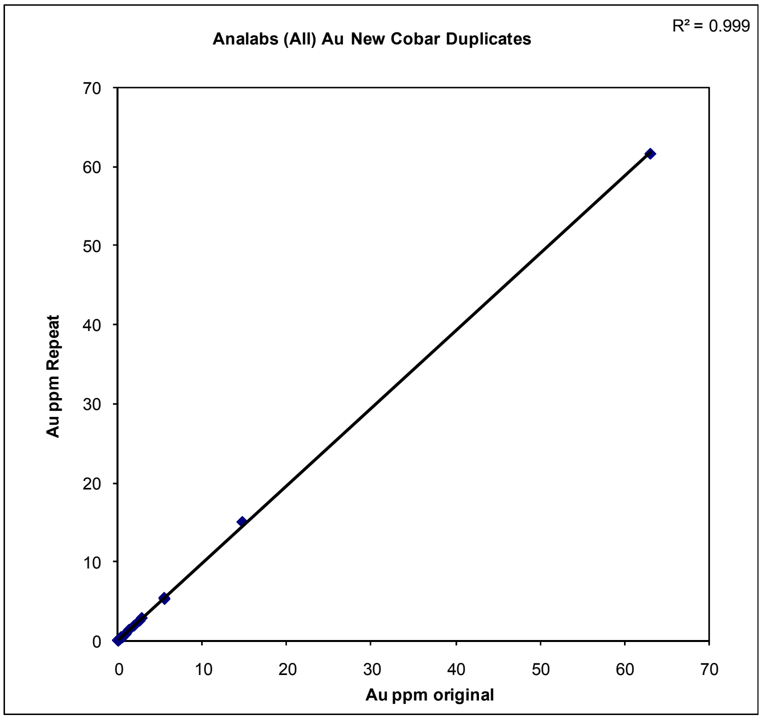




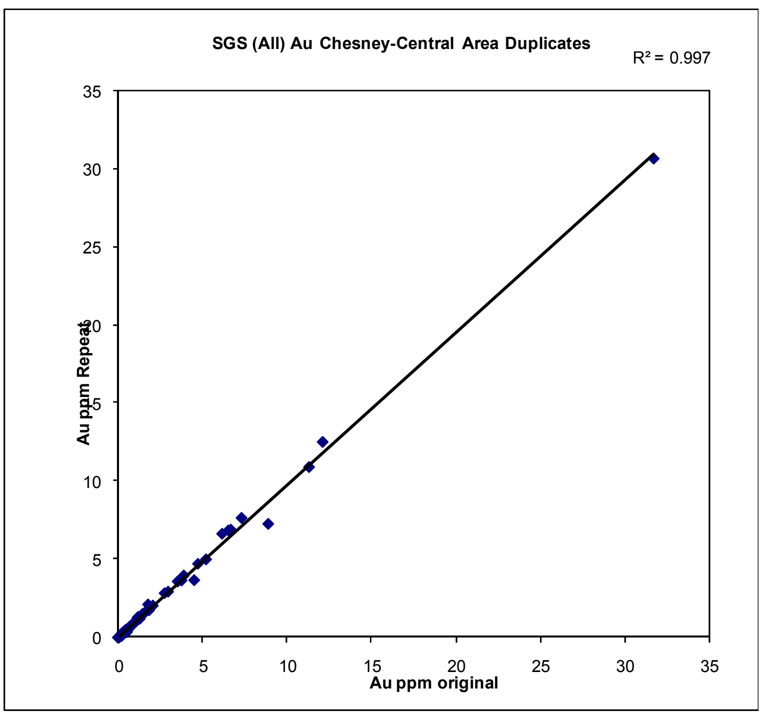

R² = 0.997
ALS (All) Au Perseverance Duplicates
Au ppm Repeat
Au ppm original
0
2
4
6
8
10
12
14
16

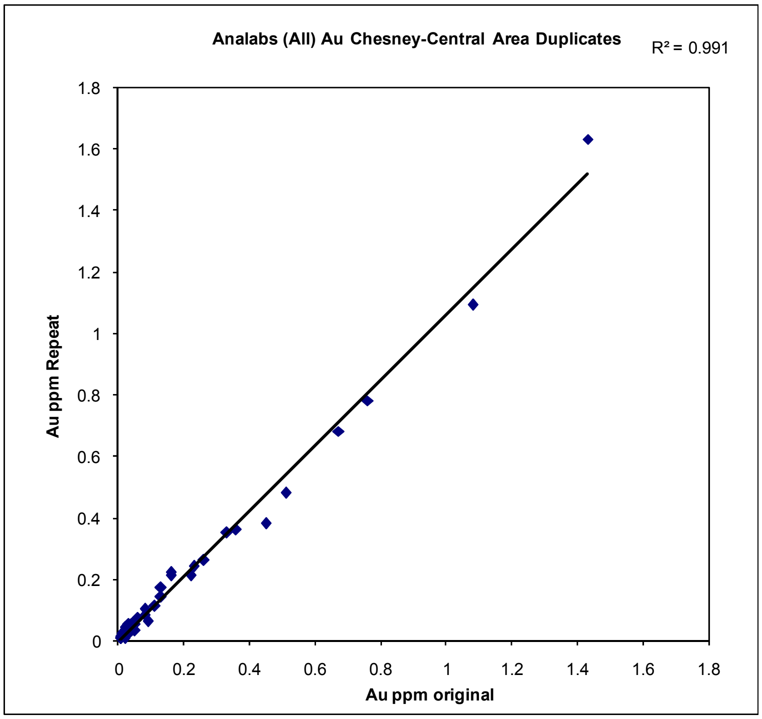
New Cobar


SGS (All) Au New Cobar Duplicates
R² = 0.954
Au ppm Repeat
Au ppm original
0
20
40
60
80
100
120
140



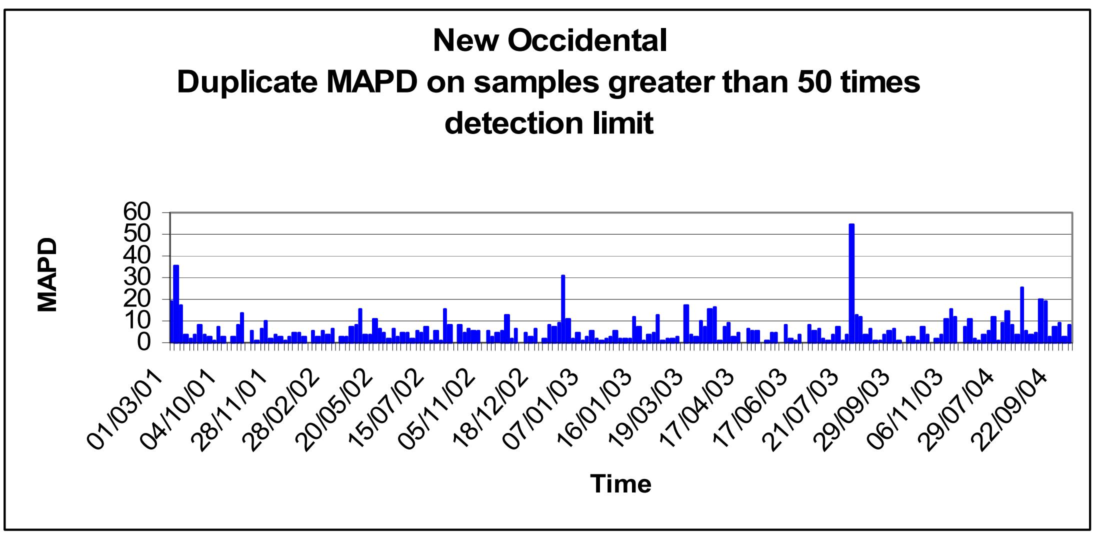
Chesney Central Area

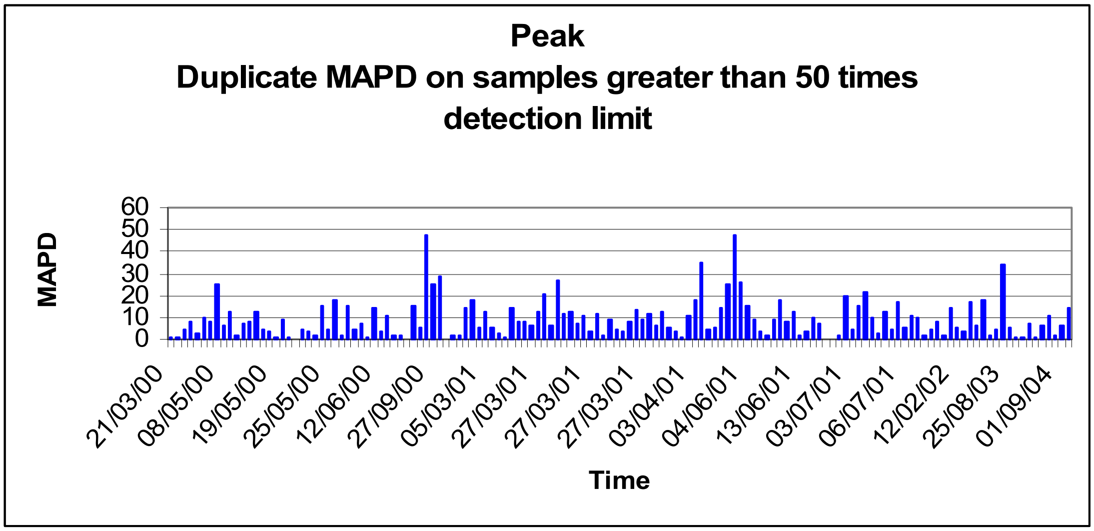


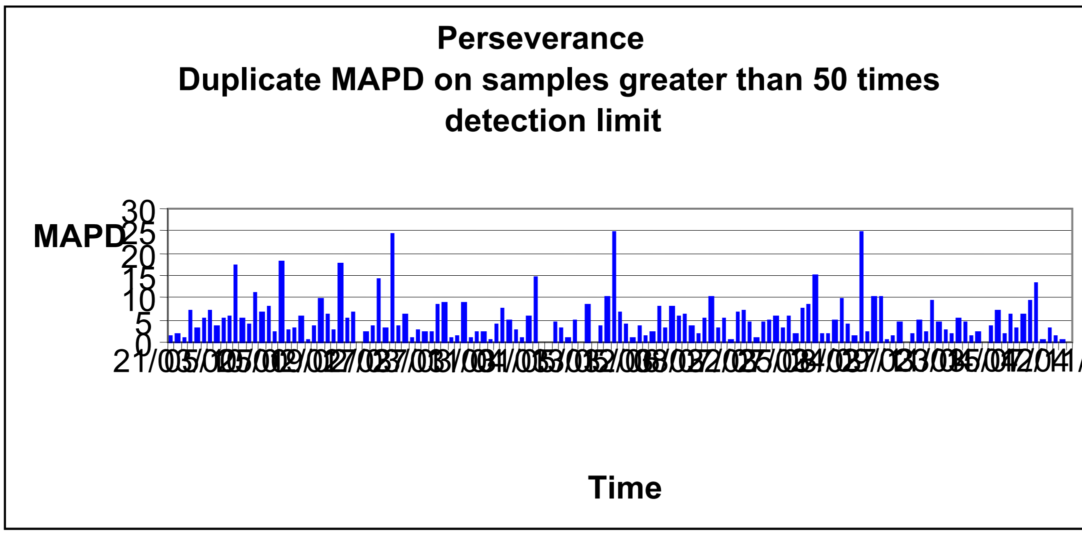


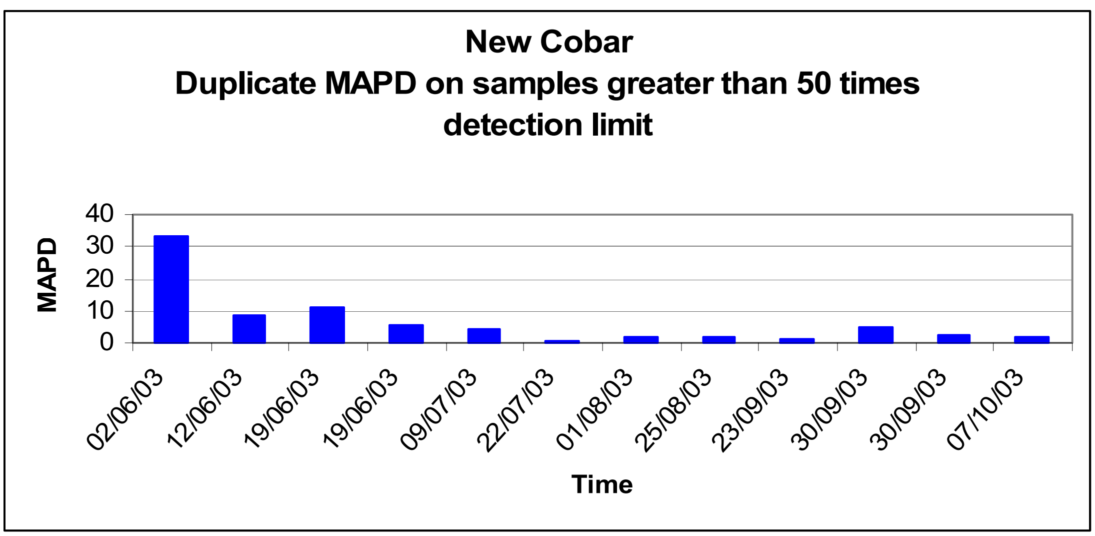
Duplicate performance on an MAPD basis over time on a per project area basis is given on the following graphs.

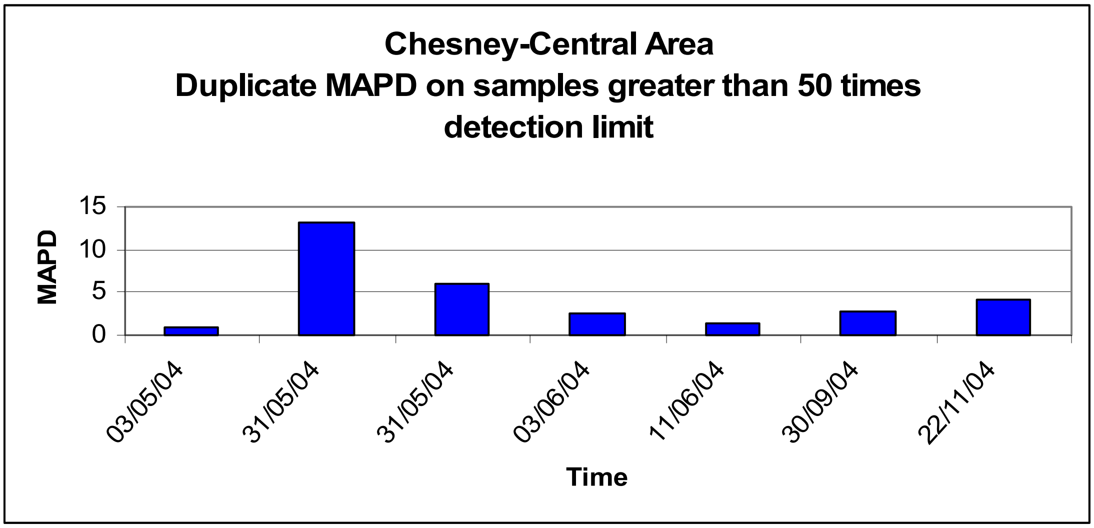
Figure 14-7; Duplicate MAPD on a per project basis




Peak
Duplicate MAPD on samples greater than 50 times detection limit
MAPD
60
50
40
30
20
10
0
21/03/00
08/05/00
19/05/00
25/05/00
12/06/00
27/09/00
05/03/01
27/03/01
27/03/01
27/03/01
03/04/01
04/06/01
13/06/01
03/07/01
06/07/01
12/02/02
25/08/03
01/09/04
Time


Perseverance
Duplicate MAPD on samples greater than 50 times detection limit
MAPD
30
25
20
15
10
5
0
Time


New Cobar
Duplicate MAPD on samples greater than 50 times detection limit
MAPD
40
30
20
10
0
02/06/03
12/06/03
19/06/03
19/06/03
09/07/03
22/07/03
01/08/03
25/08/03
23/09/03
30/09/03
30/09/03
07/10/03
Time



Duplicate performance is comparable to replicate performance indicating a high degree of homogeneity in pulps prepared by the LM50 disk pulverizer.

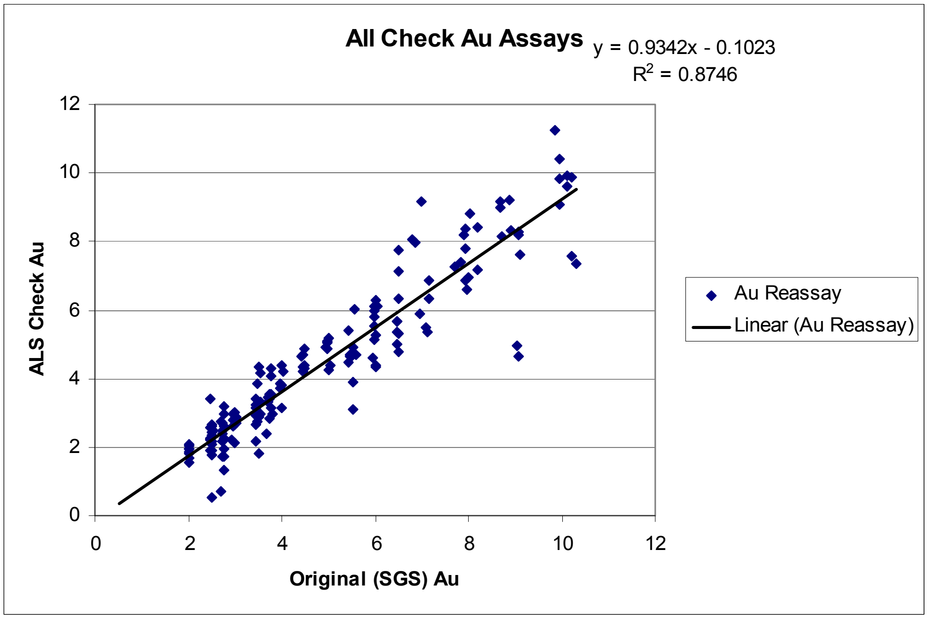
14.1.3 Umpire Assays

In early 2006, PGM used ALS in Orange as an umpire laboratory to check 166 pulps originally analysed for Au by SGS in Cobar. Summary statistics from this study are shown Table 14-6. The results from this study indicated that the original SGS Au assays had an overall positive bias with respect to the ALS Au analyses, with the mean of the original SGS Au assays 8.7% higher than the mean of the ALS check Au assays. The cause of the bias is uncertain. Figure 14-8 shows a correlation plot of the original SGS Au assay data and the ALS check assay data.

Table 14-6: Summary Statistics for 2006 Umpire Assaying

Original SGS Auppm		*ALS Check Auppm*	
Mean	4.95	Mean	4.52
Standard Error	0.18	Standard Error	0.18
Median	4.02	Median	4.21
Mode	6.48	Mode	2.63
Standard Deviation	2.35	Standard Deviation	2.35
Sample Variance	5.52	Sample Variance	5.50
Kurtosis	- 0.65	Kurtosis	- 0.24
Skewness	0.70	Skewness	0.78
Range	8.29	Range	10.72
Minimum	2.01	Minimum	0.53
Maximum	10.30	Maximum	11.25
Sum	822.24	Sum	751.12
Count	166	Count	166
Confidence Level (95.0%)	0.36	Confidence Level (95.0%)	0.36

Figure 14-8: Correlation Plot of Original Assay versus Check Assays



14.1.4 Blank Reference Samples

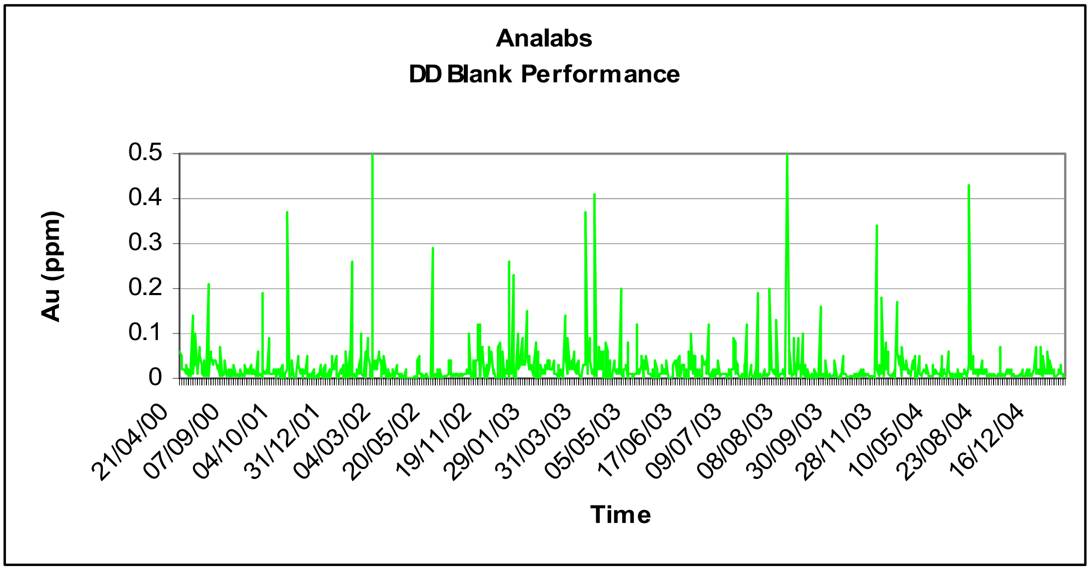
PGM currently uses intervals of core identified to be barren as material for blank samples, these samples are not certified as being barren. Figure 14-9 shows standards control plots for Au and Cu of the assaying of the blank samples (DD_Blank) from Analabs since 2000 to 2004. Figure 14-10 shows the results of blank samples from ALS to 2007. Figure 14-11 shows the results of blank samples from SGS (since 2004 to 2008). Summary statistics for the blank reference samples are shown in Table 14-7 to December 2008.

The great majority of the Cu blank sample results (>98% of the data) for all of the laboratories fall below a 0.06% Cu threshold. Most of the gold blank sample results (91%) are below a 0.08g/t Au threshold. Review of several of the assay peaks for the blank samples reveals that only a few are associated with high grade previous samples from the same drill-hole sample sequence. Many of the other anomalies for the blank sample data are associated with blank samples inserted at the start of the batch sampling stream as part of the site QAQC protocol. This suggests that there is some laboratory contamination occurring prior to the processing of the PGM samples. In addition, the source diamond core used for blank samples is not certified and may be partially mineralised above detection limits for the assay processes used.

Table 14-7: Summary Statistics for Blank Standards Results since January 2000 to December 2008

Gold Data (ppm)			
Item	Analabs	ALS	SGS
Expected Value	0.05	0.05	0.05
Expected Value Range	0.00 to 0.06	0.00 to 0.06	0.00 to 0.06
Count	1070.00	228.00	1039.00
Minimum	0.00	0.00	0.01
Maximum	1.53	1.33	2.71
Mean	0.03	0.02	0.02
Std Deviation	0.07	0.10	0.12
% in Tolerance	90.70%	92.52%	91.75%
% Bias	-47.72%	-38.88%	-45.41%
% RSD	265.15%	409.97%	416.11%
Cu Data (%)			
Item	Analabs	ALS	SGS
Expected Value	0.05	0.05	0.05
Expected Value Range	0.00 to 0.06	0.00 to 0.06	0.00 to 0.06
Count	1070.00	228.00	1030.00
Minimum	0.00	0.00	0.00
Maximum	0.23	1.70	0.69
Mean	0.01	0.01	0.01
Std Deviation	0.02	0.12	0.03
% in Tolerance	98.67%	98.67%	98.45%
% Bias	-83.59%	-69.27%	-77.51%
% RSD	203.01%	937.06%	309.54%

Figure 14-9 DD Blank Performance for Au and Cu for Analabs (2000 to 2004)



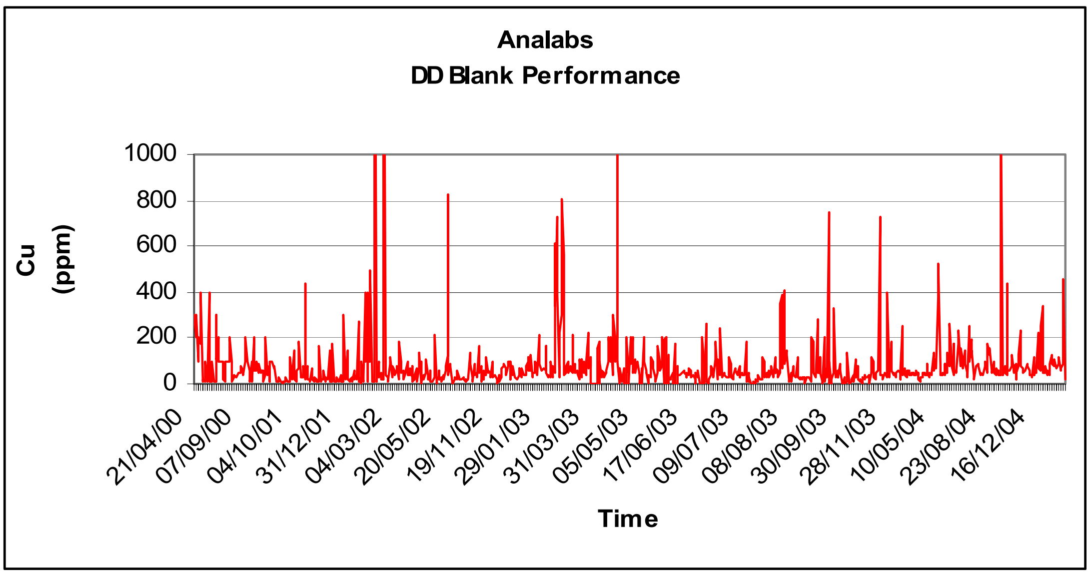
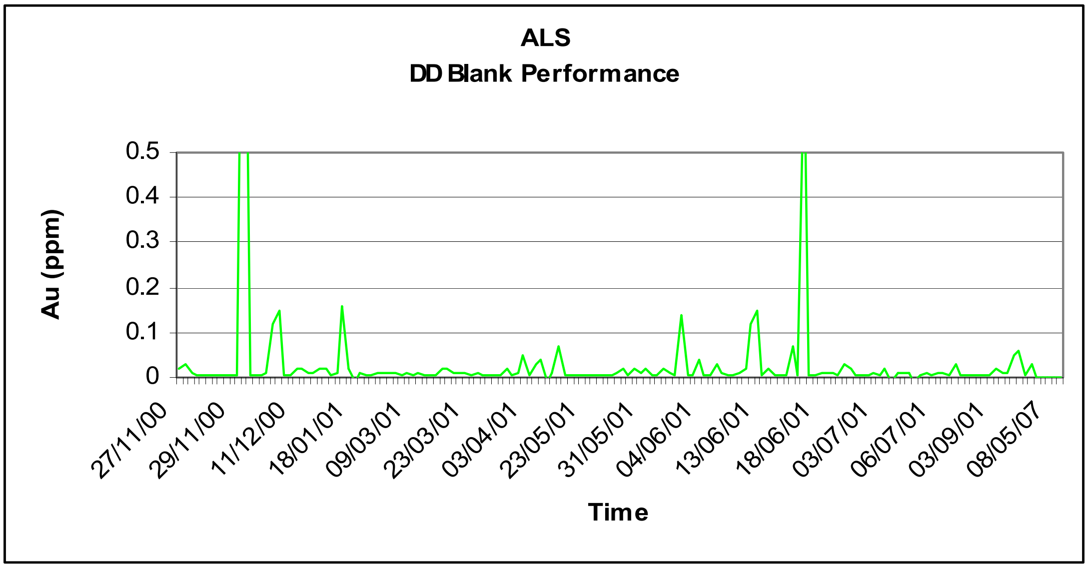


Figure 14-10: DD Blank Performance for Au and Cu for ALS (2000 to 2007)



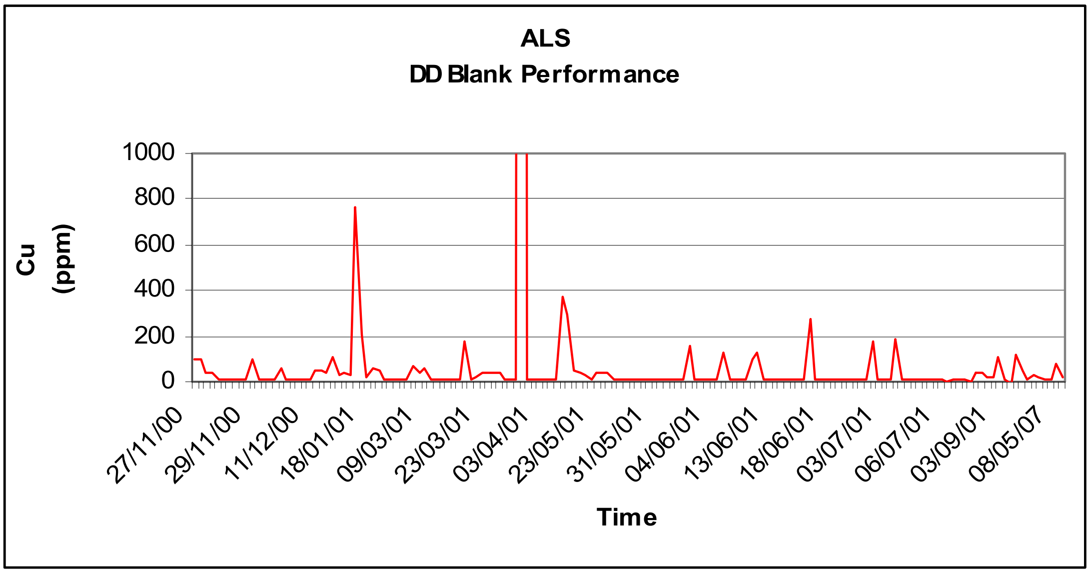
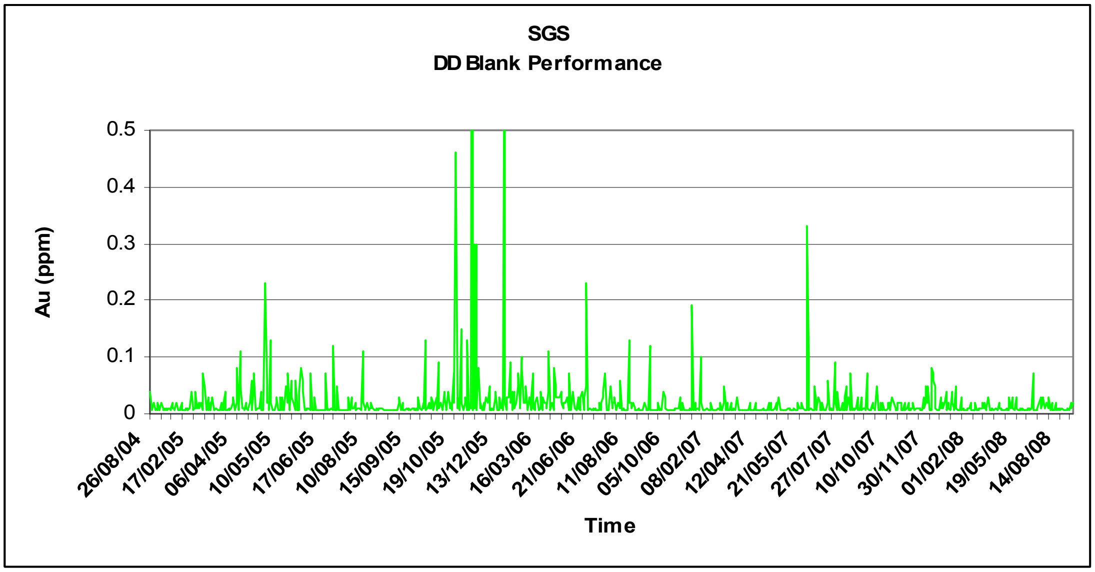
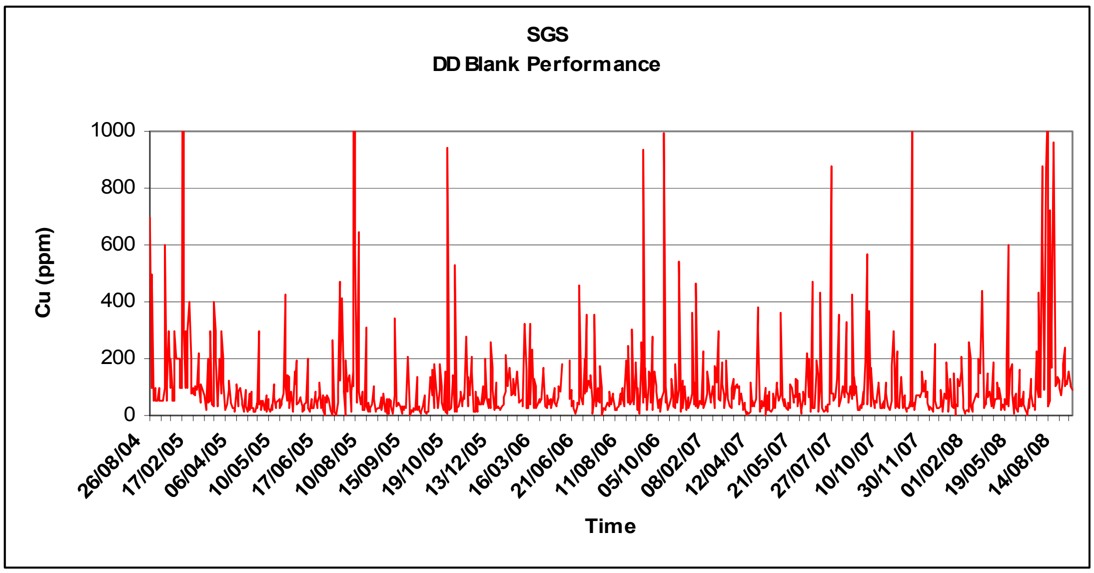


Figure 14-11: DD Blank Performance for Au and Cu for SGS (2004 to 2008)

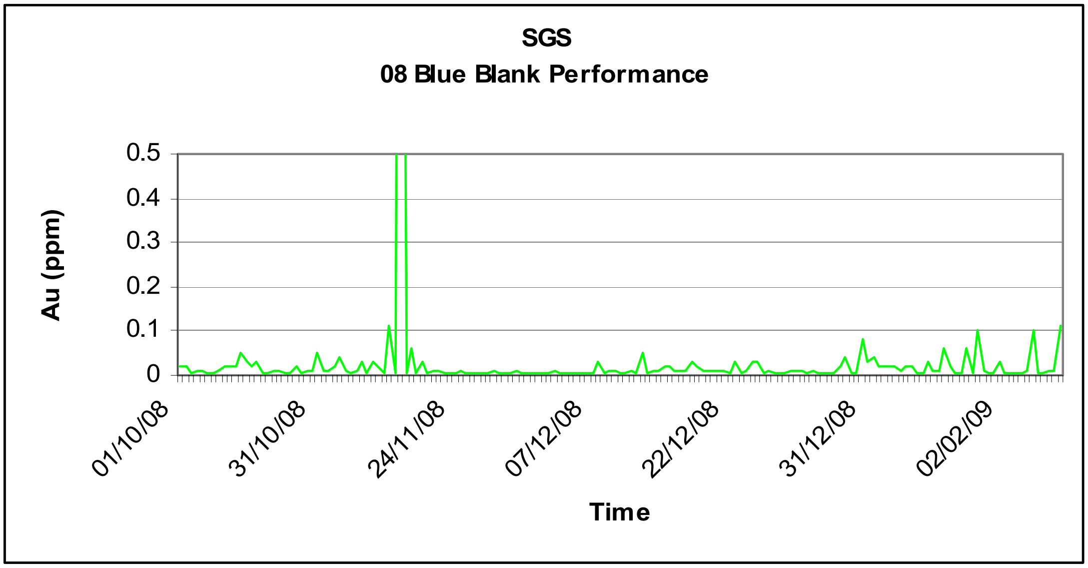


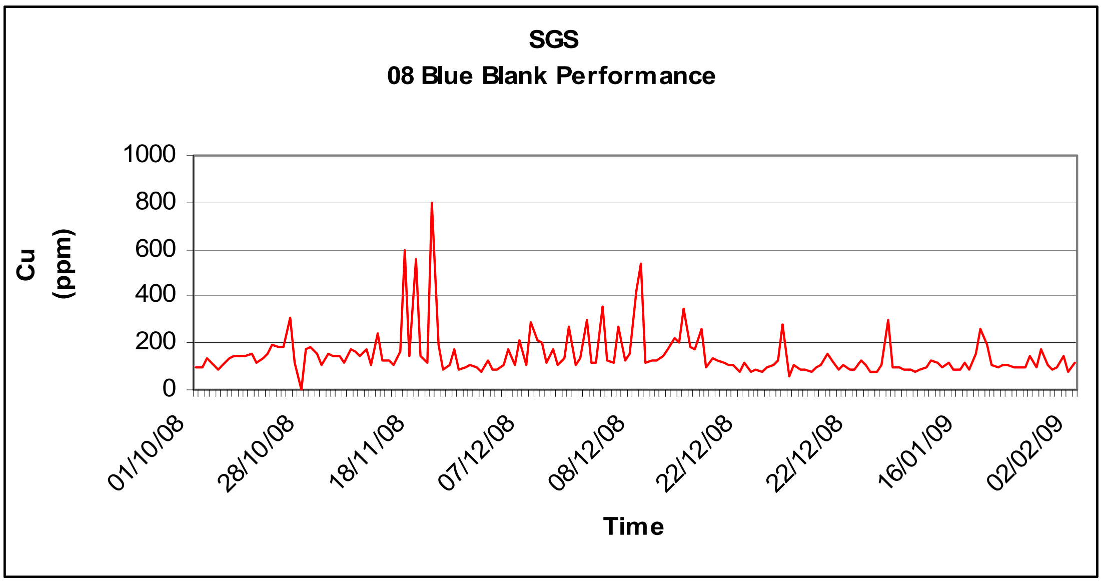


Midway through 2008 the blank medium was changed from selected barren core to crushed weakly altered diorite (drill hole stemming). Although core selected for blank material was generally barren the majority of this core was sourced from areas marginal to economic ore deposits and realistically occasional samples were likely to contain anomalous gold. The appearance of anomalous gold values in the blanks could be attributed to either poor lab hygiene or anomalous core. By using material known to be barren any anomalous gold detected in the blanks was likely to be an artifact of the sample preparation regime.

The low noise in the Gold 08 Blue Blank plot (Figure 14-12) in comparison to the Gold DD Blank plot Figure 14-11 can be clearly seen.

Figure 14-12: Blue Blank Performance Au and Cu for SGS (2008)





14.2 Project Database

PGM use a customised database management system to capture all details regarding the drilling and sampling of drill holes on site. This system automatically generates sample numbers for drill hole sampling, and controls the insertion points for standards and blanks. All samples in the database are tracked by a unique sample number. Since 1995, all assays have been electronically loaded into the database. The Drillview database contains a comprehensive record of the drill hole surveying, QAQC data, sampling schemes and laboratories for every sample in the database.

Write access to the Drillview database is password protected (currently limited to access by the Principal Geologist, Geology Superintendent and Senior Geologists). Validation reports are generated regularly, checking areas such as drill hole overlaps, hole name validation and end of hole depth mismatches. After loading of the drill hole data, site geologists visually check its spatial position using Vulcan mining software.

14.3 Bulk Density Determination

Bulk density measurements have been routinely recorded since 1989. A total of 100,895 density readings have been recorded in the database since 2000. The current site practice includes regular check measurements using standard samples and direct recording of the standard results into a computerised system.

Bulk densities are determined using the wet balance method unless the technique cannot be applied due to condition of the drill core (broken core or presence of voids). Scales are calibrated once a year as part of QAQC procedures applied to the on site lab.

Chronologically ordered plots of bulk density standards are presented below for the two standards used (SG9901 & SG9902) over the period from the beginning of 2000 until the end of 2008. The acceptable range limits are given on the charts as red lines. These lines lay 1.9% and 1.1% from the mean of the acceptable ranges for SG9901 and SG9902 respectively.

Figure 14-13: SG9901 SG standard graphs

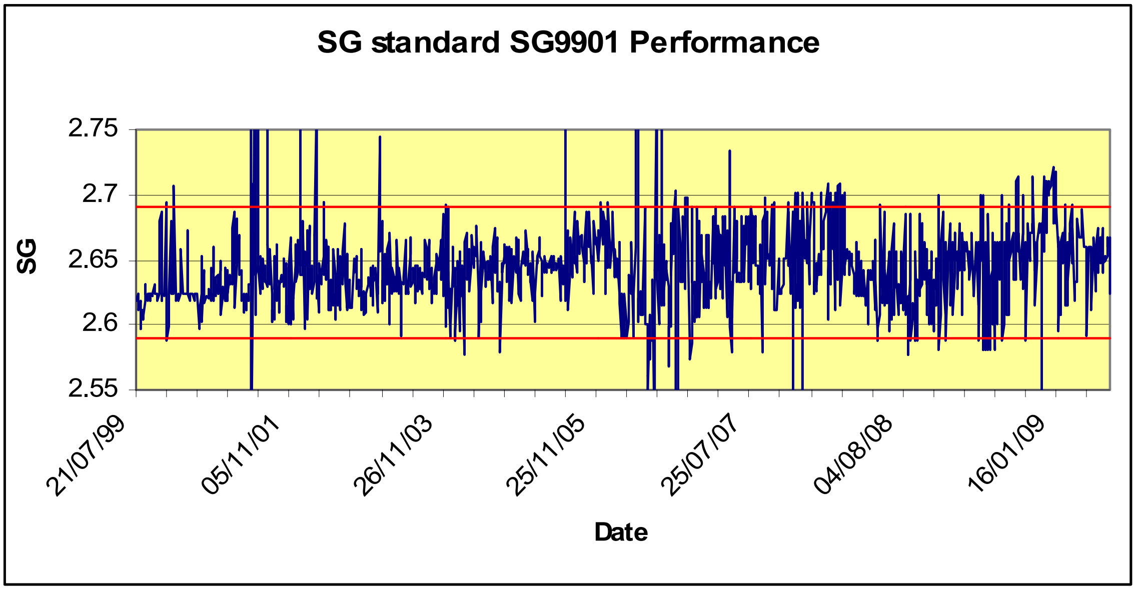
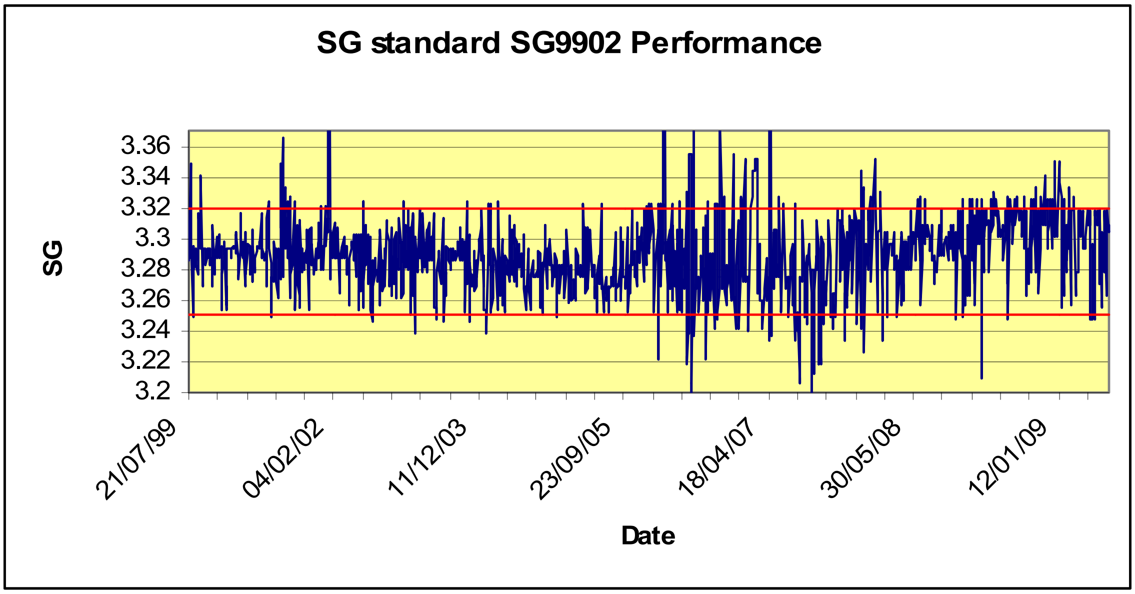


Figure 14-14: SG9902 SG standard graphs



14.4 Drill Hole Recovery

As discussed previously, the bulk of all mineralised samples exhibit excellent recovery, with over 98% of the mineralised gold intervals (>1g/t Au) with recorded recoveries of 100%.

14.5 Survey Control

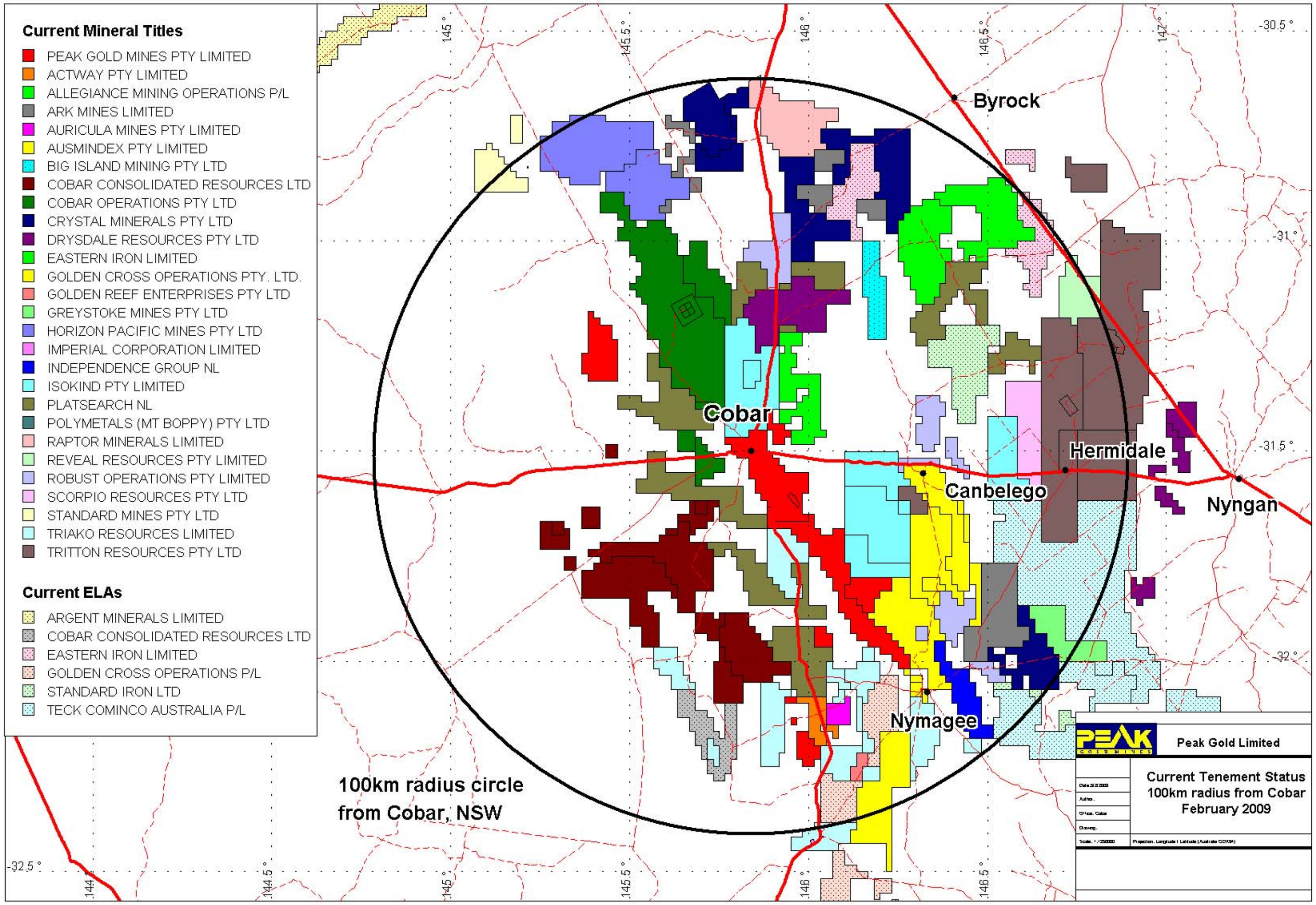
Survey control for the area has been well established. A detailed topographic surface has been generated for the project areas based on a digital ortho-photo survey and is considered adequate for the current investigations. Site surveyors accurately locate all underground drill holes and near-mine surface drilling. All surface mine and underground mine development have been picked up by site surveyors. Current detailed survey methods and results are considered to be of industry standard.

15 ADJACENT PROPERTIES

PGM controls a large block of ground in and surrounding the Cobar Gold Field comprised of exploration licenses and mining leases. The land position extends over an approximately 70-km strike length of prospective structures and lithologies. Figure 15-1 shows the numerous active and abandoned mines, prospects, and exploration licenses. No adjacent properties have a material effect on the value of the properties held directly by PGM.

PGM has farm-in agreements on two exploration licenses (EL 5982 and 6127) that are described in Section 4.2 of this report.

Figure 15-1: Cobar Regional Exploration Licences and Applications



16 MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical section was prepared with the assistance of David Hall, Mill Manager of Peak Gold Mines.

The PGM Life of Mine production plan considers treating nine different feed materials between 2009 and 2017. The materials identified include New Occidental, Perseverance, New Cobar (POX), New Cobar, Chesney, Process Water Dam material, Great Cobar, Hulk and Hercules and Peak. Feed through the plant will include batch treating as well as blending. New Cobar (POX) material treated is from an existing stockpile and is planned to be depleted by early 2009.

16.1 Mineralogy

Following is a summary of mineralogical characterization studies performed on selected samples of different ores treated at Peak Gold Mine. The results are based on the specific samples analysed and may not represent the entire deposits. Certain mineral assemblages may not have been identified or analysed. Reports from which the information is sourced are identified in Section 22.

16.1.1 New Cobar

Gold occurs as both native gold and electrum (Ag, Au). Gold is predominantly fine grained (10-100μm). Bismuth, selenium and arsenic are impurity elements. Bismuth minerals include native bismuth (Bi), bismuthinite (Bi_2S_3) and paraguanajuatite ($Bi_2(Se, S)_3$). Bismuth sulphide and selenide minerals are likely to have similar flotation properties to chalcopyrite. Chalcopyrite was found to be the only copper mineral present.

16.1.2 New Occidental

The New Occidental ore is predominantly characterized by fine grained gold mineralization associated with bismuth minerals and stilpnomelane in a strongly quartz veined and silicified quartz breccia host. Gold recoveries:

- Do not appear to be affected by bismuth content.
- Appear to be adversely affected by the presence of stilpnomelane $(Fe, Mg)_8(Si,Al)_{12}(O,OH)_{12}$.
- Appear to be retarded when mineralization is strongly quartz veined, suggesting that some gold is entrapped in inert silica.
- Do not appear to be influenced by iron sulphide content.

A small part of the ore body appears to contain appreciable chalcopyrite mineralization (0.2–0.8% Cu) and the gold associated with this mineralization appears to demonstrate higher gold recoveries along with higher sodium cyanide consumptions. A small portion of the ore body is interpreted to contain erratically distributed coarse grained gold that is slow leaching.

16.1.3 Perseverance

Samples of mineralized material from Perseverance indicate that the nature of the gold mineralization varies and a variety of discrete precious metal phases can be present. Metallic gold and/or silver rich varieties of electrum usually represent the dominant Au bearing phase which may be accompanied by trace amounts of one or more maldonite, aurostibite and rare precious metal tellurides. Pyrrhotite represents the dominant sulphide mineral. It is often associated with subordinate amounts of chalcopyrite and can host electrum particles containing 10-30% by weight of Ag in solution.

The gold mineralization can be classified into four major types; defined by the composition of the metallic gold and electrum as well as the nature of the associated ore minerals. The gold and electrum particles in the ores are commonly fine grained and contain a significant proportion of small particles (<25μm) that are often intimately intergrown with sulphides, notably pyrrhotite and chalcopyrite. Subordinate, but variable amounts of smaller electrum particles (typically <10μm) are also present as refractory inclusions in these sulphides.

16.2 Metallurgical Test work

The PGM life of mine production plan considers the treatment of New Occidental, New Cobar, POX, Peak, Perseverance, Hulk & Hercules (a sub-section of the Perseverance orebody) and Chesney ores.

The plan also includes the treatment of Process Water Dam Material. This material was reclaimed in 2002 from an onsite water dam. The material had accumulated in the dam over a period of time due to issues flocculating the leach and tail thickeners. New Occidental, New Cobar, POX, Peak, Perseverance and a small amount of Hulk & Hercules ores are currently being treated. With the exception of Hulk & Hercules ore, there is a history of plant operating data for all ores currently being treated. Processing of Chesney development ore commenced in late 2008 and stope ore will come on line in early 2009. Following is a summary of relevant metallurgical test work on different ores where available. Ores such as New Occidental and New Cobar have been treated for some time and there is a history of plant operating data. Other ores have had test work performed to establish processing parameters and determine variability in metallurgical responses. Only minor metallurgical test work has been conducted on Hulk & Hercules ore. It has been assumed that the ore will behave similarly to the main Perseverance ore body. Metallurgical test work on the Process Water Dam material has been completed and some material has been processed by blending with New Cobar and stockpile ore. Metallurgical criteria for budgeting purposes are based on a combination of historical data and metallurgical test work on ore samples representative of the treatment plan.

16.2.1 POX

POX ore is presently treated by flotation to produce a copper concentrate. The flotation tails is not treated through the CIL circuit due the high levels of cyanide soluble copper. Test work has been performed to investigate reducing the cyanide soluble copper in the flotation tails to acceptable levels for processing through the CIL plant while producing a minimum 20% copper concentrate. Results indicate a sulphide rougher concentrate can produce a 20% grade copper concentrate with a recovery of around 30% copper. Gold recovery into this concentrate is in the range of 55% to 60%. Only a minor amount of POX is remaining and will be processed in early 2009.

16.2.2 Perseverance

Metallurgical test work on two composites (Type 1 a low copper sample and Type 2 a high copper sample) from the Perseverance deposit indicated the following:

- Total gold recovery for the Type 1 sample was 94% and for the Type 2 sample was 97%.
- Type 1 material has a Bond Work Index of around 17.7kWh/t.
- Type 2 material has a Bond Work Index of around 17.3kWh/t.
- Gravity circuit gold recoveries were 48.3% and 49.2% for Type 1 and 2 respectively.
- Copper and gold recoveries from flotation were 63% and 32.3% respectively for Type 1. For Type 2 the copper and gold recoveries were 70.4% and 68.2% respectively. Higher copper recoveries were achievable with longer residence times.
- Cyanide leaching for 48 hours produced gold recoveries of 79.9% for Type 1 and 80.9% for Type 2. Extending the leach time to 60 hours increased gold recovery to 82.7% and 81.8% for Type 1 and Type 2 respectively.
- Leaching for 48 hours resulted in cyanide consumptions of 2.05kg/t material for Type 1 and 3.2kg/t material for Type 2.

16.2.3 Chesney

Chesney ore consists of oxide, transitional and sulphide rock types. Metallurgical test work on samples of these ore types indicates the following:

- Oxide is mostly suitable as leach feed.
- Transitional samples show very poor flotation copper recovery. Separate metallurgical treatment is required to recover oxidized copper. Gravity gold recovery is acceptable.

- The treatment route for the sulphide rock type varies. Some of it is suitable as plant feed. Other portions will need to be blended for effective treatment.

16.3 Performance and Recovery Predictions

Historical metallurgical operating data is presented in Table 16-1. Annual data for 2004 through 2008 are shown.

Table 16-1: Historical Metallurgical Operating Data

Description	Unit	Annual				
		2004	2005	2006	2007	2008
Grinding						
Tonnes Milled	t	663,441	657,923	702,777	709,244	768,592
Grind Size	p80	72.33	80.85	85.49	75	85
Gold Grade	g/t	7.40	6.97	6.08	5.87	4.67
Copper Grade	%	0.58	0.50	0.58	0.65	0.62
Silver Grade	g/t	6.76	5.69	2.15	6.53	5.80
Utilisation * Changed Calc.	%	95.48%	95.16%	94.26%	97.7%	93.4%*
Throughput rate	tph	79.1	80.7	85.1	89.1	94.1
Gravity						
Au recovered	kg	911	1189.70	1155.43	1208.1	754
Au recovered	oz	29,274	38,250	37,148	37,576	24,246
Recovery	% of feed	18.5	25.9	27.0	28.1	21.0
Flotation						
Cu Conc	t	16,703	12,900	13,651	15,076	17,263
Conc Grade	% Cu	18.19	19.74	21.90	22.5	21.7
Metal Content Cu	t	3037.90	2546.24	2989.81	3397	3,741.7
Conc Grade	g/t Au	79.73	84.56	88.52	74.2	61.2
Metal Content Au	kg	1,331.77	1,090.80	1,208.41	1,118.64	1,055.6
Silver Grade	g/t	126.41	122.74	110.67	122.3	114.8
Metal Content Ag	kg	2,111.44	1,583.33	1,510.85	1,843.79	1981.6
Conc Grade	ppm Bi	5,614.76	3,899.75	3,096.43	3341	3709
Metal Content Bi	kg	93,784.67	50,305.32	42,270.46	50,368.92	64,028
Tail Grade	% Cu	0.13	0.12	0.16	0.17	0.14
	g/t Au	4.13	3.57	2.77	2.71	
Recovery	% Cu	78.69	76.96	73.32	73.9	78.6
	% Au	27.11	23.79	28.27	26.9	29.4
Recovered Au	kg	1,331.77	1,090.80	1,208.41	1,169.2	1055.6
	oz	42,817.43	35,070.04	38,851.23	35,983	33,939
CIL						
Recovered Au	oz	70,609.95	60,092.59	46,586.74	42,929	42,307
Recovery	% Au	44.71	40.76	33.90	32.0	36.6
Final Tail						
Final Tail Grade	g/t Au	0.73	0.68	0.67	0.78	0.62
Final Tail Grade	% Cu	0.13	0.12	0.16	0.17	0.14
Elution						
No of Strips		519.00	561.00	476.00	350	321
Barren Carbon	ppm Au	127.83	185.69	99.08	200	241
	ppm Ag	79.76	109.93	152.67	187	280
Loaded Carbon	ppm Au	2,152.63	2,676.07	2,376.73	2828	3304

	ppm Ag	779.61	958.43	1,275.50	1043	1263
Au Strip Efficiency	%	94.08	93.35	95.39	93.4	92.7

Between 2004 and 2008 plant throughput has increased from around 79tph to 94tph or 19%. Overall gold recovery has been between 87 to 90% and copper recovery has been between 73.3 to 78.7%, dependant on POX throughput.

Gold recovery in the gravity circuit has increased from an average of 18.5% in 2004 to 22% in 2008. In 2006, Gemini tables in the gravity circuit were replaced with an intensive leach reactor and this could explain the slight improvements. Between 2004 and 2008 copper concentrate grades have ranged from around 18% Cu to 22.5% Cu. Copper feed grades were consistently between 0.5% Cu and 0.58% Cu for 2004 to 2006. 2008 saw a slight increase to 0.62% Cu. Copper flotation recoveries have decreased slightly since 2004 but are in line with increases in concentrate grades achieved during the same time frame. Gold recoveries in the flotation concentrate are consistently in the mid to high 20% values. High bismuth concentrations are consistently reported in copper concentrates. Bismuth is a penalty element and concentrations in excess of 500ppm incur penalties in concentrate treatment charges. Peak has a policy of blending concentrates in an attempt to control bismuth levels in concentrates for marketing.

Gold recovery in the CIL circuit has decreased from around 45% Au in 2004 to 36.6% Au in 2008. During the same time frame, gold content in the plant feed has decreased from around 7.4g/t Au to 4.67g/t Au. The reported recoveries are in line with the decrease in head grade.

Elution circuit efficiencies have been between 92.7% and 95.4% between 2004 and 2008. Processing plant throughput for 2009 is planned to be 752,350 tonnes. The 2009 planned throughput is similar to 2008 throughput. Flash flotation is expected to be installed to assist with higher copper head grades and resulting volume of concentrates.

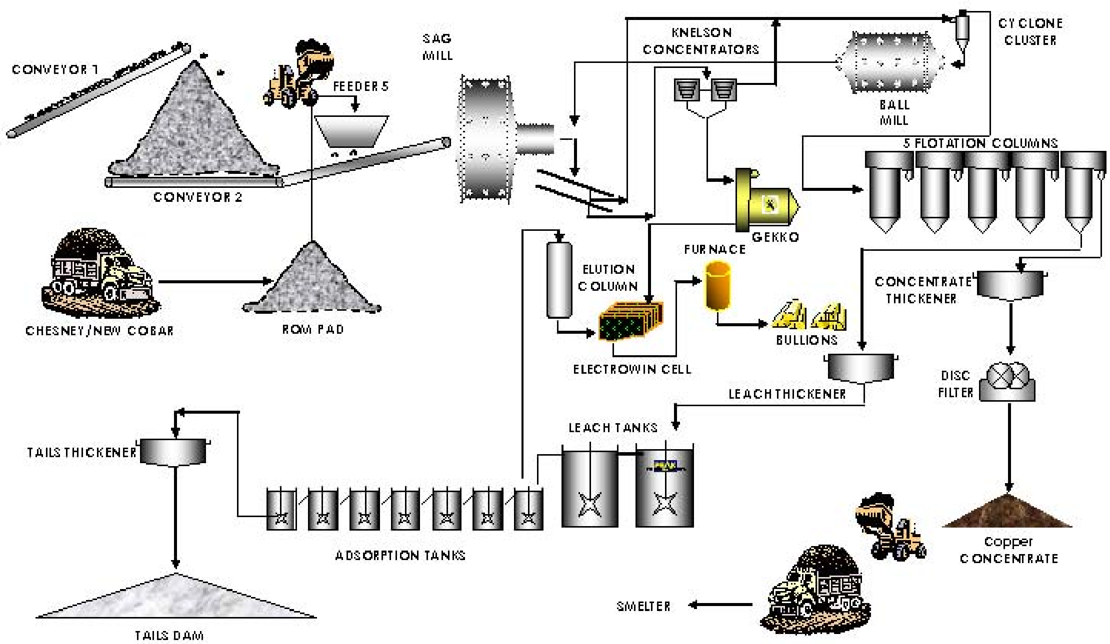
The life of mine plan considers treating New Occidental, Perseverance, Hulk & Hercules, Great Cobar, New Cobar (Underground), Chesney sulphide and Peak ores. Also considered is the Process Water Dam Material which has limited tonnage.

Budgeted metallurgical recoveries are determined from a combination of operating history and, where necessary, metallurgical test work. Recoveries for New Occidental, Perseverance, New Cobar (Underground) and Peak ores are predominantly based on historical operating data. Peak has a system which considers the metallurgy of the individual ores and then produces a predicted recovery based on the feed mixes. The predicted metallurgical recoveries for New Occidental, Perseverance, New Cobar (Underground) and Peak ores are consistent with those from historical operating data.

16.4 Process Flow sheet

A schematic diagram of the Peak Gold Mine processing route is shown in Figure 16-1

Figure 16-1: Schematic of Peak Gold Mine Processing Route



Processing ore through the Peak Gold Mine processing plant involves grinding, cyclone classification, gravity separation, flotation, concentrate filtration, CIL leaching, elution, carbon regeneration, electrowinning and smelting. Feed to the plant is crushed underground; suitable for SAG mill feed.

Ore from the Perseverance and New Occidental deposits is crushed underground and hoisted to the surface via the Peak mine shaft. It is discharged onto a surface stockpile from where it is reclaimed via feeders under the stockpile and fed to the SAG mill.

Ore from the New Cobar and Chesney mines is trucked to a ROM pad stockpile. It is reclaimed by a front end loader and fed via a hopper onto the SAG mill feed conveyor.

Feed rate to the SAG mill is around 95tph. SAG mill discharge feeds onto a double deck vibrating screen with a 16mm top deck screen aperture size and 2mm bottom deck aperture size. Plus 16mm material reports to a scats pile. Minus 2mm material is pumped to Knelson concentrators where a gold concentrate containing free coarse gold is recovered and sent to an intensive leach reactor. The gold rich eluate is then pumped to a dedicated electrowinning cell in the gold room.

Tails from the Knelson concentrator is combined with the -16mm +2mm material from the vibrating screen and pumped to a bank of hydrocyclones. Cyclone underflow reports to a ball mill for further grinding. Ball mill discharge combines with SAG mill discharge. Cyclone overflow with a $P_{80} \approx 75\mu m$ reports to the flotation circuit consisting of five column cells. Flotation concentrate is thickened, filtered and discharged onto concentrate storage pads. From there it is loaded into trucks and transported off site.

Flotation tails reports to a thickener and is then fed to the carbon in leach circuit. In the leach circuit gold is dissolved and loaded onto carbon. The gold loaded carbon is sent to the elution circuit where it is recovered into a gold rich eluate and pumped to the gold room for electrowinning in a dedicated electrowinning cell. The stripped carbon is regenerated and returned to the leach circuit. The leach circuit tails is sent to a thickener and pumped to the tailings dam.

16.5 Plant Design

The original process plant was designed to treat 450,000 tonnes per year (56.3tph) and produce separate copper, lead and zinc concentrates. Gravity gold was originally recovered by a combination of Knelson concentrators and Gemini tables. Leachable gold was treated in a CIL circuit.

In 2001 the plant flow sheet was changed to treat new ores. The upgrade was referred to as the "New Occidental Project". Plant capacity was increased to 600,000 tonnes per year (73tph). As part of this upgrade the grinding circuit was modified to include a ball mill and additional cyclones. No additional flotation capacity was added, however, the duties of the flotation columns were changed to produce a single copper concentrate. Two additional leach tanks were installed for gold leaching.

In 2006 the Gemini tables were removed from the gravity circuit and an Intensive Leach Reactor was added.

The current plant capacity is a nominal 750,000 tonnes per year (92tph). In 2015 it is planned to increase this to 800,000 tonnes per year (96tph). Modifications to the tailings piping system are planned in order to achieve this.

16.6 Smelting

Copper concentrate is not treated at the Peak Gold Mine site. It is sold and smelting/refining is the responsibility of the purchaser.

Pregnant solutions from the CIL elution and gravity circuits are electrowon in separate electrowinning cells. The gold is recovered onto steel wool cathodes. The cathodes are cleaned weekly to remove the gold bearing sludge. The sludge is then filtered and dried. For accounting purposes the dry sludges from the gravity and CIL circuits are smelted individually. Doré bars from the gravity circuit are labelled "PG" while bars from the CIL circuit are labelled "PE".

16.7 Concentrate Specifications

Concentrate is blended by Peak Gold Mine to meet contract specifications. Bismuth is particularly monitored for blending purposes. Concentrate specifications along with a sample of indicative concentrate assays are shown in **Table 16-2**.

Table 16-2: Concentrate Specifications and Assays

Element	Unit	Contract	2008
Ag	ppm	135.5	115
Al_2O_3	%	1.52	
As	ppm	34.5	
Au	ppm	71.02	61
Ba	ppm	18.5	
Bi	ppm	3,650	3709
CaO	%	0.095	
Cd	ppm	26	
Cl	%	0.02	
Co	ppm	106	
Cr	ppm	<5	
Cu	%	16	21.7
F	ppm	280	
Fe_2O_3	%	53.9	
MgO	%	0.36	
MnO	%	0.04	
Mo	ppm	47.5	
Na_2O	%	0.06	
Ni	ppm	31.5	
P_2O_5	%	0.04	
Pb	%	2.715	2.03
S	%	30.9	
Sb	ppm	28.5	
SiO_2	%	7.93	

Sr	ppm	<10	
TiO_2	%	0.04	
V	ppm	<10	
Zn	%	0.88	2.12
Zr	ppm	13.5	

Complete minor element assays are not available for the shipped concentrates and as such a complete comparison can not be made. Based on the limited assays above, the minimum gold, copper and silver contents are being exceeded in the produced concentrates. Zinc content is higher than permitted. The combination of lead and zinc contents will incur penalties if their combined assays are in excess of 2.5%. Bismuth levels in the concentrates are high and incur penalties for levels in excess of 500ppm.

16.8 Ramp-up Schedule

The Peak processing plant has been in operation for around 17 years. Minor increases in throughput are planned for in 2009. These are planned to be achieved by modifying plant pumping and piping at tails.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

PGM currently has six deposits that have published Mineral Resources and/or Mineral Reserves within the Cobar gold field. The deposits are as follows:

- New Cobar;
- Chesney Oxide and Sulphide (2 separate resource estimates);
- New Occidental;
- Peak; and
- Perseverance.

All of the PGM resources have been estimated using similar 3D methods utilising either Ordinary Kriging (OK) or Multiple Indicator Kriging (MIK) estimation methodologies.
For the MIK estimates, the E-type mean or whole block grade was used to report the gold grades. The resources have been estimated either by site personnel or through the use of consultants Hellman & Schofield Pty Ltd (H&S). PGM also use H & S to audit their resource estimations.

The estimation and classification of the resources by PGM are in accordance with the guidelines set out in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves of December 2004 as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

The resource classification is also consistent with criteria laid out in the Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects of December 2005 (the Instrument) and the classifications adopted by CIM Council in December 11, 2005. The reporting of reserves differs in the use of the term "Proved" in JORC Code rather than "Proven" in the CIM Definition Standards. The reporting of resource classification under the JORC Code and the Canadian NI 43-101 systems are essentially identical, the notable difference being the requirement to report Inferred Mineral Resources separate from the totalled Measured and Indicated Mineral Resources under NI 43-101.

The PGM resource estimates for the New Cobar, Chesney Oxide, Chesney Sulphide, New Occidental, Peak and Perseverance deposits are managed by Mr. Rex Berthelsen who is a member of the Australian Institute of Mining and Metallurgy, and has more than five years experience in the estimation of resources in gold and copper deposits. Mr. Berthelsen is both a "Competent Person" and a "Qualified Person" with respect to the JORC Code and CIM Standards respectively for the Mineral Resource estimates.

17.1 Database Validation

Peak Mine Grid is a metric grid defined to be exactly 15° 30' 40.00" west of AMG (AGD66) grid north. A common point, PK PM1, is located in the Peak Mine core yard. The datum is located 80 metres northwest of the Peak Mine head frame at 10,256.87 PMG RL/256.451 AHD RL. Peak Mine Grid has a mean combined scale factor of exactly 0.999701 over the Cobar area. PMG to AMG and AMG to PMG transformations are described in Table 17-1.

Table 17-1: PMG to MGA (GDA94) Translation

Common Point				
Base Peg	**PMG E**	**PMG N**	**MGA E**	**MGA N**
PK PM1	25,649.975	10,280.005	393569.718	6507097.031
Transformation				
Transformation	**Rotation**	**Scale Factor**	**Origin of Grid Transformed System**	
			MGA E	**MGA N**
PMG to AMG	-15°.31' 38.72201"	0.999696398	371614.525	6490330.826

17.2 Geological Interpretation and Modelling

17.2.1 New Cobar

PGM modelled two mineralised zones for the New Cobar grade estimate, the Main Zone and the Western Zone. Both the Main and West Zones were modelled based upon a combination of geology and grade; the latter is nominally 0.1g/t Au. Underground workings and voids were modelled and coded into the block model for depletion of the resource. The Main zone was further split into 2 sub zones based on dip. The upper part of the Main zone dips east at 80° while the lower part dips steeply east at 87°.

17.2.2 Chesney Oxide

The Chesney Oxide mineralisation was modelled by H&S in 2003 (Van de Heyden, 2003). The estimate covers the upper portion of the historical Chesney underground mine, above the 10,120mRL (~150m below surface). The deposit was modelled with 2 lenses, a thicker main lens and a thin eastern lens, both of which are sub-vertical.

PGM supplied H&S with a preliminary interpretation for the gold mineralisation based on a nominal 0.5g/t Au threshold, while the copper mineralisation was based on a nominal 0.4% Cu threshold. H&S expanded the gold mineralised zone interpretation to a nominal 0.1g/t Au boundary based on statistical analysis. The H&S interpretation of the copper mineralisation was based on 0.1% Cu but was broadly similar to the preliminary interpretation by PGM.

The mineralised zone was further divided into 2 sub-domains. These were the eastern limb which includes the upper eastern limb and lower stem of mineralisation (Domain 1) and the upper western limb (Domain 2). PGM supplied a base of oxidation DTM that was based upon drill-hole interpretation. Underground workings and voids were modelled and coded into the block model for depletion of the resource.

17.2.3 Chesney Sulphide

PGM modelled three mineralised domains. These were the Chesney Main Zone Copper (CHM), the Chesney Main Lode Gold (CHA) and the Eastern Gold Lode (CHE). The CHM domain was defined using a nominal 0.1% copper threshold and modified only when continuity between holes and sections was improved. The CHA domain was modelled based on a combination of a 0.1g/t Au threshold and/or a 25-50 ppm Bi threshold. This provided reasonable continuity between holes and sections and prevented the smearing of higher grades into non gold mineralised areas.

The CHE domain was modelled on a similar basis to the CHA domain. The gold mineralising event is understood to be synchronous in each of the structures. Both structures display similar geochemical signatures. The base of complete oxidation and top of fresh rock were modelled. Only the non oxidized (NOX) mineralised domain was modelled for the underground resource.

Underground workings and voids were modelled and coded into the resource for depletion of the resource.

17.2.4 New Occidental

Four domains were defined for the purpose of resource estimation. These were the New Occidental Main Zone (OXM), Barren Zone (BAR), Gossan Lode (GOS) and Albion Lode (ALB).

OXM was defined in drill core by the presence of crypto-crystalline silica and Fe-rich chlorite alteration. BAR was defined by weakly altered, unmineralised country rock within the OXM domain. GOS was defined geologically by its position on the late bucky type 4 breccia and its proximity with base metal sulphide. It was also defined by gold grade using an approximate lower 0.1g/t Au threshold. ALB occurs as quartz vein hosted gold mineralisation and is located just east of OXM within the Chesney sandstone. Due to limited drilling it is defined by a grade contour of approximately 0.1g/t Au. The lens occurs adjacent to the historical workings, extending from 150 metres to approximately 300 metres vertical depth.

The three dimensional shapes for each domain were based on section and plan interpretation. Underground workings and voids were modelled and coded into the block model for depletion of the resource.

17.2.5 Peak

Six main geological domains were modelled for the purpose of resource estimation based on a combination of geology and grade (Table 17-2). Due to the arcuate geometry of the Copper/Western

Lead Zinc (CWLZ) Lens and East lenses, they were split into sub-lenses to allow for better grade interpolation.

Table 17-2: Peak- Geological Domains

Main Domains	
Main Geological Lens	**Details**
Copper/Western Lead Zinc	Main shear hosted mineralisation
East	Rhyolite hosted mineralisation
Deeps	Continuation of CWLZ lens at depth
Regolith	Mineralisation hosted within weathered rock
Contact	Rhyolite/sediment contact hosted mineralisation
Waste	All other areas
Sub Domains	
Sub-lens	**Details**
East upper	Upper East lens
East lower	Lower East lens
CWLZ North	Northern Copper/Western Lead Zinc lens
CWLZ South	Southern Copper/Western Lead Zinc lens

17.2.6 Perseverance

The Perseverance Zone D gold deposit essentially consists of a number of higher grade shoots within the sub-vertical Perseverance Shear Zone (PSZ). The dimensions of Zone D is approximately 500 metres north-south by 500 metres vertically. Zone D is typically around 10 metres thick, although it can be up to 40 metres thick near Zone A.

Mineralisation tends to occur within the PSZ in the vicinity of a rhyolite unit in the hangingwall. There is no significant mineralisation above the position of the rhyolite but it does tend to continue some distance down plunge from the rhyolite. At the southern end of Zone D, the mineralisation leaves the PSZ and occupies a subsidiary shear splay within the rhyolite unit to the east (in the hangingwall). The interpretation of Zone D used for the resource estimate was based on geology (lithology and stratigraphy) and grade (typically >0.1g/t Au).

The Hulk Lens starts near the rhyolite contact just above Zone D, then swings eastwards to a position close to the fold hinge within the rhyolite and then runs approximately parallel to Zone D at a distance of 80 metres to the north, leaving the rhyolite. The Hulk Lens contains both copper and gold mineralisation. The dimensions of the Hulk Lens is approximately 100 metres long (north-south) by 350 metres vertically and averages around 30 metres in thickness (east-west).

Zone B is a broad zone containing a number of narrow lenses of gold mineralisation entirely within the rhyolite. It occurs on the fold hinge in the rhyolite, between 15 and 90 metres east of Hulk Lens. The dimensions of Zone B is approximately 120 metres north-south by 100 metres vertically and 70 metres thick (east-west).

The Hercules Lens is a zone of copper mineralisation mostly within the eastern limb of the rhyolite, located approximately 100 metres east of Zone B. The dimensions of the Hercules Lens is approximately 175 metres north-south by 150 metres vertically and 60 metres thick (east-west). The interpreted shape of the Hercules Lens is based on geology and copper grade (typically >0.1% Cu).

17.3 Statistical Analysis

For the underground deposits, New Cobar, Chesney Sulphide, New Occidental, Peak and Perseverance, 1 metre composites were used for grade estimation. The raw samples were flagged by the interpreted mineralised domain wireframes and composited within each domain. Composites less than 0.5 metres were discarded. No high grade cuts were applied to the composite data.

For the Chesney Oxide open pit grade estimate, 2 metre composite lengths were used with a minimum composite length of 1 metre. The raw samples were flagged by zone using the domain wireframe models and then composited within each zone.

17.3.1 New Cobar

Data Distribution

Data spacing from the 10,175m RL (approximately 100 metres below surface) down to the 10,040m RL (approximately 240 metres below surface) is nominally 15 metres by 20 metres (northing and elevation) spacing. Below the 10,040m RL to the 9,850m RL, the data is spaced approximately 50 metres by 40 metres based on surface diamond and underground drilling. Below the 9,850m RL and to the limit of data (approximately 9,600m RL), the drilling is spaced roughly 70 metres by 40 metres based on surface diamond drilling.

Statistics

Summary statistics for the Main and Western Zones are shown below in Table 17-3 and Table 17-4. Conditional statistics were generated for the MIK estimation, and are shown in Table 17-5 and Table 17-6.

Table 17-3: New Cobar - Summary Statistics for Main Zone

	Au g/t	Cu %	Pb %	Zn %	Ag g/t	Bi ppm	Fe %	Density t/m³
Number of Data	7063	6978	5698	6352	6683	4571	4592	2735
Minimum:	0.002	0.001	0.001	0.001	0.1	0.5	0.36	2.274
Maximum:	106.579	11.7	9.1	1.823	147	5900	44.04	3.69
Mean:	1.972	0.39	0.041	0.028	2.728	62.657	7.71	2.811
Median	0.362	0.103	0.004	0.01	1.106	20.6	7.43	2.804
Variance:	22.685	0.496	0.052	0.009	25.481	38784.48	12.252	0.009
Coefficient of Variation	2.416	1.806	5.542	3.453	1.85	3.143	0.454	0.035

Table 17-4: New Cobar - Summary Statistics for West Zone

	Au g/t	Cu %	Pb %	Zn %	Ag g/t	Bi ppm	Fe %	Density t/m³
Number of Data	6484	6458	6349	6479	6484	6462	6322	6484
Minimum:	0.005	0.001	0.001	0.001	0.25	2.5	1.04	1.096
Maximum:	175	10.14	7.071	10.924	610	4600	44.7	4.049
Mean:	1.414	0.324	0.028	0.049	2.963	31.136	9.935	2.834
Median	0.17	0.164	0.003	0.012	1	10.36	9.46	2.823
Variance:	38.701	0.229	0.03	0.043	119.42	9707.328	11.84	0.009
Coefficient of Variation	4.401	1.479	6.265	4.24	3.688	3.164	0.346	0.033

Table 17-5: New Cobar – Conditional Statistics for Main Zone 1 metre Gold Composites

Threshold Number	Grade Threshold (g/t Au)	Cumulative Probability	Class Mean (g/t Au)	Class Median (g/t Au)	Mean Above Threshold	Class Data
1	0.02	0.1	0.005	0.006	2.296	651
2	0.06	0.2	0.038	0.04	2.578	651
3	0.12	0.3	0.089	0.089	2.934	652
4	0.24	0.4	0.173	0.17	3.394	651
5	0.44	0.5	0.328	0.326	4.008	652
6	0.76	0.6	0.595	0.599	4.861	651
7	1.336	0.7	1.025	1	6.138	651
8	1.805	0.75	1.545	1.531	7.057	326

9	2.573	0.8	2.154	2.13	8.284	326
10	3.68	0.85	3.121	3.12	9.999	325
11	5.632	0.9	4.525	4.48	12.737	326
12	9.6	0.95	7.432	7.34	18.041	326
13	13.4	0.97	11.29	11	22.519	130
14	22.6	0.99	16.771	15.8	33.842	130
15	66.577	1	33.842	31.1	-99	66

Table 17-6: New Cobar – Conditional Statistics for West Zone 1 metre Gold Composites

Threshold Number	Grade Threshold (g/t Au)	Cumulative Probability	Class Mean (g/t Au)	Class Median (g/t Au)	Mean Above Threshold	Class Data
1	0.015	0.1	0.004	0.005	1.293	1307
2	0.04	0.2	0.026	0.027	1.452	1308
3	0.07	0.3	0.055	0.054	1.651	1308
4	0.12	0.4	0.09	0.09	1.911	1308
5	0.19	0.5	0.15	0.15	2.264	1308
6	0.309	0.6	0.243	0.24	2.769	1307
7	0.53	0.7	0.406	0.4	3.556	1308
8	0.72	0.75	0.619	0.615	4.143	654
9	1	0.8	0.852	0.85	4.966	654
10	1.462	0.85	1.212	1.21	6.218	654
11	2.314	0.9	1.822	1.779	8.416	654
12	4.72	0.95	3.287	3.2	13.544	654
13	7.13	0.97	5.752	5.64	18.719	261
14	15.264	0.99	10.319	9.84	35.521	262
15	175	1	35.521	23.5	-99	131

17.3.2 Chesney Oxide

Data Distribution

At Chesney, the topographic surface occurs typically around the 10,270mRL. Shallow mineralisation extending down to approximately the 10,150mRL is drilled on a nominal 20 metre by 20 metre spacing. The hole spacing is wider around the old workings and on the extremities of the deposit to the north and south. The material down to the 10,120mRL has been estimated as part of the Oxide resource.

Within the historic underground levels, back and/or face samples were compiled from original linen plans. These samples are spaced approximately 30 metres apart on sub level intervals. A number of underground diamond holes drilled from 1943 to 1950 intersect the original orebody within the old workings.

Below the base of the old underground workings, drill hole spacing is on a nominal 40 metre by 40 metre pattern from holes generated by several phases of drilling down to 9750mRL. The majority of holes which are used in the estimation in this area are from programs that were completed in 1997, 2004 and 2005. Below 9750mRL, the drilling is considerable more widely spaced.

A drill hole selection file was used to select the drill holes for data analysis and subsequent grade interpolation. The UD61 series drill holes were not included for the Au estimate, as the Au assays were not considered to be representative of the narrow gold shoots due to the samples being composited over wide intervals and a relatively high lower-detection limit for the Au assays. The holes were used however for the Cu estimate.

Statistics

The summary statistics for copper by lens and oxidation are shown in Table 17-7. Table 17-8 contains the summary statistics for Au by zone and oxidation state and for the 2 metre composites. Domaining of the mineralisation has significantly reduced the coefficient of variation (standard deviation divided by mean grade) for mineralisation compared to the total sample population. Comparison of the mean grade between the fresh and transitional zones indicates that the mean gold grade does not vary substantially by oxidation state, and examination of drill hole data shows no obvious changes in grade at oxidation boundaries. H&S considered it appropriate to estimate gold grade without considering the oxidation boundaries.

Table 17-7: Chesney Oxide – Summary Statistics for Copper 2 metre Composites

Zone	Type	Number of Data	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Standard Deviation	Coefficient of Variation
11	Main Fresh	442	0.0012	7.3	0.92	1.00	1.08
12	East Fresh	176	0.005	3.1	0.51	0.54	1.06
13	Rest Fresh	307	0.0005	1.4	0.07	0.11	1.62
21	Main Trans	523	0.0004	16.7	1.11	1.59	1.43
22	East Trans	158	0.0021	8.9	0.52	0.92	1.78
23	Rest Trans	413	0.0006	0.6	0.08	0.08	1.10
30	All Oxide	1566	0.0029	4.0	0.07	0.18	2.63
Total		**3585**	**0.0004**	**16.7**	**0.37**	**0.86**	**2.33**

Table 17-8: Chesney Oxide – Summary Statistics for Gold 2 metre Composites

Zone	Type	Number of Data	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Standard Deviation	Coefficient of Variation
Summary Statistics By Domain							
1	Main Lens	1118	0.0025	23.4	0.65	1.89	2.92
2	East Lens	128	0.005	15.2	1.06	2.26	2.13
3	Remainder	2340	0.0025	1.8	0.04	0.07	2.01
Total		3586	0.0025	23.4	0.26	1.19	4.49
Summary Statistics By Domain and Oxidation State							
11	Main Fresh	442	0.0012	7.3	0.92	1.00	1.08
12	East Fresh	176	0.005	3.1	0.51	0.54	1.06
13	Rest Fresh	307	0.0005	1.4	0.07	0.11	1.62
21	Main Trans.	523	0.0004	16.7	1.11	1.59	1.43
22	East Trans.	158	0.0021	8.9	0.52	0.92	1.78
23	Rest Trans.	413	0.0006	0.6	0.08	0.08	1.10
30	All Oxide	1566	0.0029	4.0	0.07	0.18	2.63
Total		**3585**	**0.0004**	**16.7**	**0.37**	**0.86**	**2.33**

Conditional statistics (Table 17-9) were generated based on deciles to 70%, then at 5% intervals to 95%, then 97 and 99% for use with MIK estimation.

Table 17-9: Chesney Oxide - Conditional Statistics for Main Lens Gold 2 metre Composites

Threshold Number	Grade Threshold (g/t Au)	Cumulative Probability	Class Mean (g/t Au)	Class Median (g/t Au)	Class Data
1	0.010	0.10	0.007	0.005	111
2	0.040	0.20	0.028	0.030	112
3	0.090	0.30	0.064	0.060	112
4	0.130	0.40	0.111	0.110	112
5	0.180	0.50	0.151	0.150	112
6	0.250	0.60	0.216	0.220	111
7	0.370	0.70	0.310	0.310	112
8	0.465	0.75	0.409	0.405	56
9	0.580	0.80	0.521	0.520	56
10	0.780	0.85	0.676	0.680	56
11	1.23	0.90	0.990	0.990	56
12	2.53	0.95	1.80	1.80	56
13	3.69	0.97	3.14	3.21	22
14	8.51	0.99	5.54	5.22	22
15	23.4	1.00	15.7	17.4	12

17.3.3 Chesney Sulphide

Statistics

Summary statistics for the 1 metre composites from the mineralised zones used for the resource estimation are shown in Table 17-10. Multivariate analysis of the composite data indicated that Au does not exhibit a significant correlation with the other elements, although a general spatial relationship exists between Au and Bi (see Table 17-11).

There is a weak relationship between the Pb and Zn, but these elements are present in minor quantities only. The elemental pair which has the highest correlation is Cu and Ag. This relationship is also notable at Gladstone to the west of Chesney. The relationship between Au and Bi is weak in the eastern lens, but more prominent than that of the main lens. The most significant relationships identified were between Ag-Cu and Bi–Cu.

Table 17-10: Chesney Sulphide- Summary Statistics for 1metre Composite Data

CHA – Main Lode Gold Domain							
	Density (t/m^3)	Au (g/t)	Cu (%)	Pb (%)	Zn (ppm)	Ag (ppm)	Bi (ppm)
Number of Data	892	5839	5859	1892	1892	1892	1642
Minimum	2.16	0.002	0	0	0.001	0.1	0.5
Maximum	3.44	186.911	11.3	0.74	0.875	37	3650
Mean	2.86	1.68	1.68	0.02	0.03	4.46	84.07
Standard Deviation	0.102	6.985	1.440	0.046	0.049	4.70	186
Coefficient of Variation	0.036	4.167	0.859	2.469	1.823	1.054	2.212
CHM – Main Lode Copper Domain							
	Density (t/m^3)	Au (g/t)	Cu (%)	Pb (%)	Zn (ppm)	Ag (ppm)	Bi (ppm)
Number of Data	357	527	527	494	494	494	487
Minimum	2.4	0.005	0.005	0	0.001	0.1	0.5
Maximum	3.135	31.1	6.1	0.331	0.13	11	279
Mean	2.84	0.11	0.82	0.00	0.01	2.30	21.97
Standard Deviation	0.066	1.376	0.830	0.017	0.010	1.60	24
Coefficient of Variation	0.023	12.694	1.010	3.439	0.652	0.698	1.100

CHE – Eastern Gold Lens							
	Density (t/m^3)	Au (g/t)	Cu (%)	Pb (%)	Zn (ppm)	Ag (ppm)	Bi (ppm)
Number of Data	271	491	491	483	483	483	470
Minimum	2.667	0.005	0	0	0	0.05	0.5
Maximum	3.424	32.5	6.22	0.285	0.558	25	2130
Mean	2.88	0.86	0.68	0.01	0.02	3.15	182.61
Standard Deviation	0.088	3.221	0.815	0.025	0.027	3.30	273
Coefficient of Variation	0.031	3.727	1.205	2.194	1.665	1.050	1.497

Table 17-11: Chesney Sulphide - Correlation Matrix for 1metre Composites

Chesney Main Lens						
	Density	***Au***	***Cu***	***Pb***	***Zn***	***Ag***
Au	0.20					
Cu	0.21	0.09				
Pb	-0.04	0.09	0.04			
Zn	0.10	0.12	0.17	0.55		
Ag	0.14	0.14	0.62	0.23	0.30	
Bi	0.15	0.04	0.27	-0.02	-0.01	0.06
Chesney Eastern Gold Lens						
	Density	***Au***	***Cu***	***Pb***	***Zn***	***Ag***
Au	0.003					
Cu	0.33	0.02				
Pb	0.07	-0.01	0.04			
Zn	0.19	-0.01	0.07	0.22		
Ag	0.31	0.14	0.83	0.07	0.10	
Bi	0.23	0.34	0.31	0.05	-0.01	0.42

Conditional statistics were also conducted for the Chesney Main Lens (CHA) for use with MIK estimation (Table 17-12).

Table 17-12: Chesney Sulphide – Conditional Statistics for Chesney Main Lens 1 metre Gold Composites

Threshold Number	Grade Threshold (g/t Au)	Cumulative Probability	Class Mean (g/t Au)	Class Median (g/t Au)	Class Data
1	0.011	0.1	0.006	0.005	372
2	0.061	0.2	0.017	0.02	373
3	0.081	0.3	0.04	0.04	374
4	0.139	0.4	0.079	0.08	370
5	0.232	0.5	0.128	0.13	373
6	0.353	0.6	0.184	0.18	372
7	0.547	0.7	0.281	0.281	373
8	0.778	0.75	0.384	0.38	186
9	1.041	0.8	0.499	0.49	184
10	1.505	0.85	0.658	0.65	187
11	2.561	0.9	0.936	0.92	183
12	4.884	0.95	1.645	1.61	185
13	8.661	0.97	2.855	2.74	74
14	16.322	0.99	5.357	5.19	74
15	40.114	1	18.991	13.9	37
				Total	**3717**

17.3.4 New Occidental

Data Distribution

Data distribution varies throughout the deposit. From the 106 Level (1,050 metres below surface) up to the base of old underground workings; the drill-hole data has a nominal spacing of 15 metres by 20 metres (northing and elevation). Below the 106 Level, the data has a nominal spacing of approximately 50 metres by 75 metres for the next 100 metres.

Statistics

Data analysis and grade estimation for New Occidental was performed based upon 1 metre composites. Summary statistics for Au are shown in Table 17-13 and summary statistics for the base metals and density are shown in Table 17-14. Analysis of the multivariate statistics in Table 17-15 indicates that the Main Lens has moderate Au-Bi and Pb-Ag relationships as well as a moderate to weak Pb-Zn relationship.

Table 17-13: New Occidental – Gold Summary Statistics for 1 metre Composites

	Zone			
Statistic	**OXM**	**GOS**	**ALB**	**BAR**
Number of Data	6385	1023	299	197
Minimum (g/t Au)	0.002	0	0.005	0.005
Maximum (g/t Au)	206.63	524	254.686	11.4
Mean (g/t Au)	6.219	3.854	3.384	0.615
Median (g/t Au)	2.3	0.6	0.144	0.32
Variance	140.015	444.111	323.415	1.529
Coefficient of Variation	1.903	5.469	5.315	2.011

Table 17-14: New Occidental – Base Metal and Density Summary Statistics for 1 metre Composites

Statistic	Cu (%)	Pb (%)	Zn (%)	Ag (ppm)	Bi (ppm)	Density (t/m^3)
Number of Data	22940	19590	22722	15947	18797	18697
Minimum	0	0.001	0.001	0.003	0.5	2.166
Maximum	7.6	5.027	4.1	566	27000	3.568
Mean	0.117	0.02	0.019	2.12	269.908	2.777
Variance	0.023	0.003	0.008	1	75	2.776
Coefficient of Variation	0.092	0.016	0.007	39.101	330848.3	0.003

Conditional statistics by lens (waste, OXM, GOS), for later use with the Au MIK estimation, were performed using 14 cumulative grade probability thresholds based on deciles to 70%, 5% intervals to 95%, then at 97% and 99%. Additional thresholds at the upper spectrum of the Au range were used to better define the high grade tail of the Au distribution. The conditional statistics for the Main Lens (OXM) and the Gossan Lens (GOS) are shown in Table 17-16 and in Table 17-17 respectively.

Table 17-15: New Occidental – Correlation Matrices for 1 metre Composites

All Data							
	Density	**Cu**	**Bi**	**Pb**	**Zn**	**Ag**	**Au**
Density							
Cu	0.108364						
Bi	0.14391	0.099072					
Pb	0.064008	0.275364	0.112322				
Zn	0.048322	0.08603	0.07771	0.352282			
Ag	0.078973	0.308789	0.212112	**0.515413**	0.202084		
Au	0.008719	0.043817	**0.44249**	0.07707	0.064044	0.151798	
Main Lens (OXM)							
	Density	**Cu**	**Bi**	**Pb**	**Zn**	**Ag**	**Au**
Density							
Cu	0.108364						
Bi	0.14391	0.099072					
Pb	0.064008	0.275364	**0.112322**				
Zn	0.048322	0.08603	**0.07771**	0.352282			
Ag	0.078973	0.308789	**0.212112**	0.515413	0.202084		
Au	0.008719	0.043817	**0.44249**	0.07707	0.064044	0.151798	

Table 17-16: New Occidental – Conditional Statistics for Main Lens 1 metre Gold Composites

Threshold Number	Grade Threshold (g/t)	Cumulative Probability	Class Mean (g/t)	Class Median (g/t)	Class Data
1	0.15	0.1	0.067	0.07	638
2	0.47	0.2	0.285	0.28	639
3	0.89	0.3	0.667	0.65	638
4	1.5	0.4	1.178	1.18	639
5	2.3	0.5	1.858	1.85	638
6	3.47	0.6	2.864	2.85	639
7	5.25	0.7	4.306	4.286	638
8	6.55	0.75	5.86	5.82	319
9	8.5	0.8	7.468	7.4	320
10	11.3	0.85	9.918	9.96	319
11	16	0.9	13.361	13.2	319
12	24.8	0.95	19.754	19.6	319
13	32.8	0.97	28.592	28.5	128
14	58.9	0.99	42.344	41.1	128
15	206.63	1	85.553	73.7	64
				total	6385

17.3.5 Peak

Data Distribution

Data distribution varies throughout the deposit. From the 715 Level (9,540m RL, approximately 715 metres below surface) up to the 300 Level (9,950m RL, approximately 300 metres below surface) the data spacing is nominally 10 metre by 12 metre (northing and elevation). Below the 715 Level, the data is nominally on a 50 metre by 50 metre spacing. From the 300 Level up to the base of oxidation (at approximately 10,150m RL – 110 metres below surface) the data is spaced from 25 metres by 25 metres up to 50 metres by 50 metres. Above the base of oxidation through to surface the data is spaced at approximately 20 metres by 20 metres.

Statistics

Table 17-17 : Peak – Summary Statistics for 1m Gold Composites

	Au (g/t)						
	All	Waste	Regolith	Contact	East	Copper	Deeps
Number of samples	52,655	8,680	383	38,577	48,087	2,601	52,655
Minimum	0.00	0.00	0.01	0.00	0.00	0.01	0.00
Maximum	193.50	143.00	1,680.00	1,052.34	2,269.65	275.00	193.50
Mean	0.08	0.29	9.18	1.35	2.69	0.59	0.08
Median	0.01	0.04	0.05	0.07	0.12	0.04	0.01
Variance	1.27	4.10	7,913.03	128.75	539.11	43.89	1.27
Coefficient of Variation	14.59	6.88	9.69	8.40	8.64	11.27	14.59

Drill-hole data for Au was composited into the Waste, Regolith, Contact, East, Copper and Deeps domains. Cu, Pb, Zn and Ag were composited separately into Waste, Regolith, East and a combined Copper/Deeps domain. Density data was composited separately into Waste, East and a combined Copper/Deeps domain. There was no bulk density data for the Regolith domain.

Conditional statistics for the CWLZ and East lenses, for use with the Au MIK estimation, were generated using 14 cumulative grade probability thresholds based on deciles to 70%, 5% intervals to 95%, then at 97% and 99%. Additional thresholds at the upper spectrum of the Au range were used to better define the high grade tail of the Au distribution. The conditional statistics for the CWLZ and East lenses are shown in:

Table 17-18: Peak – Summary Statistics for 1m Gold Composites

	Au (g/t)						
	All	Waste	Regolith	Contact	East	Copper	Deeps
Number of samples	52,655	8,680	383	38,577	48,087	2,601	52,655
Minimum	0.00	0.00	0.01	0.00	0.00	0.01	0.00
Maximum	193.50	143.00	1,680.00	1,052.34	2,269.65	275.00	193.50
Mean	0.08	0.29	9.18	1.35	2.69	0.59	0.08
Median	0.01	0.04	0.05	0.07	0.12	0.04	0.01
Variance	1.27	4.10	7,913.03	128.75	539.11	43.89	1.27
Coefficient of Variation	14.59	6.88	9.69	8.40	8.64	11.27	14.59

Table 17-19 : Peak – Summary Statistics for 1m Composites by domain

Waste Domain						
	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Density (t/m³)
Number of samples	52,655	52,030	51,337	53,759	51,051	5,866
Minimum	0.001	0.001	0.001	0.001	0.01	2.097
Maximum	193.5	9.353	40	34.655	456.916	3.852
Mean	0.077	0.044	0.078	0.085	1.935	2.774
Median	0.01	0.005	0.004	0.011	1.487	2.77
Variance	1.267	0.038	0.181	0.258	31.543	0.008
Coefficient of Variation	14.587	4.41	5.44	5.964	2.902	0.033

Peak East						
	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Density (t/m³)
Number of samples	38,577	38,567	38,562	38,564	36,694	4,535
Minimum	0.001	0.001	0.001	0.001	0.01	2.151
Maximum	1052.344	14.5	24.5	44.474	1065	4.495
Mean	1.351	0.261	0.347	0.432	6.09	2.802
Median	0.07	0.07	0.034	0.02	2.501	2.75
Variance	128.748	0.353	0.912	3.66	669.809	0.04
Coefficient of Variation	8.4	2.278	2.755	4.425	4.25	0.071

Copper / Western Lead Zinc						
	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Density (t/m³)
Number of samples	48,087	48,041	47,954	47,931	43,379	14,632
Minimum	0.001	0.001	0.001	0.001	0.01	2.069
Maximum	2269.651	15.3	45	43.9	740.948	4.405
Mean	2.688	0.326	0.425	0.514	4.449	2.783
Median	0.12	0.06	0.077	0.08	2.501	2.759
Variance	539.11	0.602	1.486	2.16	201.887	0.01
Coefficient of Variation	8.638	2.381	2.871	2.861	3.194	0.036

Peak Deeps						
	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Density (t/m³)
Number of samples	2,601	2,597	2,591	2,601	2,601	1,834
Minimum	0.005	0.001	0.001	0.001	0.02	2.281
Maximum	275	6.26	14.5	23	74	3.538
Mean	0.588	0.397	0.318	0.523	3.113	2.797
Median	0.04	0.14	0.045	0.094	2	2.772
Variance	43.889	0.4	0.802	1.706	18.165	0.011
Coefficient of Variation	11.265	1.593	2.813	2.498	1.369	0.037

Regolith						
	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Density (t/m³)
Number of samples	8,680	8,349	8,377	8,439	8,360	841
Minimum	0.002	0.001	0.001	0.001	0.25	2.245
Maximum	143	3.5	7.65	1.01	2700	3.054
Mean	0.294	0.02	0.048	0.028	6.091	2.665
Median	0.04	0.007	0.01	0.016	1	2.697
Variance	4.099	0.01	0.049	0.003	2827.76	0.018
Coefficient of Variation	6.879	4.908	4.606	1.758	8.73	0.05

Table 17-20: Peak - Conditional Statistics 1m Gold Composites

CWLZ Lens					
Threshold Number	Grade Threshold (g/t)	Cumulative Probability	Class Mean (g/t)	Class Median (g/t)	Class Data
1	0.01	0.1	0.005	0.005	4782
2	0.02	0.2	0.015	0.013	4783
3	0.04	0.3	0.032	0.03	4782
4	0.072	0.4	0.057	0.06	4783
5	0.12	0.5	0.096	0.099	4782
6	0.22	0.6	0.165	0.16	4783
7	0.421	0.7	0.306	0.3	4782
8	0.62	0.75	0.514	0.51	2391
9	0.94	0.8	0.765	0.76	2392
10	1.629	0.85	1.238	1.21	2391
11	3.236	0.9	2.281	2.21	2391
12	9.284	0.95	5.44	5.11	2391
13	17.6	0.97	12.809	12.5	957
14	49.4	0.99	28.851	26.8	956
15	2269.651	1	128.813	76.655	479
Eastern Gold Lens					
Threshold Number	Grade Threshold (g/t)	Cumulative Probability	Class Mean (g/t)	Class Median (g/t)	Class Data
1	0.005	0.10	0.005	0.005	3838
2	0.010	0.20	0.007	0.006	3838
3	0.020	0.30	0.015	0.010	3838
4	0.040	0.40	0.030	0.030	3838
5	0.070	0.50	0.053	0.050	3838
6	0.120	0.60	0.090	0.090	3838
7	0.210	0.70	0.157	0.150	3837
8	0.290	0.75	0.249	0.250	1920
9	0.450	0.80	0.363	0.360	1919
10	0.760	0.85	0.587	0.580	1919
11	1.450	0.90	1.049	1.020	1918
12	3.960	0.95	2.393	2.240	1919
13	7.590	0.97	5.410	5.220	768
14	24.10	0.99	13.284	12.420	768
15	1052.34	1.00	72.588	48.000	384

17.3.6 Perseverance

Data Distribution

The data spacing at Zone D is irregular and varies widely, from 20 metres by 40 metres in the better drilled upper areas to 80 metres by 40 metres in areas down dip and along strike or wider in other areas. There are some wide gaps in the drilling. Zone A is defined in the new model as the area north of 9800mN and above 9060mRL. Some Zone A intersections were included with the Zone D data because the Perseverance lode at Zone A is continuous with Zone D. The data spacing at the Hulk Lens varies from around 15 metres by 20 metres in better drilled areas to 30 metres by 60 metres or wider in other areas. Zone B has some very close spaced drilling because some fans of holes directed at Zones A and D and the Hulk Lens started inside Zone B. Otherwise, the drill hole spacing is approximately 20 metres by 20 metres in the better drilled areas of Zone B. Hole spacing is approximately 20 metres by 30 metres in the central part of the Hercules Lens, with no drilling outside of this area.

Statistics

Summary statistics for the 1 metre Au composites from the modelled zones are shown in Table 17-21. Conditional statistics based on deciles to 70%, then at 5% intervals to 95%, then 97% and 99% were generated for use during the MIK estimation from Zone D (Table 17-22). Based on analysis of the summary statistics and composite histograms, a high grade cut to the Au data for the OK estimations for Zones other than Zone D. For estimation purposes, Zone D was further split into two separate domains; a northern end which had a steep easterly dip, and a southern end which has a steep westerly dip that follows the rhyolite contact. Table 17-23 summarises the high grade cuts for Au data that were used. No high grade cuts were applied for the estimation of the other elements. Table 17-24 shows the summary statistics by zone for the other items estimated in the model (Cu, Ag, Pb, Zn, Bi, Fe and Density).

Table 17-21: Perseverance - Summary Statistics for 1m Gold Composites

Zone	Number of data	Minimum	Maximum	Average	Median	Variance	Coefficient of Variation
Waste	24,326	0.001	1,730.00	0.18	0.01	126.20	63.62
Zone B	4,089	0.002	8,228.16	3.00	0.01	17,775.77	44.41
Hulk	3,287	0.005	2,711.00	2.11	0.06	2,300.63	22.77
Zone D	2,043	0.005	2,960.00	6.43	0.13	7,061.78	13.07
Zone A	10,087	0.001	2,701.54	2.93	0.12	1,243.63	12.03
Hercules	1,183	0.003	59.18	0.21	0.02	4.44	10.01

Table 17-22: Perseverance – Zone D Conditional Statistics 1m Gold Composites

Threshold Number	Cumulative Probability	North Domain				South Domain			
		Grade Threshold (g/t Au)	Class Mean (g/t Au)	Class Median (g/t Au)	Class Data	Grade Threshold (g/t Au)	Class Mean (g/t Au)	Class Median (g/t Au)	Class Data
1	0.1	0.005	0.005	0.005	154	0.01	0.006	0.005	141
2	0.199	0.016	0.009	0.01	154	0.03	0.019	0.02	141
3	0.3	0.03	0.022	0.02	155	0.05	0.037	0.04	141
4	0.4	0.047	0.036	0.037	154	0.07	0.058	0.06	142
5	0.5	0.076	0.06	0.06	155	0.12	0.095	0.09	141
6	0.6	0.132	0.101	0.1	154	0.2	0.157	0.16	141
7	0.699	0.25	0.184	0.18	154	0.36	0.267	0.26	142
8	0.75	0.375	0.312	0.31	78	0.5	0.424	0.422	70
9	0.8	0.61	0.481	0.477	77	0.92	0.672	0.64	71
10	0.85	1.02	0.78	0.77	77	1.609	1.236	1.22	71
11	0.9	1.99	1.37	1.33	77	3.52	2.362	2.225	70
12	0.949	9.418	4.749	4.1	77	8.05	5.744	5.45	71
13	0.97	23.104	15.344	14.3	31	14.803	11.498	11.764	28
14	0.99	52.84	35.572	32.204	31	49.675	27.108	22.511	28
15	1	2960	481.767	108	16	579	173.401	115	15

Table 17-23: Perseverance – High Grade Cuts Used for 1m Composite Gold Data

ZONE	1	2	8/9	12
Top 10 Sample Au Grades (g/t)	Waste	Zone B	Hulk	Hercules
1	1730	8228.16	2711	59.18
2	236	2173.87	274	28.40
3	49.4	353	175	25.06
4	43.3	324	127.98	7.20
5	42.9	171.44	125.15	7.14
6	37.6	99.3	123	7.06
7	35.7	86.4	94.61	6.1
8	35.7	57.4	76.5	5.58
9	33.5	55	69.98	5.07
10	30	53.5	67.42	5.03
Total Number of Data	24,326	4,089	3,287	1,183
High Grade Cut (Au g/t)	30	100	50	10
Number of Composites Cut	10	5	11	3
Percentage of Composites Cut	0.04%	0.12%	0.33%	0.25%

Table 17-24: Perseverance – Summary Statistics for 1 metre Composites by Zone

Data	Zone	Number of samples	Minimum	Maximum	Mean	Median	Variance	Coefficient of Variation
Au g/t	Waste	24,326	0.00	1730.00	0.18	0.01	126.20	63.62
	Zone B	4,089	0.00	8228.16	3.00	0.01	17775.77	44.41
	Hulk	3,287	0.01	2711.00	2.11	0.06	2300.63	22.77
	Zone D	2,043	0.01	2960.00	6.43	0.13	7061.78	13.07
	Zone A	10,087	0.00	2701.54	2.93	0.12	1243.63	12.03
	Hercules	1,183	0.00	59.18	0.21	0.02	4.44	10.01
Cu %	Waste	24,293	0.00	15.80	0.15	0.03	0.17	2.70
	Zone B	4,290	0.00	6.25	0.18	0.07	0.08	1.63
	Hulk	3,288	0.00	15.37	0.75	0.35	1.47	1.61
	Zone D	2,042	0.00	8.50	0.53	0.22	0.72	1.60
	Zone A	10,102	0.00	13.90	0.87	0.36	1.72	1.50
	Hercules	1,199	0.00	16.41	0.66	0.25	1.33	1.74
Pb %	Waste	22,813	0.00	21.40	0.06	0.00	0.13	5.46
	Zone B	4,280	0.00	5.00	0.12	0.03	0.08	2.36
	Hulk	3,282	0.00	17.96	0.16	0.02	0.29	3.45
	Zone D	2,035	0.00	39.70	0.13	0.01	1.07	7.76
	Zone A	9,222	0.00	14.45	0.10	0.01	0.22	4.86
	Hercules	1,199	0.00	16.51	0.41	0.16	0.88	2.29
Zn %	Waste	24,678	0.00	21.40	0.08	0.01	0.13	4.65
	Zone B	4,304	0.00	10.40	0.16	0.03	0.16	2.60
	Hulk	3,288	0.00	10.50	0.15	0.02	0.24	3.18
	Zone D	2,043	0.00	22.00	0.13	0.02	0.46	5.06
	Zone A	9,950	0.00	17.60	0.06	0.01	0.13	6.54
	Hercules	1,199	0.00	19.60	0.40	0.18	1.05	2.55
Ag g/t	Waste	24,818	0.10	516.00	1.61	1.00	27.42	3.25
	Zone B	4,307	0.10	348.55	2.74	1.00	50.53	2.60
	Hulk	3,288	0.10	210.00	8.39	4.00	210.35	1.73
	Zone D	2,043	0.10	593.00	5.65	2.00	369.89	3.40
	Zone A	10,036	0.10	1595.00	5.72	1.80	394.27	3.47
	Hercules	1,199	0.29	688.22	10.41	4.04	769.77	2.66
Bi ppm	Waste	23,858	0.50	1810.00	12.57	5.00	944.14	2.44
	Zone B	4,269	5.00	720.00	19.46	5.00	1338.51	1.88
	Hulk	3,288	0.50	11000.00	93.62	40.00	100458.88	3.39
	Zone D	2,043	0.50	4800.00	40.75	10.00	24977.25	3.88
	Zone A	8,740	0.50	3539.69	31.82	5.00	11817.02	3.42
	Hercules	1,199	5.00	1296.72	18.88	5.01	3581.18	3.17
Fe %	Waste	18,833	0.00	43.17	5.02	4.52	6.78	0.52
	Zone B	2,849	0.74	21.46	5.23	4.67	7.39	0.52
	Hulk	3,288	1.19	49.41	9.48	8.21	28.95	0.57
	Zone D	1,877	1.28	35.70	6.70	6.01	9.48	0.46
	Zone A	4,166	0.00	45.60	7.00	6.28	12.64	0.51
	Hercules	1,092	1.45	29.87	6.96	6.17	12.53	0.51
Density t/m³	Waste	16,064	2.03	3.94	2.78	2.78	0.01	0.03
	Zone B	3,574	2.01	3.77	2.75	2.75	0.00	0.02
	Hulk	3,016	2.39	4.09	2.85	2.82	0.02	0.05
	Zone D	1,817	2.21	3.88	2.80	2.79	0.01	0.04
	Zone A	9,272	2.37	4.12	2.83	2.81	0.01	0.04
	Hercules	1,158	2.44	4.85	2.82	2.80	0.02	0.05

17.4 Variography

17.4.1 New Cobar

Indicator and grade variography was performed on the Au 1 metre composite data for the New Cobar Main Zone, Lower and West Zones using both spherical and exponential models. Grade variography was carried out for the Jubilee Lode and the surrounding waste hangingwall and footwall areas. The grade variography is summarised in Table 17-25 and the indicator variography is shown in Table 17-26.

Table 17-25: New Cobar - Grade Variography for the Main and Lower Zones

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range	GS3 Rotation
Au (Main)	Nugget	0.1							z,y,x
	sph	0.75	95	-80	0	32	23	5	0
	sph	0.15	95	-80	0	100	76	42	10
									5
Au (West)	Nugget	0.2							z,y,x
	exp	0.42	44	-68.91	-44	35	13	3	0
	exp	0.28	44	-68.91	-44	20	20	7	10
	exp	0.1	44	-68.91	-44	44	24	14	5
Cu	Nugget	0.1							
	exp	0.27	90	-85	0	145	79	26	0
	exp	0.5	90	-85	0	7	3	5	5
	exp	0.13	90	-85	0	64	636	211	0
Pb	Nugget	0.17							
	Sph 1	0.13	90	-85	0	28	61	6	0
	Sph 2	0.04	90	-85	0	188	181	18	5
	Sph3	0.66	90	-85	0	3	20	4	0
Zn	Nugget	0.1							
	Sph 1	0.31	90	-85	0	11	12	2	0
	Sph 2	0.59	90	-85	0	2	24	1.5	5
									0
Ag	Nugget	0.13							
	Sph	0.2	73	-80	0	29	39	30	-17
	Sph	0.47	73	-80	0	3	2	2	10
	Sph 3	0.2	73	-80	0	154	505	50	0
Bi	Nugget	0.15							
	Exp 1	0.12	75	-85	0	3	5	5	-15
	Exp 2	0.66	75	-85	0	41	6	4	5
	Exp 3	0.07	75	-85	0	149	387	38	0
SG	Nugget	0.1							
	Exp 1	0.56	85	-80	0	14	9.5	5.5	-5
	Exp 2	0.13	85	-80	0	171	108	370	10
	Exp 3	0.21	85	-80	0	514	152	52	0
Fe	Nugget	0.1							
	Sph 1	0.45	90	-85	0	187	261	104	0
	Sph 2	0.29	90	-85	0	3	3	5	5
	Sph 3	0.16	90	-85	0	40	7	11	0

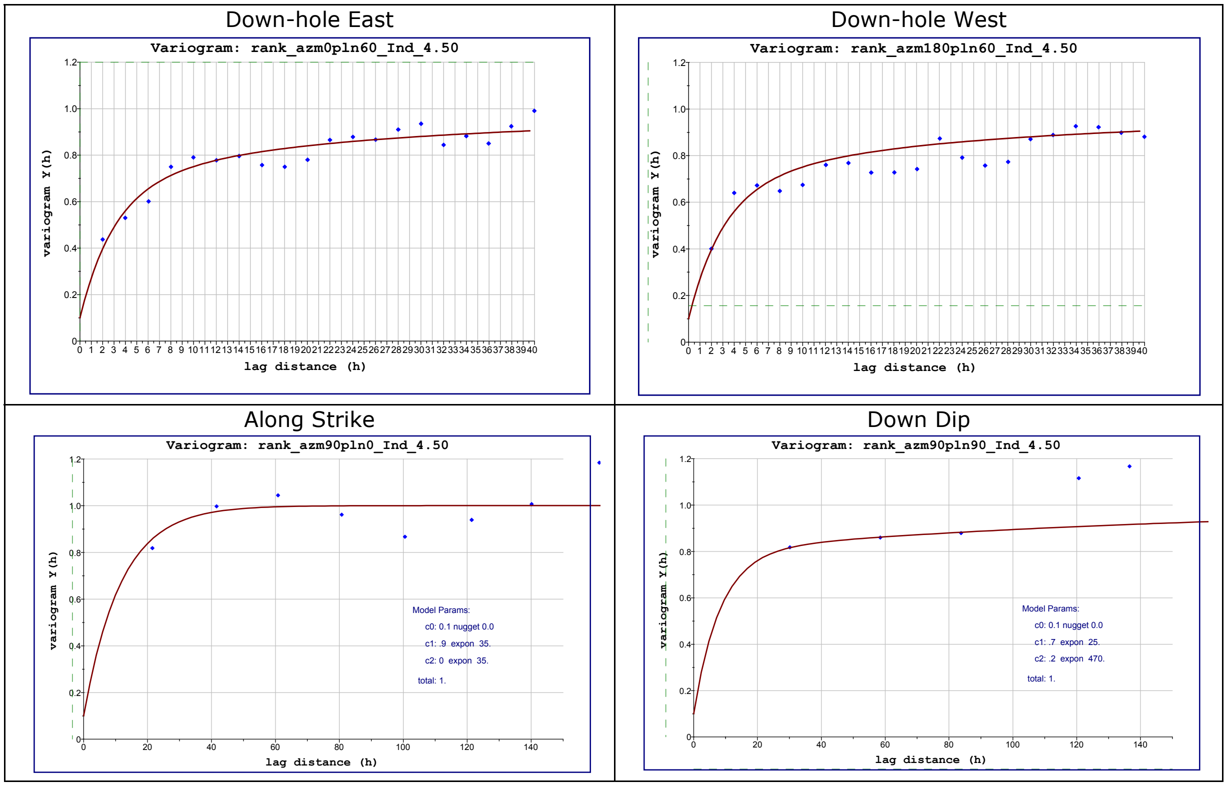
Table 17-26: New Cobar - Gold Indicator Variography for the Main and Lower Zones

Threshold	Structure	Variance	Rot Z	Rot Y	Rot X	Z range (major)	Y range (semi-major)	X range (Minor	Rotation GS3(xyz)
1	Nugget	0.1							
	exp	0.38	95	-80	0	29	20	10	0
	exp	0.21	95	-80	0	14	2	2	10
	exp	0.31	95	-80	0	111	43	11	5
2	Nugget	0.1							
	exp	0.22	95	-80	0	13	2	2	0
	exp	0.38	95	-80	0	26	10	9	10
	exp	0.3	95	-80	0	123	56	12	5
3	Nugget	0.1							
	exp	0.61	95	-80	0	15	5	6	0
	exp	0.08	95	-80	0	18	12	15	10
	exp	0.21	95	-80	0	193	102	19	5
4	Nugget	0.1							
	exp	0.19	95	-80	0	20	16	10	0
	exp	0.51	95	-80	0	12	7	7	10
	exp	0.2	95	-80	0	143	114	14	5
5	Nugget	0.1							
	exp	0.28	95	-80	0	25	25	9	0
	exp	0.4	95	-80	0	25	25	5	10
	exp	0.22	95	-80	0	85	70	18	5
6	Nugget	0.1							
	exp	0.06	95	-80	0	26	8	8	0
	exp	0.62	95	-80	0	15	8	5.5	10
	exp	0.22	95	-80	0	200	110	27	5
7	Nugget	0.1							
	sph	0.52	95	-80	0	5.5	4	2.5	0
	sph	0.22	95	-80	0	27	8	8	10
	sph	0.16	95	-80	0	200	99	30	5
8	Nugget	0.1							
	sph	0.65	95	-80	0	9	4	2.6	0
	sph	0.12	95	-80	0	71	35	35	10
	sph	0.13	95	-80	0	371	122	37	5
9	Nugget	0.1							
	sph	0.65	95	-80	0	15	4	2.6	0
	sph	0.12	95	-80	0	59	35	35	10
	sph	0.13	95	-80	0	200	122	27	5
10	Nugget	0.1							
	sph	0.06	95	-80	0	267	90	41	0
	sph	0.73	95	-80	0	31	3	3	10
	sph	0.11	95	-80	0	83	65	40	5
11	Nugget	0.15							
	sph	0.06	95	-80	0	85	15	37	0
	sph	0.72	95	-80	0	8	2.5	3	10
	sph	0.07	95	-80	0	150	115	35	5
12	Nugget	0.2							
	sph	0.62	95	-80	0	10	3	3	0
	sph	0.13	95	-80	0	26	9	9	10
	sph	0.05	95	-80	0	90	36	9	5
13	Nugget	0.2							
	exp	0.74	95	-80	0	14	2	2.5	0
	exp	0.06	95	-80	0	60	60	12	10
			95	-80	0				5
14	Nugget	0.2							
	exp	0.8	95	-80	0	12	3	3	0
									10
									5

17.4.2 Chesney Oxide

Data for Au was modelled spatially using indicator semi-variograms (Figure 17-1). The indicator variogram models are exponential, with the nugget effect increasing at higher thresholds while the variogram ranges decrease. The down-dip ranges are generally longer than ranges along strike.

Figure 17-1: Chesney Oxide – Median Indicator Variograms for Gold

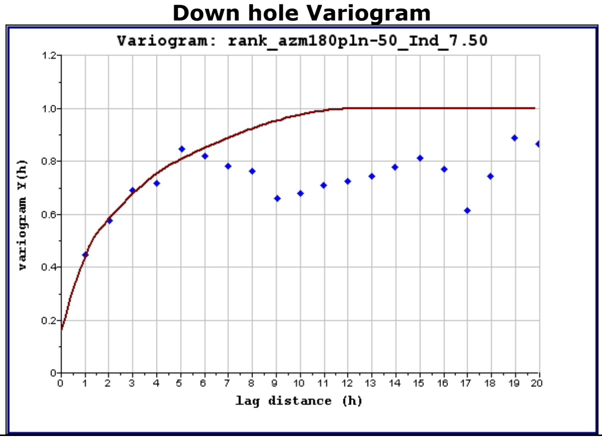


17.4.3 Chesney Sulphide

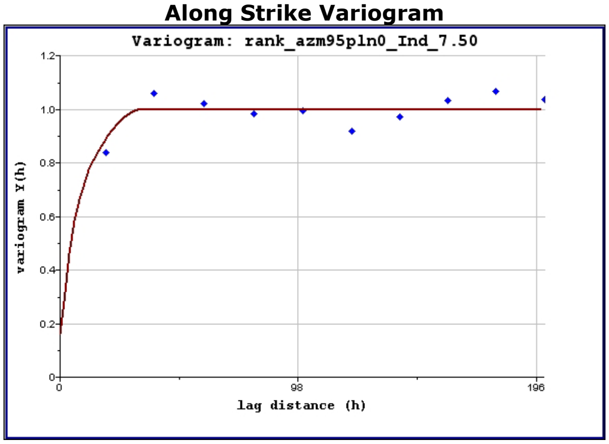
Analysis of variography for the Chesney Sulphide deposit included both indicator and grade variography for Au and grade variography for the other elements and density (Cu, Pb, Zn, Ag, Bi, and Density). Indicator variography for Au was performed using 14 cumulative grade probability thresholds based on deciles to 70%, 5% intervals to 95%, then at 97% and 99% (Table 17-27). Additional thresholds at the upper spectrum of the Au range were used to better define the high grade tail of the gold distribution. Figure 17-2 shows an example of the indicator variography for Au for the main lens (CHA). The variography indicates that the Au mineralisation is most continuous in the vertical direction and least continuous in the across strike direction (east-west). This is consistent with geological observations by site personnel. The variogram models used for the multi-element estimations are summarised in Table 17-28.

Figure 17-2: Chesney Sulphide - Typical Indicator Variograms for Gold

Down hole Variogram



Along Strike Variogram



Down Dip Variogram

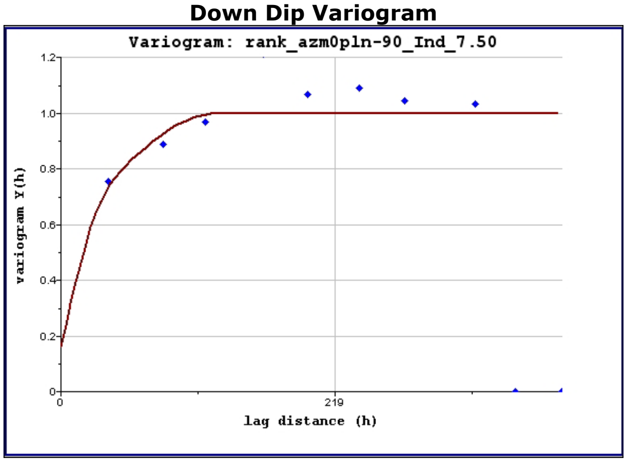


Table 17-27: Chesney Sulphide - Gold Indicator Variography for Chesney Main Lens

Indicator	Threshold (g/t Au)	Structure	Variance	Z Range (Major)	Y range (Semi Major)	X range (Minor)	Rotation (Vulcan Software)		
							Bearing (Z)	Plunge (Y')	Dip (X")
1	0.03	Nugget	0.34						
		exp1	0.23	94	31	33	85	-85	0
		exp2	0.43	41	44	25	85	-85	0
		exp3							
2	0.078	Nugget	0.15						
		exp1	0.23	16	10	4	85	-85	0
		exp2	0.1	62	33	7.5	85	-85	0
		exp3	0.52	96	21	17	85	-85	0
3	0.1	Nugget	0.19						
		exp1	0.4	16	10	3.5	85	-85	0
		exp2	0.11	62	33	11	85	-85	0
		exp3	0.3	114	56	49	85	-85	0
4	0.16	Nugget	0.19						
		exp1	0.38	16	10	2.5	85	-85	0
		exp2	0.14	62	33	5	85	-85	0
		exp3	0.29	121	34	58	85	-85	0
5	0.31	Nugget	0.19						
		exp1	0.41	16	10	3.5	85	-85	0
		exp2	0.11	60	33	16	85	-85	0
		exp3	0.29	156	40	69	85	-85	0
6	0.458	Nugget	0.19						
		exp1	0.25	16	10	3.5	85	-85	0
		exp2	0.1	62	33	16	85	-85	0
		exp3	0.46	156	40	69	85	-85	0
7	0.653	Nugget	0.25						
		exp1	0.35	16	10	3.5	85	-85	0
		exp2	0.22	60	30	16	85	-85	0
		exp3	0.18	215	38	47	85	-85	0
8	0.92	Nugget	0.25						
		exp1	0.35	16	10	3.5	85	-85	0
		exp2	0.22	60	30	16	85	-85	0
		exp3	0.18	215	38	47	85	-85	0
9	1.222	Nugget	0.25						
		exp1	0.35	16	10	3.5	85	-85	0
		exp2	0.22	60	30	16	85	-85	0
		exp3	0.18	215	38	47	85	-85	0
10	1.866	Nugget	0.3						
		exp1	0.12	17	10	20	85	-85	0
		exp2	0.36	88	41	6	85	-85	0
		exp3	0.22	113	37	43	85	-85	0
11	3.35	Nugget	0.3						
		exp1	0.2	19	12	11	85	-85	0
		exp2	0.36	88	42	6.5	85	-85	0
		exp3	0.14	180	34	30	85	-85	0
12	7.082	Nugget	0.5						
		exp1	0.24	20	11	5.5	85	-85	0
		exp2	0.01	88	41	7	85	-85	0
		exp3	0.25	168	36	24	85	-85	0
13	10.652	Nugget	0.5						
		exp1	0.29	17	10	5.5	85	-85	0
		exp2	0.11	88	41	7	85	-85	0
		exp3	0.1	223	36	22	85	-85	0
14	25.657	Nugget	0.5						
		exp1	0.12	17	7.5	5.5	85	-85	0
		exp2	0.39	80	10	4	85	-85	0

Table 17-28 Chesney Sulphide – Multi-Element Grade Variography for Chesney Main Lens

Element	Structure	Variance	Rotation (Vulcan Software) Bearing (Z)	Plunge (Y')	Dip (X'')	Z range (Major)	Y range (Semi Major)	X Range (Minor)
Au	Nugget	0.3						
	exp	0.1	85	-85		40	20	11
	exp	0.32	85	-85		8.5	11	2
	exp	0.28	85	-85		126	36	26
Cu	Nugget	0.2						
	exp	0.18	85	-85		29	176	42
	exp	0.62	85	-85		45	22	7
	exp							
Pb	Nugget	0.2						
	Exp	0.65	85	-85		29	23	5
	Exp	0.15	85	-85		119	123	36
Zn	Nugget	0.5						
	Exp	0.26	90	-90		50	30	12
	Exp	0.24	90	-90		8	8	3
Ag	Nugget	0.25						
	Exp	0.44	85	-85		23.1	10	8.5
	Exp	0.31	85	-85		220	77	19.5
Bi	Nugget	0.32						
	Exp 1	0.54	85	-85		50	27	3.5
	Exp 2	0.14	85	-85		35	60	80
	Exp 3							
SG	Nugget	0.32						
	Exp 1	0.16	85	-85		50	21	3.5
	Exp 1	0.52	85	-85		63	19	8

17.4.4 New Occidental

Variographic analysis for New Occidental was redone after the increase in drilling information in the 98 – 104 level. This included indicator and grade variography for Au and grade variography for all other elements (Cu, Pb, Zn, Ag, Bi and Density).

Indicator variography for Au by lens (Waste, OXM, GOS) was generated using the thresholds in Table 17-16. The results from the variography indicated that grades are most consistent and continuous in the vertical direction and least consistent and continuous in the across strike direction (east-west). This is deemed consistent with geological observations. The variogram models used for the grade estimates are summarised in Table 17-29 and Table 17-30.

Table 17-29: New Occidental – Gold Indicator Variogram Models Used for the Main and Gossan Lenses

Threshold	Structure	Variance	Z rot	Y rot	X rot	Z range	Y range	X range
1	Nugget	0.15						
	exp	0.17	90	-87	0	206	69	23
	exp	0.58	90	-87	0	14	12	3
	exp	0.1	90	-87	0	10	2	21
2	Nugget	0.2						
	exp	0.12	90	-87	0	48	45	8
	exp	0.56	90	-87	0	11	11	4
	exp	0.12	90	-87	0	240	38	20
3	Nugget	0.2						
	exp	0.18	90	-87	0	18	10	6
	exp	0.2	90	-87	0	73	49	14
	exp	0.42	90	-87	0	11	10	3
4	Nugget	0.2						
	exp	0.21	90	-87	0	23	10	7
	exp	0.17	90	-87	0	63	58	12
	exp	0.42	90	-87	0	14	10	2.5
5	Nugget	0.2						
	exp	0.21	90	-87	0	20	10	6
	exp	0.17	90	-87	0	106	58	8
	exp	0.42	90	-87	0	9	10	3
6	Nugget	0.2						
	exp	0.5	90	-87	0	8	9	3
	exp	0.11	90	-87	0	12	14	3
	exp	0.19	90	-87	0	135	49	9
7	Nugget	0.2						
	exp	0.4	90	-87	0	6.5	6	2.5
	exp	0.2	90	-87	0	9.5	13	7
	exp	0.2	90	-87	0	130	38	7
8	Nugget	0.2						
	exp	0.4	90	-87	0	7.5	5	2
	exp	0.2	90	-87	0	10	9	6
	exp	0.2	90	-87	0	68	36	7
9	Nugget	0.28						
	exp	0.51	90	-87	0	8	5	3.5
	exp	0.11	90	-87	0	46	34	6
	exp	0.1	90	-87	0	91	50	9
10	Nugget	0.28						
	exp	0.47	90	-87	0	3.5	5	4
	exp	0.1	90	-87	0	20	10	2
	exp	0.15	90	-87	0	100	48	9
11	Nugget	0.3						
	exp	0.1	90	-87	0	46	46	4.5
	exp	0.43	90	-87	0	3	10.5	4
	exp	0.17	90	-87	0	46	4	4.5
12	Nugget	0.3						
	exp	0.15	90	-87	0	5	18	2.5
	exp	0.35	90	-87	0	2	11	3.5
	exp	0.2	90	-87	0	45	4	3.5
13	Nugget	0.32						
	exp	0.45	90	-87	0	5	8	2.5
	exp	0.125	90	-87	0	3	25	3.5
	exp	0.11	90	-87	0	46	5	4.5
14	Nugget	0.33						
	exp	0.37	90	-87	0	7	5	1.5
	exp	0.3	90	-87	0	3	4	2.5

Table 17-30: New Occidental –Grade Variogram Models Used in the Multi-element Estimates for the Main Lens, Gossan, and Barren Lenses

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range
Au (Main)	Nugget	0.2						
	exp	0.1	95	-87	0	15	6	2
	exp	0.63	95	-87	0	12	23	3.5
	exp	0.07	95	-87	0	110	8	15
Cu	Nugget	0.15						
	exp	0.61	90	-90	0	8	2	2
	exp	0.14	90	-90	0	8	16	2
	Sph	0.1	90	-90	0	205	34	20
Pb	Nugget	0.2						
	Sph 1	0.66	90	-90	0	12	7.5	2
	Sph 2	0.02	90	-90	0	94	14	16
	Sph 3	0.12	90	-90	0	99	53	9.5
Zn	Nugget	0.4						
	Exp 1	0.42	90	-90	0	12	11	4
	Sph 2	0.14	90	-90	0	149	69	14
	Sph 3	0.04	90	-90	0	187	71	8
Ag	Nugget	0.2						
	Sph	0.52	90	-90	0	12	8.5	2.5
	Sph	0.18	90	-90	0	26	15.5	13.5
	Sph	0.1	90	-90	0	178	57	22
Bi	Nugget	0.15						
	Exp 1	0.44	90	-90	0	10	8	2
	Sph	0.24	90	-90	0	20	15	3.5
	Sph	0.17	90	-90	0	300	32	11
SG	Nugget	0.2						
	Exp	0.38	90	-90	0	7	9	5
	Exp	0.25	90	-90	0	14	9	5.5
	Sph	0.17	90	-90	0	350	66	9.5
Au (gossan)	Nugget	0.15						
	Exp	0.6	95	-87	0	16	11	2.5
	Sph	0.1	95	-87	0	16	11	6
	Sph	0.15	95	-87	0	110	28	6
Au (BAR)	Nugget	0.15						
	Exp	0.6	95	-87	0	16	11	2.5
	Sph	0.1	95	-87	0	16	11	6
	Sph	0.15	95	-87	0	110	28	6

17.4.5 Peak

Grade variography that was generated from the 2006 resource model was not recalculated for the 2007 estimate due to the relatively low amount of additional drilling which would consequently not change the variography materially. The 2006 grade variographic analysis was for Au in the Waste, Regolith, Contact, East and CWLZ Lenses zones (Table 17-31). Indicator variography was generated for Au in the CWLZ and East Lenses using the thresholds defined in Table 17-32. Based upon the variography, the gold mineralisation shows the most consistency and continuity in the vertical direction and least consistency and continuity in the across strike direction (east-west). This is consistent with geological observations.

Grade variography was also generated for other elements and density and are shown in Table 17-33.

Table 17-31: Peak – Gold Grade Variogram Models Used for the Main Lenses

Zone	**Structure**	**Variance**	**X range (Minor)**	**Y range (Semi-Major)**	**Z range (Major)**	**Rotation (Vulcan Software)**		
						Bearing (Z)	**Plunge (Y')**	**Dip (X'')**
Waste	Nugget							
	Spherical	0.32	90	-90	0	10	10	1.5
	Spherical	0.08	90	-90	0	15	15	5
Regolith	Nugget	0.1						
	Spherical	0.6	90	-90	0	11	3	3.5
	Spherical	0.2	90	-90	0	11	10	4
	Spherical	0.1	90	-90	0	30	20	4
Contact	Nugget	0.6						
	Spherical	0.4	90	-90	0	12	10	3.5
East Upper	Nugget	0.56	-	-	-			
	Sph 1	0.33	1	11	11	80	-74	0
	Sph 2	0.11	6	16	22	80	-74	0
East Lower	0.56							0.56
	0.33	80	-74	0	11	11	1	0.33
	0.11	80	-74	0	22	16	6	0.11
CWLZ	Nugget	Nugget	0.5					
	Sph 1	Sph	0.42	90	-85	0	12	10
	Sph 2	Sph	0.08	90	-85	0	46	30

Table 17-32: Peak - Gold Indicator Variogram Models Used for the CWLZ Lens (North and South)

Threshold	Structure	Variance	Z rot	Y rot	X rot	Z range	Y range	X range
1	Nugget	0.55						
	sph	0.16	90	-85	0	16	16	6
	sph	0.1	90	-85	0	22	30	21
	sph	0.19	90	-85	0	100	82	40
2	Nugget	0.5						
	sph	0.18	90	-85	0	16	19	6.5
	sph	0.11	90	-85	0	22	48	26
	sph	0.21	90	-85	0	100	88	38
3	Nugget	0.38						
	exp	0.24	90	-85	0	23	19	6.5
	exp	0.13	90	-85	0	16	48	28
	exp	0.25	90	-85	0	146	88	40
4	Nugget	0.4						
	exp	0.18	90	-85	0	13	13	5.5
	exp	0.13	90	-85	0	24	70	21
	exp	0.29	90	-85	0	175	68	41
5	Nugget	0.4						
	exp	0.19	90	-85	0	13	13	5
	exp	0.16	90	-85	0	37	65	21
	exp	0.25	90	-85	0	90	70	50
6	Nugget	0.4						
	exp	0.3	90	-85	0	14	22	14
	exp	0.16	90	-85	0	66	71	200
	exp	0.14	90	-85	0	34	70	200
7	Nugget	0.45						
	exp	0.13	90	-85	0	11	12	8.5

Threshold	Structure	Variance	Z rot	Y rot	X rot	Z range	Y range	X range
	exp	0.16	90	-85	0	13	17	3.5
	exp	0.26	90	-85	0	144	76	40
8	Nugget	0.45						
	exp	0.13	90	-85	0	11	11	5.5
	exp	0.16	90	-85	0	15	15	3.5
	exp	0.26	90	-85	0	140	70	37
9	Nugget	0.49						
	exp	0.17	90	-85	0	44	40	8.5
	exp	0.19	90	-85	0	9.5	20	10
	exp	0.15	90	-85	0	295	80	66
10	Nugget	0.5						
	exp	0.24	90	-85	0	14	14	6.5
	exp	0.21	90	-85	0	100	71	27
	exp	0.05	90	-85	0	270	55	40
11	Nugget	0.5						
	exp	0.22	90	-85	0	18	13	9
	exp	0.18	90	-85	0	63	54	12
	exp	0.1	90	-85	0	200	60	40
12	Nugget	0.5						
	exp	0.32	90	-85	0	16	8.5	7
	exp	0.07	90	-85	0	100	50	14
	exp	0.11	90	-85	0	100	65	18
13	Nugget	0.54						
	exp	0.33	90	-85	0	17	10	5
	exp	0.13	90	-85	0	94	66	21
14	Nugget	0.53						
	exp	0.37	90	-85	0	17	10	4
	exp	0.1	90	-85	0	63	36	10

Table 17-33: Peak – Grade Variogram Models Used for Contact Zone

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range
Au	Nugget	0.6						
	Spherical	0.4	90	-90	0	12	10	3.5
Cu	Nugget	0.1						
	Spherical	0.37	90	-90	0	12	10	2.5
	Spherical	0.17	0	0	0	20	15	12
	Spherical	0.36	90	-90	0	180	86	38
Pb	Nugget	0.35						
	Exponential	0.51	90	-90	0	10	10	3.5
	Spherical	0.14	90	-90	0	125	70	20
Zn	Nugget	0.35						
	Exponential	0.48	90	-90	0	10	10	5.5
	Exponential	0.17	90	-90	0	145	90	28
Ag	Nugget	0.2						
	Exponential	0.42	90	-90	0	26	5.5	3
	Spherical	0.26	90	-90	0	44	14	14
	Spherical	0.12	90	-90	0	92	12	14
SG	Nugget	0.15						
	Spherical	0.37	90	-90	0	11	10	4
	Spherical	0.24	90	-90	0	56	44	12
	Spherical	0.24	0	0	0	135	86	43

Table 17-34: Peak – Grade Variogram Models Used for Waste Zone

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range
Au	Nugget							
	Spherical	0.32	90	-90	0	10	10	1.5
	Spherical	0.08	90	-90	0	15	15	5
Cu	Nugget	0.1						
	Spherical	0.37	90	-90	0	12	10	2.5
	Spherical	0.17	0	0	0	20	15	12
Pb	Nugget	0.35						
	Exponential	0.51	90	-90	0	10	10	3.5
	Spherical	0.14	90	-90	0	125	70	20
	Sph 3							
Zn	Nugget	0.35						
	Exponential	0.48	90	-90	0	10	10	5.5
	Exponential	0.17	90	-90	0	145	90	28
Ag	Nugget	0.2						
	Exponential	0.42	90	-90	0	26	5.5	3
	Spherical	0.26	90	-90	0	44	14	14
	Spherical	0.12	90	-90	0	92	12	14

Table 17-35: Peak – Grade Variogram Models Used for Regolith Zone

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range
Au	Nugget	0.1						
	Spherical	0.6	90	-90	0	11	3	3.5
	Spherical	0.2	90	-90	0	11	10	4
	Spherical	0.1	90	-90	0	30	20	4
Cu	Nugget	0.1						
	Exponential	0.67	0	0	0	7.5	7	3.5
	Spherical	0.23	90	-90	0	56	52	21
Pb	Nugget	0.01						
	Spherical	0.48	90	-90	0	13	10	2
	Spherical	0.51	90	-90	0	37	29	7
Zn	Nugget	0.01						
	Spherical	0.48	90	-90	0	13	10	2
	Spherical	0.51	90	-90	0	37	29	7
	Sph 3							
Ag	Nugget	0.1						
	Spherical	0.66	90	-90	0	10	6	4
	Spherical	0.24	90	-90	0	15	28	4
	Sph							

Table 17-36: Peak – Grade Variogram Models Used for Copper WLZ Zone

Element	Structure	Variance	Rot Z	Rot Y	Rot X	Z range	Y Range	X Range
Au COPWLZ	Nugget	0.5						
	Sph	0.42	90	-85	0	12	10	2.5
	Sph	0.08	90	-85	0	46	30	8
	Sph							
Au COPWLZ Nth	Nugget	0.5						
	Spherical	0.42	280	-85	0	12	10	2.5
	Spherical	0.08	280	-85	0	46	30	8
	Sph							
Au COPWLZ Sth	Nugget	0.5						
	Spherical	0.42	260	-85	0	12	10	2.5
	Spherical	0.08	260	-85	0	46	30	8
	Sph							
Cu COP/WLZ	Nugget	0.1						
	Sph	0.4	90	-90	0	12	10	3.5
	Sph	0.14	90	-90	0	15	25	13
	Sph	0.36	90	-90	0	180	100	44
Cu COP/WLZ North	Nugget	0.1						
	Sph	0.4	280	-85	0	12	10	3.5
	Sph	0.14	280	-85	0	15	25	13
	Sph	0.36	280	-85	0	180	100	44
Cu COP/WLZ South	Nugget	0.1						
	Sph	0.4	260	-85	0	12	10	3.5
	Sph	0.14	260	-85	0	15	25	13
	Sph	0.36	260	-85	0	180	100	44
Pb	Nugget	0.35						
	Exponential	0.51	260	-85	0	10	10	3.5
	Spherical	0.14	260	-85	0	125	70	20
Zn	Nugget	0.35						
	Exponential	0.48	260	-85	0	10	10	5.5
	Exponential	0.17	260	-85	0	145	90	28
	Sph 3							
Ag	Nugget	0.2						
	Exponential	0.42	270	-85	0	26	5.5	3
	Spherical	0.26	270	-85	0	44	14	14
	Spherical	0.12	270	-85	0	92	12	14
Ag Nth	Nugget	0.2						
	Exponential	0.42	280	-85	0	26	5.5	3
	Spherical	0.26	280	-85	0	44	14	14
	Spherical	0.12	280	-85	0	92	12	14
Ag Nth	Nugget	0.2						
	Exponential	0.42	260	-85	0	26	5.5	3
	Spherical	0.26	260	-85	0	44	14	14
	Spherical	0.12	260	-85	0	92	12	14

17.4.6 Perseverance

Grade variography and variogram models were generated for all attributes within each domain for Zone D by H&S (Table 17-37). Analysis of indicator variograms and variogram models were also generated using the 1 metre gold composites for the 2 sub-domains in Zone D (Table 17-38 and Table 17-39). PGM generated Au indicator variograms and variogram models for Zone A (Table 17-40 and Table 17-41). The indicator variography for Zone D was based upon the thresholds shown in Table 17-22. Grade variography was also generated for other attributes within each domain for Zone D (Table 17-37). Existing PGM variogram models for the Zone A, Zone B and Hercules Lenses were used but Hulk was revised (Table 17-40).

Table 17-37: Perseverance–Zone D Grade Variogram Models Used in the Multi-element Estimates

			North Domain			**South Domain**			
Element	**Structure**	**Variance**	**Z range**	**Y Range**	**X Range**	**Variance**	**Z range**	**Y Range**	**X Range**
Au	Nugget	0.2				0.2			
	exp	0.1	50	23	1.5	0.1	13	10	1.5
	exp	0.63	10	21	3	0.63	5.5	20	3.5
	exp	0.07	8	2	2	0.07	5	49	2
Cu	Nugget	0.38				0.17			
	exp	0.41	50	23	2	0.58	23	6.5	2
	exp	0.14	39	17	34.5	0.23	93	19	3
	Sph	0.07	44	18	38	0.02	44	18	38
Pb	Nugget	0.38				0.17			
	Sph 1	0.4	12	7.5	2	0.58	23	6.5	2
	Sph 2	0.11	94	14	16	0.23	93	19	3
	Sph 3	0.11	99	53	9.5	0.02	44	18	38
Zn	Nugget	0.4				0.17			
	Exp 1	0.42	12	11	4	0.19	330	34.5	33
	Sph 2	0.14	149	69	14	0.62	50	19	2.5
	Sph 3	0.04	187	71	8	0.02	50	18	3
Ag	Nugget	0.2				0.17			
	Sph	0.52	10	11	5.5	0.58	24	7.5	1
	Sph	0.18	2	21	9	0.23	50	19	2.5
	Sph	0.1	50	8	84	0.02	50	18	3
Bi	Nugget	0.15				0.15			
	Exp 1	0.58	24	1	9.5	0.81	2	1.5	2
	Sph	0.16	2	21	9	0.02	22.5	7	22
	Sph	0.11	49	8	84	0.02	2	2	2
Fe	Nugget	0.2				0.2			
	Exp	0.63	5	14	49	0.02	50	25	36
	Exp	0.17	18	9	1.5	0.43	78	75	13.5
SG	Nugget	0.2				0.26			
	Exp	0.38	7	9	2	0.22	50	25	34
	Exp	0.25	15	9	2.5	0.15	3	21	4
	Sph	0.17	28	66	9.5	0.37	4	11	12

Table 17-38: Perseverance - Gold Indicator Variogram Models Used for Zone D – North Domain

Threshold	Structure	Variance	Z	Y	X	Z range	Y range	X range	Rotation GS3	Rotation Vulcan
1	Nugget	0.1							x,y,z	z,y,x
	exp	0.36	105	-85	0	14	7	2.5	0,5,15	105,-85,0
	exp	0.3	105	-85	0	28	24	15		
	exp	0.24	105	-85	0	18	15	13		
2	Nugget	0.15								
	exp	0.31	105	-85	0	20	11	2	0,5,15	105,-85,0
	exp	0.28	105	-85	0	25	22	7.5		
	exp	0.26	105	-85	0	50	20	8		
3	Nugget	0.18							0,5,15	105,-85,0
	exp	0.18	105	-85	0	20	11	1		
	exp	0.38	105	-85	0	25	22	4.5		
	exp	0.26	105	-85	0	50	20	8		
4	Nugget	0.18							0,5,15	105,-85,0
	exp	0.18	105	-85	0	20	11	1		
	exp	0.38	105	-85	0	25	22	5.5		
	exp	0.26	105	-85	0	50	20	8		
5	Nugget	0.18							0,5,15	105,-85,0
	exp	0.18	105	-85	0	20	11	0.5		
	exp	0.38	105	-85	0	25	22	5.5		
	exp	0.26	105	-85	0	50	20	8		
6	Nugget	0.14							0,5,15	105,-85,0
	exp	0.18	105	-85	0	20	11	4		
	exp	0.38	105	-85	0	25	22	2.5		
	exp	0.3	105	-85	0	50	20	8		
7	Nugget	0.14							0,5,15	105,-85,0
	exp	0.18	105	-85	0	16	11	0.5		
	exp	0.38	105	-85	0	25	22	2		
	exp	0.3	105	-85	0	50	20	13		
8	Nugget	0.18							0,5,15	105,-85,0
	exp	0.21	105	-85	0	35	14	1		
	exp	0.42	105	-85	0	20	24	5		
	exp	0.19	105	-85	0	23	6	5.5		
9	Nugget	0.18							0,5,15	105,-85,0
	exp	0.21	105	-85	0	35	23	1		
	exp	0.42	105	-85	0	20	7	8		
	exp	0.19	105	-85	0	23	6	9.5		
10	Nugget	0.18							0,5,15	105,-85,0
	exp	0.21	105	-85	0	35	14	1		
	exp	0.42	105	-85	0	20	24	5		
	exp	0.19	105	-85	0	23	6	5.5		
11	Nugget	0.18							0,5,15	105,-85,0
	exp	0.25	105	-85	0	35	27	2.5		
	exp	0.45	105	-85	0	6	15	4		
	exp	0.12	105	-85	0	97	9	9.5		
12	Nugget	0.18							0,5,15	105,-85,0
	exp	0.25	105	-85	0	37	27	1		
	exp	0.45	105	-85	0	19	12	3.5		
	exp	0.12	105	-85	0	15	13	9.5		
13	Nugget	0.15							0,5,15	105,-85,0
	exp	0.28	105	-85	0	29	26	2		
	exp	0.45	105	-85	0	20	12	3.5		
	exp	0.12	105	-85	0	15	13	9.5		
14	Nugget	0.4							0,5,15	105,-85,0
	exp	0.11	105	-85	0	28	25	8		
	exp	0.19	105	-85	0	12	30	1		
	exp	0.3	105	-85	0	27	13	6		

Table 17-39: Perseverance - Gold Indicator Variogram Models Used for Zone D – South Domain

Threshold	Structure	Variance	Z rot	Y rot	X rot	Z range	Y range	X range	Rotation GS3
1	Nugget	0.1							x,y,z
	exp	0.19	270	-83	0	38	18	5.5	0,-7,0
	exp	0.41	270	-83	0	34	37	10	
	exp	0.3	270	-83	0	16	41	0.5	
2	Nugget	0.1							0,-7,0
	exp	0.19	270	-83	0	38	18	5.5	
	exp	0.41	270	-83	0	34	37	7	
	exp	0.3	270	-83	0	16	41	2	
3	Nugget	0.1		-83		-	-	-	0,-7,0
	exp	0.19	270	-83	0	38	18	1.5	
	exp	0.41	270	-83	0	34	37	7	
	exp	0.3	270	-83	0	16	41	2	
4	Nugget	0.1		-83		-	-	-	0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	6.5	
	exp	0.3	270	-83	0	16	41	2	
5	Nugget	0.1		-83		-	-	-	0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	4.5	
	exp	0.3	270	-83	0	16	41	2	
6	Nugget	0.1		-83		-	-	-	0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	4.5	
	exp	0.3	270	-83	0	16	41	2	
7	Nugget	0.1		-83		-	-	-	0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	4	
	exp	0.3	270	-83	0	16	41	2	
8	Nugget	0.1							0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	3	
	exp	0.3	270	-83	0	16	41	2	
9	Nugget	0.1							0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	4	
	exp	0.3	270	-83	0	16	41	2	
10	Nugget	0.1							0,-7,0
	exp	0.19	270	-83	0	37	18	0.5	
	exp	0.41	270	-83	0	34	37	4.5	
	exp	0.3	270	-83	0	16	41	2	
11	Nugget	0.25							0,-7,0
	exp	0.18	270	-83	0	37	18	0.5	
	exp	0.27	270	-83	0	34	37	5	
	exp	0.3	270	-83	0	16	41	2	
12	Nugget	0.24							0,-7,0
	exp	0.18	270	-83	0	37	18	0.5	
	exp	0.27	270	-83	0	34	37	5	
	exp	0.31	270	-83	0	16	41	2	
13	Nugget	0.3							0,-7,0
	exp	0.18	270	-83	0	37	18	0.5	
	exp	0.27	270	-83	0	34	37	5	
	exp	0.25	270	-83	0	16	41	2	
				-83					
14	Nugget	0.4							0,-7,0
	exp	0.6	270	-83	0	23	19	3	

Table 17-40: Perseverance – Other Zones -Grade Variogram Models Used in the Multi-element Estimates

Zone	Element	Model	Variance	Rotation Z	Rotation Y	Rotation X	Major Axis	Semi-Major	Minor
Zone A	Au	Nugget	0.15						
		Exponential	0.75	265	-55	0	13	20	2.5
		Exponential	0.1	265	-55	0	60	20	8
Zone A	Cu	Nugget	0.1						
		Spherical	0.33	90	-90	0	14	13	2
		Spherical	0.3	90	-90	0	14	13	8.5
		Spherical	0.27	90	-90	0	100	52	21
Zone A	Pb	Nugget	0.3						
		Spherical	0.29	90	-90	0	14	13	1.5
		Spherical	0.25	90	-90	0	14	13	5
		Spherical	0.16	90	-90	0	100	62	11
All Zones	Zn	Nugget	0.4						
		Spherical	0.37	90	-90	0	14	13	1
		Spherical	0.13	90	-90	0	14	13	4.5
		Spherical	0.1	90	-90	0	100	62	5
Zone A	Ag	Nugget	0.2						
		Spherical	0.55	90	-90	0	14	10	1.5
		Spherical	0.11	90	-90	0	14	10	9
		Spherical	0.14	90	-90	0	100	46	18
Zone A	Bi	Nugget	0.25						
		Exponential	0.6	90	-90	0	14	10	2.5
		Spherical	0.08	90	-90	0	14	10	5
		Spherical	0.07	90	-90	0	73	20	10
All Zones	SG	Nugget	0.1						
		Exponential	0.7	90	-90	0	24	20	2.7
		Exponential	0.2	90	-90	0	90	20	16
All Zones	Fe	Nugget	0.1						
		Exponential	0.55	90	-90	0	24	27	3
		Exponential	0.35	90	-90	0	110	27	20
Zone B	Au	Nugget	0.5						
		Exponential	0.4	80	-85	0	13	20	2.5
		Exponential	0.1	80	-85	0	60	20	8
Zone B, Waste East	Cu	Nugget							
		Spherical	0.33	90	-90	0	14	13	2
		Spherical	0.3	90	-90	0	14	13	8.5
		Spherical	0.27	90	-90	0	100	52	21
Zone B, Waste East, Hulk and Hercules	Pb	Nugget	0.3						
		Spherical	0.29	90	-90	0	14	13	1.5
		Spherical	0.25	90	-90	0	14	13	5
		Spherical	0.16	90	-90	0	100	62	11
Zone B, waste	Ag	Nugget	0.2						
		Spherical	0.55	90	-90	0	14	10	1.5
		Spherical	0.11	90	-90	0	14	10	9
		Spherical	0.14	90	-90	0	100	46	18
Waste, Hulk Zone B and Hercules	Bi	Nugget	0.25						
		Exponential	0.6	90	-90	0	14	10	2.5
		Spherical	0.08	90	-90	0	14	10	5
		Spherical	0.07	90	-90	0	73	20	10
Hercules	Au	Nugget	0.15						
		Exponential	0.75	90	-90	0	13	20	2.5
		Exponential	0.1	90	-90	0	60	20	8
Hercules	Cu	Nugget	0.1						
		Spherical	0.33	90	-90	0	14	13	2
		Spherical	0.3	90	-90	0	14	13	8.5
		Spherical	0.27	90	-90	0	100	52	21

Table 17-41: Perseverance - Gold Indicator Variogram Models Used for Zone A

Threshold	Structure	Variance	X range	Y range	Z range	Rotation z,y,x Subzone 1	Rotation z,y,x Subzone 2	Rotation z,y,x Subzone 3
1	Nugget	0.15	-	-	-			
	Exp 1	0.5	2.5	16	18	265	265	100
	Exp 2	0.15	30	16	18	-55	-81	-85
	Exp 3	0.2	3	16	120	0	0	0
2	Nugget	0.15						
	Exp 1	0.37	2	18	21	265	265	100
	Exp 2	0.28	18	18	21	-55	-81	-85
	Exp3	0.2	170	18	18	0	0	0
3	Nugget	0.15						
	Exp 1	0.45	3	22	23	265	265	100
	Exp 2	0.2	18	22	23	-55	-81	-85
	Exp 3	0.2	18	22	180	0	0	0
4	Nugget	0.15						
	Exp 1	0.4	2	28	15	265	265	100
	Exp 2	0.23	18	28	18	-55	-81	-85
	Exp3	0.22	18	28	180	0	0	0
5	Nugget	0.1						
	Exp 1	0.5	2.5	27	14	265	265	100
	Exp 2	0.18	18	27	14	-55	-81	-85
	Exp 3	0.22	18	27	180	0	0	0
6	Nugget	0.15						
	Exp 1	0.58	2.5	29	15	265	265	100
	Exp 2	0.08	35	29	15	-55	-81	-85
	Exp 3	0.19	35	29	250	0	0	0
7	Nugget	0.15						
	Exp 1	0.6	2.5	20	16	265	265	100
	Exp 2	0.08	35	56	16	-55	-81	-85
	Exp 3	0.17	35	56	280	0	0	0
8	Nugget	0.15						
	Exp 1	0.64	2.5	20	15	265	265	100
	Exp 2	0.06	3	60	15	-55	-81	-85
	Exp 3	0.15	50	60	350	0	0	0
9	Nugget	0.2						
	Exp 1	0.5	3	16	15	265	265	100
	Exp 2	0.15	3	60	15	-55	-81	-85
	Exp 3	0.15	50	60	350	0	0	0
10	Nugget	0.2						
	Exp 1	0.5	3	16	15	265	265	100
	Exp 2	0.15	3	60	15	-55	-81	-85
	Exp 3	0.15	50	60	170	0	0	0
11	Nugget	0.25						
	Exp 1	0.4	3	15	14	265	265	100
	Exp 2	0.2	3	50	14	-55	-81	-85
	Exp 3	0.15	40	50	150	0	0	0
12	Nugget	0.3						
	Exp 1	0.3	2	15	12	265	265	100
	Exp 2	0.2	2	45	12	-55	-81	-85
	Exp 3	0.2	18	45	75	0	0	0
13	Nugget	0.4				265	265	100
	Exp 1	0.3	2.4	32	13	-55	-81	-85
	Exp 2	0.3	2.4	40	38	0	0	0
14	Nugget	0.5				265	265	100
	Exp 1	0.3	2	224	25	-55	-81	-85
	Exp 2	0.2	2	35	24	0	0	0

17.5 Block Modelling

All block model extents are in the PGM local grid.

17.5.1 New Cobar Sulphide

A rotated block model dipping 80° → 090° was created using Vulcan mining software according to the parameters in Table 17-42. The parent block size (2 metres by 12 metres by 15 metres) was chosen to best reflect drill hole spacing in the deposit. Sub-celling to 1 metre by 6 metres by 7.5 metres was performed to allow for adequate volume definition on zone contacts.

Table 17-42: Chesney Oxide – Block Model Parameters

	X (Easting)	Y (Northing)	RL
Origin	24,960	15,900	9,370
Offset	170	720	975
Parent Block Dimension	2	12	15
Minimum Sub-Cell Dimension	1	6	7.5

17.5.2 Chesney Oxide

An orthogonal (unrotated) block model was created in H&S GS3 software based upon the parameters in Table 17-43. The parent block size (2 metres by 10 metres by 5 metres) was chosen as a compromise between potential bench height requirements and drill hole spacing. Sub-celling to 1 metre by 5 metres by 5 metres was performed to allow for adequate volume definition on zone contacts.

Table 17-43: Chesney Oxide – Block Model Parameters

	Easting	Northing	RL
Origin	25050	15250	10120
Maximum Extent	25150	15620	10280
Parent Block Dimension	2	10	5
Minimum Sub-Cell Dimension	1	5	5

17.5.3 Chesney Sulphide

An orthogonal block model was used for the estimate, with blocks aligned sub-parallel to the plane of mineralisation. Model parameters are presented in Table 17-44

The parent block dimension in the Northing (Y) and RL (Z) directions was chosen to reflect the nominal drill hole spacing combined with a compromise for proposed underground sub-level intervals. Sub-celling down to 1 metre by 10 metres by 10 metres was completed to preserve the geometry of the ore domains at the boundaries. No individual sub-cells are estimated; sub-cells are assigned the grade of the parent block.

Table 17-44: Chesney Sulphide - Block Model Parameters

	Easting	Northing	RL
Origin	25,050	15,250	9,300
Maximum Extent	25,210	15,810	10,280
Parent Block Dimension	2	20	20
Minimum Sub-Cell Dimension	1	10	10

17.5.4 New Occidental

An inclined/rotated block model was used for the grade estimations, with blocks aligned parallel to the plane of mineralisation dipping at -87° towards the east (Table 17-45).

The parent block size in the Y and Z direction is a reflection of the nominal drill hole spacing. Sub-celling down to 1 metre by 6 metres by 7.5 metres was completed to preserve the geometry of the ore domains at the boundaries. No individual sub-cells are estimated; sub-blocks are assigned the grade of the parent block.

Table 17-45: New Occidental - Block Model Parameters

	X (Easting)	Y (Northing)	RL (Z)
Origin	25,000	13,300	9,000
Maximum Extent	5,200	13,804	10,200
Parent Block Dimension	2	12	15
Minimum Sub-Cell Dimension	1	6	7.5

17.5.5 Peak

An orthogonal block model was used for the grade estimations. The block model parameters are summarised in Table 17-46. The parent cell size (2 metres by 10 metres by 10 metres) was chosen to reflect the dominant drill hole spacing within the deposit. Sub-celling down to 1 metre by 5 metres by 5 metres was performed to allow for adequate volume representation on zone boundaries.

Table 17-46: Peak – Block Model Parameters

	Easting	Northing	RL
Origin	25,520	10,280	9,250
Maximum Extent	26,000	10,920	10,300
Parent Block Dimension	2	10	10
Minimum Sub-Cell Dimension	1	5	5

17.5.6 Perseverance

An orthogonal block model was used for the grade estimations. The block model parameters for all zones are summarised in Table 17-47. The parent cell size (2 metres by 20 metres by 20 metres) was chosen to reflect the dominant drill hole spacing within the deposit. Sub-celling down to 1 metre by 5 metres by 5 metres was performed to allow for adequate volume representation on zone boundaries.

Table 17-47: Perseverance – Block Model Parameters

	Easting	Northing	RL
Origin	25,550	9,400	8,600
Maximum Extent	25,900	10,000	9,400
Parent Block Dimension	2	20	20
Minimum Sub-Cell Dimension	1	5	5

17.6 Grade Estimation and Classification

17.6.1 New Cobar Sulphide

PGM used Vulcan mining software to estimate Au in the Main Zone and Western Zone using hard domain boundaries. Grade estimates used four search passes in each domain. Intra-class means were used to calculate the E-type estimate. The upper domain search ellipse had a major axis orientation of 80° → 090°, while the lower domain search ellipse had a major axis of 87° → 090°. The Western Zone has a major axis orientation of 75°→ 090° and plunging 75°N. Soft boundaries were used between the upper and lower Main zone domains. No high grade cutting or high grade limiting was used. The sample selection criteria for the MIK estimate are shown in Table 17-48.

Table 17-48: New Cobar – MIK Composite Search Parameters

Pass	Search Radii (m)			Composites		Min. No. Octants
	Major	Semi-Major	Minor	Minimum	Maximum	
1	20	15	3	8	32	4
2	40	30	6	8	32	4
3	80	60	12	8	32	2
4	120	80	20	4	32	2

Gold was also estimated using OK for check purposes only, applying soft boundaries within the Main Zone for the upper and lower sub-domains. Three passes were used for the estimate in the upper portion of the Main Zone and four passes were used for the lower domain.

The elements other than Au (Pb, Zn, Cu, Ag, Bi, Density and Fe) were estimated using OK with four search passes using soft domain boundaries. Gold was also estimated using OK in the hangingwall and footwall regions separated by hard domain boundaries. The sample search and selection criteria for all OK estimates are shown in Table 17-49. The orientation of the sample search for the OK estimates are summarised in Table 17-50.

The reported gold resource for the New Cobar Sulphide was based upon the MIK E-type estimate within the Main Zone and the OK estimates within the Hangingwall and Footwall zones.

Table 17-49: New Cobar – OK Sample Search Parameters

Pass	Search Radii (m)			Composites		Min. No. Octants	Classificatio n
	Major	Semi-Major	Minor	Minimum	Maximum		
1	20	15	3	8	32	4	Measured
2	40	30	6	8	32	4	Indicated
3	80	60	12	8	32	2	Inferred
4	120	80	20	4	32	2	

Table 17-50: New Cobar – OK Sample Search Orientations

Parameter	Search Orientation (Vulcan Software)		
	Bearing (Z)	Plunge (Y’)	Dip (X’’)
Au - Waste	90	-85	0
Au - Main Zone Upper domain	90	-80	0
Au - Main Zone – lower domain	90	-87	0
Au – West Zone	44	-68.91	-44
Pb	90	-85	0
Zn	90	-85	0
Cu	90	-85	0
Ag	73	-80	0
Bi	75	-85	0
Density	85	-80	0
Fe	90	-85	0

Classification

The New Cobar sulphide copper and gold resource was classified according to the estimation pass of the OK Au estimations. Blocks estimated during pass 1 were classified as Measured, pass 2 blocks were classified as Indicated, and pass 3 as Inferred.

Figure 17-3 is a schematic long section showing the model blocks and mineral resource classification for New Cobar. Figure 17-4 shows a similar schematic long section with the model block Au grades coloured.

Figure 17-3: New Cobar Sulphide- Long Section Showing Mineral Resource Classification for Reported Blocks

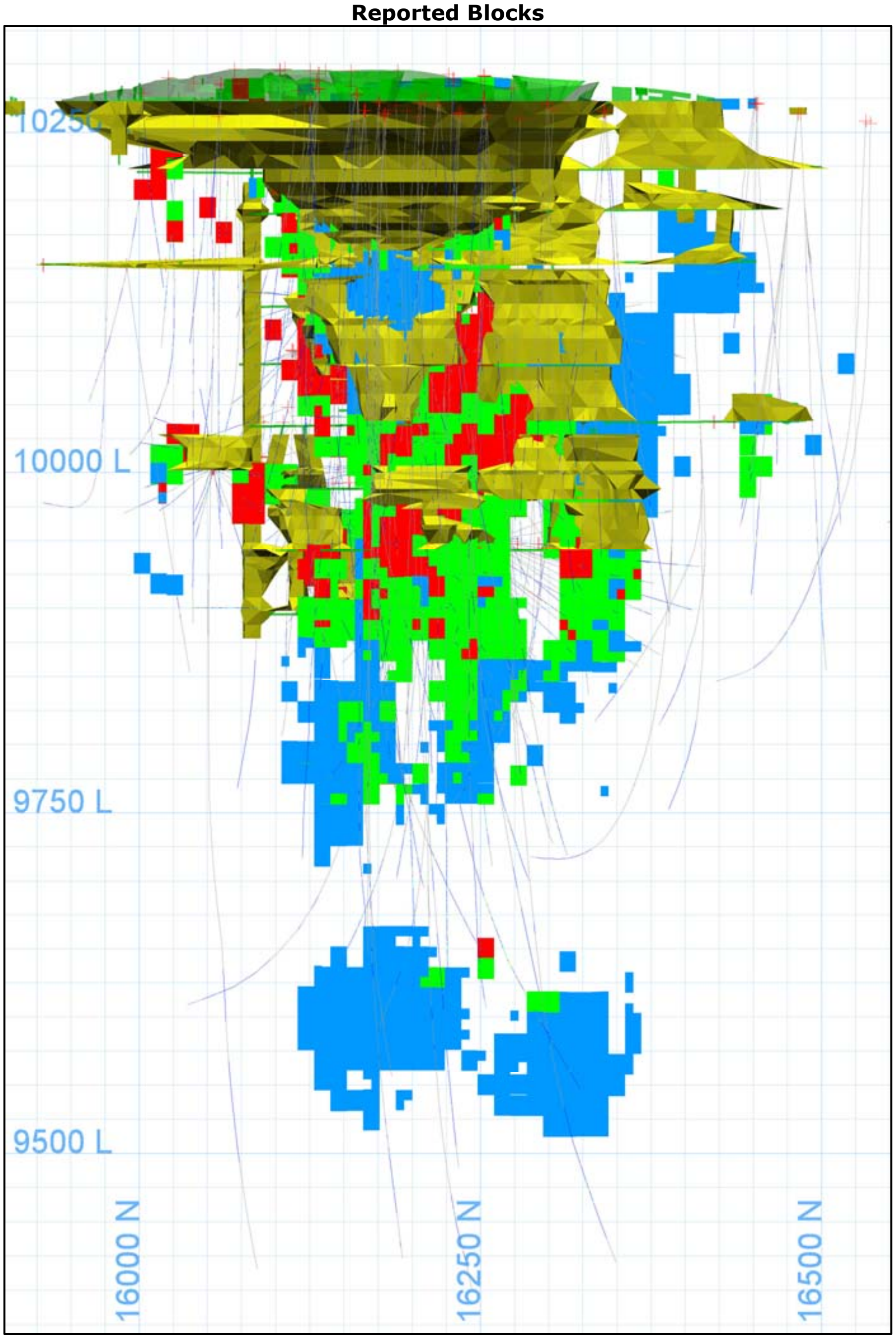
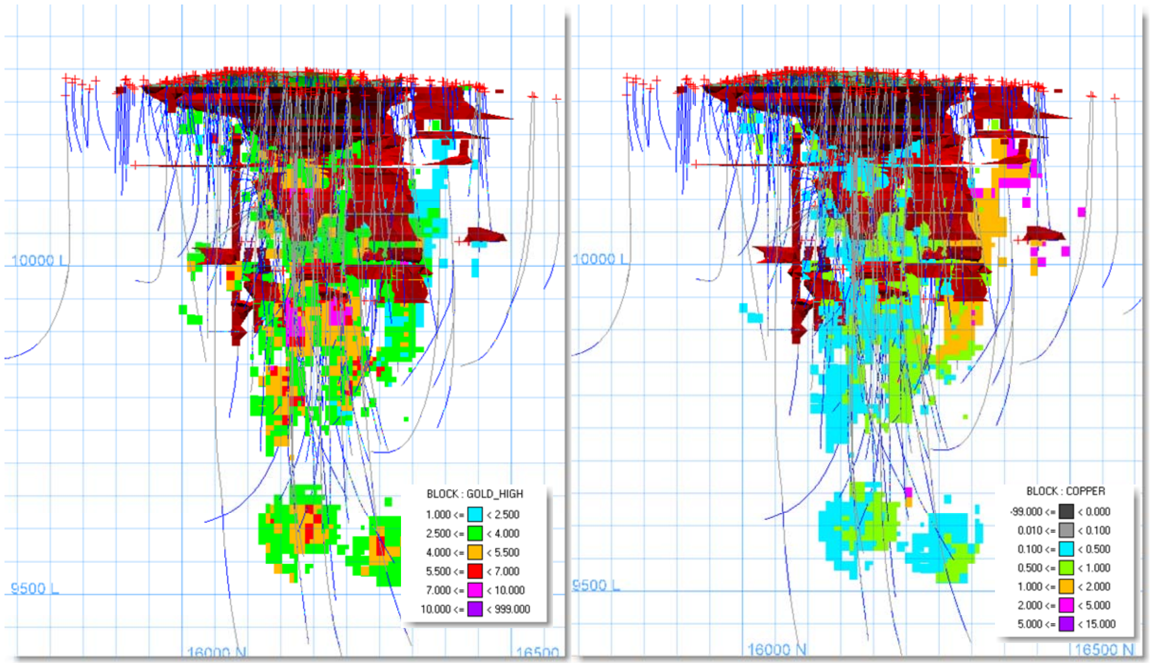


Figure 17-4: New Cobar Sulphide– Long Section Showing Gold and Copper Distribution for Reported Blocks



17.6.2 Chesney Oxide

Grade Estimation

H&S estimated Au using a three pass search strategy (Table 17-51). The lenses were estimated without using oxidation boundaries as sub-domains, as the Au grade distribution in each of the oxidation zones was considered to have a similar behaviour. The MIK estimates were produced for the parent blocks using the GS3 software package and then imported into Datamine software. The intra-class means were used during the MIK estimation process for all classes. H&S used an octant based search for sample selection with a set minimum number of octants required for each pass with a minimum of 2 samples per octant.

Copper was estimated using OK for both the mineralised lenses and adjacent zones separated by hard domain boundaries. The oxide, partial oxide and primary mineralisation zones were estimated separately for each lens using hard oxidation domain boundaries for selection of the composites. High grade cuts were not applied. The search strategy used was the same as that for the MIK model.

The reported gold resource for the Chesney Oxide deposit was based upon the MIK E-type estimate within the Lenses and for copper the OK estimates were used.

Density was assigned to the model using the average values for the oxide, transitional and primary mineralisation zones (Table 17-52).

Table 17-51: Chesney Oxide – Estimation Parameters for the Main Lens Grade Estimation

Pass	Search Radii (m)			Composites		Min. No. Octants	Classification
	Major	Semi-Major	Minor	Minimum	Maximum		
1	20	20	4	8	32	4	Measured
2	40	40	8	8	32	4	Indicated
3	80	80	16	8	32	2	Inferred

Search Direction

Axes	Minor	Semi-Major	Major
Main Lens	05→265	00→355	85→085 (Main Lens)
East Lens	05→270	00→360	85→090 (East Lens)
Surrounds	05→267	00→357	85→087 (Surrounds)

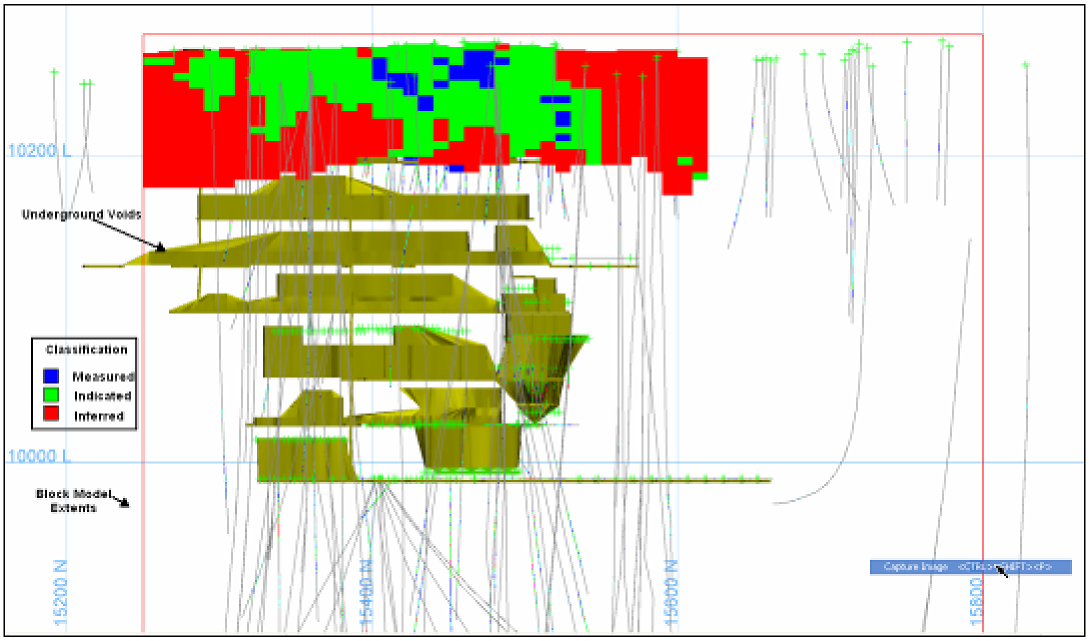
Table 17-52: Chesney Oxide – Density Used for Tonnage Estimation

OXIDATION DOMAIN	Bulk Density (t/m3)
Primary	2.78
Transitional	2.73
Oxide	2.53

Classification

The Chesney Oxide estimate was classified primarily based upon the search pass criteria. Blocks estimated during pass 1 were classified as Measured, pass 2 blocks were classified as Indicated, and pass 3 as Inferred (Figure 17-5).

Figure 17-5: Chesney Oxide – Long Section Showing Resource Classification of Model Blocks



17.6.3 Chesney Sulphide

PGM estimated Au grade using MIK to generate an E-type mean. Cu, Pb, Zn, Ag, Bi and Density were estimated using OK. As a check for the MIK E-Type estimate, Au was also interpolated using OK. OK was used to estimate the hangingwall and footwall waste zones to the mineralised domains. A five pass octant based search was used for the Au MIK estimate (Table 17-53), with a minimum of 2 samples within 4 octants required for each pass. No high grade cutting or high grade limiting was used. Soft domain boundaries were used to estimate Au within the Main Lens Au Lode (CHA) and the Eastern Au Lens (CHE).

All elements, including Au and Density, were estimated by OK within the wider Main Lode Copper (CHM) envelope and Eastern Lens (CHE) respectively, using soft domain boundaries. A final Au field in the block model was populated with Au (MIK) values from the CHA and CHE domains and with Au (OK) values from the CHM and WST (hangingwall and footwall domains respectively).

Table 17-53: Chesney Sulphide -Estimation Parameters for the Gold MIK Estimation

Parameter	Search Radii (m)			Search Type	Composites		Min No. Octants	Classification
	Minor	Semi-Major	Major		Minimum	Maximum		
Pass 1	4	20	40	Rotation	8	40	4	Measured
Pass 2	8	40	40	Rotation	8	40	4	Indicated
Pass 3	16	60	60	Rotation	8	40	4	Inferred
Pass 4	20	80	80	Rotation	4	40	4	Inferred

Search Orientation (Vulcan Software)		
Bearing (Z)	**Plunge (Y')**	**Dip (X'')**
85	-85	0

Classification

The resource classification for Chesney Sulphide was based upon the search pass criteria used for interpolation (Table 17-53). Blocks estimated during pass 1 were classified as Measured, pass 2 were classified as Indicated, pass 3 and 4 blocks were classified as Inferred. Blocks between 15,200m N and 15,400m N and below 9,600m RL that were estimated during pass 3 or pass 4 were not classified as drilling in this area is extremely sparse.

Figure 17-6 and Figure 17-7 show drilling, the resource classification and grade distribution for the reported blocks.

Figure 17-6: Chesney Sulphide – Long Section Showing Mineral Resource Classification for Reported Blocks

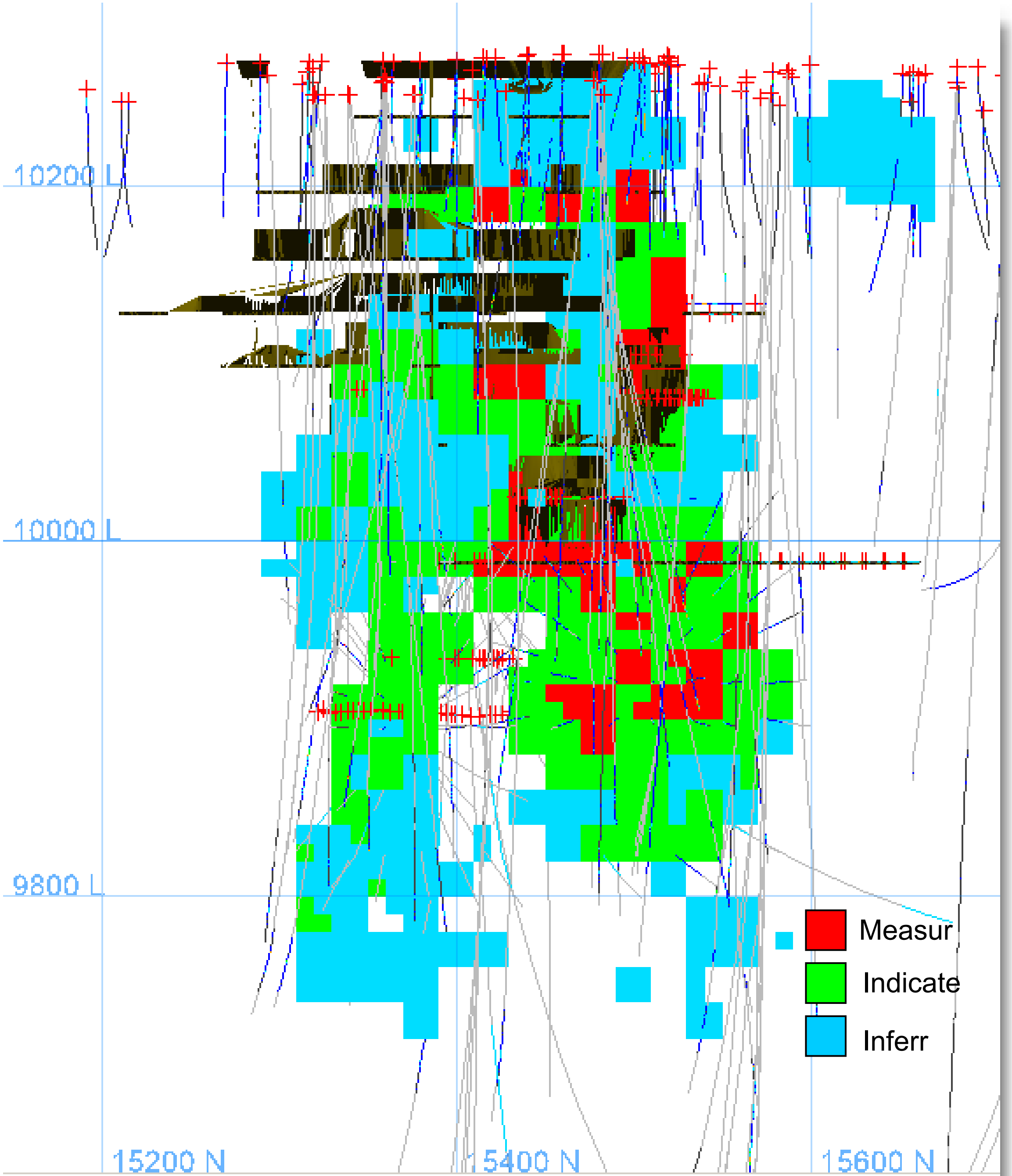
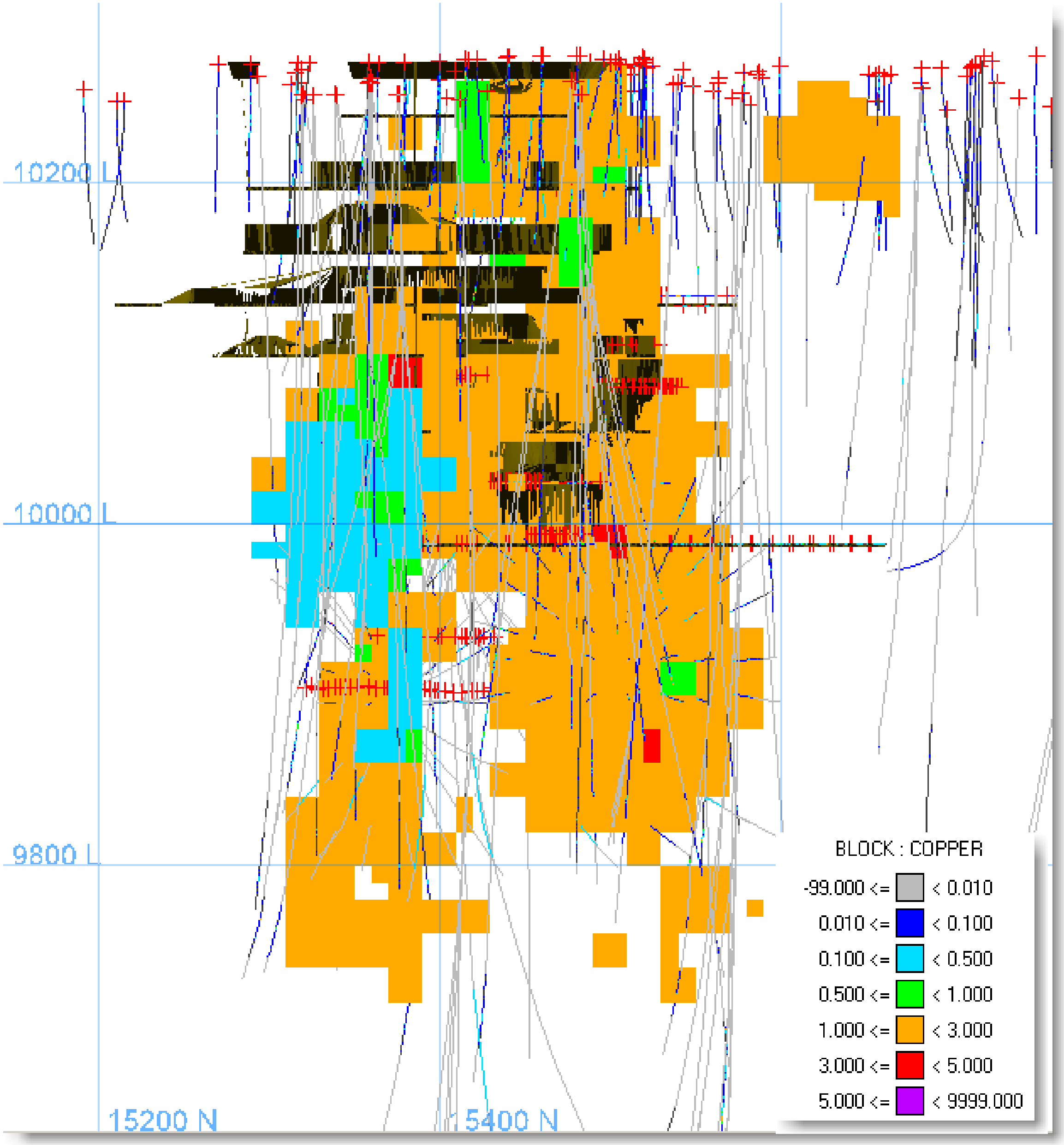


Figure 17-7: Chesney Sulphide – Long Section Showing Cu Grades for Reported Blocks



17.6.4 New Occidental

PGM estimated Au grade using MIK to generate E-Type means. Cu, Pb, Zn, Ag, Bi and Density interpolation were completed using OK. Gold was also interpolated by OK as a check on the MIK estimate. A four pass, octant based search strategy was used with a minimum of 2 samples within 4 octants required for each pass. Hard domain boundaries were used for Au and Bi, while all other variables were estimated with soft domain boundaries. Table 17-54 summarises the search strategy for the MIK and OK estimations. No high grade cutting or high grade limiting was used. Intra-class means were used for the estimation of the E-type mean for each of the zones, except for the Gossan where the median grade was used for the top bin.

Table 17-54: New Occidental – Estimation Parameters for MIK and OK Estimations

Parameter	Search Radii (m)			Search Type	Composites		Min No. Octants	Classification
	Minor	Semi-Major	Major		Minimum	Maximum		
Pass 1	3	15	20	Ellipse	4	8	4	Measured
Pass 2	6	30	40	Ellipse	4	8	4	Indicated
Pass 3	12	60	80	Ellipse	2	8	4	Inferred
Pass 4 (OK only)	18	90	90	Ellipse	2	8	4	Not Classified

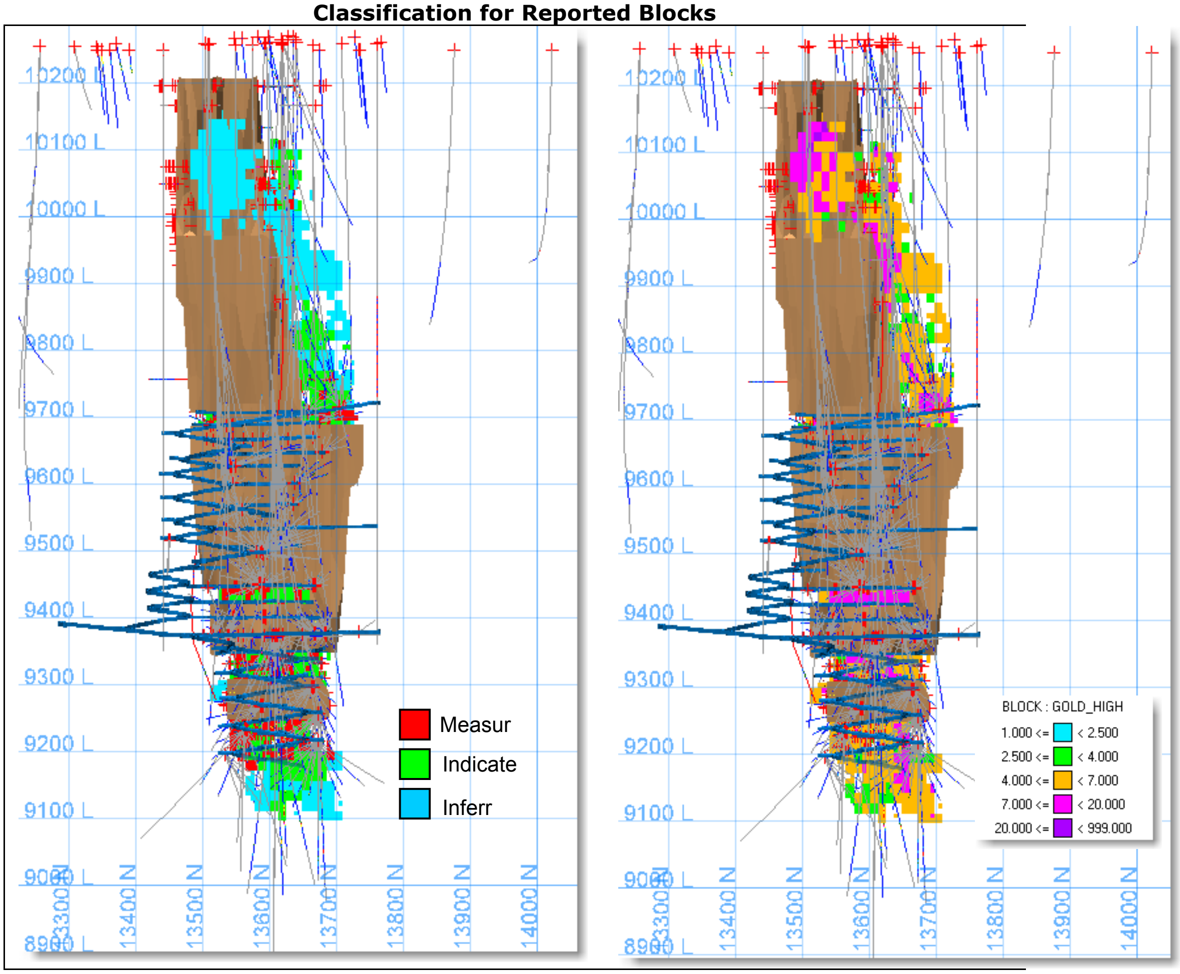
Search Orientation (Vulcan Software)			
	Bearing (Z)	**Plunge (Y')**	**Dip (X'')**
Au	95	-87	0
Other Elements	90	-90	0

Classification

The resource classification for New Occidental was based upon the search pass criteria used for interpolation (Table 17-54), where pass 1 was classified as Measured, pass 2 was classified as Indicated and pass 3 was classified as Inferred. Blocks estimated during pass 4 were not classified.

Figure 17-8 shows the Au grade and mineral resource classification for the reported blocks.

Figure 17-8: New Occidental – Long Section Showing Gold Grade and Mineral Resource Classification for Reported Blocks



17.6.5 Peak

PGM generated the Peak grade estimates using Vulcan mining software. For Au, grade interpolation was completed with both MIK (E-Type estimate) and OK. Cu, Pb, Zn, Ag and Density interpolation was completed using OK. A four pass, octant based search was used for the estimations with a minimum of 2 samples within 4 octants required for each pass (Table 17-55).

Table 17-55: Peak – General Sample Search Parameters for MIK and OK Estimations

Parameter	Search Radii (m)			Search Type	No. Samples		Min No. Octants
	X	Y	Z		Minimum	Maximum	
Pass 1	3	15	15	Ellipse	8	40	4
Pass 2	6	30	30	Ellipse	8	40	4
Pass 3	12	60	60	Ellipse	8	40	2
Pass 4 (OK only)	18	60	60	Box	4	40	2

Gold – MIK Estimation

Using MIK to generate an E-type mean, Au was estimated into the CWLZ Lens and the East Lens domains. A limit of influence was placed on high grades to prevent potential smearing of metal into low grade areas (see Table 17-56). The dimensions of the limit were based on the variography of the top indicator class for both the CWLZ Lens and East Lens. For the estimation of Au within the top indicator class, the mean value was used for both the CWLZ and East Lenses.

Table 17-56: Peak – MIK Sample Selection and Search Parameters for Gold

Search Orientation (Vulcan Software)					
Zone	**Bearing (Z)**		**Plunge (Y')**	**Dip (X'')**	
East Upper	80		-74	0	
East Lower	270		-74	0	
CWLZ North	285		-85	0	
CWLZ South	285		-85	0	
High Grade Limit (g/t Au)					
Threshold	50	50	50	50	50
Major radius	40	40	60	60	60
Semi-Major radius	30	30	50	50	50
Minor radius	4	4	6	6	6

Gold – OK Estimation

OK was also used to estimate Au into each of the modelled domains. As with the MIK estimate, a limit of influence was used to restrict high grade samples. In addition to the restriction, a high grade cut was used for the East, CWLZ, Deeps and Contact Lenses (Table 17-57). The high grade cut was used to control the influence of extreme grade values.

Table 17-57: Peak – OK Sample Selection and Search Parameters for Gold Copper, Lead, Zinc and Silver OK Estimations

Lens	**East Upper**	**East Lower**	**CWLZ North**	**CWLZ South**	**Waste**	**Regolith**	**Contact**	**Deeps**
Grade cut (g/t Au)	500	500	500	500			200	200
Search Orientation (Vulcan)								
Bearing (Z)	80	270	280	260	90	90	90	90
Plunge (Y')	-74	-74	-85	-85	-90	-90	-90	-90
Dip (X'')	0	0	0	0	0	0	0	0
Sample Selection	East/CWLZ	East	CWLZ/Deep	CWLZ /Deep	Waste/ Regolith	Waste/Rego lith/CWLZ	Contact	Deep/ CWLZ
High Grade Limit								
Threshold (g/t Au)	50	50	50	50	50		100	50
Major radius (m)	40	40	60	60	60		15	60
Semi-Major radius (m)	30	30	50	50	20		15	50
Minor radius (m)	4	4	6	6	6		3	6

Ordinary kriging was used to estimate the Cu, Pb, Zn and Ag block grades. Estimates for Cu and Pb in the Waste and Contact Lenses borrowed variography from CWLZ lens. For estimates of Zn and Ag, the Waste, Regolith and Contact lenses also borrowed the variography from the CWLZ lens. A limit of influence was used to restrict high grade samples greater than 500g/t Ag for the Ag estimate in the regolith to 30m in the major direction, 30m in the semi-major direction and 10m in the minor direction. The sample selection and search parameters are presented in Table 17-58.

Density

Density was estimated using OK. The estimate was completed using soft domain boundary sample selection with the exception of the regolith domain. The search orientation was: major 90°→090°; semi major 00°→000°; and minor 00°→090°.

Default bulk density values of 2.53t/m^3 and 2.73t/m^3 were used for the completely oxidised mineralisation (COX) and the partially oxidised mineralisation (POX), respectively.

Classification

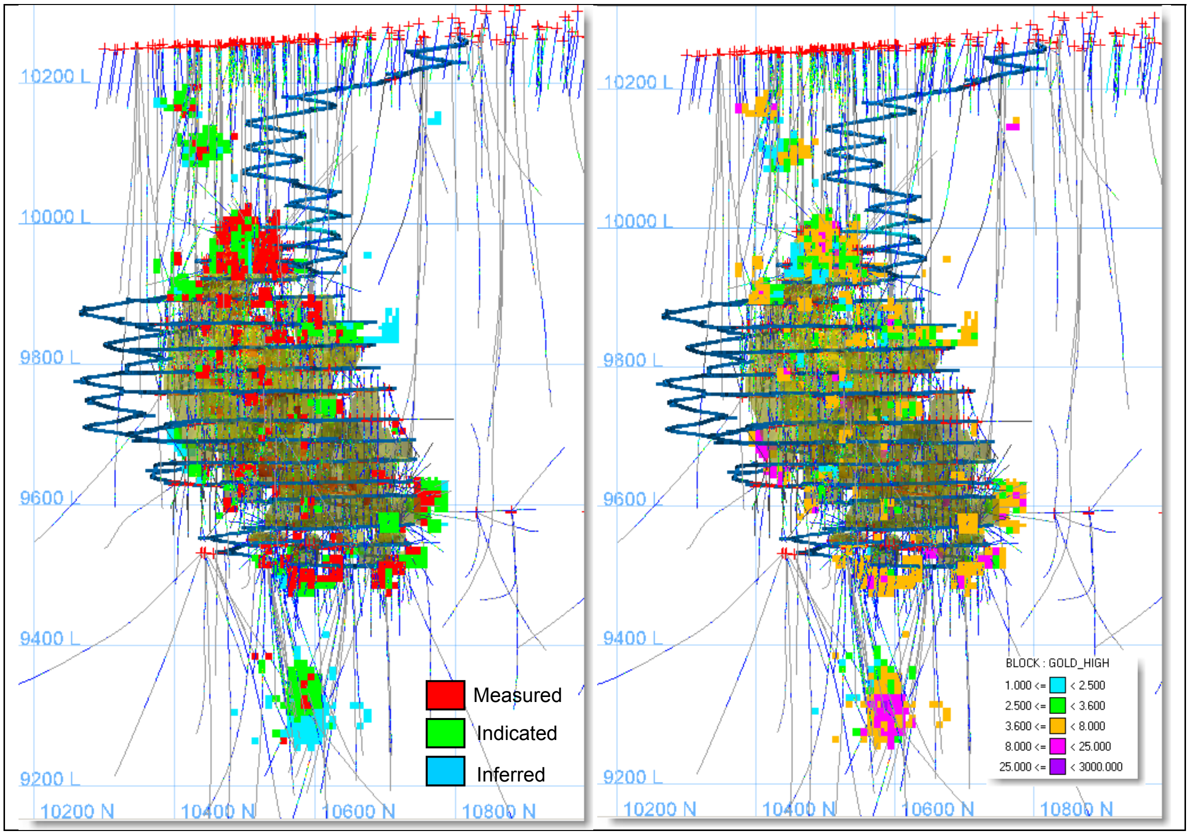
- The mineral resource was classified according to the search pass in which the block estimate was generated. Blocks coded as pass 1 were classified as Measured, blocks coded as pass 2 were classified as Indicated and block coded as pass 3 were classified as Inferred. Blocks coded as pass 4 were not classified.

Figure 17-9 shows the Au grade and mineral resource classification for the reported blocks.

Table 17-58: Peak - OK Sample Selection and Search Parameters for Copper, Lead, Zinc and Silver

Lens	East	East	CWLZ	CWLZ	Waste/Cont	Regolith	Deeps
Search Orientation (Vulcan Software)							
Copper							
Bearing (Z)	80	270	285	260	90	90	90
Plunge (Y')	-74	-74	-85	-85	-90	-90	-90
Dip (X'')	0	0	0	0	0	0	0
Sample	East/CWL	East	East/CWLZ	East/CWLZ	Waste/regolit	Waste/regoli	Deep/CWL
Lead							
Bearing (Z)	80	270	260	260	90	90	90
Plunge (Y')	-74	-74	-85	-85	-90	-90	-90
Dip (X'')	0	0	0	0	0	0	0
Sample	East/CWL	East	East/CWLZ	East/CWLZ	Waste/regolit	Waste/regoli	Deep/CWL
Zinc							
Bearing (Z)	90	270	260	260	90	90	90
Plunge (Y')	-74	-74	-85	-85	-90	-90	-90
Dip (X'')	0	0	0	0	0	0	0
Sample	East/CWL	East	East/CWLZ	East/CWLZ	Waste/regolit	Waste/regoli	Deep/CWL
Silver							
Bearing (Z)	75	255	260	280	90	90	90
Plunge (Y')	-74	-74	-85	-85	-90	-90	-90
Dip (X'')	0	0	0	0	0	0	0
Sample	East/CWL	East	East/CWLZ	East/CWLZ	Waste/regolit	Waste/regoli	Deep/CWL

Figure 17-9: Peak – Long Section Showing Gold Grade and Classification for Reported Blocks



17.6.6 Perseverance

For Zone A and Zone D, Au was estimated using MIK to generate the E-type mean, while the other elements including gold (for comparison purposes) were estimated using OK. For Zone B, Hulk, and Hercules Lenses, OK was used to estimate all elements, with high grade cuts applied to Au. An octant based search was used with a minimum of 8 octants with 2 samples per octant required for pass 1 and pass 2 and 4 octants with 2 samples per octant required for pass 3.

The search strategy for the Zone D MIK estimation is shown in Table 17-59. A four pass search strategy with octant searching. A minimum of 8 and maximum of 32 composites were required for pass 1 and pass 2. Pass 3 required a minimum of 4 samples and a maximum of 32. The search strategy for the OK estimates was similar. An octant based search was used with a minimum of 4 octants with 2 samples per octant required for passes 1 through 3 while pass 4 required only 2 octants.

Table 17-59: Perseverance – MIK Estimation Search Strategy for Zone D

Parameter	Search Radii (m)			Search Type	Samples		Min No. Octants
	Minor	Semi-Major	Major		Minimum	Maximum	
Pass 1	3	15	15	Ellipse	8	32	4
Pass 2	6	30	30	Ellipse	8	32	4
Pass 3	12	30	30	Ellipse	4	32	4
Pass 4	12	60	60	Ellipse	4	32	2

The PGM estimation search strategy for Zone A is shown in Table 17-60.

Table 17-60: Perseverance - OK Estimation Search Strategy for Zone A

Parameter	Search Radii (m)			Search Type	Samples		Min No. Octants
	Minor	Semi-Major	Major		Minimum	Maximum	
Pass 1	3	15	15	Ellipse	8	32	4
Pass 2	6	30	30	Ellipse	8	32	4
Pass 3	12	60	60	Ellipse	8	32	4
Pass 4	20	60	60	Ellipse	8	32	2

Table 17-61 summarises the orientation of the search ellipsoids and variogram models for each zone estimated. Each zone was estimated separately using hard domain boundaries. The two sub-domains within Zone D were estimated using soft boundaries.

Table 17-61: Perseverance – Search Orientations for Zone D, Zone B, Hulk and Hercules Lens

Lens	Zone D, North Domain	Zone D, South Domain	Zone B	Hulk Lens	Hercules Lens
Z Rotation	105	270	80	80	90
Y Rotation	-85	-10	-85	-75	-90
X Rotation	0	0	0	0	0
Main Axis Orientation	85°→105°	80°→270°	85°→080°	75°→080°	90°→90°

Table 17-62 summarises the search and variogram orientations for Zone A.

Table 17-62: Perseverance – Search and Variogram Orientations for PGM Zone A Estimations

Zone	Z Rot.	Y Rot.	X Rot.
Gold Zone A Search and Variography Orientations			
Sub domain 1	265	-55	0
Sub domain 2	265	-81	0
Sub domain 3	100	-87	0
Other Elements Variography			
All Domains	90	-90	0
Other Elements Search			
Ag	270	-85	0
Bi	90	-90	0
Cu - Footwall	90	-87	0
Cu – Other	90	-90	0
Pb	270	-85	0
Zn	90	-90	0
Density	90	-90	0
Fe	90	-90	0

Classification

The resource was classified based upon estimation search pass and geological continuity. For Zone A, pass 1 was classified as Measured, pass 2 as Indicated and pass 3 as Inferred. For the other zones, blocks that were estimated in pass 1 were classified as Indicated and blocks that were estimated in pass 2 or pass 3 were classified as Inferred.

Figure 17-10: Perseverance – Long Section Showing Mineral Resource Classification for Reported Blocks

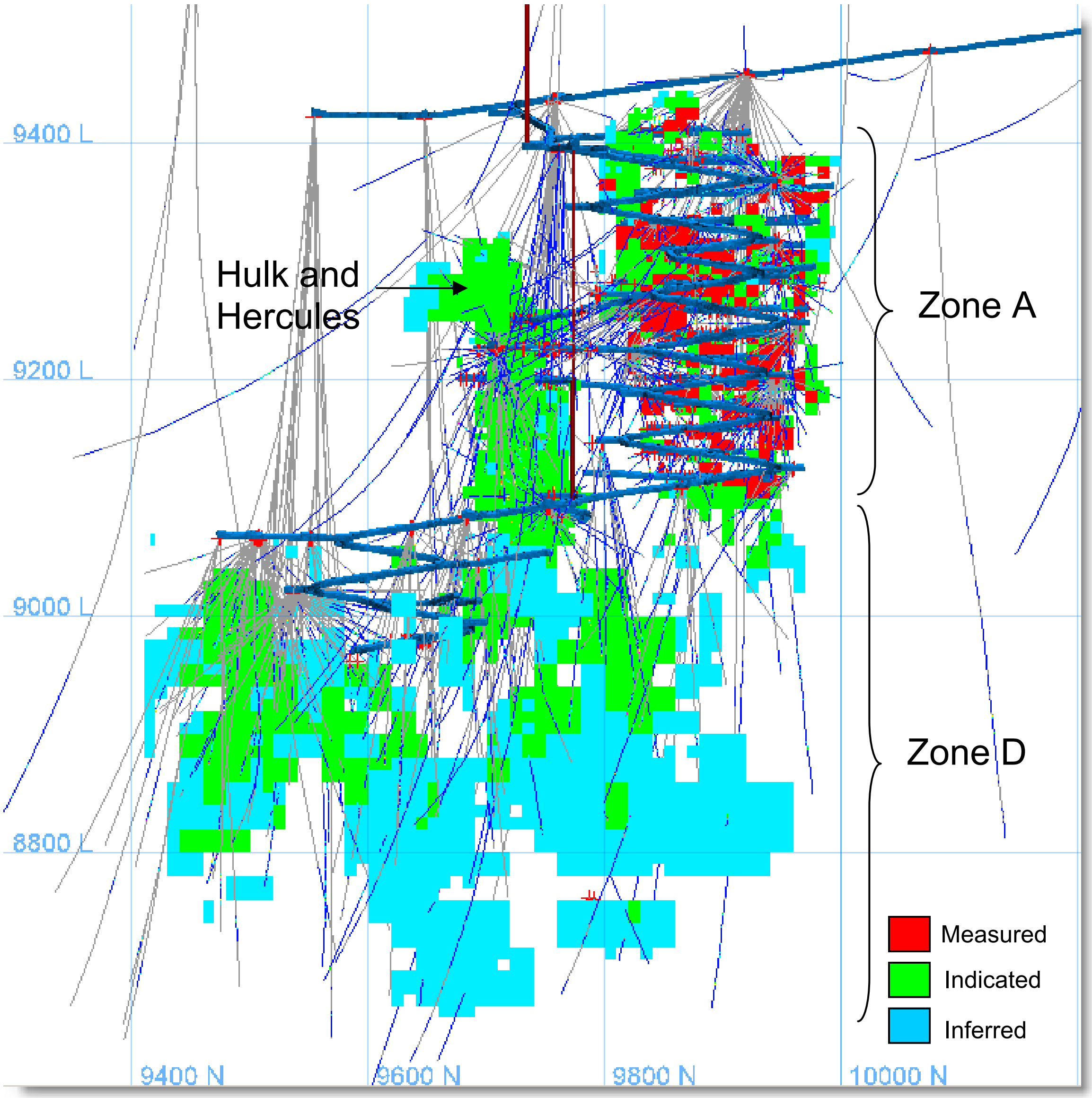
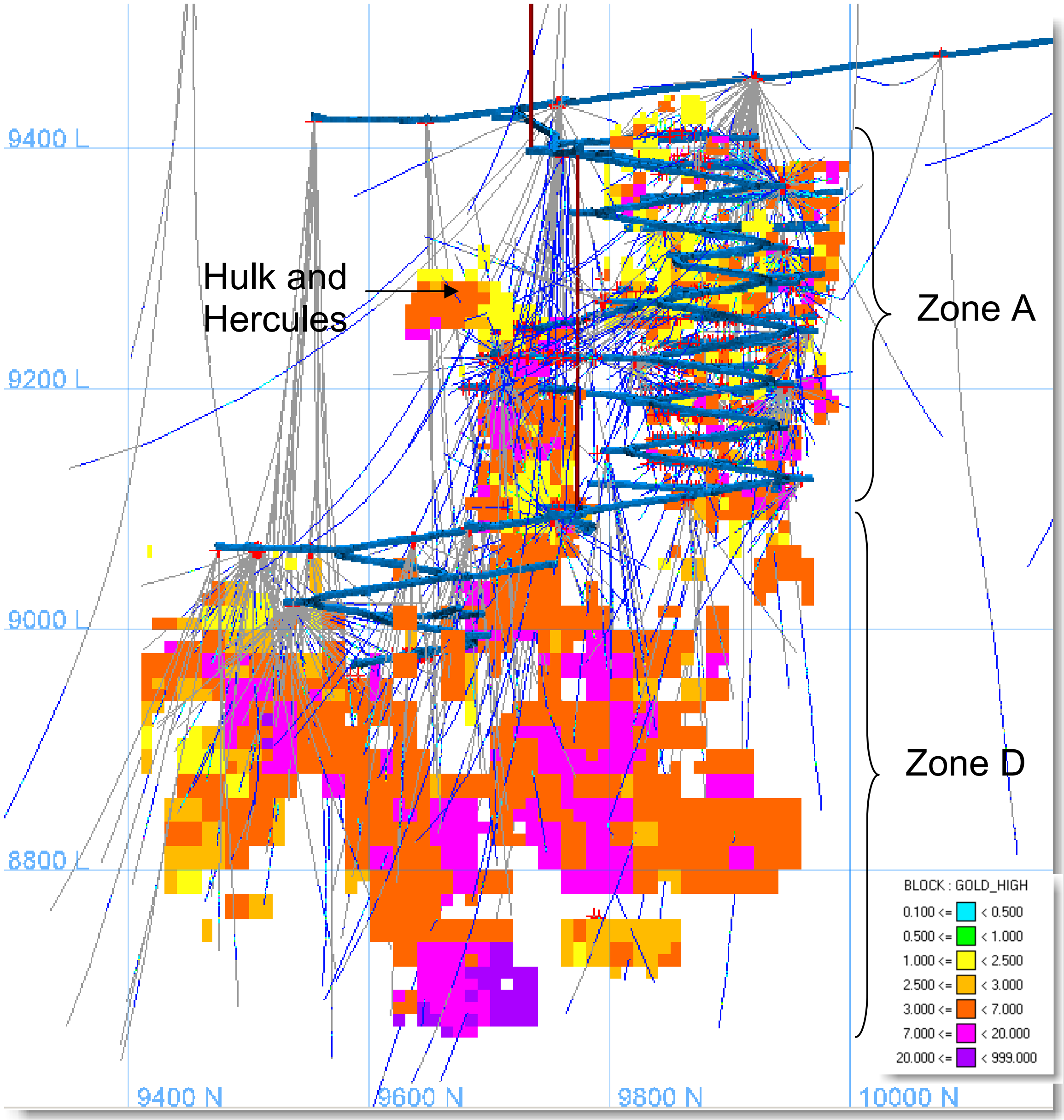


Figure 17-11: Perseverance – Long section Showing Gold Grade for Reported Blocks



17.7 Resource Reporting

The NSR value was calculated by PGM to take into account gold, copper and silver grades, metallurgical recoveries and deleterious element penalties. The NSR cut-off of AU$95 represents in-situ gold and copper grades in the range of between approximately 3.39 g/t Au, 6g/t Ag and 0.02% Cu to 1.0 g/t Au, 6 g/t Ag and 2.16% Cu.

For the Peak Upper resource, significant grades of both lead and zinc were experienced and were treated as credits in the NSR calculation. Penalty elements include Bi and Pb (for Perseverance ore). The NSR values were coded into the site Vulcan block models using scripts by PGM staff.

The assumptions used for the NSR calculation in the December 31, 2008 resource reporting are as follows:

- Gold price – US$750/oz;
- Cu price – US$2.00/lb;
- Ag price – US$10.00/oz; and
- Exchange rate – US$0.80.

For the Peak Upper Pb/Zn zone and remnants of the Peak orebody the following assumptions were used in the NSR calculation:

- Zn price – US$0.80/lb; and
- Pb price – US$0.80/lb.

In general terms, the NSR is equivalent to the revenue generating capacity of the material less the downstream costs and penalties associated with recovering the contained metal. The NSR formula incorporates parameters for multi element metal prices (e.g. Au, Ag, cu), metallurgical recoveries, smelting and refining costs and penalties, government royalties and transport costs. The NSR calculation can be summarised as:

NSR= Revenue of Recovered Metals – (Smelter Cost + Refining Cost and Penalties + Royalties +Transport Costs)

The A$95 / AU$86 cut-off grade was calculated by PGM staff using the following formulae:

Cut-off ($/t) = mining costs + processing costs + administrative costs + sustaining capital.

The US$750/oz gold price is significantly below the 31 December, 2008 spot gold price of US$865/oz and the Cu price used of US$2.00/lb is above the 31 December, 2008 spot copper price of US$1.32/lb.

The resources for the PGM properties have been classified as Measured, Indicated and Inferred Resources based upon the criteria discussed in Section 17.6. The grade tonnage reports for each deposit are provided as Table 17-63.

Table 17-63: PGM Statement of Mineral Resources (Exclusive of Reserves) as of 31 December 2008.

Deposit	Group	Class	Tonnes	Au (g/t)	Cu %	Au Oz	Cu Mlb
Chesney	**Measured + Indicated**	**Measured**	534,602	1.11	1.31	19,079	15
		Indicated	493,059	1.92	1.34	30,436	15
	M&I		**1,027,661**	**1.50**	**1.32**	**49,560**	**30**
	Inferred	**Inferred**	624,947	1.98	1.52	39,783	21
Chesney Total			**1,652,608**	**1.68**	**1.40**	**89,263**	**51**
New Cobar	**Measured + Indicated**	**Measured**	249,153	3.58	0.61	28,677	3
		Indicated	346,378	3.65	0.58	40,648	4
	M&I		**595,531**	**3.62**	**0.59**	**69,311**	**8**
	Inferred	**Inferred**	613,354	3.49	0.66	68,822	9
New Cobar Total			**1,208,885**	**3.55**	**0.62**	**137,976**	**17**
New Occidental	**Measured + Indicated**	**Measured**	94,832	6.09	0.14	18,568	0
		Indicated	138,852	5.01	0.19	22,366	1
	M&I		**233,684**	**5.45**	**0.17**	**40,946**	**1**
	Inferred	**Inferred**	196,942	5.13	0.18	32,482	1
New Occidental Total			**430,626**	**5.30**	**0.17**	**73,378**	**2**
Peak	**Measured + Indicated**	**Measured**	280,528	3.94	0.51	35,536	3
		Indicated	219,479	2.81	0.65	19,829	3
	M&I		**500,007**	**3.44**	**0.57**	**55,300**	**6**
	Inferred	**Inferred**	99,531	5.53	0.48	17,696	1
Peak Total			**599,538**	**3.79**	**0.56**	**73,055**	**7**
Perseverance	**Measured + Indicated**	**Measured**	191,569	4.88	1.50	30,056	6
		Indicated	652,026	4.54	1.02	95,173	15
	M&I		**843,595**	**4.62**	**1.13**	**125,305**	**21**
	Inferred	**Inferred**	975,054	6.27	0.66	196,556	14
Perseverance Total			**1,818,649**	**5.50**	**0.88**	**321,590**	**35**

Class	Tonnes	Au (g/t)	Cu %	Au Oz	Cu Mlb
Measured	1,350,684	3.04	0.96	132,014	29
Indicated	1,849,794	3.50	0.92	208,153	37
M & I	**3,200,478**	**3.31**	**0.94**	**340,592**	**66**
Inferred	2,509,828	4.40	0.83	355,049	46
Total	**5,710,306**	**3.79**	**0.89**	**695,808**	**112**

(1) The Mineral Resources for the Peak Mine deposits set out in the table above have been estimated by R. Berthelsen who is a qualified person under NI 43-101 and a competent person under the JORC Code. The Mineral Resources are classified as Measured, Indicated and Inferred, and are based on the JORC Code.

(2) The Mineral Resources were estimated using three-dimensional multiple indicator and ordinary kriged block models, constrained by geological and grade domains.

(3) A$95 net smelter return cut-off was applied to all in-situ Mineral Resources, along with appropriate recoveries, for Peak, Perseverance, and New Occidental and a A$86 cut-off for New Cobar and Chesney.

(4) Excluded from the Identified Mineral Resources are mined material and material unlikely to be converted to reserve status for engineering or technical reasons and remnant stope pillars, skins and other material sterilized as a result of mining as well as discontinuous mineralization.

(5) Mineral Resources are not known with the same degree of certainty as Mineral Reserves and do not have demonstrated economic viability.

(6) Numbers may not add up due to rounding.

Table 17-64: Measured, Indicated and Inferred Mineral Resources (1) (2) (3) (4) (5) (6) (including Proven and Probable Ore Reserves)

Deposit	Class	Tonnes ('000)	Au (g/t)	Cu %	Gold oz ('000)	Cu Mlb
Chesney	Measured	765	1.52	1.63	37	27
	Indicated	776	1.95	1.64	49	28
	M&I	**1,541**	**1.74**	**1.63**	**86**	**56**
	Inferred	674	1.92	1.67	42	25
Chesney Total		**2,216**	**1.79**	**1.64**	**128**	**80**
New Cobar	Measured	429	4.14	0.69	57	7
	Indicated	585	4.10	0.63	77	8
	M&I	**1,014**	**4.11**	**0.66**	**134**	**15**
	Inferred	649	3.58	0.65	75	9
New Cobar Total		1,663	3.91	0.65	209	24
New Occidental	Measured	205	6.81	0.13	45	1
	Indicated	268	5.63	0.19	49	1
	M&I	**473**	**6.14**	**0.16**	**93**	**2**
	Inferred	215	5.17	0.17	36	1
New Occidental Total		**688**	**5.84**	**0.17**	**129**	**3**
Peak	Measured	653	5.02	0.59	105	8
	Indicated	527	4.10	0.70	69	8
	M&I	**1,180**	**4.61**	**0.64**	**175**	**17**
	Inferred	125	5.60	0.47	23	1
Peak Total		**1,305**	**4.70**	**0.62**	**197**	**18**
Perseverance	Measured	243	4.47	1.76	35	9
	Indicated	1,538	6.39	1.07	316	36
	M&I	**1,781**	**6.12**	**1.16**	**350**	**46**
	Inferred	1,048	6.34	0.67	214	16
Perseverance Total		**2,830**	**6.20**	**0.98**	**564**	**61**
Stockpiles	Measured	110	2.87	0.67	10	2
Stockpiles Total		**110**	**2.87**	**0.67**	**10**	**2**
Total	**Class**	**Tonnes**	**Au (g/t)**	**Cu %**	**Gold (oz)**	**Cu Mlb**
	Measured	2,405	3.75	1.02	290	54
	Indicated	3,694	4.71	1.00	559	82
	M&I	**6,100**	**4.33**	**1.01**	**849**	**136**
	Inferred	2,713	4.45	0.87	388	52
	Total	**8,812**	**4.37**	**0.97**	**1,238**	**188**

(1) The Mineral Resources for the Peak Mine deposits set out in the table above have been estimated by R. Berthelsen who is a qualified person under NI 43-101 and a competent person under the JORC Code. The Mineral Resources are classified as Measured, Indicated and Inferred, and are based on the JORC Code.

(2) The Mineral Resources were estimated using three-dimensional multiple indicator and ordinary kriged block models, constrained by geological and grade domains.

(3) A$95 net smelter return cut-off was applied to in-situ Mineral Resources at Peak, Perseverance and New Occidental while at New Cobar and Chesney A$86 was used, along with appropriate recoveries.

(4) Excluded from the Identified Mineral Resources are mined material and material unlikely to be converted to reserve status for engineering or technical reasons and remnant stope pillars, skins and other material sterilized as a result of mining as well as discontinuous mineralization.

(5) Mineral Resources are not known with the same degree of certainty as Mineral Reserves and do not have demonstrated economic viability.

(6) Numbers may not add up due to rounding.

The Mineral Resources reported have been subjected to a test of having a reasonable expectation of being economically extracted by mining methods similar to those currently being used at PGM operations at present. There are no known factors related to mining, metallurgy or infrastructure or any other such factor that may materially affect the reasonable economic extraction of the Resources or Reserves.

Where Mineral Resources and Mineral Reserves are detailed, there are no material affects known from issues such as environmental, permitting, legal, title, taxation, socio-economic, marketing, political or any other relevant issue.

17.8 Mineral Reserve Methodology

17.8.1 Estimation of Cut Off Costs

Net Smelter Return ("NSR") is the revenue generating capacity of a particular block of material, less the downstream value adding costs. The NSR may be summarized as:

NSR= Revenue of Recovered Metals – (Smelter Cost + Refining Cost and Penalties + Royalties +Transport Costs)

PGM measures the NSR in Australian dollars and the NSR grade (NSR/tonne) is measured in Australian Dollars per tonne (A$/t).

PGM calculates the NSR within the resource model using the criteria shown in Table 17-65 to Table 17-67.

Table 17-65: Metal Sales Prices

Metal	Unit	Sales Price US$
Gold	troy oz	750
Copper	lb	2.00
Silver	troy oz	10.00
Lead	lb	0.80
Zinc	lb	0.80
Exchange rate	AUD/USD	0.80

Table 17-66: Process Recovery

Activity	Process Recovery %
Gold	89%
Copper	76%
Silver	61%

Table 17-67: Smelting, Refining Costs and Penalties, Royalties and Transport Costs

Cost Item	Unit	Unit Cost US$
Bullion Refining		
Gold	Per oz	1.78
Silver	Per oz	0.02
Concentrates		
Freight cost US$	Per t concentrate	50+(80*USD/AUD)
Smelting Treatment Charge	Per t concentrate	50.00
Copper Refining	Per lb	0.07
Gold Refining	Per oz	4.635
Silver Refining	Per oz	0.3605
Bi Penalty (each 50g>500g/t in concentrate)	Per t concentrate	2.00
Other		
Royalty	Gross Sales Value	3%
Moisture in Concentrate		9%

PGM has developed a mineral reserve cut off strategy that incorporates an opportunity cost as opposed to the breakeven NSR formula used prior to 2008. PGM has also developed a cut off for individual ore sources. The rational behind the development of a cut-off strategy for individual ore sources originates from work conducted on capacity constraints. All ore sources have been allocated costs of mining, processing, administration and sustaining capital on the basis of marginal costing.

The Cut Off Cost is estimated where:

Cut off NSR (A$/t) = Site Costs of Production + Opportunity Cost

PGM defines the site costs of production and opportunity cost as follows:

Site Costs of Production = [Operating Costs for Mine+Mill+Administration] + [Sustaining Capital for Mine + Mill Capital + Plant and Equipment+ Exploration Costs]

Opportunity Cost = Cost of Capital multiplied by Net Present Value of Operation (Optimised) divided by Annual Production Rate

Mineral Reserves for each orebody as at the December 2008 are estimated using a forecast of 2009 activity based on the 2008 actual expenditure as shown in Table 17-68.

Table 17-68: Estimate of 2009 Site Costs of Production by orebody

Cost Function	Mining Cost							Processing Cost	Administration Cost	Sustaining Capital		Project Capital	Opportunity Cost	Total	Rounded Total
Cost Function Subset	Operating Development	Longhole Drill and Blast	Bogging	Hauling	Hoisting	Surface Haulage	Delineation Drilling			Capital Development	Plant and Equipment	Exploration			(used for reserves)
OreBody															
POX						$ 3.26		$ 24.29	$ 8.06		$ 5.00	$ 3.13	$ 6.34	$ 50.07	$ 50.00
Upper Occidental Fall Material			$ 8.98	$ 11.83	$ 8.33			$ 24.29	$ 8.06		$ 11.76	$ 3.13	$ 6.34	$ 82.72	$ 80.00
Peak	$ 12.00	$ 9.39	$ 8.98		$ 8.33		$ 3.70	$ 24.29	$ 8.06	$ 22.94	$ 11.76	$ 3.13	$ 6.34	$ 118.91	$ 120.00
Percy Zone A	$ 12.00	$ 9.39	$ 8.98	$ 11.83	$ 8.33		$ 3.70	$ 24.29	$ 8.06	$ 22.94	$ 11.76	$ 3.13	$ 6.34	$ 130.74	$ 130.00
Hulk	$ 10.00	$ 9.39	$ 8.98	$ 11.83	$ 8.33		$ 3.70	$ 24.29	$ 8.06		$ 11.76	$ 3.13	$ 6.34	$ 105.80	$ 105.00
Hercules	$ 12.00	$ 9.39	$ 8.98	$ 11.83	$ 8.33		$ 3.70	$ 24.29	$ 8.06	$ 22.94	$ 11.76	$ 3.13	$ 6.34	$ 130.74	$ 130.00
Lower New Occidental	$ 12.00	$ 9.39	$ 8.98	$ 11.83	$ 8.33		$ 3.70	$ 24.29	$ 8.06	$ 22.94	$ 11.76	$ 3.13	$ 6.34	$ 130.74	$ 130.00
Percy Zone D	$ 12.00	$ 9.39	$ 8.98	$ 11.83	$ 8.33		$ 3.70	$ 24.29	$ 8.06	$ 22.94	$ 11.76	$ 3.13	$ 6.34	$ 130.74	$ 130.00
Chesney	$ 15.08	$ 10.25	$ 7.87	$ 5.62		$ 3.26	$ 3.70	$ 24.29	$ 8.06	$ 12.43	$ 11.76	$ 3.13	$ 6.34	$ 111.78	$ 112.00
New Cobar	$ 15.08	$ 10.25	$ 7.87	$ 5.62		$ 3.26	$ 3.70	$ 24.29	$ 8.06	$ 12.43	$ 11.76	$ 3.13	$ 6.34	$ 111.78	$ 112.00

17.8.2 Stope Planning, Dilution and Ore Losses

Vulcan Mine Planning software system is utilized at PGM for the design and planning of stoping operations. The block model is cross sectioned at 5m to 10m intervals and stope boundaries are digitised on screen.

Minimum planned stope width is 4m. Where Mineral Resources are less than 4m in width, additional adjacent material is included in the stope tonnage and grade such that minimum stope width, prior to the effects of external dilution, is 4m.

A stope may be re-designed several times in order to establish an optimum shape, suitable for mining. The digitised outlines on sections are then used to create a 3D solid model of each stope. A second stope shape is created that includes dilution; these diluted shapes are evaluated against the block model. An estimate of ore recovery is applied to each diluted stope followed by an estimate of rockfill dilution, at zero grade, that will be delivered to the process plant. The stope models are analysed for tonnes and grade using an NSR as defined by the Peak Mine cutoff policy for Mineral Reserves.

The dilution, ore losses and modifying factors estimated for the mining of the Mineral Reserves are shown in Table 17-69.

Table 17-69: Dilution, Ore Losses and Modifying Factors

Description	Average Modifying Factor
Dilution (By Area)	
New Occidental	15%
Perseverance	21%
Peak	7%
New Cobar	8.5%
Chesney	7%
Ore Recovery	95%
Dilution from Rockfill	3%

17.9 Mineral Reserve Statement

The Mineral Reserve Statement in compliance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 31, 2008) is shown in Table 17-70.

Table 17-70: Peak Gold Mines Pty Ltd –Mineral Reserve Statement dated 31 December 2008 (1) (2) (3) (4)

Deposit		Grade		Contained Metal	
	Tonnes	Au g/t	Cu %	Au Oz	Cu lb
New Occidental					
Proven	162,433	6.26	0.13	32,692	465,535
Probable	165,377	5.15	0.17	27,383	619,811
Total	327,811	5.70	0.15	60,074	1,084,048
Perseverance					
Proven	80,864	2.45	2.17	6,370	3,868,542
Probable	1,164,737	5.70	0.94	213,449	24,137,350
Total	1,245,601	5.49	1.02	219,858	28,009,986
Peak					
Proven	359,086	4.73	0.51	54,607	4,037,402
Probable	389,590	4.77	0.70	59,747	6,012,285
Total	748,676	4.75	0.61	114,335	10,068,327
New Cobar					
Proven	172,136	4.08	0.70	22,580	2,656,468
Probable	318,594	4.20	0.64	43,021	4,495,223
Total	490,730	4.16	0.66	65,634	7,140,368
Chesney					
Proven	239,278	1.58	1.94	12,155	10,233,830
Probable	448,208	2.18	1.83	31,414	18,082,766
Total	687,486	2.00	1.87	44,206	28,342,586
Surface Stockpiles					
Proven	34,176	4.55	0.85	4,995	640,434
Probable	-	0.00	0.00	-	-
Total	34,176	4.55	0.85	4,999	640,434
Reclaim Stockpiles					
Proven	-	0.00	0.00	-	-
Probable	70,210	2.00	0.65	4,515	1,006,110
Total	70,210	2.00	0.65	4,515	1,006,110
Total					
Proven	1,048,000	3.96	0.95	133,000	21,902,000
Probable	2,557,000	4.60	0.96	380,000	54,354,000
Total	3,605,000	4.40	0.96	514,000	76,292,000

1) The Mineral Reserves have been reported with the classification criteria of the CIM Definition Standards.
2) The Mineral Reserve was prepared by Eric Strom, a full-time employee of PGM who is a qualified person as defined under NI 43-101.
3) Tonnes and grade of Mineral Reserves are stated on the basis of delivery to plant.
4) Additions subject to the effects of rounding.

18 OTHER RELEVANT DATA AND INFORMATION

PGM is not aware of any relevant data or information that is not presented in this report.

19 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

19.1 Mining Operations

Current mine production operations are located in two underground districts. The local around Cobar area has been the subject of mining since the 1890's. Current mining is from zones, which are contiguous to, or nearby, earlier mined out areas.

The New Occidental and Perseverance mining zones are accessed via infrastructure originally developed to mine the Peak deposit. A ramp connection from surface was completed 2007 to reduce the reliance on the original shaft. Open Pit mining operations ceased at New Cobar in early 2004. The currently active New Cobar mining zone is below the open pit and is accessed via a decline mined from within the open pit. Development mined from the New Cobar decline has accessed the adjacent Chesney orebody with production scheduled to commence in early 2009. The underground workings for access to the New Cobar and Chesney zones are separate from those workings that access the Peak, New Occidental and Perseverance zones.

Production operations have been underway at New Occidental since 2001, Perseverance since 2003 and the underground operations at New Cobar since 2005.

An updated Mining Operation Plan for Peak, New Occidental, Perseverance, New Cobar and Chesney for the period 2005-2010 was approved by the Department of Primary Industries in 2005.

19.1.1 Underground Mining Operations

The New Occidental and Perseverance orebodies are accessed from the Peak Mine infrastructure. Historically, the New Occidental zone was mined from 56 Level (560 metres below surface) to surface, prior to 1952. In 2001, PGM commenced mining from the 92 Level (920 metres below surface) towards the old workings above. Current production operations are located both in the pillar remaining below 56 Level and below 92 Level. The lower mining sequence started at 102 Level, advancing upwards to the 92 Level, and is currently located at 96 Level. The Occidental decline has now progressed to the 108L and development is planned down to the 112L. Mining from the 112L to 104L stoping block is planned to commence in the third quarter of 2009.

The Perseverance deposit was unexploited prior to the commencement of PGM's production operations in 2003 with the top of the orebody being approximately 900 metres below the surface.

The zones are accessed from the Peak infrastructure by declines along the general strike direction of the Peak Shear and Great Cobar Fault. A decline, of some 800m, connects the Perseverance zone, while access to the New Occidental zone is via a 1:10 decline of some 3km from the Peak infrastructure. Both declines, which are 5m x 5.5m excavations, provide for up to 55-t truck haulage from the two zones to the Peak crushing and hoisting infrastructure.

The Peak Mine infrastructure includes a 5.3m diameter concrete-lined shaft to a depth of 740m. The hoisting system is designed to provide capacity of up to 800,000 tonnes per year. The main winder is a ground-mounted, 814kW friction winder running a single 10t pay load skip, counterweighted by a 26-man cage connected by four 28mm head ropes and two 40mm tail ropes.

Ore from New Occidental and Perseverance is hauled to the Peak Mine crushing station, where a 1200mm x 1050mm Jaques double toggle jaw crusher is installed, located between the 630 and 680 Levels. Crushed ore is loaded into the 10t skip and hoisted to the surface, where it is stockpiled for milling.

Peak Mine

The mining of ore in and around the Peak workings, re-commenced in 2005. A ramp was extended upwards to access Reserves up to the 320m Level. PGM has completed development of the decline from a portal on surface and created a connection of the ramp system between the underground mine and surface in December 2007.

New Occidental

The 3km decline, from the Peak Mine infrastructure, provides access to the New Occidental mineralization intersecting at approximately 900 metres below surface (90 Level). The 90 Level is the main level for the zone, and includes dewatering sump, service bay, refuge station, and lunch room.

Stoping operations at New Occidental commenced in 2001 on the 90 Level and have progressed upwards towards the old workings. The orebody is a tabular lens that dips at 85° east, plunges north at 80-85°. The lens has a strike length of about 200m and is up to 22m wide, with an average thickness of some 9 m. The New Occidental lens remains open at depth. The hangingwall is formed by Chesney Sandstones and the footwall by Great Cobar Slate. The orebody geometry and geotechnical attributes of the ore and host rocks resulted in the selection of conventional long-hole stoping, with post waste rock backfill, as the most suitable mining method for ore extraction. Longitudinal stopes are utilized along the orebody strike length of some 200 m. Stope width is determined by the width of the ore, subject to a minimum 4m stope width. The 5m high stope drilling drives are on 20m vertical centres.

The operations consist of large size development headings (up to 5m high x 5.5m wide), with access development in the hangingwall sandstones. Development utilizes Twin-boom Tamrock electro-hydraulic jumbos, with stope drilling carried out by an Atlas Copco Simba M6C rig with additional ground support drilled with a Tamrock Cabolter rig. Standard development support is provided by 2.4m split sets, supplemented by mesh where necessary and 6m cable bolts at intersections. Shotcrete has recently been introduced to the Peak mines with the surface decline providing access for shotcrete delivery.

ANFO is the primary explosive used with stick cartridges and bulk emulsion used where necessitated by wet holes. Non electric millisecond delay detonators are used for initiation. Ore loading utilizes Caterpillar R2900 ($8m^3$) scoops with truck haulage, via the internal ramp and the decline, to the crusher at the Peak shaft. The scoops are equipped with remote control capability to permit access into stopes, for recovery of broken ore under unsupported openings. The ore is loaded by scoop into 45t and 55t Caterpillar trucks for delivery to the 630 Level crusher. A separate waste pass is available on 630 Level for the hoisting of uncrushed waste that is not directly backfilled into stopes or as an alternative system for ore handling.

Prior to backfilling, stopes walls are surveyed using a laser Optech instrument. The surveys are used for individual stope reconciliation.

Initially, high dilution and ore losses of up to 42% and 11% respectively were experienced when mining commenced in 2001. This was principally due to blasting damage on the weak stope walls, caused by the large diameter blast holes initially used.

The blast-hole size was reduced from 102 mm to 89 mm. Additional, lightly charged, pre-splitting blast holes were drilled parallel to the hangingwall to minimise damage to the hangingwall waste rocks. Cable bolts were installed from the stope drilling drives in both the hangingwall and footwall of the stopes using a cablebolting jumbo. These improvements in mining practices resulted in decreased dilution and improved recovery to around 18% and 8% respectively.

Currently, mining of the pillar remaining between the PGM and historic workings is taking place. Ore recovery in this area so far has exceeded expectations. Below 92 Level production began from 102 level in Late 2007 and continues upwards, currently at the 96 Level.

PGM has commenced development below 108 Level and plans to recommence production mining of the New Occidental Deeps area on the 112 Level, advancing upwards to the 104 Level. The dilution control techniques developed in previous mining will also be adopted for the mining of stopes in this area.

Perseverance

Three zones of mineralization have been identified within the upper portion of the Perseverance system. Zones A, B and Hulk Lens are adjacent to each other and lie in the upper half of the system above the 9100m Level with an elevation of 1,110 metres below surface, whilst Perseverance Deep (Zone D) lies beneath and trends to the south. Production from the upper portion of Perseverance commenced production in 2003. Three lenses are currently mined; Perseverance, Corvus and Hulk.

The Perseverance Lens plunges steeply to the south, dipping at between 75 and 80° to the West. The lens has a strike length up to 150m and varies in width from 1.4m to 9m, with an average of 3.7 m. Reserves extend from the middle of the lens at the 9355m Level to the 8860m Level. The Perseverance Lens is open at depth and to the south. The western hanging wall consists of Chesney Formation sandstone below the 9300m Level, with the hanging wall sediments progressively becoming finer grained as the Chesney Formation grades upwards into the Great Cobar Slate. The footwall consists of Chesney

Formation sandstone and the Peak Rhyolite. The wall rocks are generally fair to good with some specific poor areas from a geotechnical perspective.

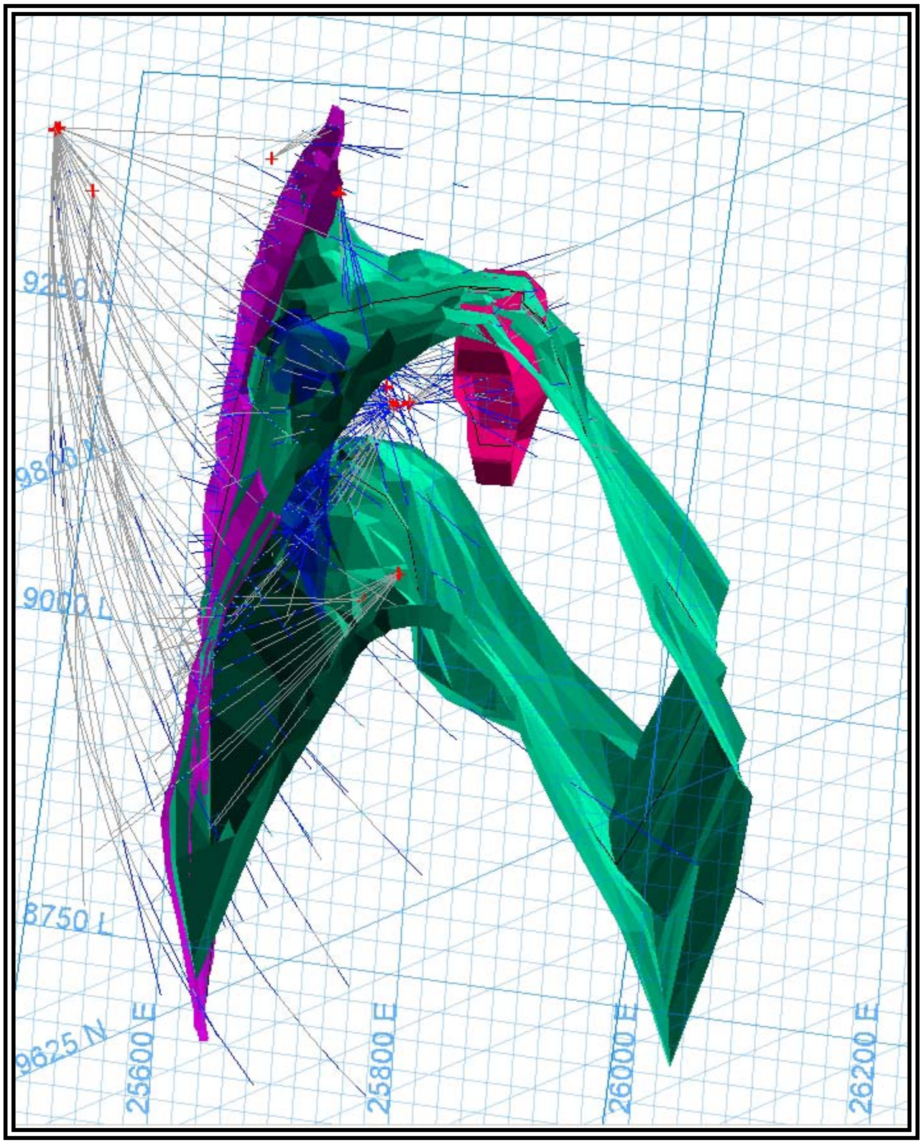
In the Perseverance Zone A the separation between the Perseverance and Corvus lenses varies from 1m to 25 m, increasing with depth. Corvus Lens Reserves are located between 9280m Level and 9160m Level. The lens dips between 75^{o} to the west to vertical, with a strike length of up to 150m and a width between 1 and 4.5m (mean 2.5m). The Corvus lens is hosted exclusively within the rhyolitic intrusive and Chesney Formation sandstone at depth.

The Hulk orebody is located approximately 100m south of the Perseverance Zone A and is accessed from the Perseverance Zone A decline. The Hulk plunges steeply, dipping between 75-80^{o} to the West. The lens has a strikelength up to 80 metres and varies in width from 4m to 21m with an average of 9m. Reserves extend from the top of the zone 9180m Level down to the 9060mLevel. The western hanging wall consists of Chesney Formation sandstone below the 9300m Level, with the hanging wall sediments progressively becoming finer grained as the Chesney Formation grades upwards into the Great Cobar Slate. The footwall consists of Chesney Formation sandstone and the Peak Rhyolite.

Mining methods, similar to those used at New Occidental, were adopted for Perseverance and Corvus lenses. Due to the wider dimensions of the Hulk orebody, up to 20m, open stoping with footwall drawpoints is the stoping method. The stoping sequence was planned from bottom to top. Dilution is around 17% with ore recovery of about 90%.

The first mining panel for Perseverance Zone D is located between the 8960m Level and the 9050m Level and the orebody width varies from 7m up to 16m. The lens is nearly vertical and the strikelength of the ore is around 300m spread over a total strikelength of 450m. A combination of bench retreat and sublevel longhole stoping will be the mining methods employed in Perseverance Zone D. Development for Zone D has progressed down to the 8960m Level which is the location of the base of the first mining panel in the southern extension of the orebody. Accesses are currently being developed on the 8980m Level and the 9000m Level and production is planned to commence in the second quarter of 2009.

Figure 19-1: Perseverance and Hulk orebodies



New Cobar

The New Cobar open pit ceased operations in 2004. A feasibility study for the New Cobar underground mine was completed in May 2004. A 1:6.5 gradient decline was mined to access Reserves adjacent to historic workings from underground. Mining of the decline commenced from the east wall within the open pit at an elevation of 10180mRL, some 20m above the bottom of the open pit.

Mining methods used are similar to those used at New Occidental and Perseverance. Mine plans call for production to continue adjacent to and below the historic underground workings (approximately 200 metres below the surface) to a depth of approximately 600 metres. These workings were dewatered from the surface via the original mine's shaft. Thereafter, controlled dewatering of the historic workings has continued as the decline progressed.

Stoping at New Cobar commenced in October 2005 and is planned to continue until 2014.

Chesney

Mining of a 700m drive to connect New Cobar with the underground resource at Chesney to provide access for production has been completed. Dewatering of the old Chesney shaft and workings has been completed. The Chesney Incline was started in June 2008 and has progressed up to the 9950m Level. The 9900mL which is the bottom of the first mining panel has been completed and the next sublevel on the 9935m Level is 50% complete.

Typical orebody widths at Chesney range from 6m up to 25m in the north so a combination of bench retreat and open stoping has been designed for 35m sublevel heights. Due to the competent nature of the host rocks the northern stopes will be mined as double lifts (70m vertical height) and extracted on the 9900m Level. Mining of the Chesney production ore is planning to start in the first quarter of 2009.

19.1.2 Mining Equipment

Peak mine has both surface and underground equipment maintenance facilities. The facilities are well equipped. The mobile equipment in use varies in age. PGM replaces equipment at the end of economic life. Several pieces of equipment currently in operation have been recently purchased. Fleet availabilities and hourly costs are within normal ranges.

A list of underground mobile equipment in use is provided in Table 19-1.

Table 19-1: Mine Mobile Equipment Fleets

Description	Equipment Type	Peak	New Cobar
Development Jumbo	Tamrock H205D	1	
	Tamrock Axera D07	2	1
Longhole Jumbo	Simba M6C	1	
	Simba 1357		1
Cablebolt Jumbo	Tamrock Cabolter H530	1	
Scooptrams	Caterpillar R2900	4	1
	Toro 1400	1	
	Caterpillar R1700		1
Trucks	Caterpillar 45t	1	1
	Caterpillar 55t	4	1
Utility	Jacon Shotcrete	1	1
	Caterpillar IT18	1	
	Caterpillar 924G IT	5	2
	ISUZU Flatdeck Truck	4	
	Caterpillar 12G Grader	1	1
	Kubota R410 Forklift	3	
	Toyota Landcruiser	13	4

19.1.3 Grade Control

Development drives in ore are located just inside the hangingwall contact. This permits pre-split drilling parallel to the hangingwall to minimize blasting damage to the waste hangingwall and therefore minimize dilution. The ore hangingwall and footwall contacts are not visible therefore assay grade control from chip sampling and diamond drilling is required to maintain the optimum position relative to the hangingwall. Return of assay results is typically 24 hours.

PGM operates a drawpoint cut-off grade based on an NSR calculated to cover all site operating costs after charging. All underground loader operators are trained to take grab samples which are then assayed and the NSR is calculated. Grab sample results are used as a method of reconciliation to determine the performance of the grade models versus mill production on a monthly basis. Grab samples, in conjunction with the Optech laser surveys, are used for final reconciliation of completed stopes.

19.1.4 Ventilation

Three additional major airways have been developed in order to ventilate the workings of the New Occidental and Perseverance operations. A 4.1m diameter airway at Perseverance, and 3.0m intake and 4.1m diameter return airways at New Occidental were raise-bored from surface over lengths of some 850 m.

An exhaust fan on surface provides the primary ventilation flow at the Perseverance with a volume of approximately 150m^3/s. PGM has installed a refrigeration plant in the Perseverance Deeps area (9010 Level) to alleviate heat issues in the Perseverance Decline experienced during the summer months.

An exhaust Fan on surface provides the primary ventilation flow at the New Occidental with a volume of approximately 170m^3/s

At New Cobar a 2.4m return raise was raise bored from surface. Fans located at the return raise on 16 Level provide a primary ventilation with a volume of approximately $100m^3/s$. At Chesney a fan has been installed on the existing 6m concrete lined shaft and provides a primary ventilation flow of approximately 140m^3/s.

19.1.5 Dewatering

All historic workings in the Cobar field flooded after the conclusion of mining operations. Dewatering of the New Occidental, New Cobar and Chesney orebodies has been successful by the use of submersible borehole pumps until underground pump stations could be established. Where new development extends downwards, adjacent to old workings, drainage holes are drilled to permit the controlled drainage of the historic workings. Pressure measurements are taken to measure drainage progress and PGM ensures that adjacent workings are drained prior to the commencement of production operations. There are no major aquifers in the Peak Mine area and little rainfall to re-charge the historic workings. Notwithstanding, there is minor re-charge of the historic workings and occasional intersection of minor water flows by active development. All water is handled by the mine pumping systems at a rate of 550,000 litres per day.

19.1.6 Open Pit Design and Considerations

New Cobar

The New Cobar open pit was operated between June 2001 and February 2004. Ore recovered totalled 1,017,000t @ 3.55g/t gold (including 230,000t of tailings fill), with 440,000t @ 0.72% copper inclusive. This equates to 116,100oz gold and 315t copper. The pit ultimate floor reached 10155mRL, 150 meters below surface. Final pit walls had an overall slope angle of between 55 and 60°.

On completion of the pit a decline was mined from the western pit wall, 20m above pit bottom to access reserves at depth. The western pit wall therefore requires maintenance to protect access to the decline portal. The bench wall adjacent to the decline portal has been shotcreted and bolted. Monitoring of pit wall stability is ongoing. Pit walls are visually inspected on a twice-yearly basis and are scaled as required.

19.1.7 Mine Planning

Vulcan Mine Planning software system is utilized at PGM for the planning of underground development and stoping operations. Drilling and blasting layout preparation is also computerised. The block model is cross sectioned at 5m intervals and these sections are used, in conjunction with geological cross section showing the ore domain boundaries. Stope boundaries are digitised on screen. Grade contours, chip samples and diamond drill holes are also used as a check. A stope may be re-designed several times in order to establish an optimum shape, suitable for mining. The digitised stopes on sections are then used to create a 3-D solid model of each stope. Modifying factors to account for dilution and recovery are applied. The stope models are analysed for tonnes and grade using an orebody NSR value as a cut-off grade for reserves.

19.1.8 Mining Performance

Actual activity and cost data for the year 2008 is compared to budget in Table 19-2.

Table 19-2: 2008 Mine Activity and Cost Data

		2008	
		Actual	**Budget**
Ore Hoisted	t	667,261	647,785
Milled Au grade	g/t	4.83	4.9
Milled Cu grade	%	0.61	0.95
Total Development	m	5194	6823
Operating Cost	AU$/t	51.99	57.79
Development Capital	AU$/t	26.16	18.29
Total U/G Mining Cost	AU$/t	78.15	76.08

19.2 Tailings and Waste Facilities and Management

At New Cobar waste is placed on the surface waste dump or in completed stopes as backfill.

At Peak Mine the majority of waste mined in development is also usually placed into completed stopes as rockfill. However, in late 2006 and continuing to present a shortage of open stopes underground has resulted in a requirement to hoist waste surface. This material is trucked from the waste skip discharge bins in the shaft headframe to the tailings dams where rock piers are being constructed inside the dams to aid operation. PGM plans that all waste hoisted to surface will be utilized in this manner or in rehabilitation of historical tailings dumps.

The Peak Tailings Dam is located approximately 1km west of the Peak processing plant. The tailings dam has been in operation since recent mining commenced in 1992 and currently holds in excess of 9.6Mt.

19.2.1 Tailings Storage Design and Operations

Thickened mine tailings (up to 62% solids) are pumped via 2 tailings lines to the tailings stack located in the centre of the tailings dam. Between the plant site and the tailings dam, tailings lines are bunded to contain any tailings line failure. The tailings line traverses two drainage channels, one being an ephemeral creek. These channels are well bunded and the tailings line passed through a pipe across the creek to ensure no material can enter the creek.

Tailings are retained by the Tailings Filter Dam, and three saddle dams. An access ramp constructed of waste rock provides vehicular access to the tailings stack. The tailings lines are located on a bench paralleling the vehicular access on a lower elevation, providing ease of access.

The Tailings Runoff Dam, also called the Decant Dam or the Tailings Overflow Dam, is located south of the tailings stack. The purpose of the Tailings Runoff Dam is to store bleed water from the tailings stack and excess runoff from the tailings storage area during storm events. There is also facility to pump excess water from the recycle water dam to this area if necessary.

According to the August 2000 Report on Seismic Stability Assessment by Australian Tailings Consultants Pty Ltd, the embankment dam has a maximum height of 5m. The embankment was constructed of sandy clay above insitu soil of medium to highly plastic red brown clay. The crest and upstream slope of the embankment were protected by gravels. At the downstream toe of the embankment a short length of toe drainage was constructed using gravels.

A report entitled "Report for Surveillance Inspection of Dams – Annual Audit Inspection" dated February 2009 and prepared by GHD Pty Ltd ("GHD") observed that the Tailings Runoff Dam appeared in good condition. GHD recommended that the maximum operation water level in the Runoff Dam be reassessed to take into account the change in the characteristics of the catchment and the configuration of the tailings stack, using current hydrological data. GHD is currently being commissioned to undertake this work. Other recommendations are minor and are mainly associated with inspection criteria and frequency. These recommendations are currently being reviewed and undertaken.

Three piezometers are installed within the tailings impoundment and four along the Tailings Runoff Dam. GHD reported that monitoring results do not indicate any unsafe or abnormal conditions in the dams.

PGM concentrator operations personnel carry out a complete inspection of the tailings impoundment system twice per shift. These inspections are being altered currently to comply with ANCOLD guidelines. Additional inspections by management and environmental personnel are also conducted.

19.2.2 Water Supply and Management

It is essential to maximize water use efficiently in Cobar's semi arid environment. There are no permanent water bodies in the area, the Cobar Shire relies on water pumped 140km from Nyngan to supply the town and local mines. Drought conditions have existed in the area since approximately 2003.

PGM is represented on the Cobar Water Board and is entitled to a raw water allocation of 1,000 million litres per year. The raw water allocation is net of evaporation, which at over 2.5m per year can account for losses of up to 50%. PGM adopts a strategy for recycling and reclaiming water to minimize raw water requirement. PGM collects all water pumped from the mines and process water for treatment and re-use.

The contaminated water management scheme is operated on a closed circuit basis with two storage structures; the recycled (process) water dam and the tailings dam. Freeboard requirements are maintained on each dam; 500mm at the recycled water dam and 700mm at the tailings facility.

Surface run-off within the plant site is classified as dirty water and enters the recycle water dam. This includes surface runoff around the surface portal and servicing roads.

Mine water from underground is treated in sediment cells, where flocculant is added to depress sediments. From 2007, water is directed to the Recycled Water Dam from the Chesney Shaft and associated workings. This water is routinely monitored for pH, EC, major anions, cations and metals in addition to oils and greases. Sediment is transferred for storage on the tailings dam. After this treatment the water is considered to be of adequate quality and is transferred to the recycle water dam. De-silting of the recycle water dam has been undertaken previously, but better management of the silts from underground has reduced the necessity for this operation.

In an attempt to reduce evaporation and therefore conserve water on-site, an evaporation inhibitor is added automatically to the Recycled Water Dam.

Raw water from off site is delivered to the site raw water tanks. Water from the recycle water dam and excess water from the New Cobar de-watering circuit is also fed to the site raw water tanks with the aim of minimizing off-site water requirements. Water for the process plant is drawn partially from the site raw water tanks and partially directly from the recycle water dam. Mine services water for underground dust suppression and surface water requirements for the ROM pad and dust suppression for haulage roads is drawn from the site raw water tanks.

A potable water allocation is provided through Cobar Shire Council. Potable water usage is approximately 9.5 million litres per year.

Peak site water usage is monitored using the CITEC plant control system, which allows real time readings to be collected. Water usage and monitoring of quality at key locations around the site are included in the Annual Environmental Management Report which is described in Section 19.5.1.

19.2.3 Surface Run-off and Control

The locations of the process plant, tailings dam and water dams were selected to permit the collection and containment of all run-off from the site followed by reclamation and re-use.

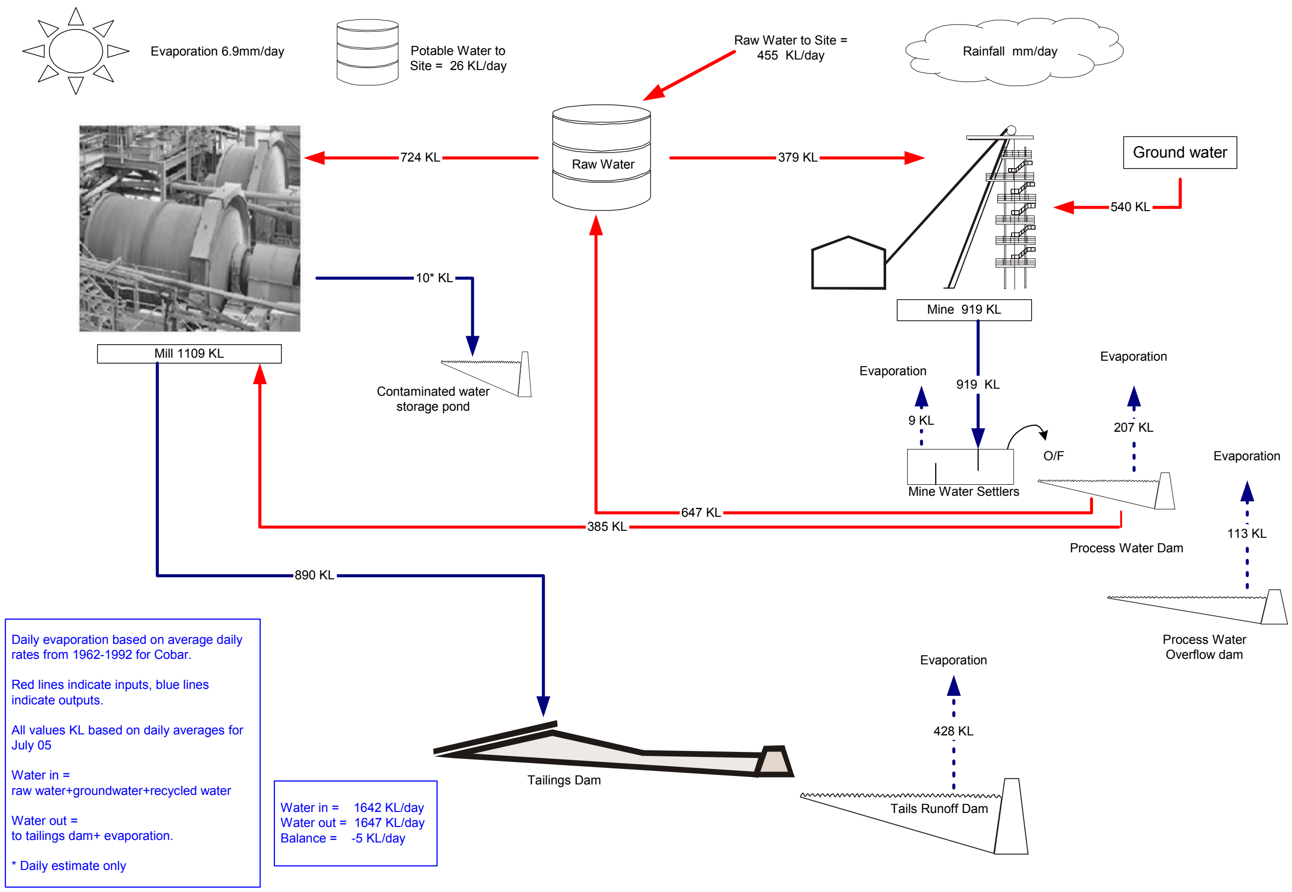
Clean run-off from the slopes above the mine site is diverted into drainage culverts around the perimeter of the site. The culverts discharge in the South Stock pond for re-use.

Drainage around the Peak and New Occidental sites are checked by the Environmental Department six monthly and after each significant rain event to ensure that culverts are free of obstruction and that the drainage system is working effectively. Currently, this inspection regime is being extended to New Cobar.

19.2.4 Water Balance

The water balance at PGM operations is shown in Figure 19-2. It does not include water from the New Cobar / Chesney operations.

Figure 19-2: Water Balance



19.3 Product Transport and Sales

19.3.1 Concentrate Handling and Transport

Concentrate from the concentrate stockpile adjacent to the process plant is loaded by front end loaders to road haulage trucks. The trucks are weighed on a weighbridge to determine the mass of concentrate trucked. The sealed trucks travel on public roads to the point of sale at the CSA mine process facility, located 5km north of Cobar, a distance of approximately 15km. Concentrates are generally trucked from Peak Mine on a monthly basis. At CSA the concentrates are and stockpiled until a Shippable parcel of 5,000t or 10,000t cargo lot has accumulated. Approximately three shipments are made from CSA per year. The port alternatives include Port Kembla and Newcastle.

19.3.2 Concentrate Marketing

Copper concentrates with enriched grades of gold and silver are currently sold under contract to Glencore International AG of Baar, Switzerland. Two contracts are currently in force, the first contract No. 101-05-11224-P terminates effective December 2008. The contract terms have been periodically amended during the contract period. The second contract No. 101-04-12959-P terminates on 30 June 2009 subject to PGM completing delivery of 18,000 wet metric tonnes of copper concentrate by that date.

A new copper concentrate contract is in the process of being negotiated.

The point of sale is at the CSA mine processing facility which is located 5km north of Cobar, whereupon the buyer takes responsibility for the product notwithstanding a contractual obligation for PGM to pay stipulated charges for shipment to overseas smelters. PGM also pays agreed concentrate treatment charges and refining charges. The contract terms are typical of those used at other operations elsewhere producing copper concentrates. Payable metals are gold silver and copper whilst penalty elements are lead, zinc and bismuth.

Ninety percent of the provisional material value is paid to PGM one month following shipment. Final settlement is effected when final assays are received and metal prices confirmed.

19.3.3 Bullion Handling and Transport

Gold doré is refined by AGR Matthey in Perth, Western Australia. Transportation of the doré is performed under contract to X Keys Pty Ltd.

19.3.4 Bullion Marketing

Bullion is sold under contract to AGR Matthey, a partnership between Western Australian Mint, Australian Gold Alliance Pty Ltd and Johnson Matthey (Aust) Ltd of Perth, Western Australia. The current contract was dated January 2005 and expires at end December 2007. The contract provides for the refining of all doré produced at the PGM operation.

Contract payments and terms are typical of similar contracts for the sale of doré at other operations elsewhere.

19.4 Infrastructure and Services

19.4.1 Introduction

The infrastructure and services required by PGM are typical for a medium sized mine and processing operation. Water supply is described in Section 19.2.2.

19.4.2 Power Supply

Power is supplied to the PGM operations under an agreement dated December 2004 with Country Energy. Maximum electricity consumption demand is 9MW and annual consumption is approximately 79GWh.

Power is provided via a 132kV transmission line, to a substation adjacent to the Peak Mine, where it is converted to 11kV for use on site, or transformed on site to lower voltages.

Power supply was regular with occasional outages resulting from storms and unforeseen occurrences. PGM operates two 0.8MW and one 0.65MW diesel generating units to provide emergency power, sufficient for emergency mine egress and to clear some of the processing lines.

19.4.3 Buildings

At New Cobar there is office space, ablution blocks, change rooms and workshops. The active area is fenced with security access and a CCTV camera.

The main buildings are located around the Peak Mine. Spacious administration and technical offices are available together with change house facilities, workshops, stores and plant buildings.

The plant site is ring fenced. A permanently manned security office is located at the main gate. Access to the site is secure and well controlled via an access card operated turnstile system for pedestrians and operator controlled vehicular gates.

19.4.4 Communications

All normal forms of communications including telephones, satellite and internet are available at the site. Underground, communications are by radio and fixed telephone lines. Though 2009, fibre optic cables and networks will be progressively commissioned in the mine. While principally a data back-bone, we expect significant communications infrastructure to be progressively incorporated.

19.4.5 Roads

Access to the site is by sealed roads to the national road system. Some internal site roads are unsealed but these are graded and watered for dust suppression as necessary.

19.4.6 Freight

Some freight arrives to Cobar on the national rail system, however all equipment and supplies are finally delivered to site by road.

19.5 Environmental Management and Heritage Impact

19.5.1 Introduction

Descriptions regarding Environmental Management and Heritage Impact are taken from the "Annual Environmental Management Report (AEMR)" prepared by PGM and dated August 2008 and the Mining Operations Plan (MOP) prepared by PGM and dated August 2005.

The reports were prepared according to the New South Wales Department of Mineral Resources Guidelines to the Mining, Rehabilitation and Environmental Management Process, 2002 ("MREMP"). These guidelines were issued by the Department of Mineral Resources now the Department of Primary Industries.

The reporting mechanism consists of two documents:

- The "Mine Operations Plan" ("MOP") that provides a projected summary of planned and proposed operations of the mine for the next five to seven years. (Reference Section 21).
- The AEMR that reports yearly environmental and rehabilitation performance against actions forecast in the MOP.

The AEMR documents details of compliance with EPA licence conditions, development consent conditions as well as other statutory environmental requirements that PGM operates under. The AEMR is provided to the following bodies:

- Department of Primary Industries;
- Department of Lands (Formerly DIPNR);
- Department of Environment and Climate Change (formerly the EPA);
- Dams Safety Committee; and
- Cobar Shire Council.

19.5.2 Licences

The environmental and operating licences held by PGM are documented in Table 4-2.

19.5.3 Environmental Controls

The AEMR documents procedures and monitoring for the following activities and materials:

- Water Management;
- Waste Management;
- Recyclable Materials;
- General Waste;
- Sewage Treatment;
- Hydrocarbons;
- Tailings and Tailings Dam
- Waste Rock;
- Hazardous Material Movement;
- Fuel Containment;
- Noise, Vibration and Dust;
- Groundwater, Surface Water, Process Water and Potable Water; and
- Flora and Fauna.

During the year July 2007 to June 2008, 51 environmental incidents occurred and were reported to the EPA if required by PGM's licence conditions. Of the 51 incidents, 46 were classified as "Low Consequence". No high ranking environmental consequence incidents were identified.

An External Environmental Audit of PGM operations was undertaken by Enesar Consulting Pty Ltd in July and August 2006 with a subsequent report issued in February 2007. The report noted a number of significant improvements since the previous audit in 2004, most notably in water recycling, implementation of dust suppression measures and upgrading of the tailings pipelines. Three issues classified as high importance were identified relating to tailings management. Of these issues one has been corrected and measures have been taken to improve practices related to the two other issues (Archaeology and Heritage).

No significant Aboriginal heritage issues have been identified in relation to PGM's operations.

The principal cultural and natural heritage issues associated with PGM's operations relate to historical mining activities that have taken place in the area dating back to the 1870's. PGM is aware of the significance of many of its historic sites and has embarked upon a consultative process with Cobar Shire Council and has encouraged the Council takes ownership of the land and develop the sites. PGM holds a Community Consultation Meeting so that community members can voice concerns over heritage, environmental or other issues.

An interpretive walk has been established near the mine site to allow visitors to appreciate Cobar's mining heritage in the context of the nearby modern mining operation. As well as this, a lookout has been opened to the public at Fort Bourke overlooking the open pit at New Cobar.

19.5.4 Closure Plan

A report entitled "Conceptual Closure Plan" was prepared by PGM dated November 2006. This report was essentially an update of the "2001 Closure Plan" developed for PGM by Enesar Pty Ltd (previously NSR Environmental Consultants Pty Ltd). Subsequently, this plan has been reviewed yearly to ensure currency.

As the holder of CMLs 6 to 9, PGM has a responsibility under law for addressing the impacts of historic as well as current mining activities on its leases. Determining the practical extent to which past mining impacts need to be remediated before lease relinquishment, is ultimately the responsibility of the Department of Primary Industries (DPI).

The potential for government, corporate and community expectations to change before final decommissioning, final closure criteria cannot be predicted with certainty. Regular review of the closure strategy, in conjunction with relevant government departments and the local community is therefore an integral part of the closure planning process.

As part of the ongoing process of closure planning review, PGM has sought legal clarification of the company's obligations for closure and lease relinquishment, for both historic and active mine areas. In particular, advice has been sought on the transfer of liability from PGM to the government upon lease relinquishment, and the implications such a transfer of liability might have on government closure requirements.

Regulatory controls pertaining to the closure of PGM's operations are:

- The New South Wales Mining Act 1992,with particular reference to:
    - The protection of flora and fauna, features of Aboriginal and archaeological importance.
    - The rehabilitation of lands damaged by mining by levelling and re-grassing or re-forestation.
    - The filling, seal or fencing of excavations shafts and tunnels.

- Schedules of Conditions for the CML's related to rehabilitation and closure with particular reference to:
    - Topsoil to be set aside for re-topsoiling at a later date.
    - Revegetation using indigenous species to a density in keeping with the surrounding area.
    - Leaseholder to fill in or secure any abandoned cuts, workings or shafts.
    - Temporary access roads to be ripped topsoiled and re-vegetated.
    - Machinery and buildings to be removed from site.

The areas for which PGM bears responsibility for rehabilitation are shown in Figure 19-3.

Figure 19-3: Location of PGM's Operations

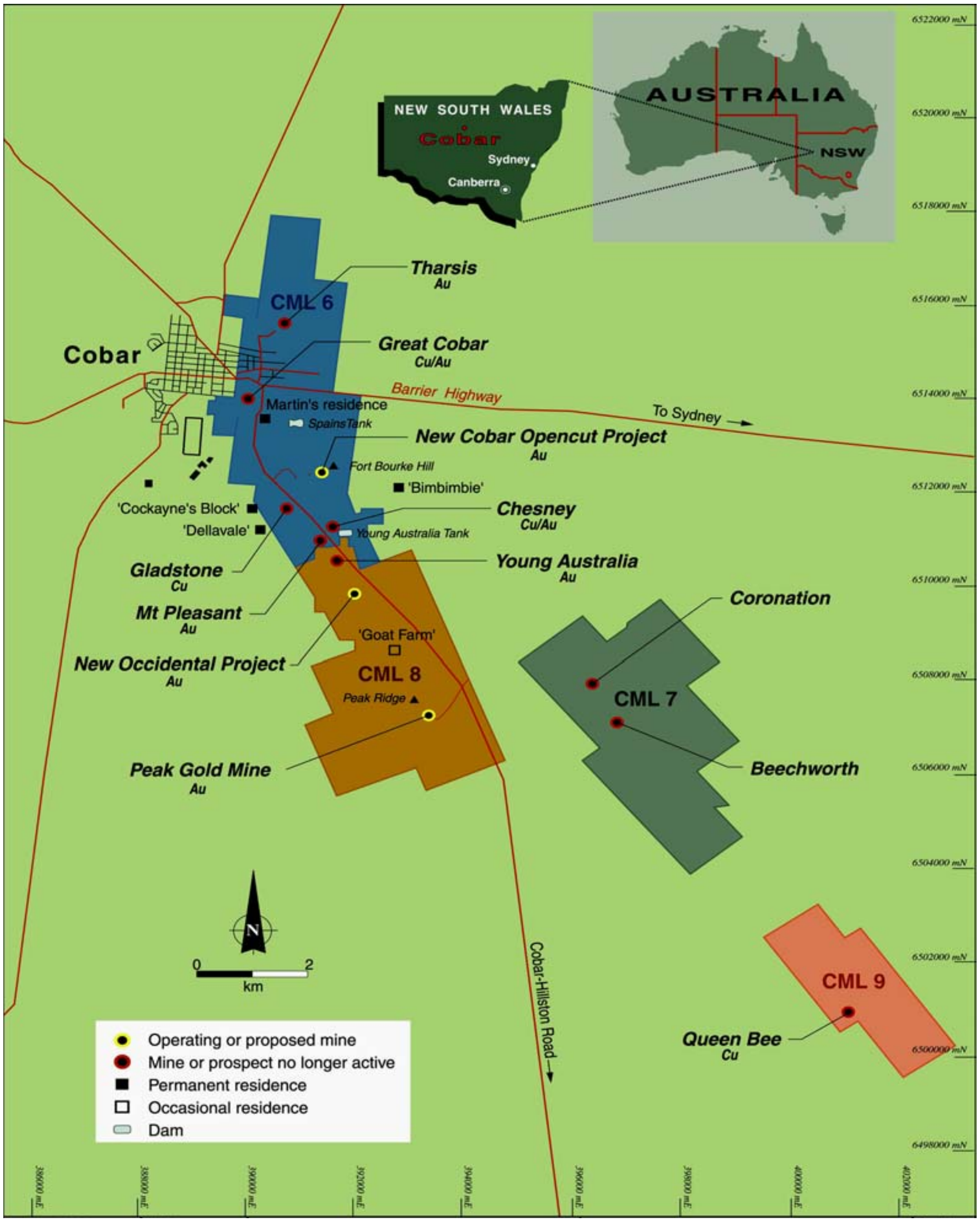


Table 19-3 summarizes the anticipated closure costs estimated in 2008, excluding costs for progressive rehabilitation, retrenchment and asset sales. PGM has a performance bond in favour of the Minister of Mineral Resources (New South Wales). The bond value was increased to A$10.1 million in 2008.

Table 19-3: November 2007 Estimate of Total Closure Costs

Location/Component	Cost A$
Peak Gold Mine	6,964,669
Perseverance	37,287
New Cobar	948,168
New Occidental Mine Site	508,294
Tharsis Mine - rehabilitation complete	0
Great Cobar Mine	24,936
Chesney Mine	538,053
Water Disposal Dams	61,006
Gladstone Mine	10,865
Mount Pleasant and Young Australia	11,080
Queen Bee Mine	33,320
Coronation & Beechworth	2,740
General Costs	1,249,659
Contingency (10% of total)	1,039,008
TOTAL	**11,429,085**

Conceptual Closure Costs, PGM, Nov 2008

19.5.5 Comments

As with all closure cost estimates, there are uncertainties involved. Specifically, the greatest uncertainties for closure requirements, and hence cost of closure, of the PGM tenements are:

- Community and government expectations of closure.
- The end land use required and the level of closure commitment necessary to ensure public safety, particularly from historic liabilities.
- Control of acid mine drainage from waste rock and tailings areas.
- Soil and water contamination.

19.6 Labour, Organisation, Occupational Health and Safety and Training

19.6.1 Labour Requirements

Table 19-4 shows the number of employees and contractors working at the PGM operations in February 2007. The Australian and world minerals industry labour market has eased from the boom conditions experienced in 2008. PGM has recruited internationally for key managerial and technical positions. Some 12 staff are working on 457 visas or applying for permanent residency. Turnover rates have dropped significantly during the later part of 2008 and early 2009.

Table 19-4: PGM Employees and Contractor Labour

Department	PGM Employees	Contractors - Services	Contractors - Labour	Total
Manager & PA	2	0	0	2
Peak Mining	94	0	5	99
New Cobar U/G	31	0	2	33
Metallurgy	43	0	2	45
ESH	12	1	0	13
Commercial	15	0	0	15
Exploration/Geology	12	8	0	20
Maintenance	50	2	0	52
TOTAL	259	11	9	278

19.6.2 Organization

PGM's operational management structure is shown in Figure 19-4.

19.6.3 Occupational Health and Training

All PGM employees work under the terms of the following policies and codes:

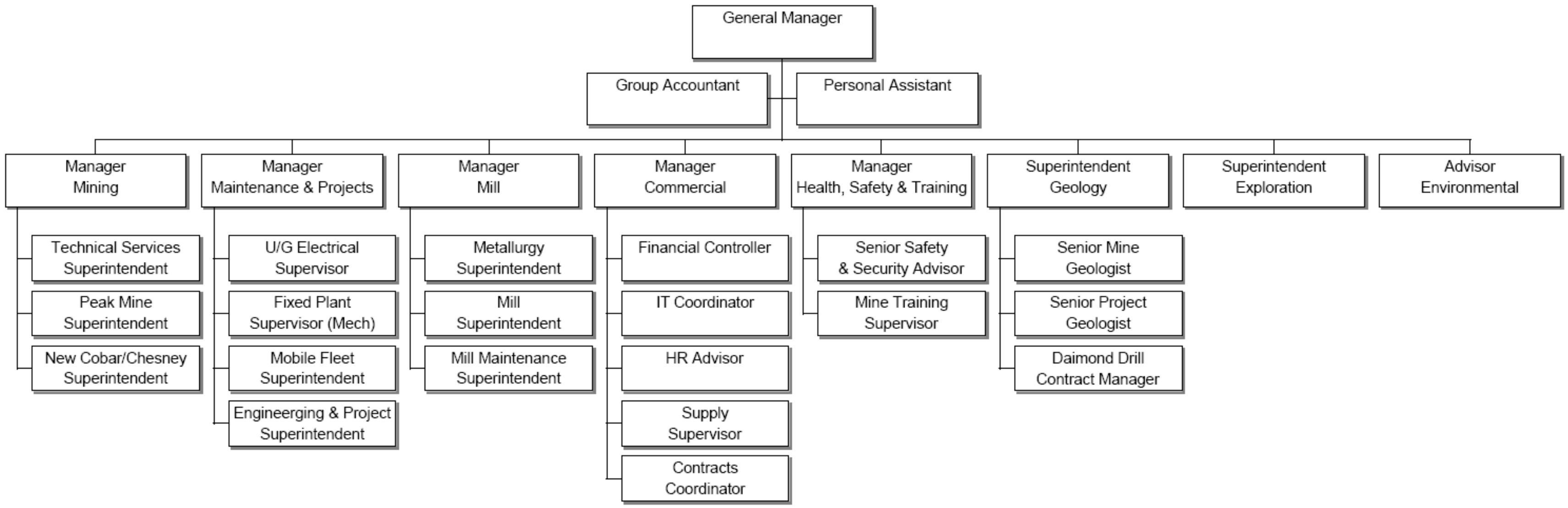
- General Conditions for Employment;
- Code of Conduct; and
- Drugs and Alcohol Policy

These documents describe the terms and conditions of employment, which are essentially similar to those provided at other operations in a similar economic environment.

PGM operates a safety and training department to coordinate all training programs throughout its operations. Formal job training is provided in conjunction with continuing on-the-job training in the workplace. A variety of other training programs are made available to employees ranging from driving awareness to Business Leadership programs and handling, computer courses and training in hazard recognition and control.

A broad range of specialist training programs are conducted on a regular basis including hazard identification, basic fire fighting, industrial first aid, occupational first aid, computer skills, confined spaces, safe use of explosives and workplace training and assessment.

Figure 19-4: PGM Operational Management Structure



19.6.4 Safety

PGM operates a safety management system using the legislation and Australian Standards as guidance. The "Peak Safety Standards" were designed as a resource for business units to assist with day-to-day operations. The standards assist with the management of hazards such as confined spaces, working at heights, mobile equipment standards, change-management, fitness for work and use and handling of explosives. The safety management system is regularly audited both internally and externally.

Peak has an Occupational Health and Safety Committee comprised of elected employee representatives and management representatives. The committee meets monthly to review site safety statistics and investigations, make recommendations for improvement to workplace safety and facilitate consultation between management and employees on matters affecting safety at the mine site. To support line management, Peak currently employs safety and training professionals that work with business units to provide an advisory and audit function.

Table 19-5: PGM Injury Statistics as at December 2008

	INJURY STATISTICS														
	HOURS		MONTH					YEAR TO DATE							Days LTI Free
	Hours Month	Hours YTD	Report Only	First Aid	Medical Aid	DI	LTI	Report Only	First Aid	Medical Aid	DI	LTI	**LTIFR**	**AIFR**	
Admin	7,110	72,602	0	0	1	0	0	3	7	1	0	1	**2.8**	**5.5**	202
Geology	4,978	73,339	1	1	0	0	0	3	18	0	0	0	**0.0**	**0.0**	1069
Maintenance	7,691	93,373	0	3	0	0	0	1	8	1	0	0	**0.0**	**2.1**	-
Mill	9,543	125,236	2	1	0	0	1	13	31	5	0	1	**1.6**	**9.6**	16
Mining	24,137	275,289	1	1	2	0	0	15	34	16	1	2	**1.5**	**13.8**	166
Contractors	8,468	148,868	1	2	0	0	0	5	17	2	1	0	**0.0**	**4.0**	1069
Employees	53,459	639,839	4	6	3	0	1	35	98	23	1	4	**1.3**	**7.5**	16
SITE	**61,927**	**788,707**	**5**	**8**	**3**	**0**	**1**	**40**	**115**	**25**	**2**	**4**	**1.0**	**6.8**	**16**

SITE FREQUENCY RATES	**Report Only Frequency Rate**	**First Aid Frequency Rate**	**Medical Aid Frequency Rate**	**Disabling Injury Frequency Rate (DIFR)**	**Lost Time Injury Frequency Rate (LTIFR)**	**All Injury Frequency Rate (AIFR)**	**Total Injury Frequency Rate (TIFR)**
12 Mth Rolling Average	10.14	29.16	6.34	0.51	1.01	6.85	47.17
CAL YTD	10.14	29.16	6.34	0.51	1.01	6.85	47.17

PGM maintains a well equipped 25 member mine rescue team with employees drawn from all areas of the operation. The team maintains a relationship with the NSW Rural Fire Service that allows utilisation of the fire service paging network to provide greater coverage over the district.

19.7 Economic Analysis

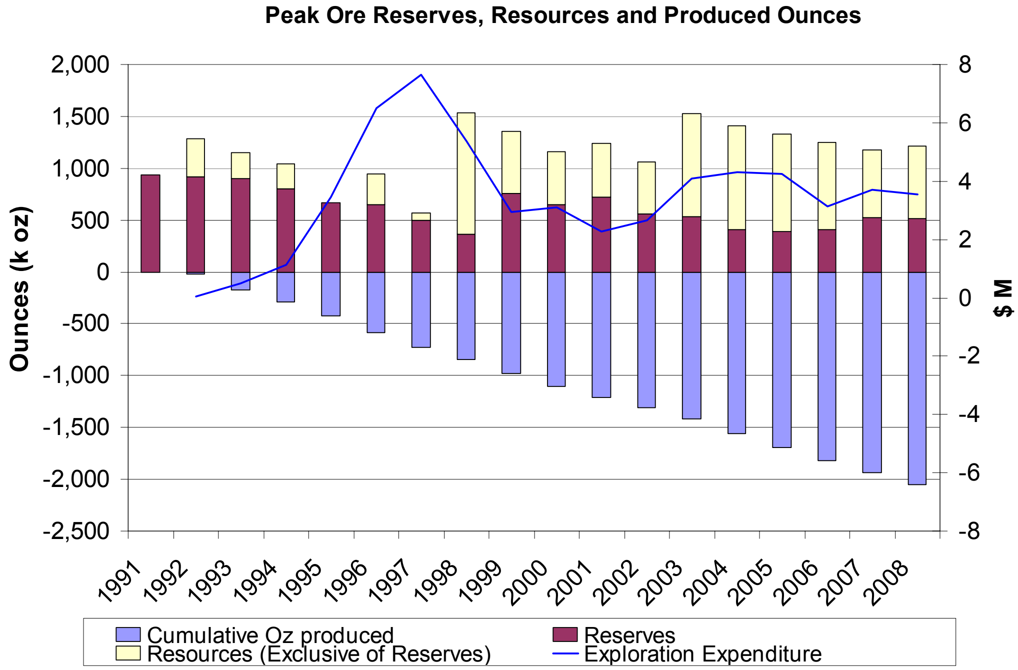
19.7.1 Life of Mine Plan

The economic stope shapes at PGM are built based on various metal prices from the budget process. Stopes may be extended, or new stopes created, to mine unplanned material. This material includes

- Measured and Indicated Mineral Resources that did not meet Mineral Reserve breakeven NSR criteria and/or;
- Material that has not been included in the previous Reserve report due to the time span between discovery and mining of this material (typically 3-6 months) being shorter than the time span between reserve reports (one year).

PGM carries out both exploration and delineation diamond drilling for resource and reserve replacement. PGM's Life of Mine Plan includes ongoing expenditure in the order of A$3M per year for exploration and delineation. PGM's historical success in discovery of new resources, converting resources to reserves is compared to exploration expenditure in Figure 19-5.

Figure 19-5: Historical Conversion of PGM Reserves, Resources and Produced Gold



For Budget purposes, PGM uses a Life of Mine Plan that includes the mining of Reserves, Inferred Resources and material from advanced exploration projects (Table 19-7). CIM Definition Standards require that the economic analysis should be derived by the mining of Mineral Reserves only. A second Life of Mine Plan that comprised the mining of Mineral Reserves only has therefore been created for the purposes of this report (Table 19-6). In Section 19.7, an estimate of the financial effects of the Life of Mine Plan comprising Mineral Reserves only is provided.

Table 19-6: Life of Mine Production Plan (Mineral Reserves Only) [(1)]

	2009	2010	2011	2012	2013	2014
New Occidental						
Tonnes	95,820	25,000	120,000	86,991		
Au g/t	4.44	6.00	6.30	6.17		
Cu %	0.19	0.02	0.15	0.15		
Perseverance						
Tonnes	332,830	220,000	220,000	220,000	252,771	
Au g/t	5.23	5.59	5.58	5.58	5.58	
Cu %	1.17	0.96	0.97	0.97	0.97	
Peak						
Tonnes	70,898	150,000	150,000	150,000	150,000	77,778
Au g/t	3.81	4.77	4.87	4.87	4.87	4.87
Cu %	0.51	0.62	0.63	0.63	0.61	0.61
New Cobar						
Tonnes	96,557	125,000	100,000	130,000	39,173	
Au g/t	4.19	4.14	4.16	4.16	4.16	
Cu %	0.81	0.68	0.60	0.60	0.60	
Chesney						
Tonnes	157,826	200,000	160,000	169,660		
Au g/t	1.74	2.21	2.00	2.00		
Cu %	1.78	1.90	1.90	1.89		
Surface Stockpiles						
Tonnes		34,176				
Au g/t		4.55				
Cu %		0.85				
Reclaim Stockpiles						
Tonnes					70,210	
Au g/t					2.00	
Cu %					0.65	
Total						
Tonnes	753,931	754,176	750,000	756,651	512,154	77,778
Au g/t	4.13	4.26	4.60	4.46	4.77	4.87
Cu %	1.06	1.06	0.92	0.95	0.79	0.61
Au Oz	100,215	103,211	110,923	108,507	78,587	12,178
Cu lb	17,618,147	17,606,518	15,209,673	15,864,579	8,946,969	1,045,973

(1) Production is shown before metallurgical recoveries are applied;

The proportion of ore in Resources and Reserves is reasonable for a deep underground mine. The conversion of Resources to Reserves is dependent on drilling density to increase confidence levels. Due to the orebody nature, access to drill platforms for close spaced drilling is restricted until development approaches the ore zone. This implies that conversion of Resources to Reserves will lag until the mine matures.

Whilst PGM's historical success in discovering new Mineral Resources and converting these to Mineral Reserves is noted as shown in Figure 19-5, there can be no assurance that PGM's operational Life of Mine Plan (Table 19-7) will be achieved. The author cautions that the inclusion of Inferred Resource and material from advanced exploration projects is by definition material of lower confidence and therefore introduces additional risk in any economic analysis.

Table 19-7: PGM Operating Life of Mine Plan Including Measured, Indicated, Inferred Resources and material from advanced exploration projects.

NEW OCCIDENTAL DEEP	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	95,820	83,803	32,994	-	-	-	-	-	-
Au	4.45	4.18	4.02	-	-	-	-	-	-
Cu	0.19	0.13	0.10	-	-	-	-	-	-
HULK AND HERCULES	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	56,629	94,158	123,273	97,883	22,428	21,150	-	-	-
Au	2.02	2.32	2.31	3.68	3.66	3.68	-	-	-
Cu	1.41	1.35	1.32	1.03	0.52	0.96	-	-	-
PEAK	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	70,898	56,155	64,532	127,810	57,076	42,369	12,568	-	-
Au	3.81	3.80	5.66	5.25	5.46	3.30	6.05	-	-
Cu	0.51	0.83	0.74	0.43	0.60	1.33	0.21	-	-
PERSEVERANCE ZONE A	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	65,269	16,515	-	-	-	-	-	-	-
Au	2.54	2.64	-	-	-	-	-	-	-
Cu	1.97	1.56	-	-	-	-	-	-	-
PERSEVERANCE ZONE D	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	210,932	244,551	273,270	181,241	171,096	185,739	172,356	117,041	-
Au	6.93	6.96	5.83	9.22	11.04	7.87	6.34	5.62	-
Cu	0.86	0.79	0.74	0.51	0.45	0.51	0.51	0.55	-
PEAK SHAFT SUMMARY	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	499,547	495,182	494,069	406,934	250,600	249,257	184,924	117,041	-
Au	4.88	5.10	4.81	6.64	9.11	6.74	6.32	5.62	-
Cu	0.89	0.81	0.84	0.61	0.49	0.69	0.49	0.55	-
NEW COBAR	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	96,557	47,688	78,979	87,563	68,139	42,877	-	-	-
Au	4.19	4.66	5.25	4.95	4.42	4.50	-	-	-
Cu	0.81	0.57	0.48	0.56	0.45	0.43	-	-	-
CHESNEY	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	157,826	204,528	173,235	249,046	189,501	149,571	-	-	-
Au	1.74	1.74	1.99	1.74	2.71	3	-	-	-
Cu	1.78	2.01	1.86	1.90	1.89	1	-	-	-
NC MINING SUMMARY	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	252,922	250,125	249,913	332,873	254,798	190,204	-	-	-
Au	2.63	2.24	2.93	2.49	3.03	3.19	-	-	-
Cu	1.36	1.65	1.34	1.45	1.41	1.03	-	-	-
ALL MINING SUMMARY	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	752,469	745,306	743,982	739,807	505,398	439,462	184,924	117,041	-
Au	4.13	4.14	4.18	4.78	6.04	5.20	6.32	5.62	-
Cu	1.05	1.10	1.01	0.99	0.95	0.83	0.49	0.55	-
STOCKPILE - PROCESS WATER DAM	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	-	-	-	-	74,768	-	-	-	-
Au	-	-	-	-	2.00	-	-	-	-
Cu	-	-	-	-	0.65	-	-	-	-
MINERALISATION	2009	2010	2011	2012	2013	2014	2015	2016	2017
Total	-	0	0	0	163000	304000	613000	682000	725766
Au	-	0	0	0	2.84	2.84	2.84	2.84	2.84
Cu	-	0	0	0	1.41	1.41	1.41	1.41	1.41
Total to the mill									
Total	752,469	745,306	743,982	739,807	743,166	743,462	797,924	799,041	725,766
Au	4.13	4.14	4.18	4.78	4.93	4.24	3.65	3.25	2.84
Cu	1.05	1.10	1.01	0.99	1.02	1.07	1.20	1.28	1.41

Note: The above Life of Mine plan is preliminary in nature and includes inferred mineral resources and material from advanced exploration projects that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the operational Life of Mine Plan will be realized.

19.7.2 Economic Assumptions

The overall economic potential of the PGM operation has been evaluated using conventional cash flow techniques in constant end-2009 Australian dollars. This procedure has been used for the purpose of estimating the financial returns expected to accrue to PGM as the owner of 100 % of the Peak mining and milling operations.

The production and grade profile is based on ore reserves only, and as such, is viewed as more conservative than the internally utilised Life of Mine Plan, which does include a proportion of resources and mineralisation. Consequently the Ore Reserve only plan depicts a mine life finishing in 2013 whereas the internal LOM Plan has an estimated mine closure in 2017.

The estimated closure costs are currently estimated at A$11.4M plus redundancy payments of A$4.1M. It is assumed that these costs will be funded by asset sales at the end of the mine life, which would include 50 company houses, 16 apartments, mobile fleet and fixed plant assets.

The cash flow analysis is based on reasonable estimates of anticipated costs, revenue and activities. Forecast cash flow excludes income tax payments, although royalty payments on revenue are included.

The economic analysis and sensitivities were updated by Bruce Kennedy, Peak Gold Mine's Commercial Manager. Bruce has a Bachelor of Commerce (Accounting Finance Systems) and is a Certified Practicing Accountant. The analysis, including assumptions, has been reviewed and checked by Peter Lloyd, the General Manager of Peak Gold Mines.

Metal Price Assumptions

Approximately 80% of gross revenue from PGM's operations is derived from the production of gold. Less than 1% of gross revenue is derived from the sale of silver and the balance is derived from copper.

PGM's metal price assumptions used for financial analysis are provided in Table 19-8.

Table 19-8: PGM Metal Price Assumptions

Year	Gold US$/oz	Copper US$/lb
2009	750	2.00
2010	750	2.00
2011	750	2.80
2012 Onwards	750	2.50

The gold price assumption is currently conservative based on average January and February 2009 prices of greater than US$900/oz.

Treatment Charges

Treatment charges, deductions and penalties for the processing of gold doré and copper concentrates have been applied as anticipated in future PGM sales contracts.

Other Economic Assumptions

Other economic factors and assumptions used in the analyses are shown in Table 19-9.

Table 19-9: Other Economic Assumptions

Description	Value
Exchange Rate A$/US$ 2009 and 2010	0.70
Exchange Rate A$/US$ post 2010	0.85
Net Effective Royalty	3% of Metal Sales Value
Taxation	30%

19.7.3 Capital Cost Estimate

PGM has estimated its capital expenditure requirements in detail for the year 2009. In subsequent years more general estimates are applied based on anticipated mine life and operational requirements.

19.7.4 Operating Cost Estimate

Budgeted Life of Mine unit operating costs are used by PGM to estimate its future operating expenditure. The average unit costs applied to the respective analyses are shown in Table 19-10.

Table 19-10 : Average Unit Operating Costs (A$/t)

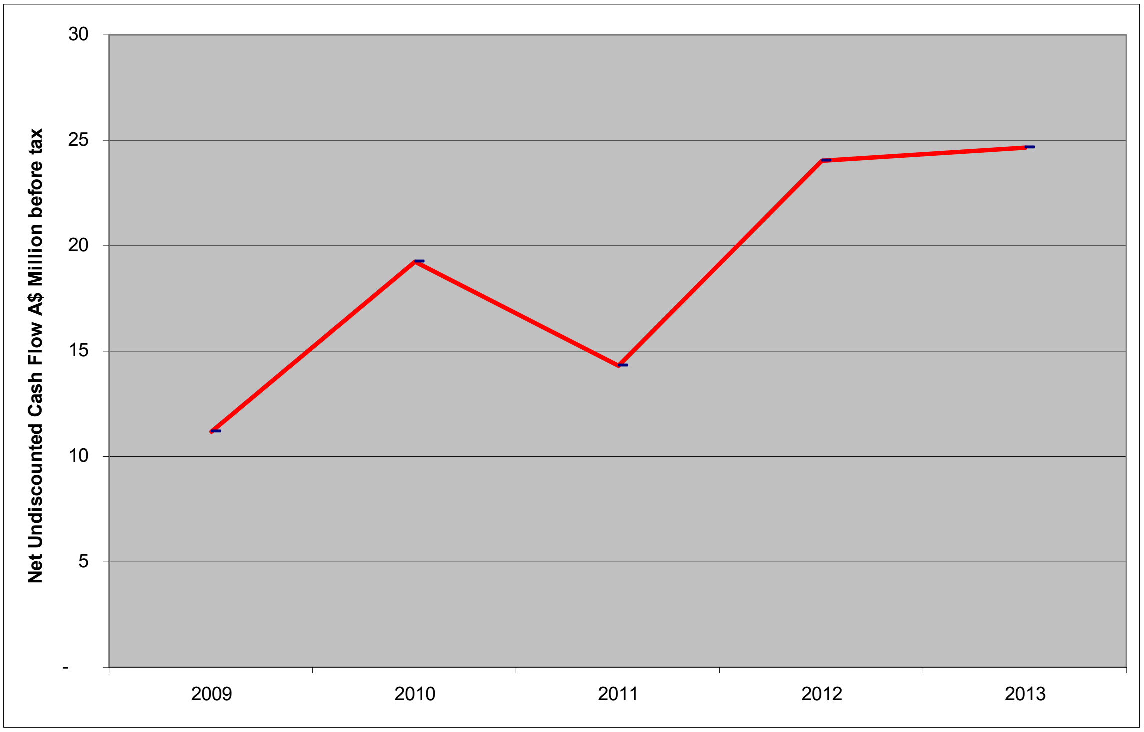
	Actual 2008	Budget 2009
Mining	55.20	63.17
Mill	23.53	26.91
Commercial	8.82	9.74
Total	87.55	99.82

19.7.5 Economic Analyses Results

Life of Mine Plan (Mineral Reserves Only)

Anticipated cash flows in A$ before company tax payments are shown in Figure 19-7. A major impact on the cashflow forecast is the reduction in capital expenditure towards 2013 due to the selective use of Reserve only tonnage.

Figure 19-6: Life of Mine Plan (Mineral Reserves Only) -Estimate of Annual Undiscounted Net Cash flow



Sensitivity Analysis

Sensitivity to changes in gold price, operating costs and capital costs are shown in Figure 19-7 to Figure 19-9.

Figure 19-7: Life of Mine Plan (Mineral Reserves Only) – Gold Price Sensitivity

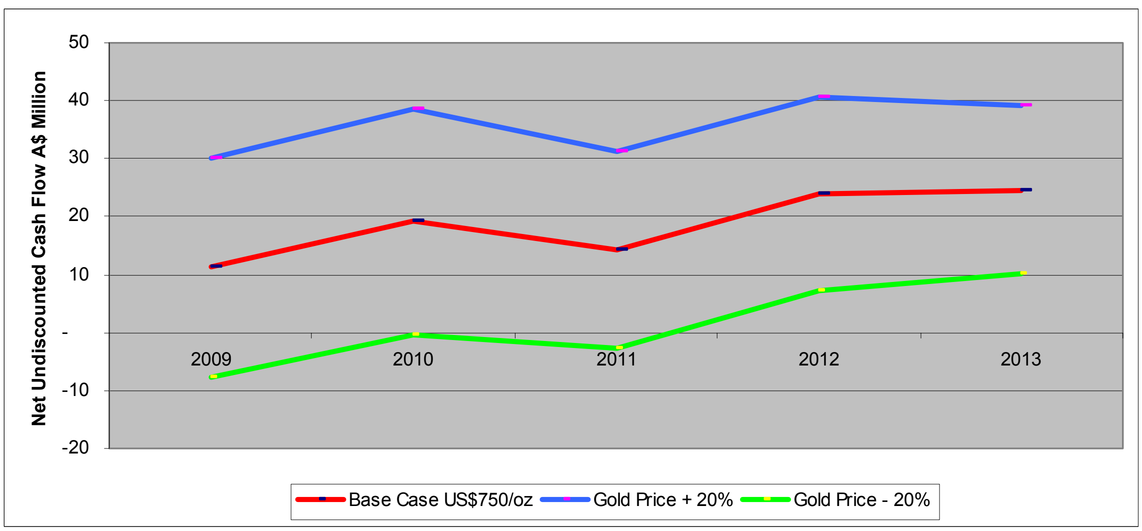


Figure 19-8: Life of Mine Plan (Mineral Reserves Only) – Operating Cost Sensitivity

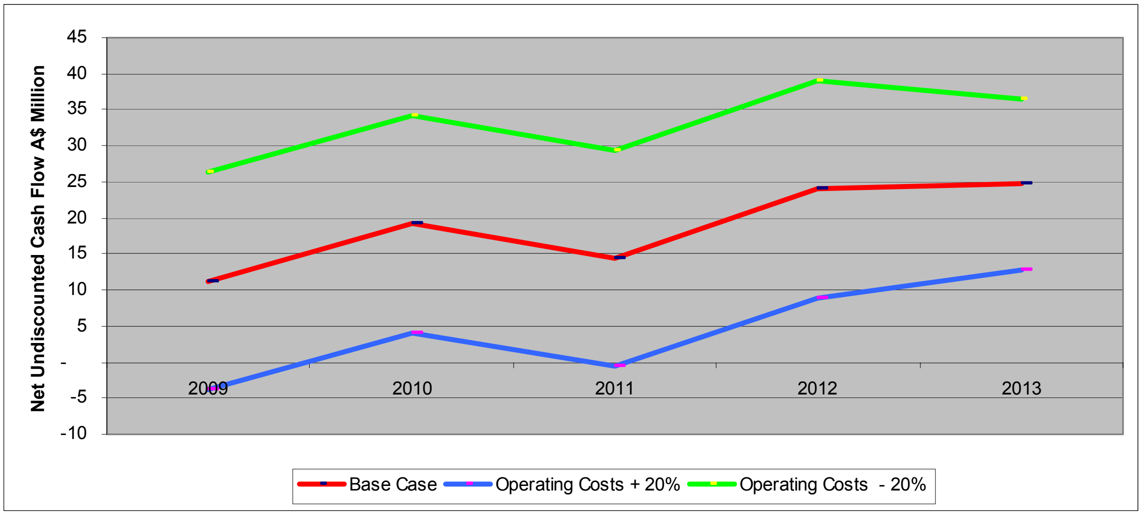
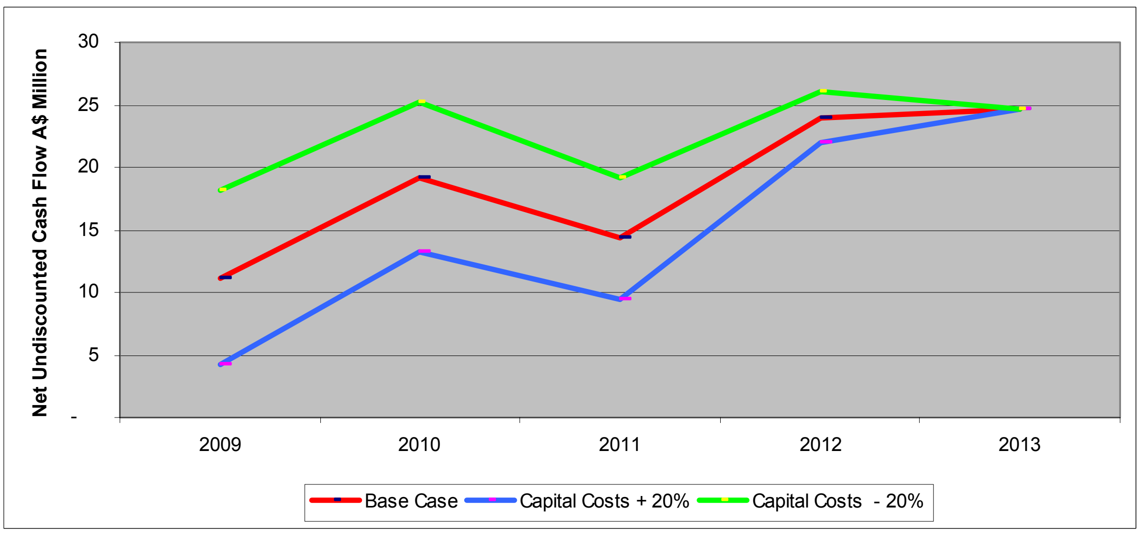


Figure 19-9: Life of Mine Plan (Mineral Reserves Only) – Capital Cost Sensitivity



19.8 Risk Analysis

19.8.1 Data Collection

The data collection and assay quality to be of low risk. PGM follows industry standard practices in the collection and analysis of the analytical data and checks made have not identified any material issues. The QAQC controls are considered to be sound and able to identify any areas which would require correction or repeats actions from the offsite laboratories.

19.8.2 Resources

The estimation methods are of industry standard and checks made have not identified any material issues with the current resource estimates and is therefore estimated to be a low risk. The current use of independent consultants (H&S) to conduct audits on resources and estimation procedures is good practice.

19.8.3 Mining and Reserves

The methodology applied by to Mineral Reserve estimation to be satisfactory.

19.8.4 Geotechnics and Hydrology

Controlled dewatering of historic workings at the New Cobar and Chesney deposits will continue. Monitoring of water levels will continue to be required. There are no known major aquifers in the current mining area. Continued vigilance will be required to ensure that development operations are controlled such that they do not intersect water bearing workings or drill holes. PGM has been operating under these conditions in recent years and there is no reason to believe that PGM will not continue to manage the conditions effectively.

PGM has demonstrated its ability to adapt and improve support practices when required. PGM has, given the success of shotcrete at New Cobar, embraced the process and commissioned a new Batch plant at the Peak mine. The increased depth of operation when mining commences in the Perseverance Deeps area may provide some challenges in terms of increased rock stress and potential for additional dilution that may require management. PGM has employed a full time Geotechnical Engineer and installed seismic monitoring systems in high risk areas.

19.8.5 Tailings Storage Facility

There is adequate tailings storage capacity available for the Reserves only Life of Mine production plan.

19.8.6 Metallurgy and Processing

The current age of the processing plant is around 17 years. The planned plant availabilities are in excess of 95%. Maintenance standards are high and corrosion issues are being addressed on a campaign basis. . In 2009, a major electrical / PLC upgrade is planned to modernise the control system and meet more stringent electrical standards.

19.8.7 Concentrate Transport and Sales

All of Peak Gold Mine's concentrate is sold to Glencore International AG. Currently the concentrate is trucked to the Glencore owned CSA Mine in Cobar (a distance of approximately 10 km), and then railed to the Newcastle port. Shipping then occurs to smelters, usually to China. The current contract is set to terminate in the first half of 2009 subject to meeting some production targets. A new contract arrangement may enter new risks and or costs to the business.

19.8.8 Infrastructure and Services

The current infrastructure and level of services are adequate for the future operational plans.

19.8.9 Environmental Impacts and Permitting

An External Environmental Audit of the PGM operation was undertaken in 2006 with a subsequent report issued in February 2007. The report noted a number of significant improvements since the previous audit in 2004. Three issues classified as of high importance were identified relating to tailings management which have been addressed. No issues classified of very high importance were identified.

No significant Aboriginal heritage issues have been identified in relation to PGM's operations.

PGM updated its Closure Plan and re-estimated closure costs at A$11.4M in 2008. As with all closure cost estimates, there are uncertainties involved. The greatest uncertainties involve changes in closure requirements, and hence cost of closure. This is reviewed annually to ensure currency and to take in to account any new disturbance, progressive rehabilitation or other factor effecting closure.

19.8.10 Project Economics

The current Mineral Reserve allows for five years of economic operation.

PGM also produces an operational Life of Mine Plan which includes the mining of Inferred Resources and mineralization from advance exploration projects as well as the Reserves which indicates a nine year mine life. PGM's operational plan is preliminary in nature and includes material that is too speculative geologically to have the economic considerations applied to them, and therefore there is no certainty that the operational Life of Mine Plan will be realized.

20 INTERPRETATIONS AND CONCLUSIONS

Geology, Data and Resources

PGM has estimated a total Mineral Resource (exclusive of Mineral Reserves) of 3.2Mt grading 3.31g/t Au and 0.94% Cu in the Measured and Indicated categories and 2.5Mt grading 4.4g/t Au and 0.83% Cu in the Inferred category for the Peak, New Occidental, Perseverance, New Cobar, and Chesney deposits.

Drilling data density is considered reasonable and adequate prior to the point of designing stopes and mining ore. In areas which are further away from existing infrastructure, the data density decreases. As development approaches these areas, drilling is completed and more data is collected as time progresses.

The Geology Department has detailed plans for future near mine and regional exploration efforts.

Mining

Mining conditions are well understood by PGM. The mining methods and equipment fleets are appropriate for the deposits and forecast rates of production. Infrastructure in place at the mine is appropriate for a mine of its size. Refrigeration has been installed in the Perseverance Deeps area and initial results have been good.

A mine plan based on Proven and Probable Reserves only is sufficient for five years mine life. PGM is operating to a nine year Life of Mine Plan 2009-2017 that includes an increasing level of Inferred Mineral Resources and mineralisation as the plan matures. Continued focused attention is required on exploration to identify Mineral Resources and delineation for identification of economically minable material.

PGM's operational plan includes Inferred Mineral Resources and mineralisation from advanced exploration projects that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the operational Life of Mine Plan will be realized.

Most of the production is derived from ore mined at the underground operations from the New Occidental, Perseverance, Chesney and New Cobar zones. Underground production has been supplemented by open cut stockpile material. This material is planned to be exhausted in early 2009. Production from underground will have to increase to meet the shortfall. Chesney is flagged to fill this void.

Metallurgy

Budgeted metallurgical recoveries are determined from a combination of operating history and, where necessary, metallurgical test work. Recoveries for New Occidental, Perseverance, New Cobar (POX), New Cobar (Underground) and Peak ores are predominantly based on historical operating data. Peak has a system which considers the metallurgy of the individual ores and then produces a predicted recovery based on the feed mixes. The risk of adverse metallurgical responses (such as from any as yet unidentified mineralogical assemblages) can be reduced by performing standard metallurgical tests on composite samples representative of the production plant.

Given the small amount of operating history with Chesney ore, budgeted metallurgical responses need to be reinforced by metallurgical test work. Supporting metallurgical test work is required to confirm predicted metallurgical recoveries for these ores.

The current age of the processing plant is around 17 years. The planned plant availabilities are in excess of 95%. The practice of refurbishing the plant to address corrosion should continue. PGM has a major upgrade of the Mill electrical and control systems planned for 2009. This is required to meet current Australian Standards.

Environmental and Closure Plan

An External Environmental Audit of PGM operations was undertaken in 2006 with a subsequent report issued in February 2007. The report noted a number of significant improvements since the previous audit in 2004, most notably in water recycling, implementation of dust suppression measures and upgrading of the tailings pipelines. Three issues classified as of high importance were identified relating to tailings management. No issues classified of very high importance were identified.

No significant Aboriginal heritage issues have been identified in relation to PGM's operations.

PGM updated its Closure Plan and re-estimated closure costs in 2008 at A$11.4M. As with all closure cost estimates, there are uncertainties involved. The greatest uncertainties involve changes in closure requirements, and hence cost of closure.

21 RECOMMENDATIONS

Rehabilitation

PGM has estimated the closure cost to be A$11.4M. This cost can be substantially reduced by progressive rehabilitation. Opportunities exist to rehabilitate areas with the waste products from mining as well as in the mining process by utilising waste products as underground fill.

Mining

The Peak deposit has associated with it zones of lead and zinc that are potentially economic. These zones can be processed with the lead / zinc depressed and treated as penalty elements in the NSR script. Alternately, a lead and zinc circuit can be introduced to the process plant. Lead and zinc could then be recovered and provide an additional income stream. Further work needs to be done to determine the economics of recovering lead & zinc from Peak ores. The AU$35,000 that has been budgeted in 2009 for engineering studies should be progressed and possibly increased.

Copper Flotation

PGM is proposing to introduce Flash Flotation in 2009. The Flash Flotation is proposed to increase copper recovery, gain the ability to handle higher grade feeds and to increase the copper grade of the copper concentrate. These are sound reasons and it is recommended that a justification be developed for introducing this process. A$1.2M has been budgeted for this upgrade in 2009.

Regional Exploration

PGM holds arguably the best ground in the Cobar district. A project of 3D alteration modelling may provide additional regional exploration targets which should be ranked along side existing targets and investigated as a priority. Capital should be allocated based on prospectivity.

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23 DATE AND SIGNATURE PAGE

The undersigned prepared this Technical Report, titled *Technical Report on the Peak Gold Mines, New South Wales, Australia,* dated January 1, 2009. The format and content of the report are intended to conform to Form 43-101F1 of National Instrument 43-101 of the Canadian Securities Administrators.

Signed,

"Peter Lloyd"

Peter Lloyd, FAusIMM — 30th day of March 2009

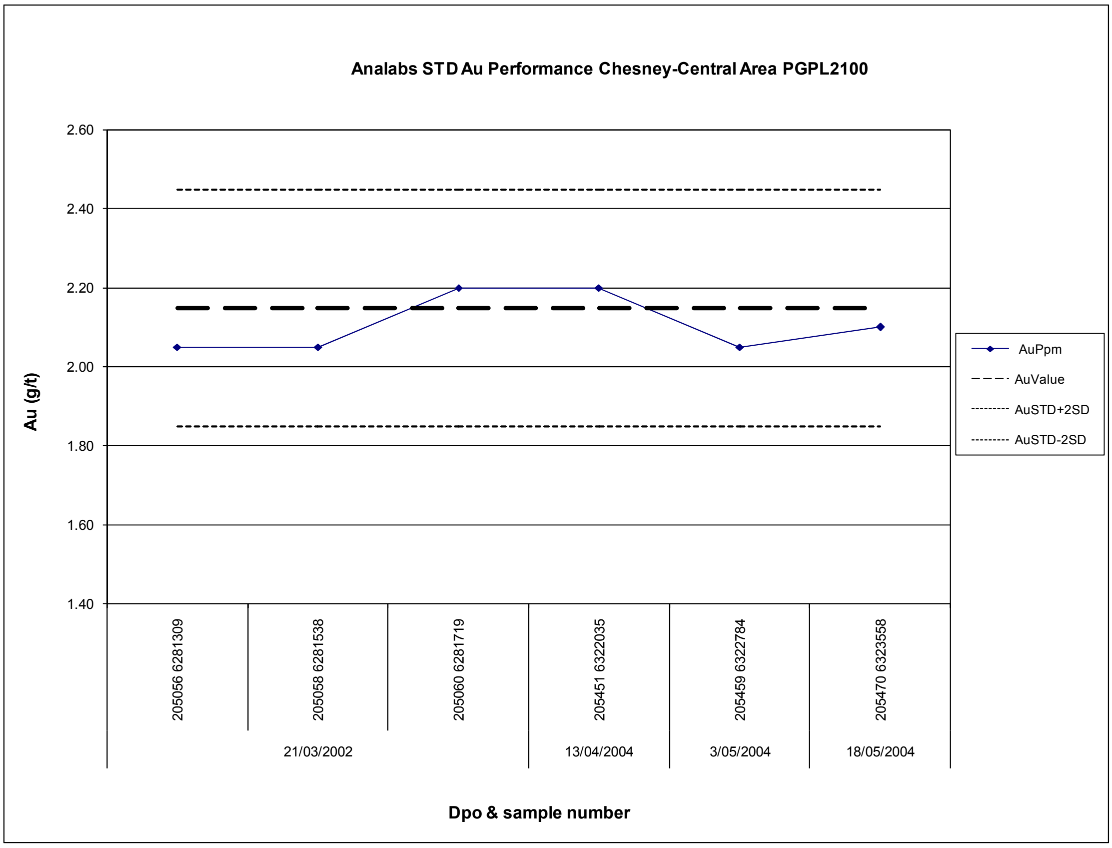
"Rex Berthelsen"

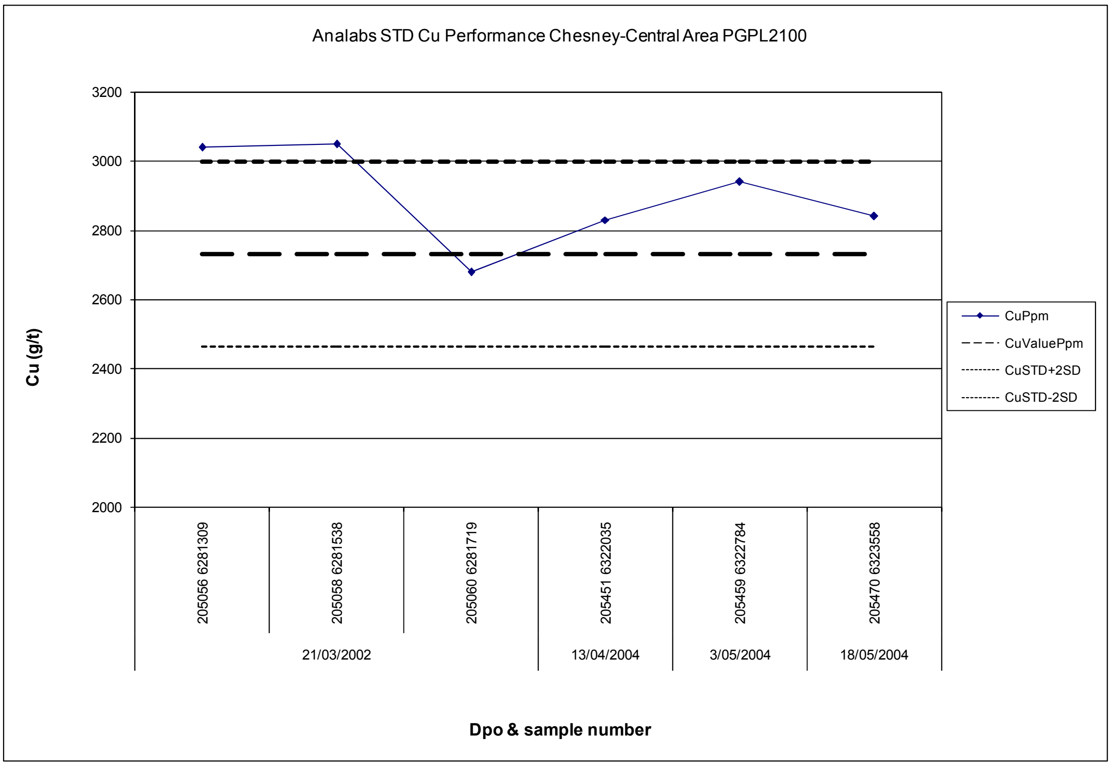
Rex Berthelsen, MAusIMM — 30th day of March 2009

"Eric Strom"

Eric Strom, P.Eng — 30th day of March 2009

Appendix A - Standards





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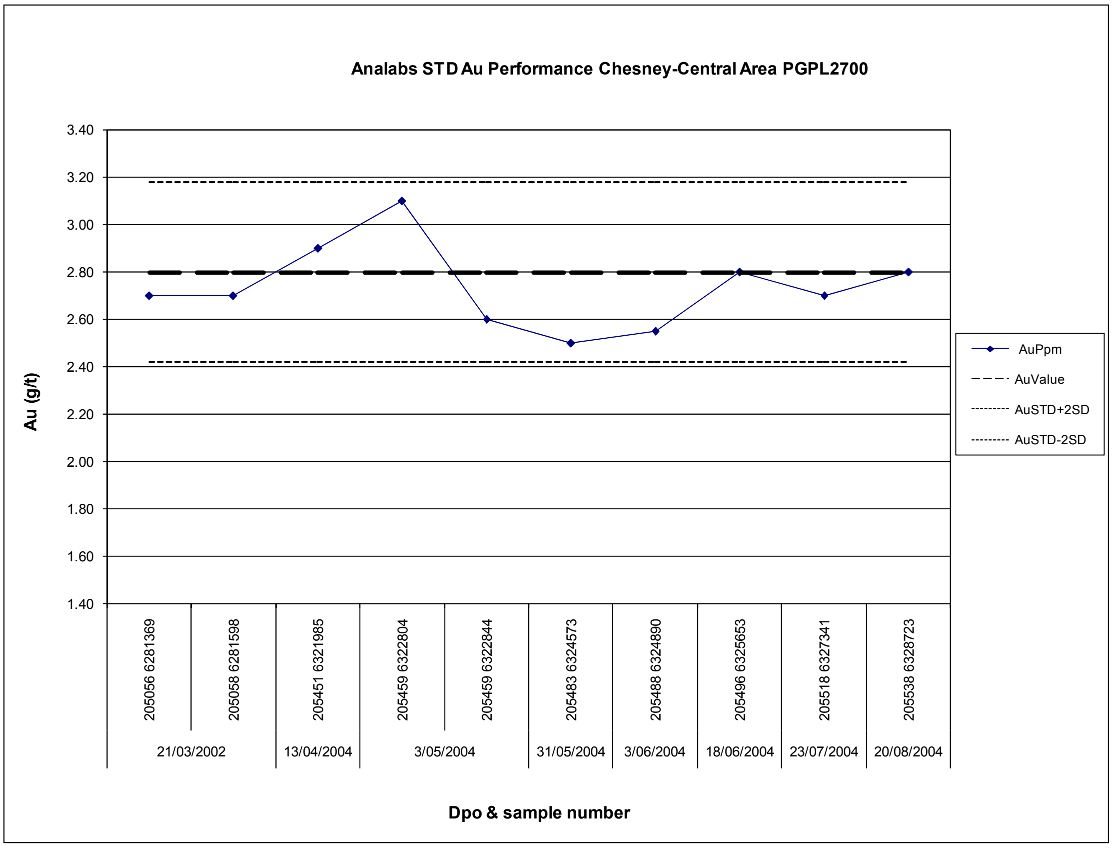

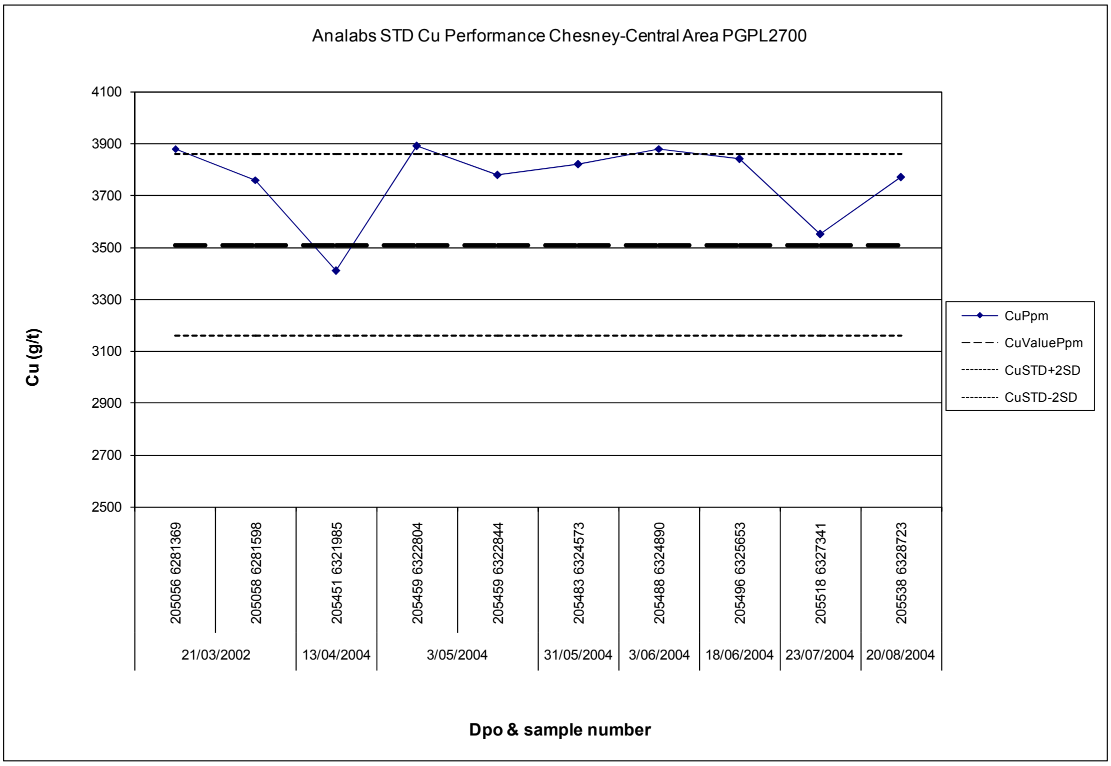

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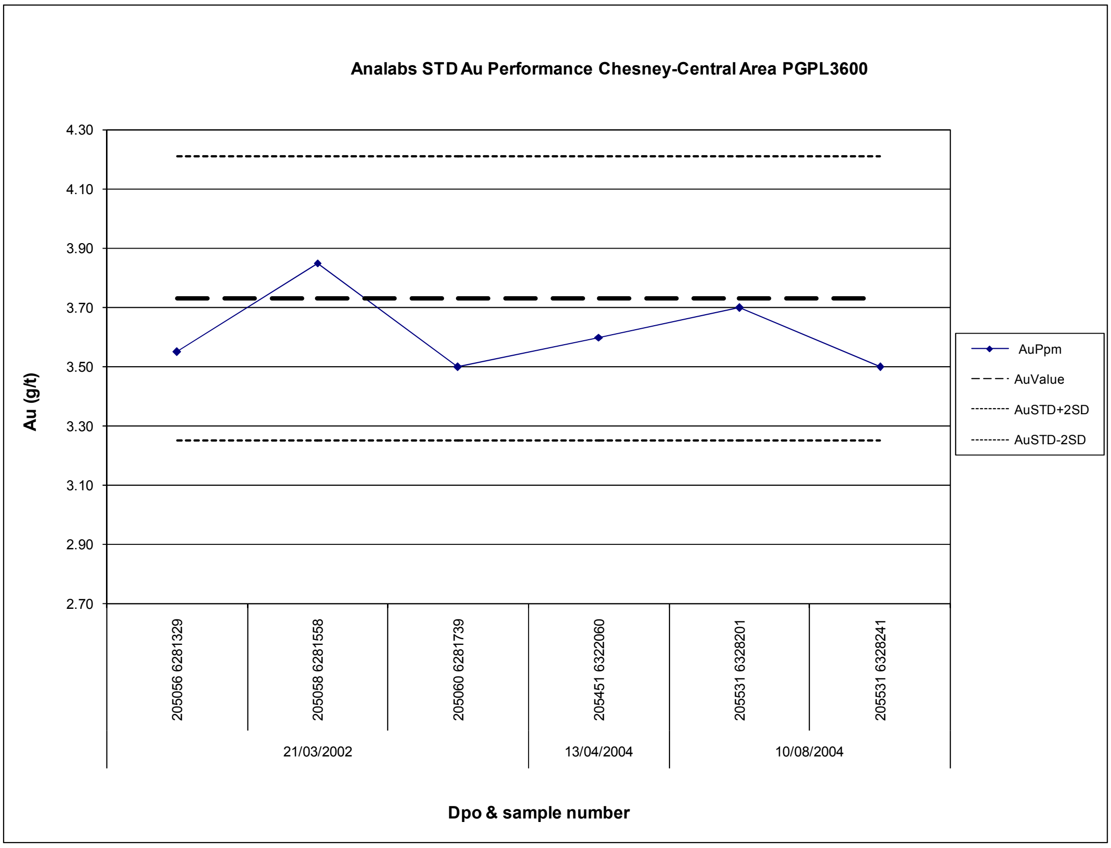

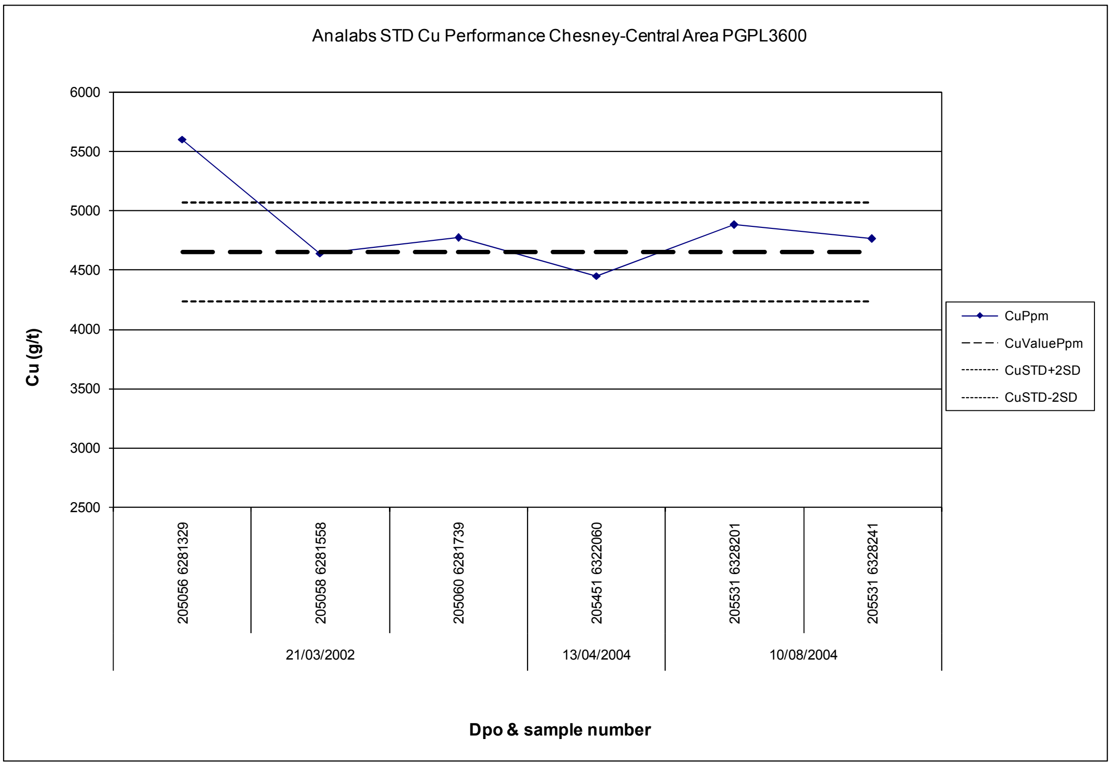

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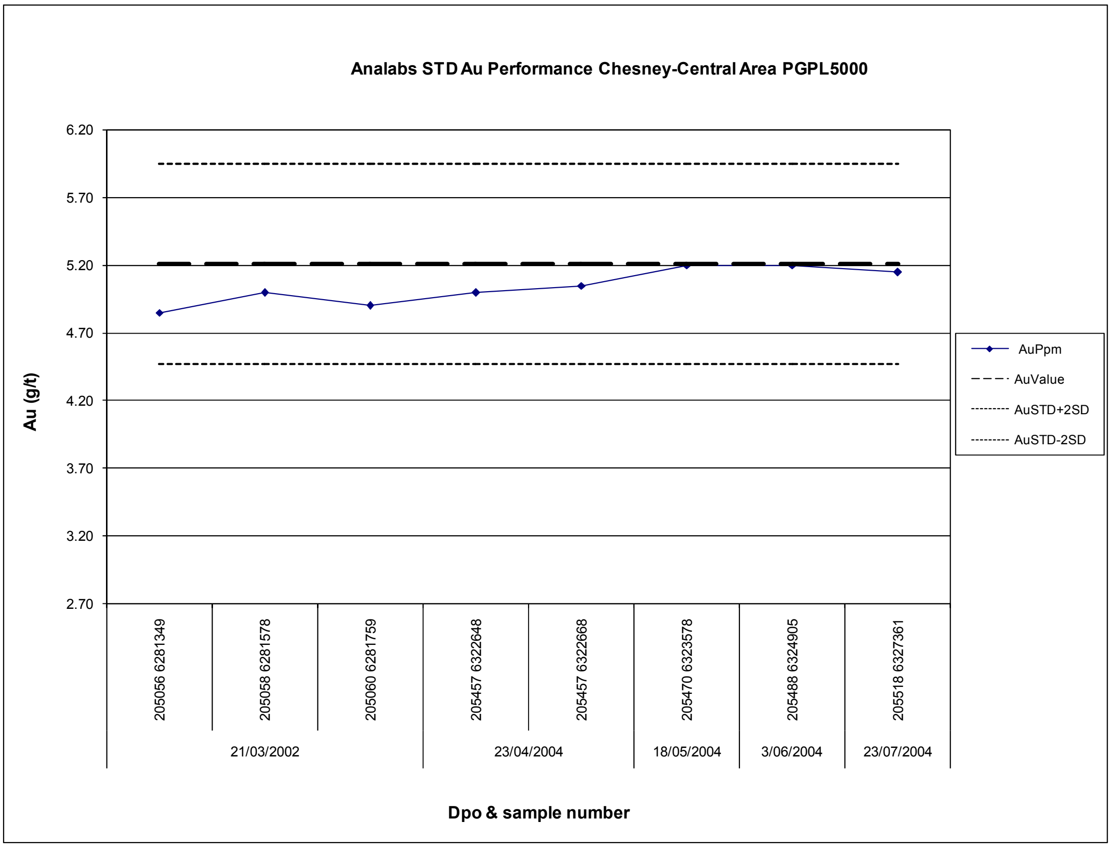

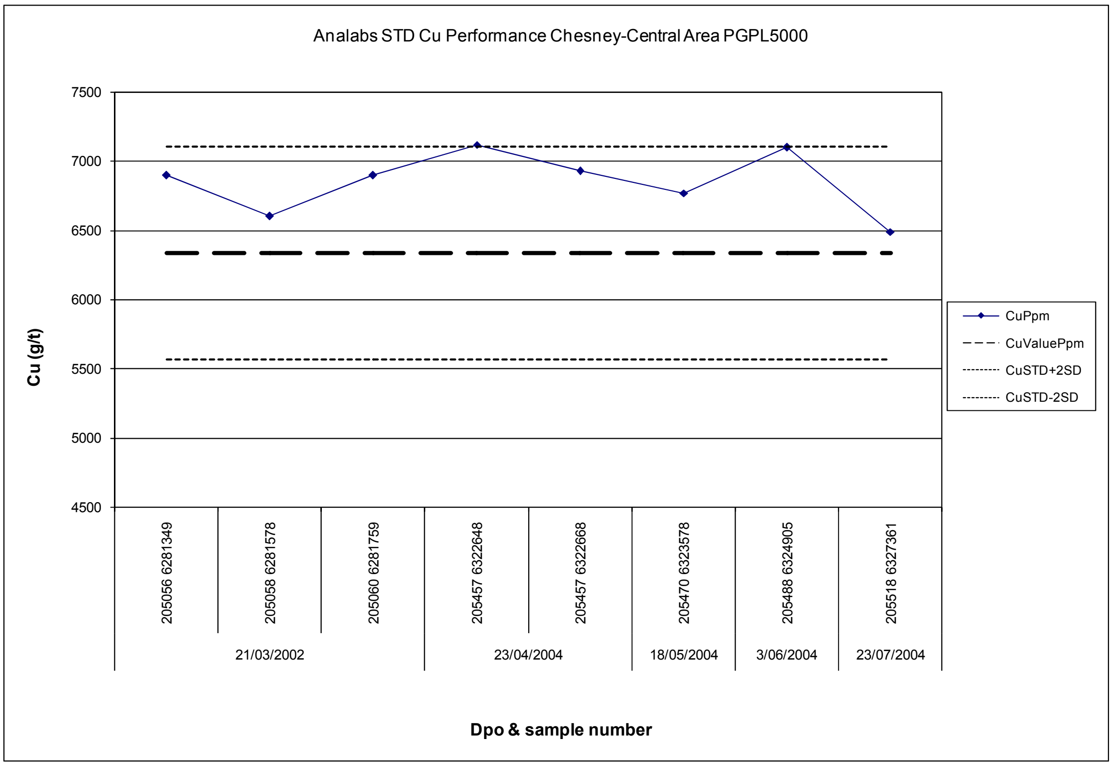

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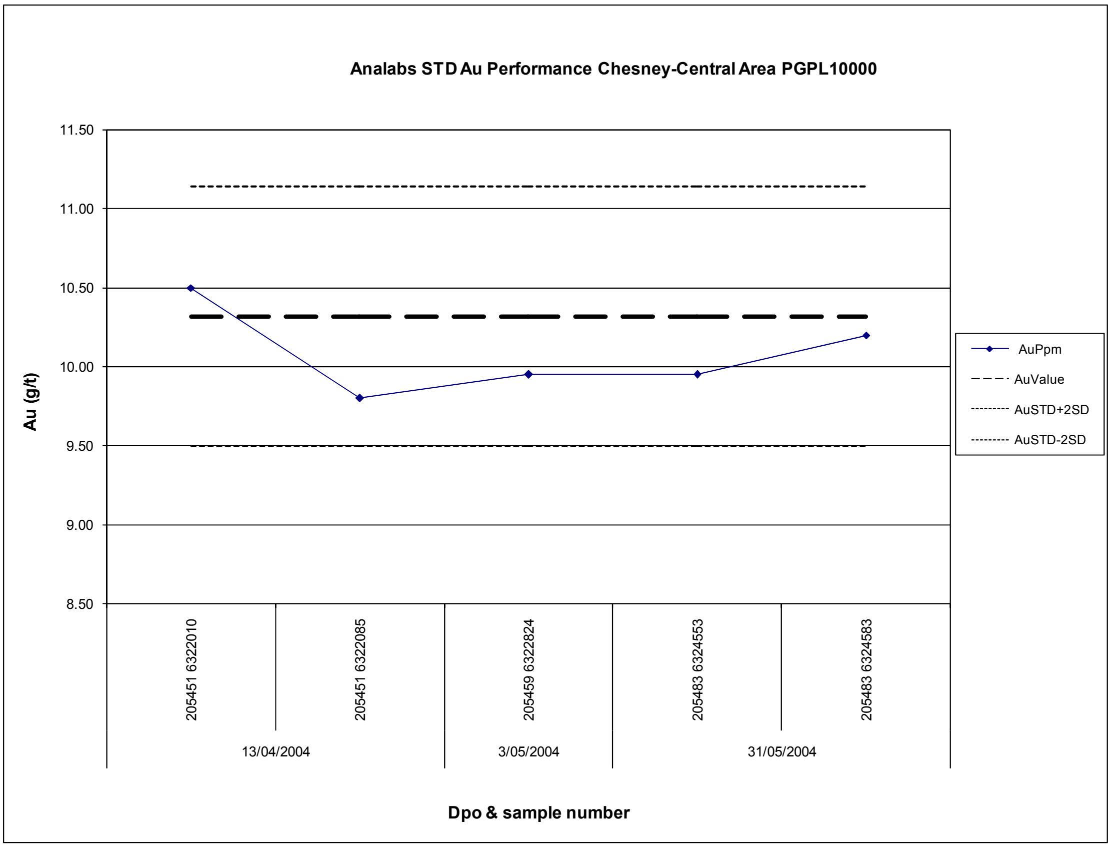

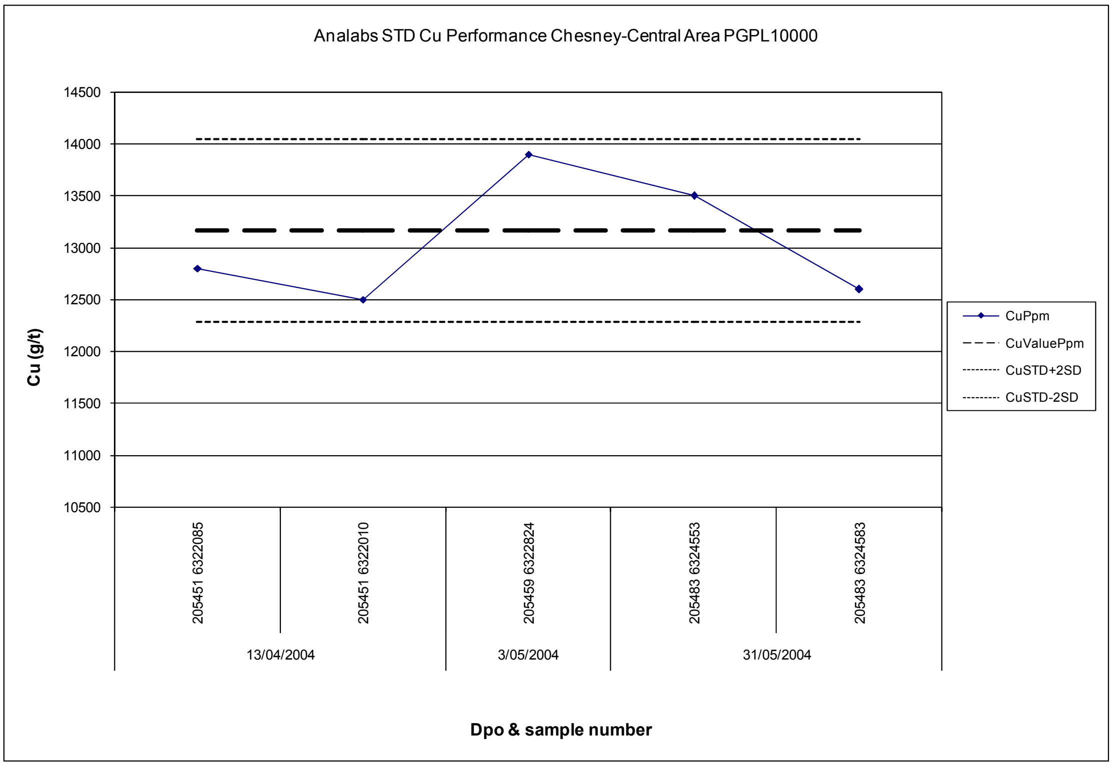

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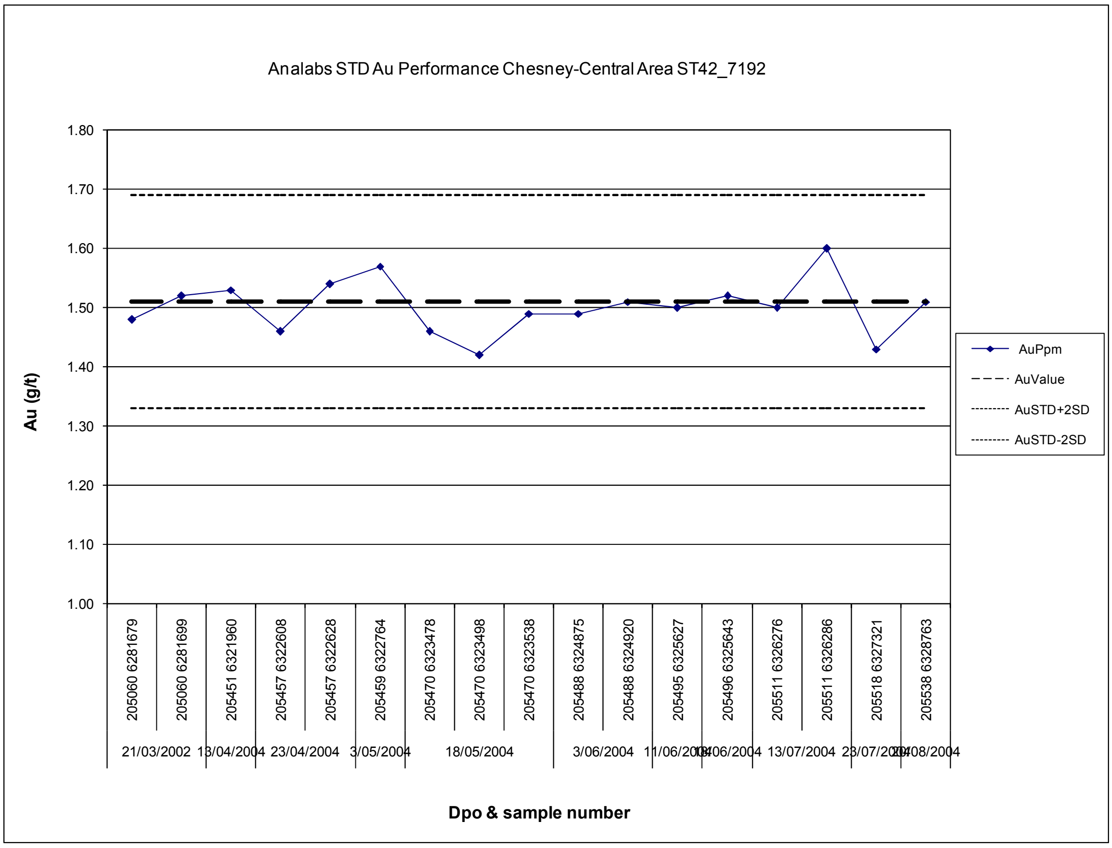

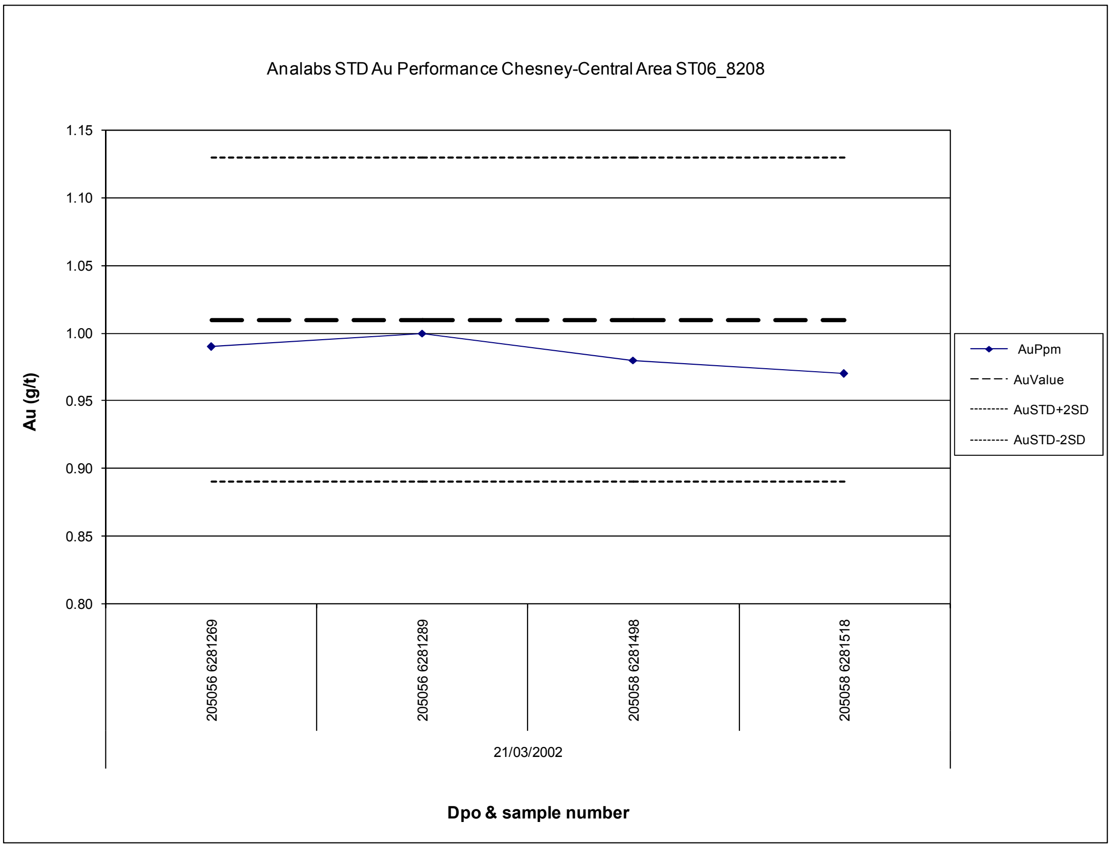

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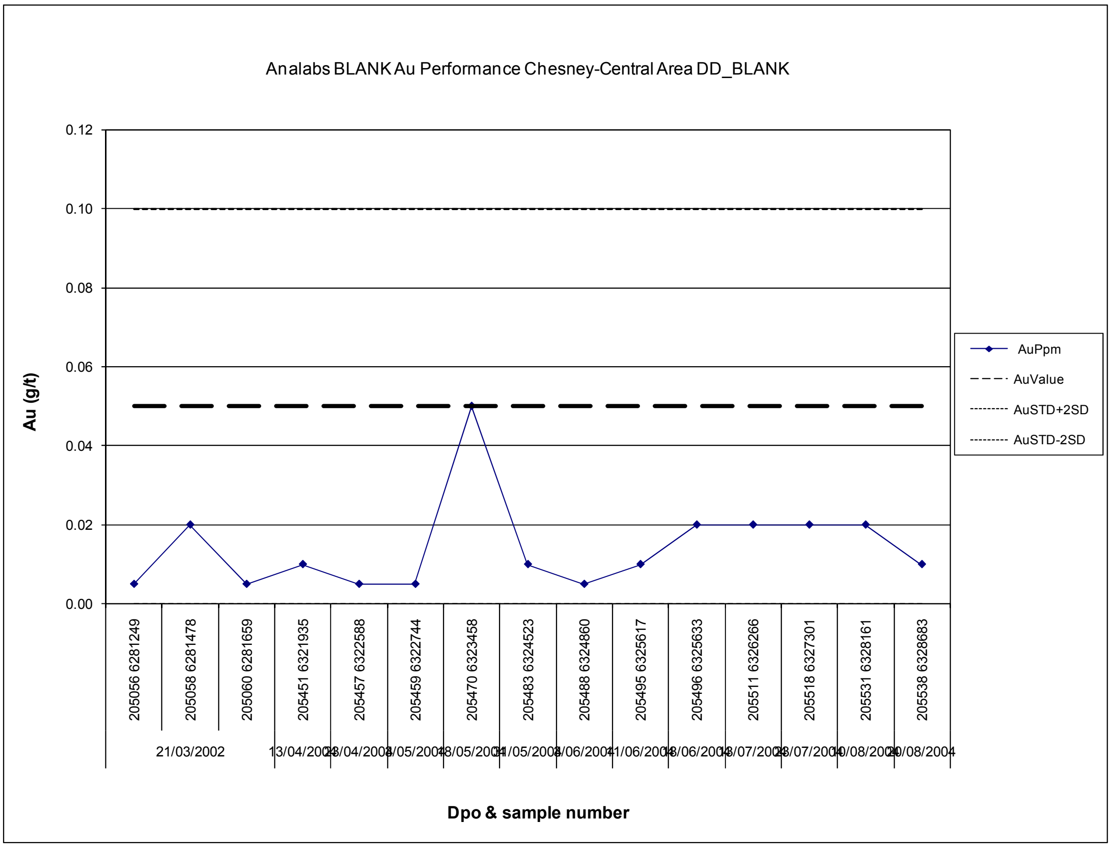

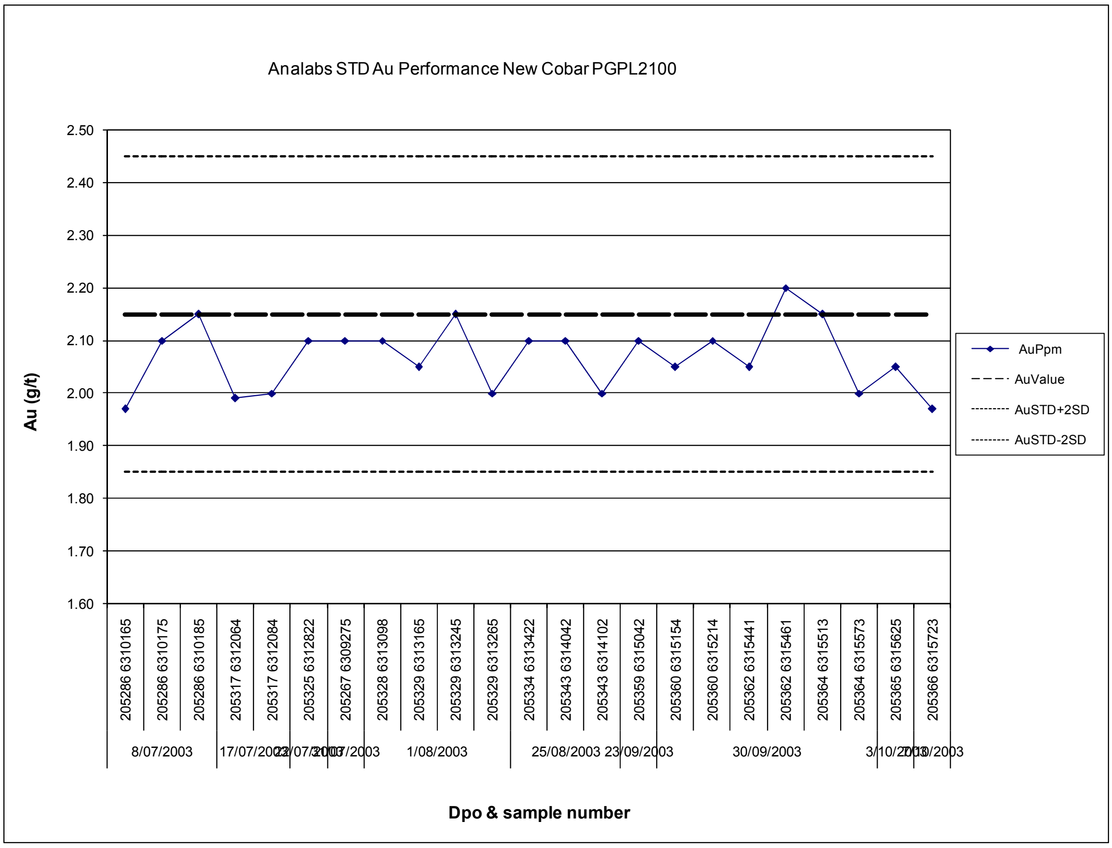

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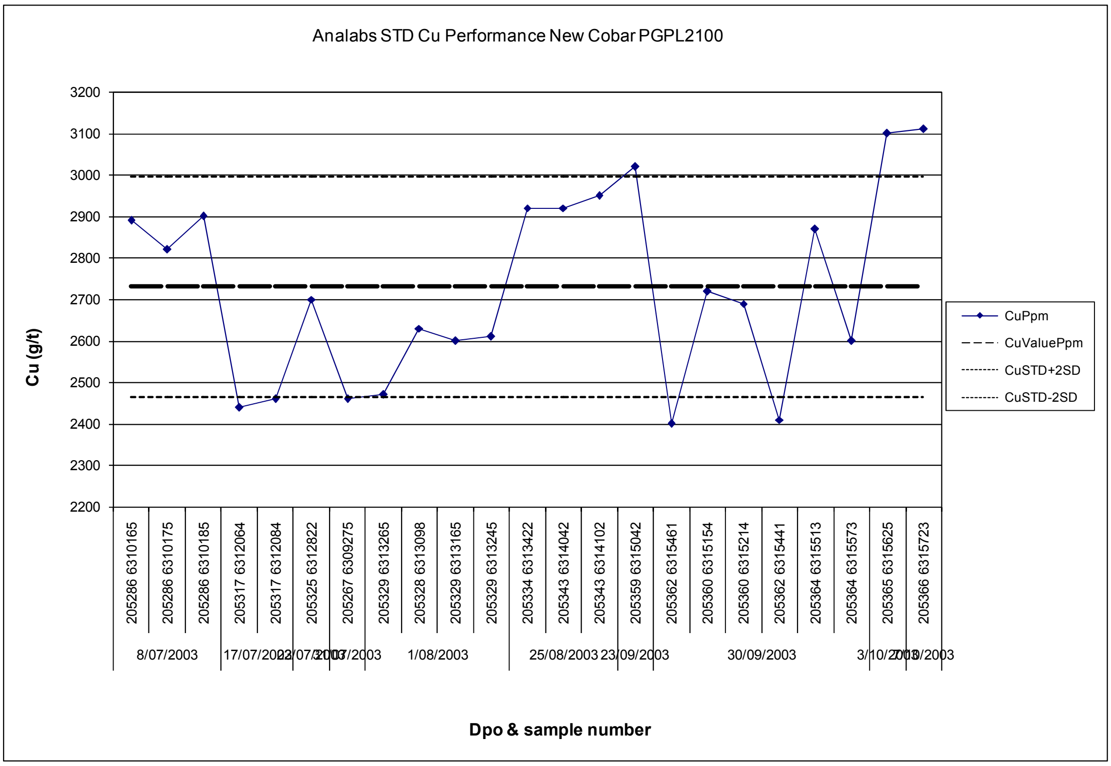

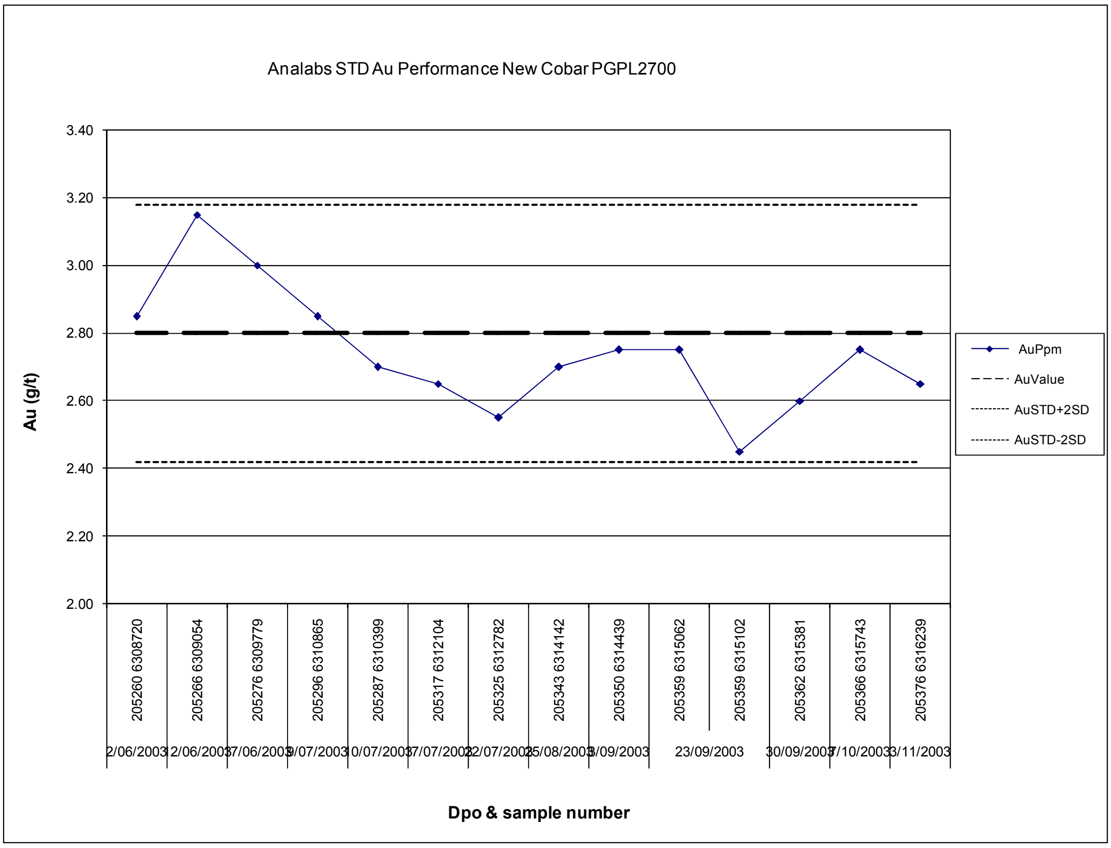

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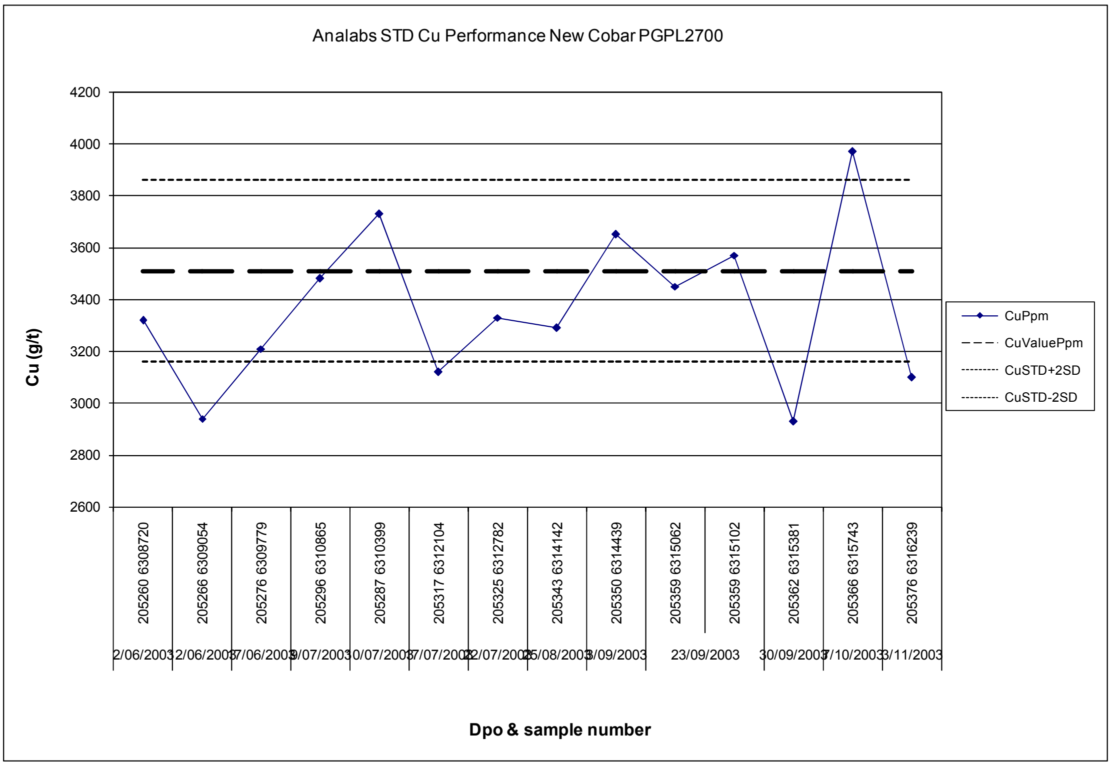

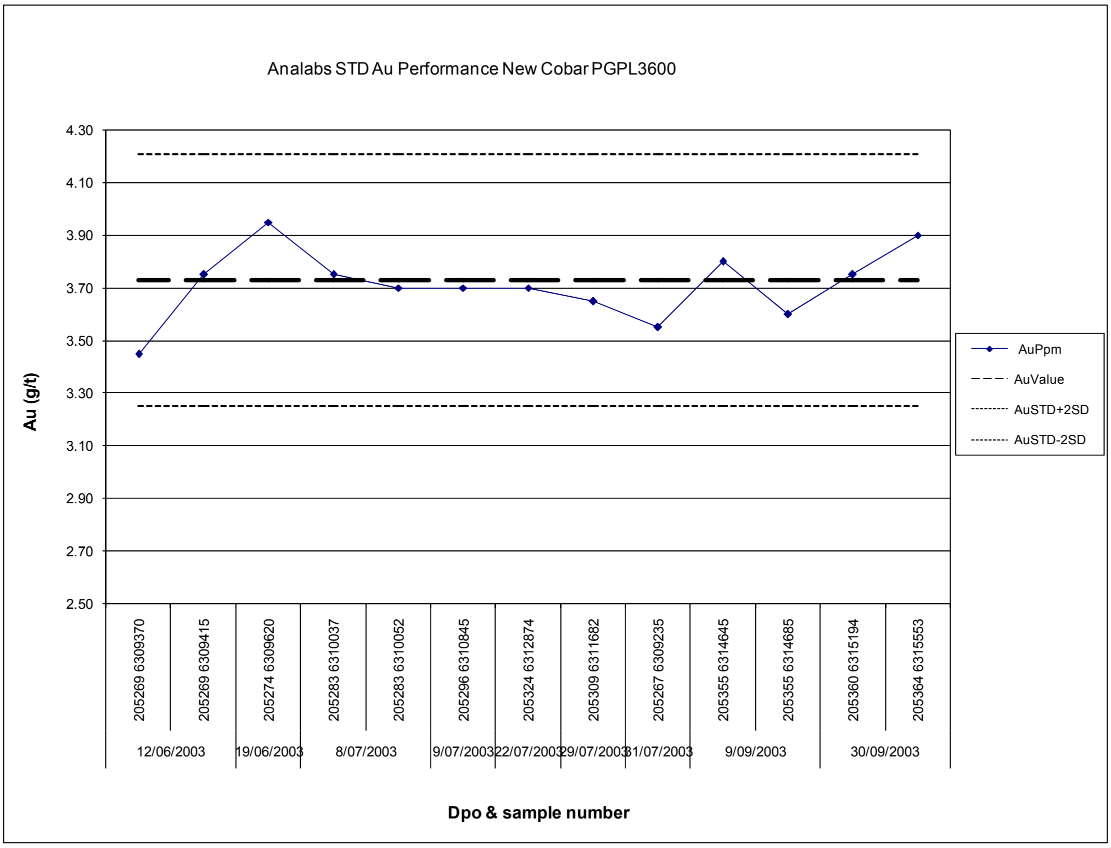

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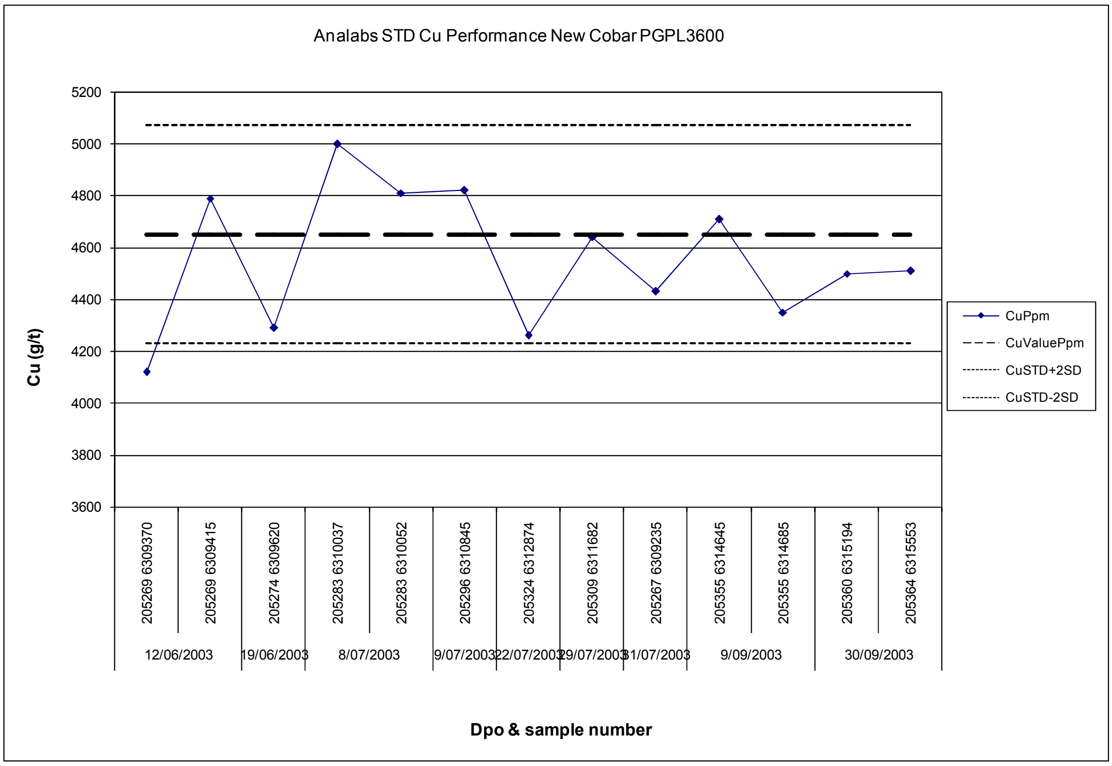

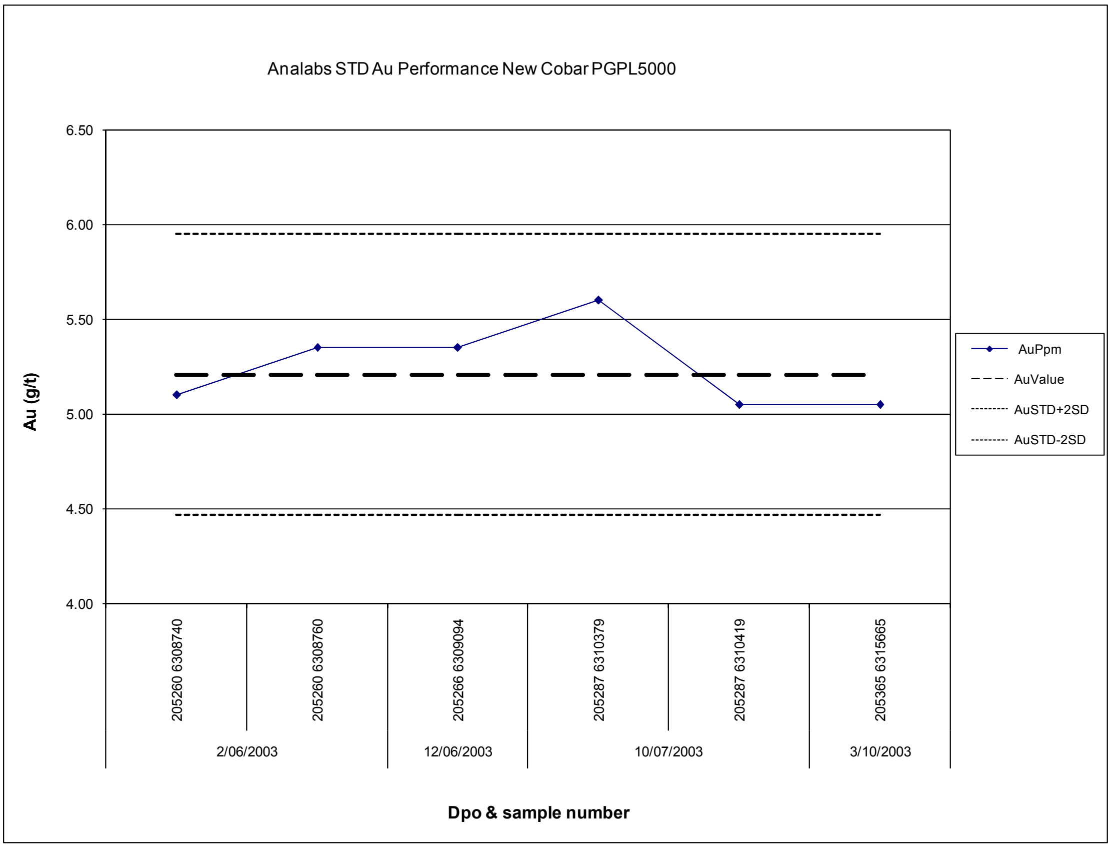

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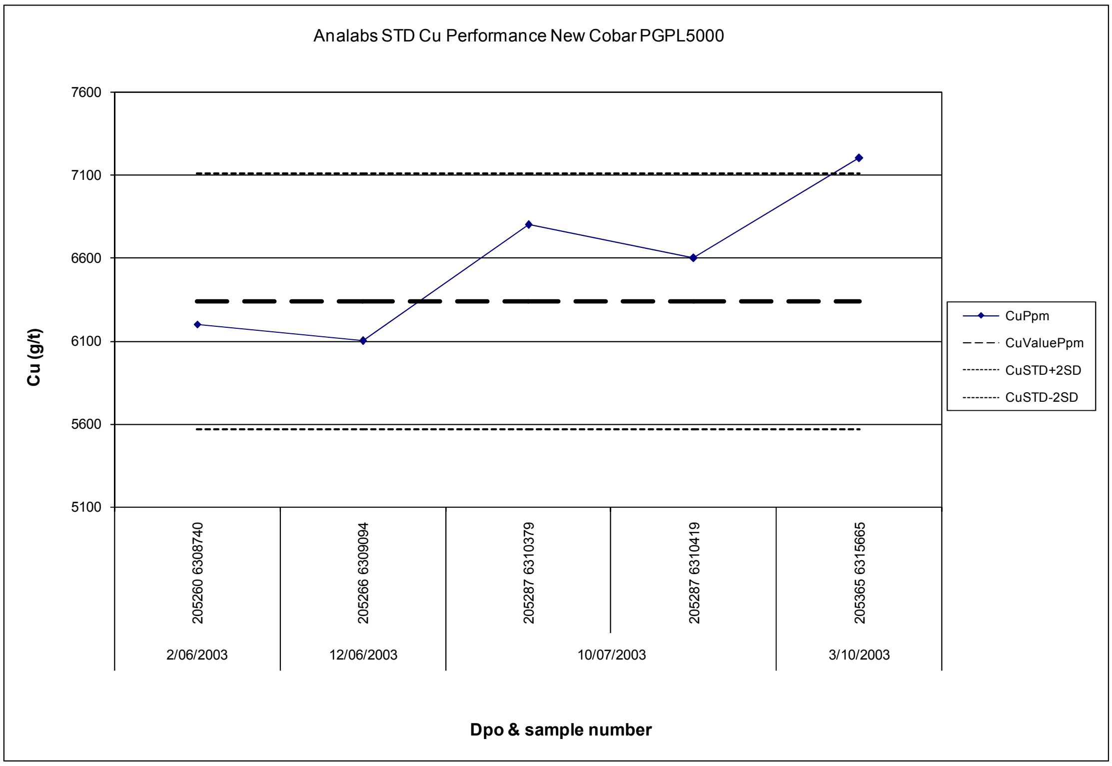

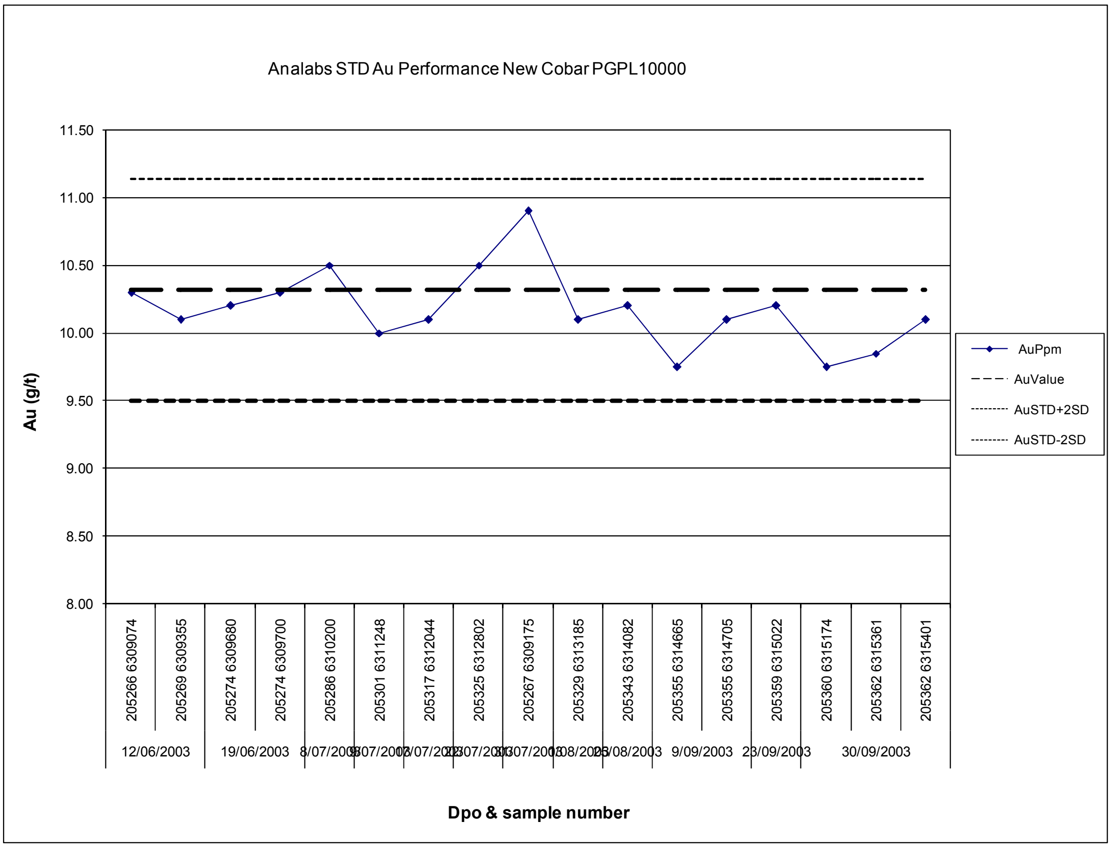

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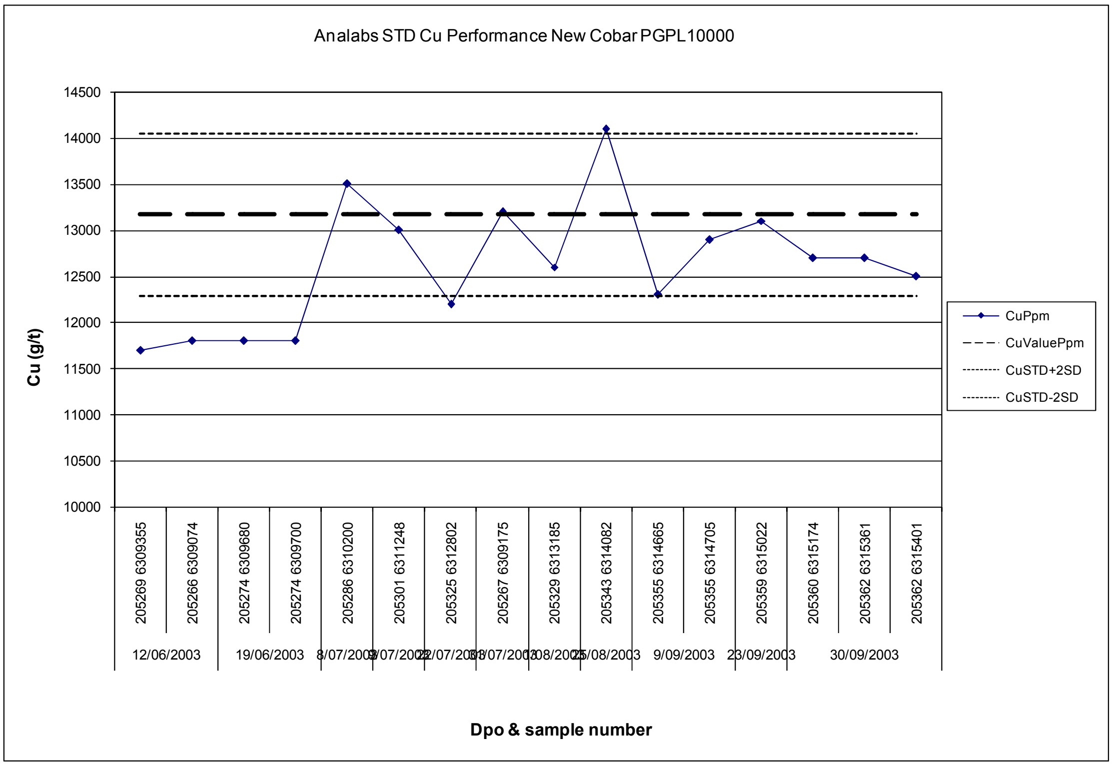

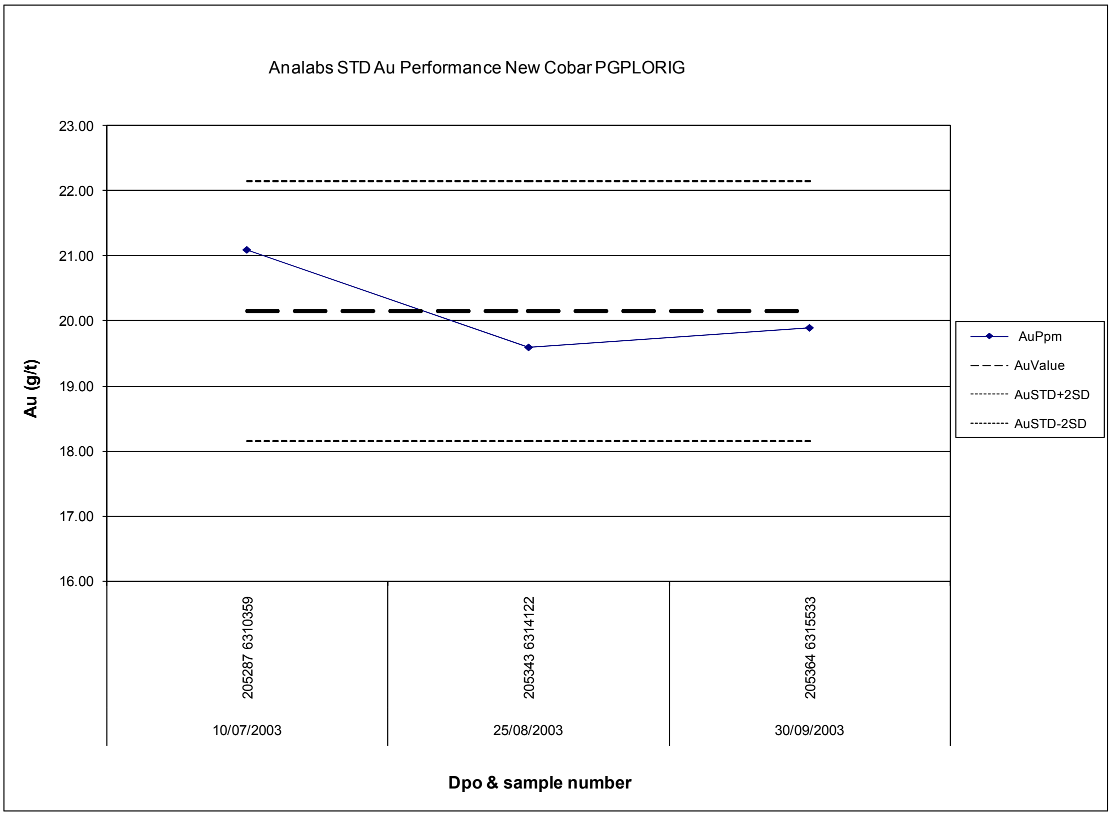

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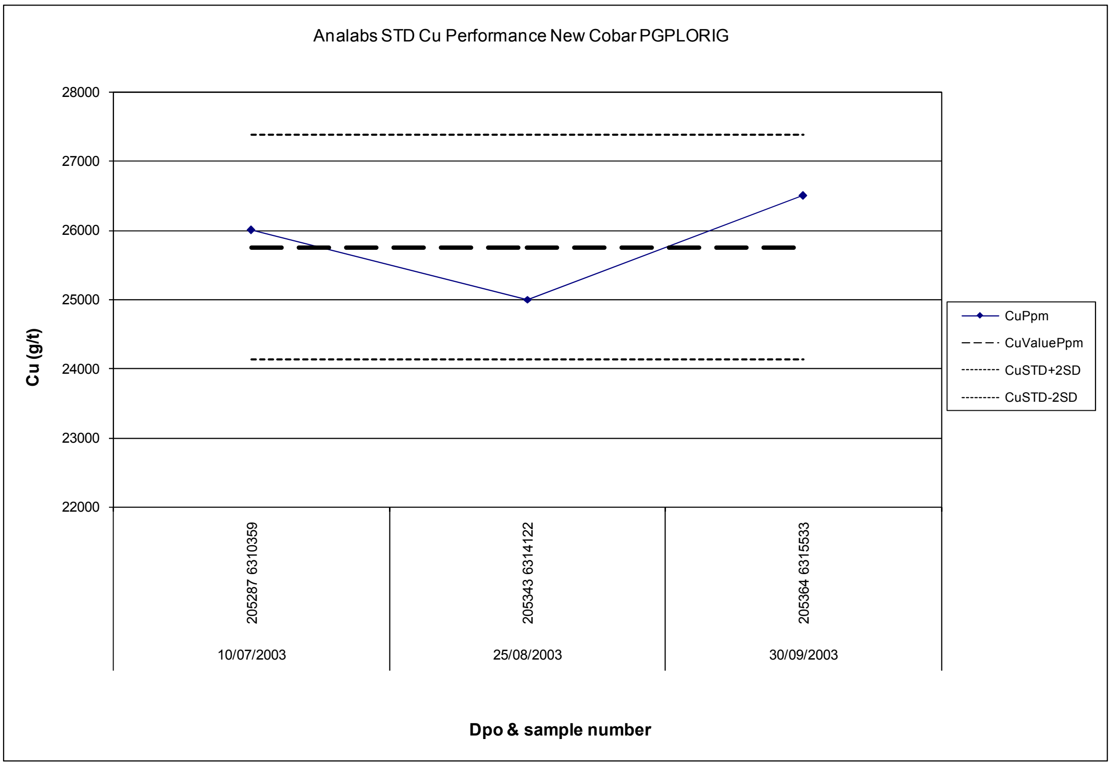

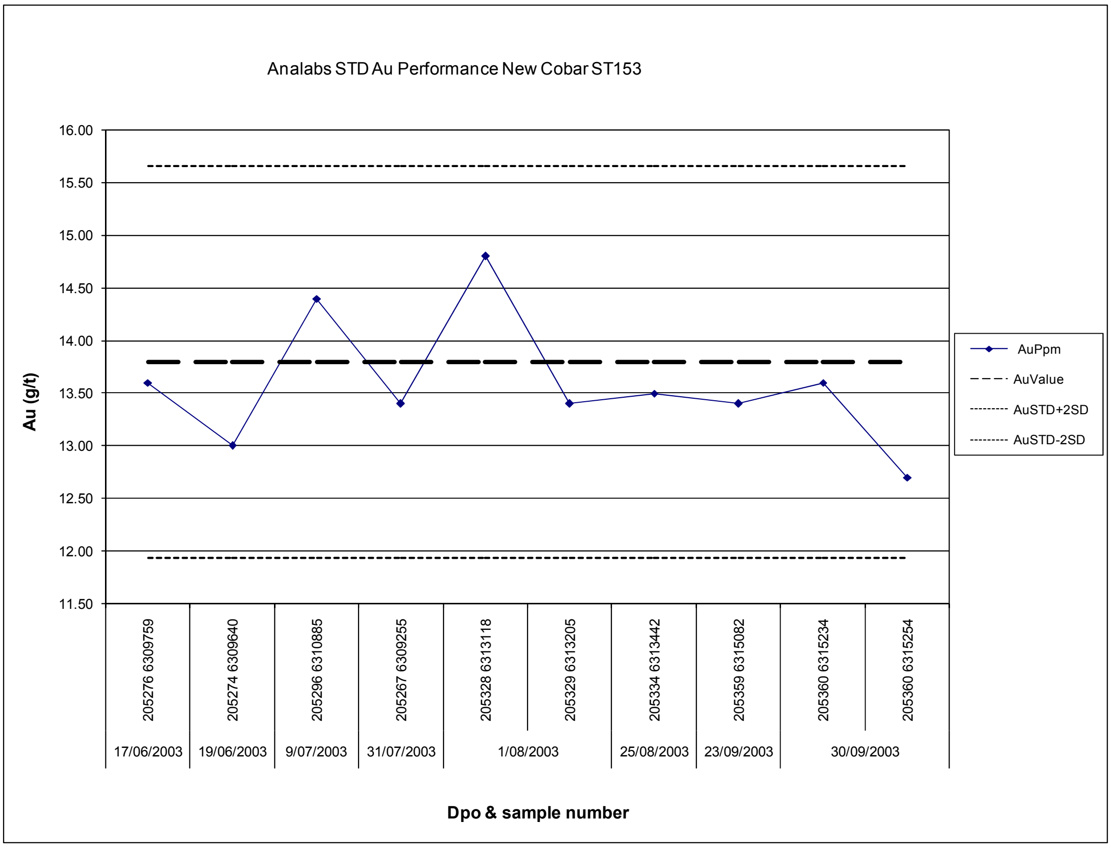

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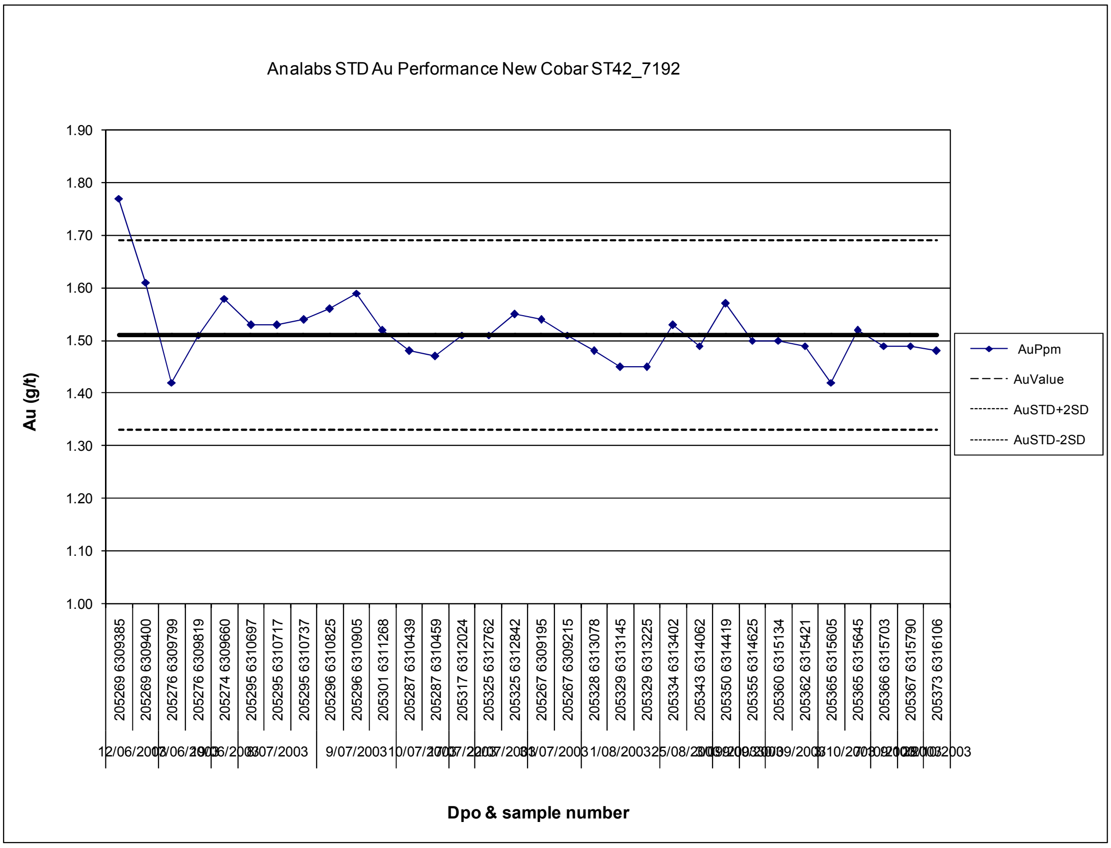

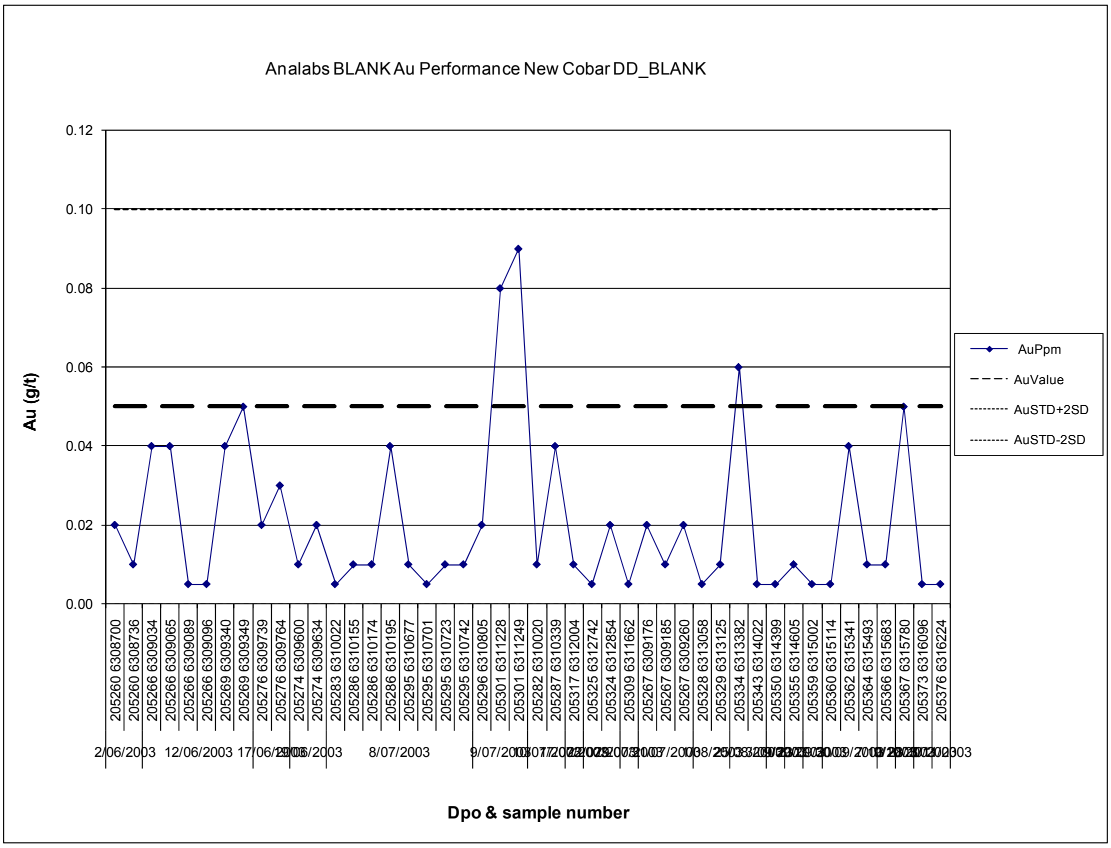

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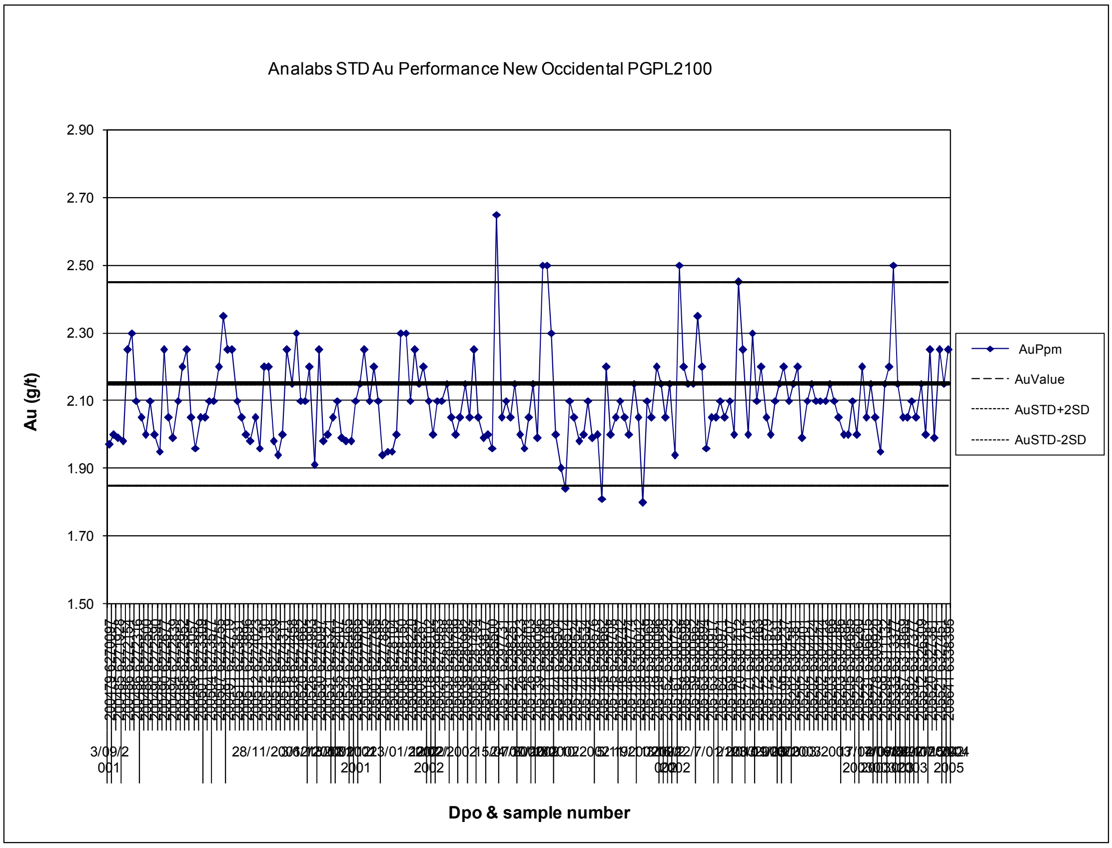

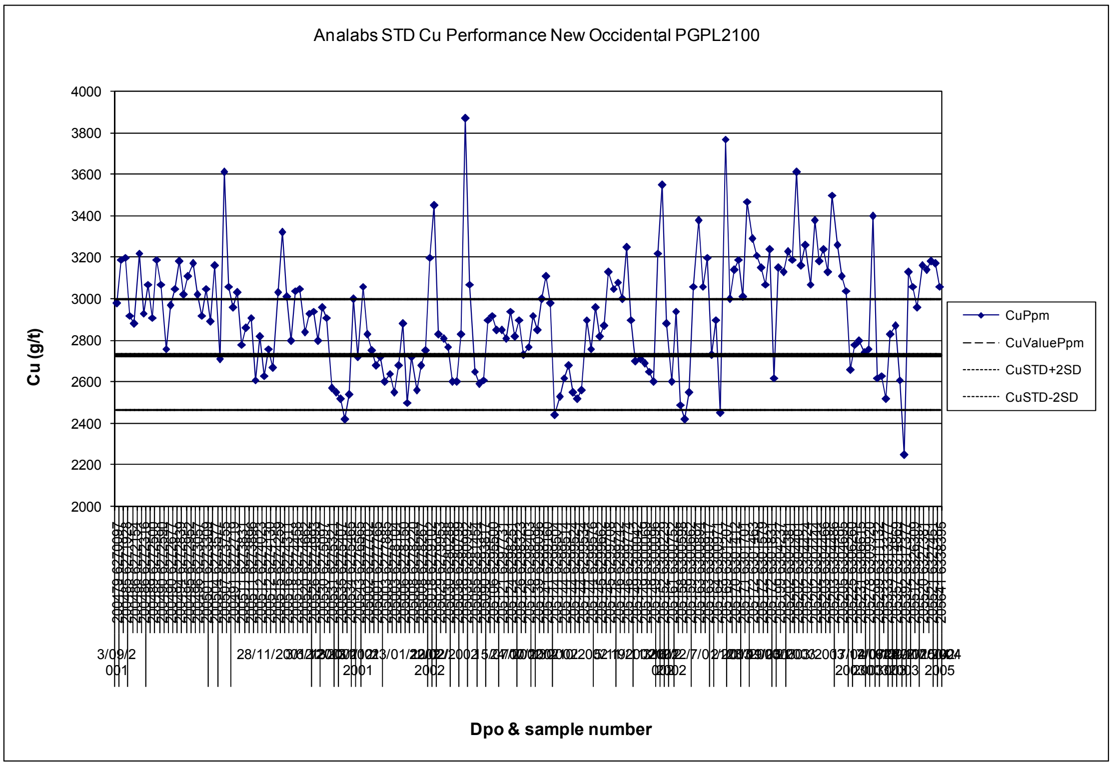

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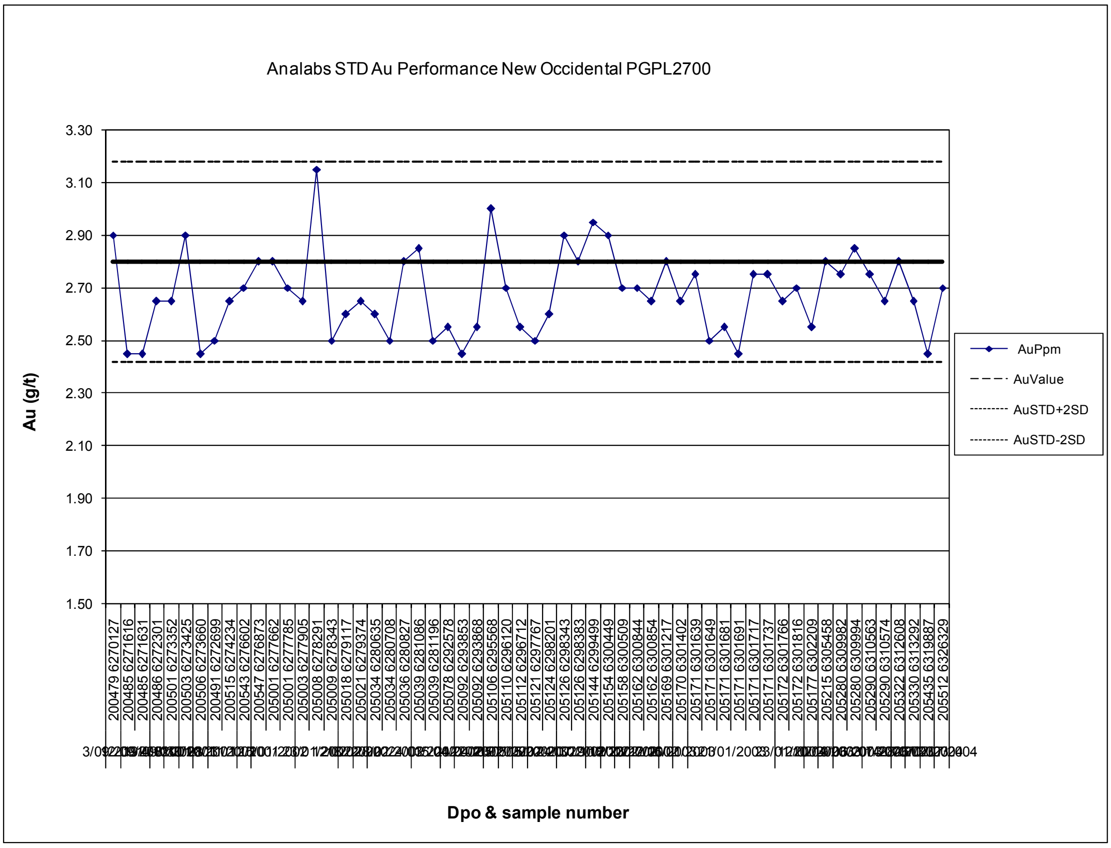

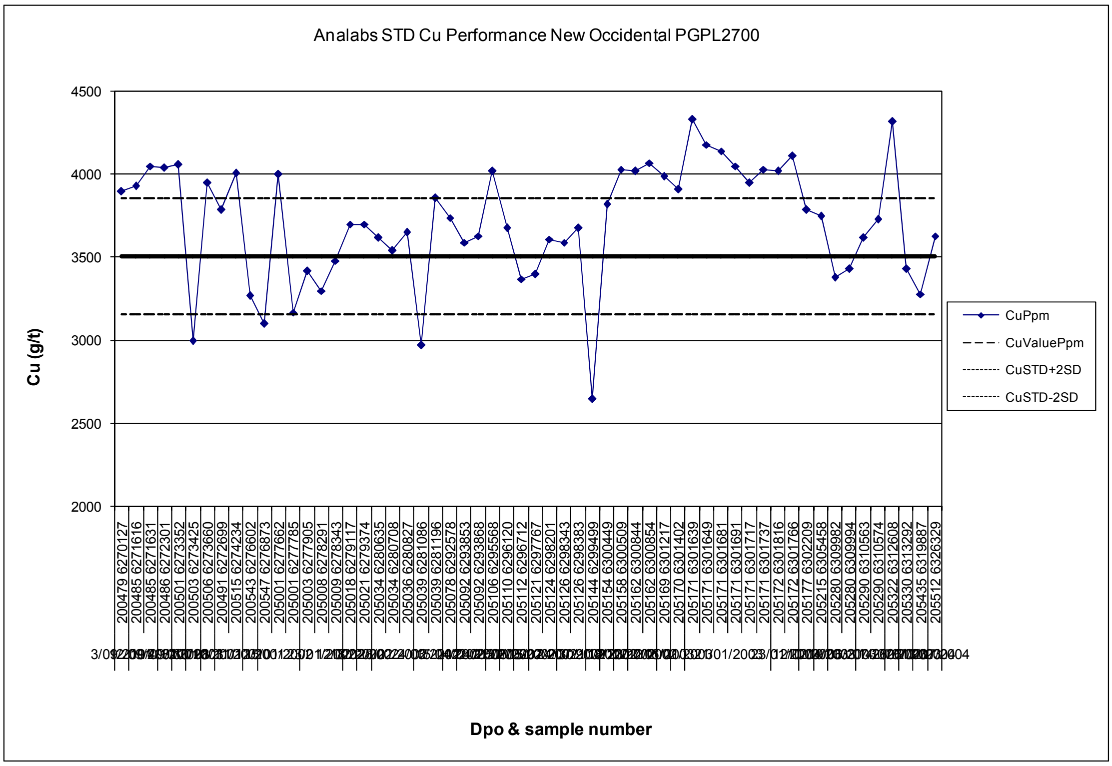

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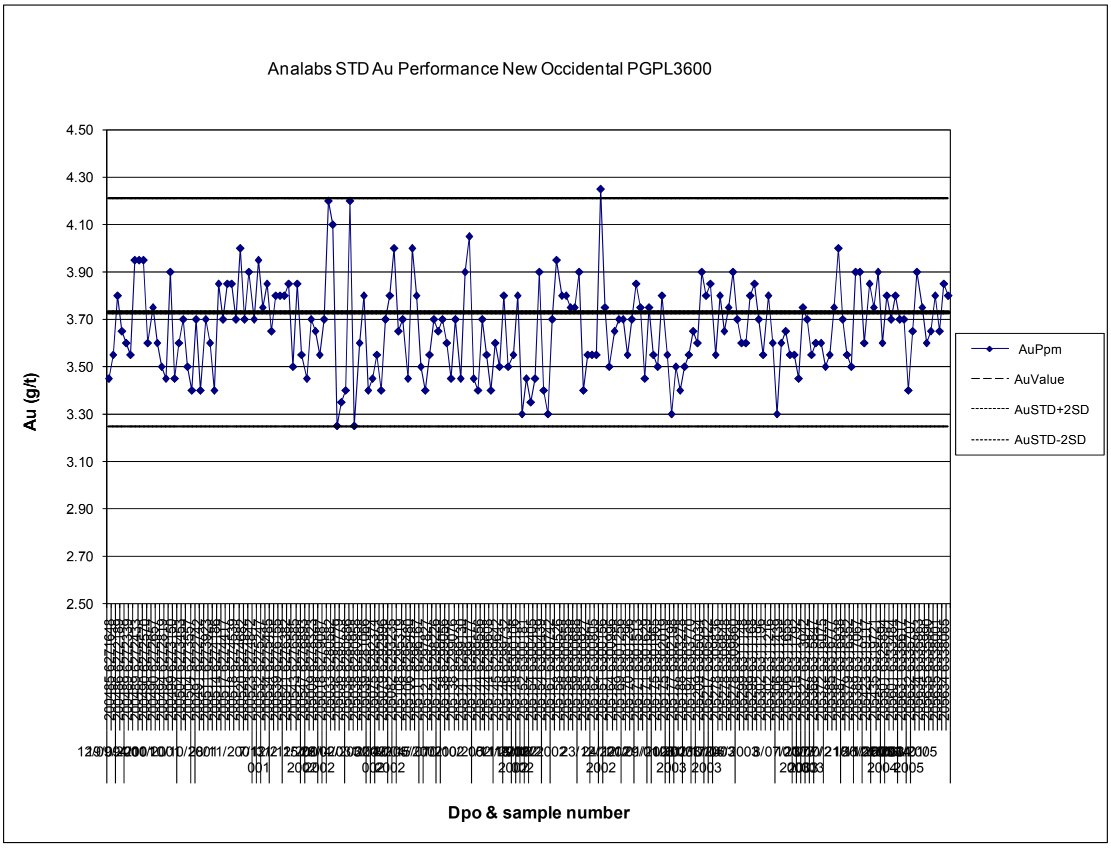

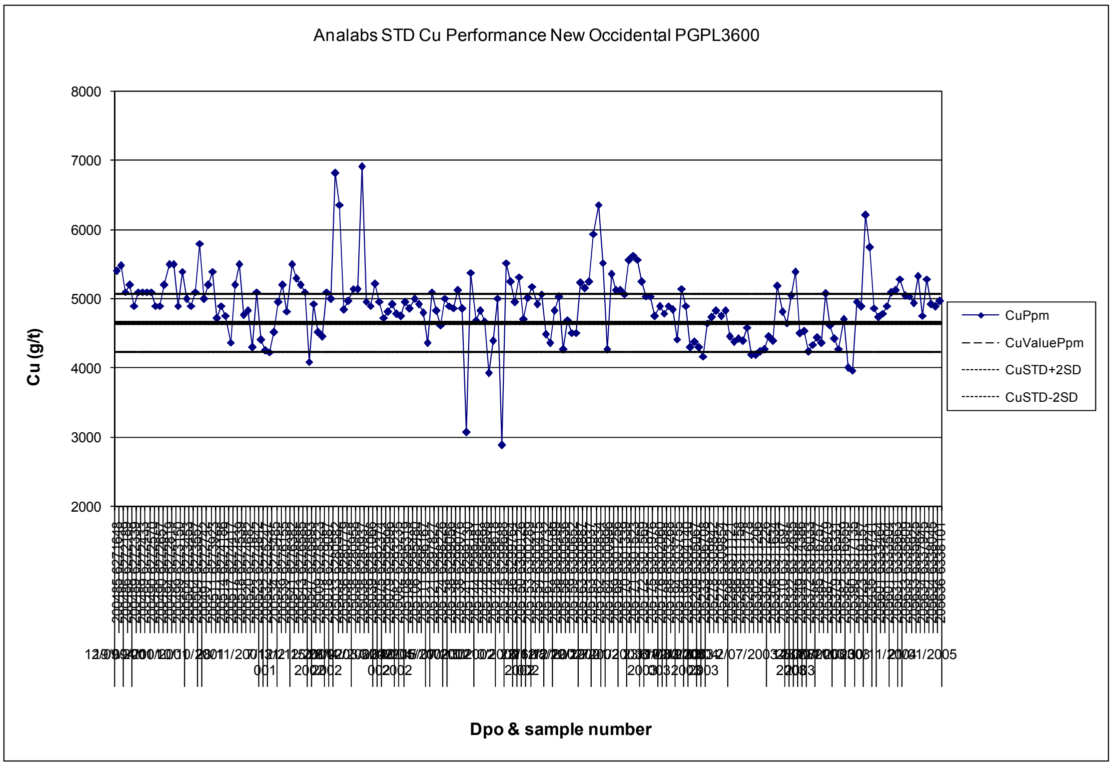

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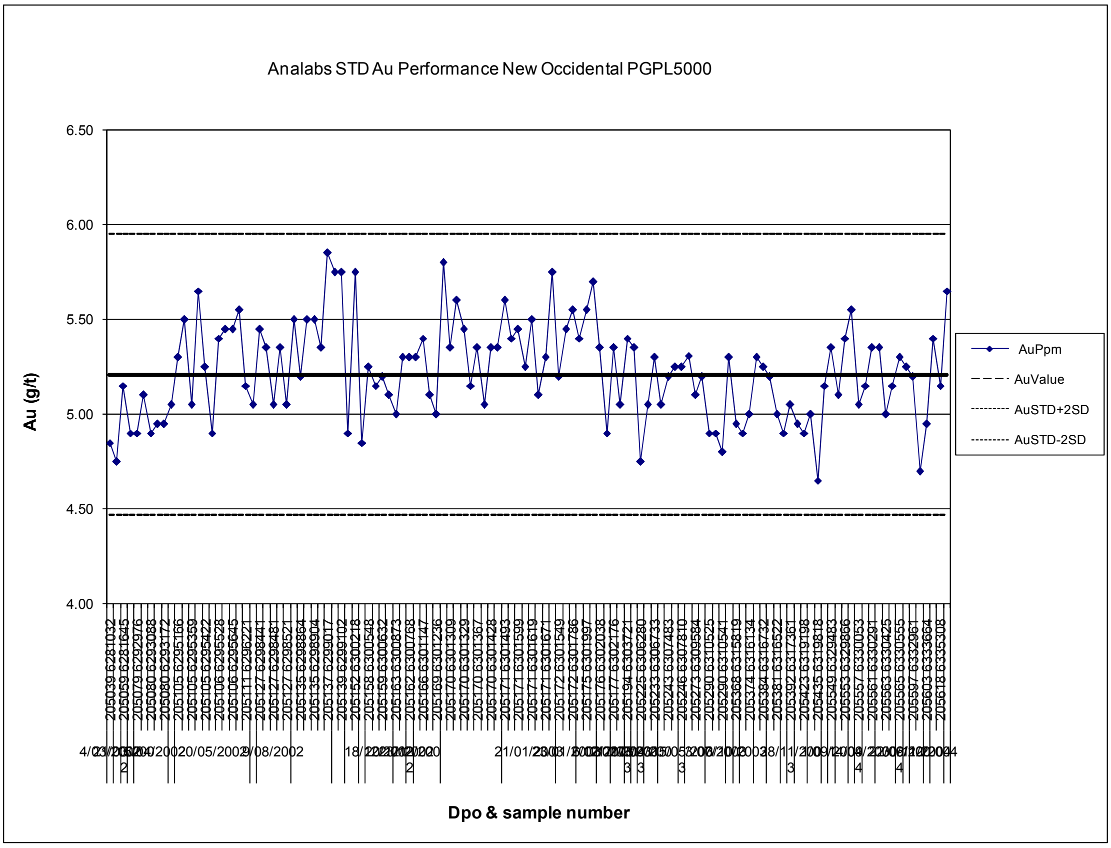

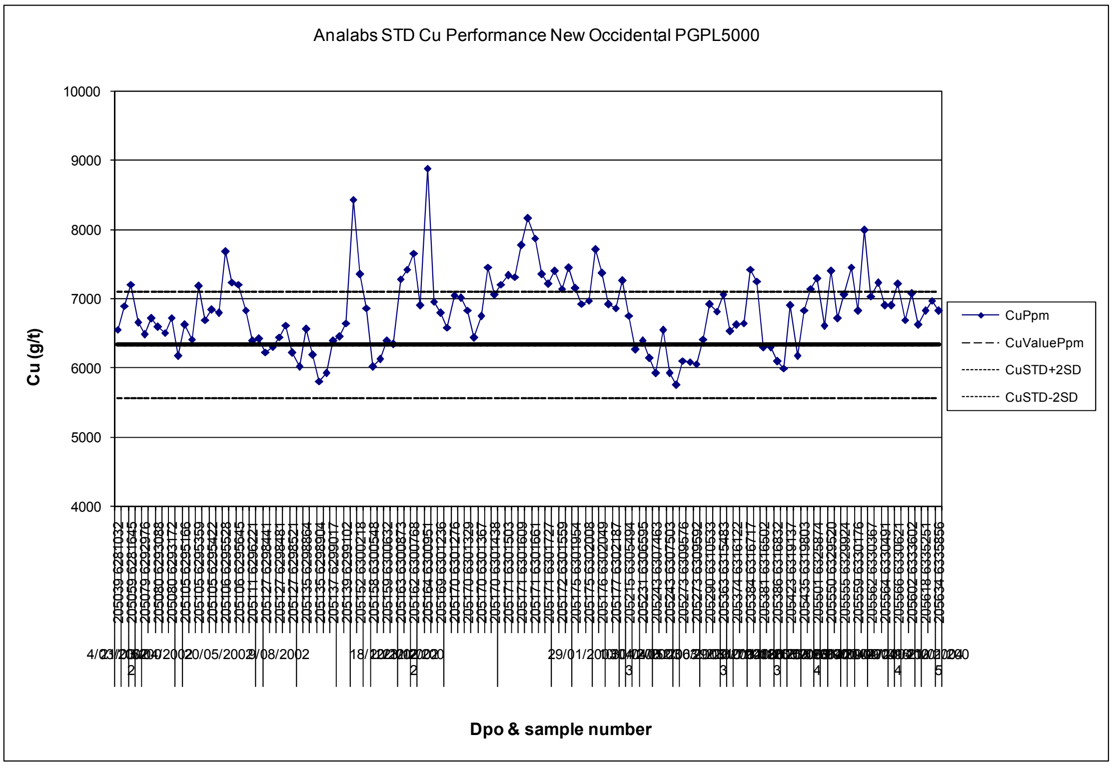

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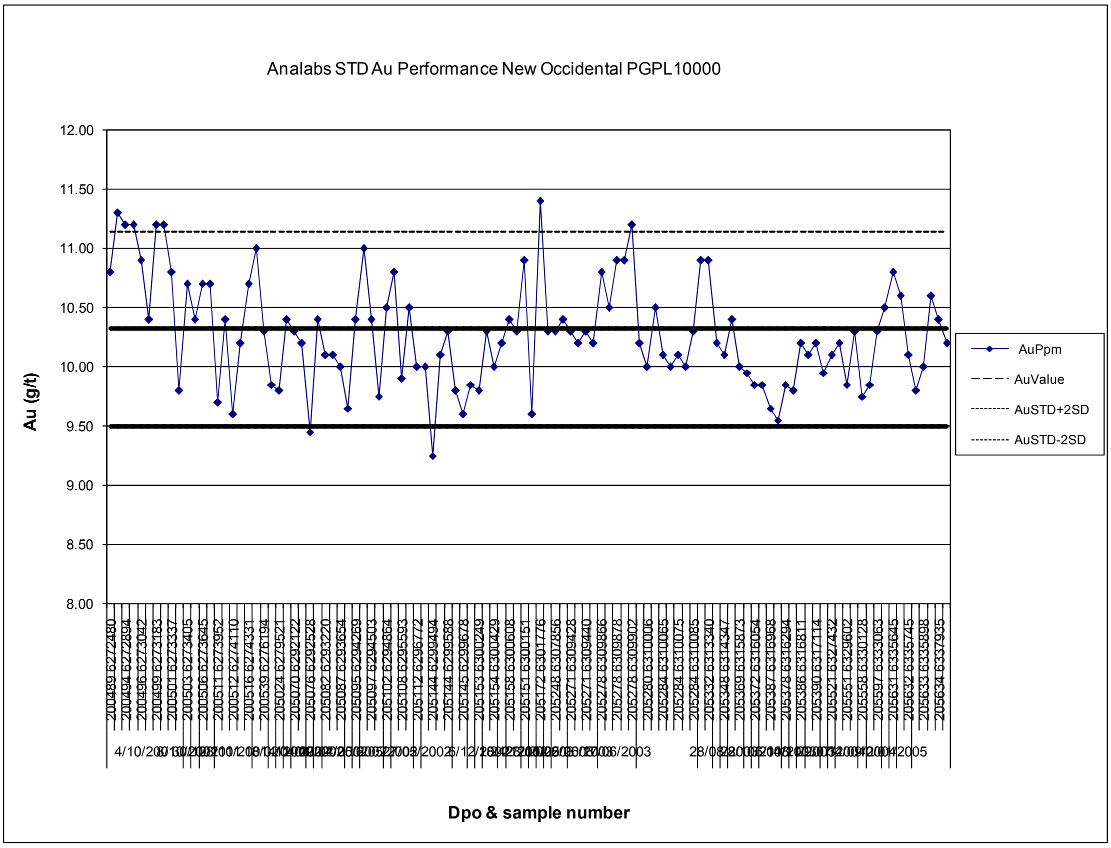

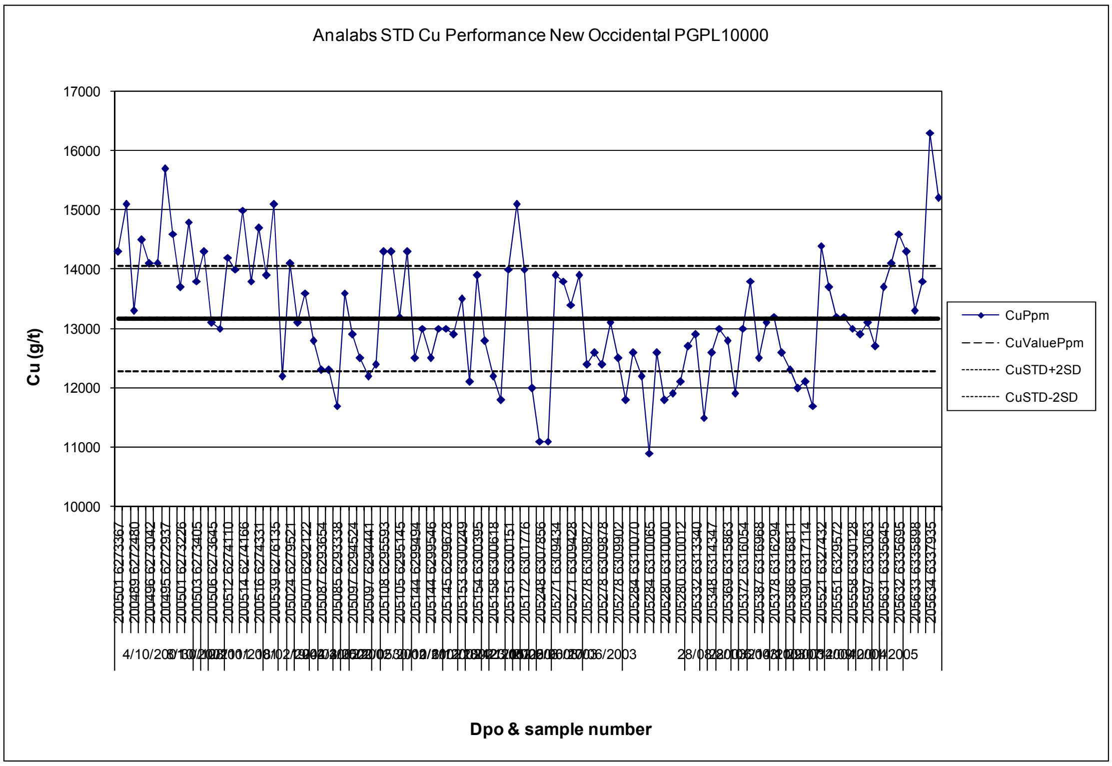

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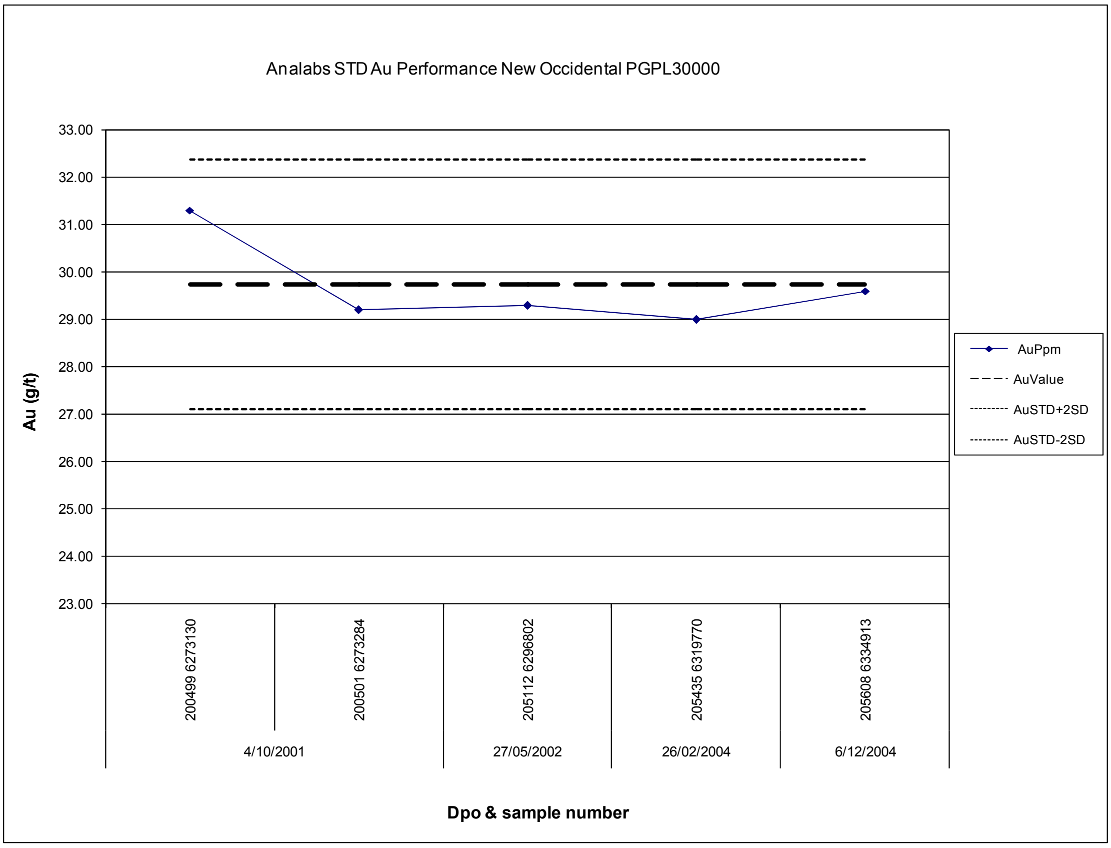

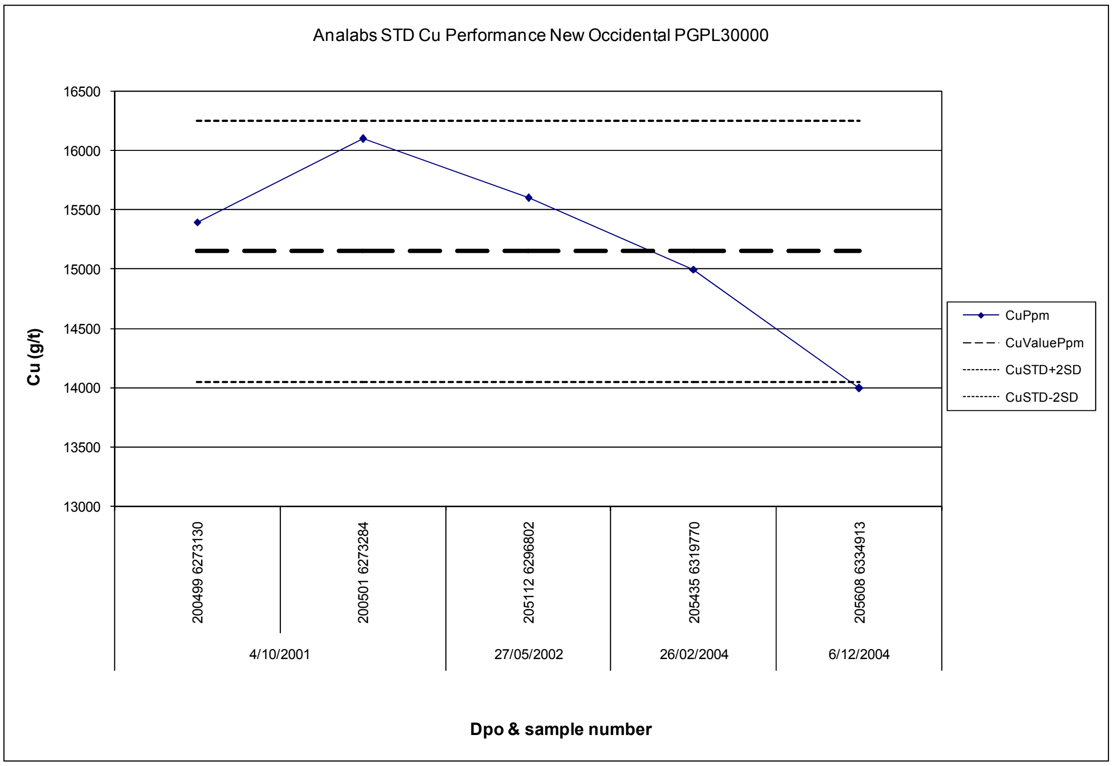

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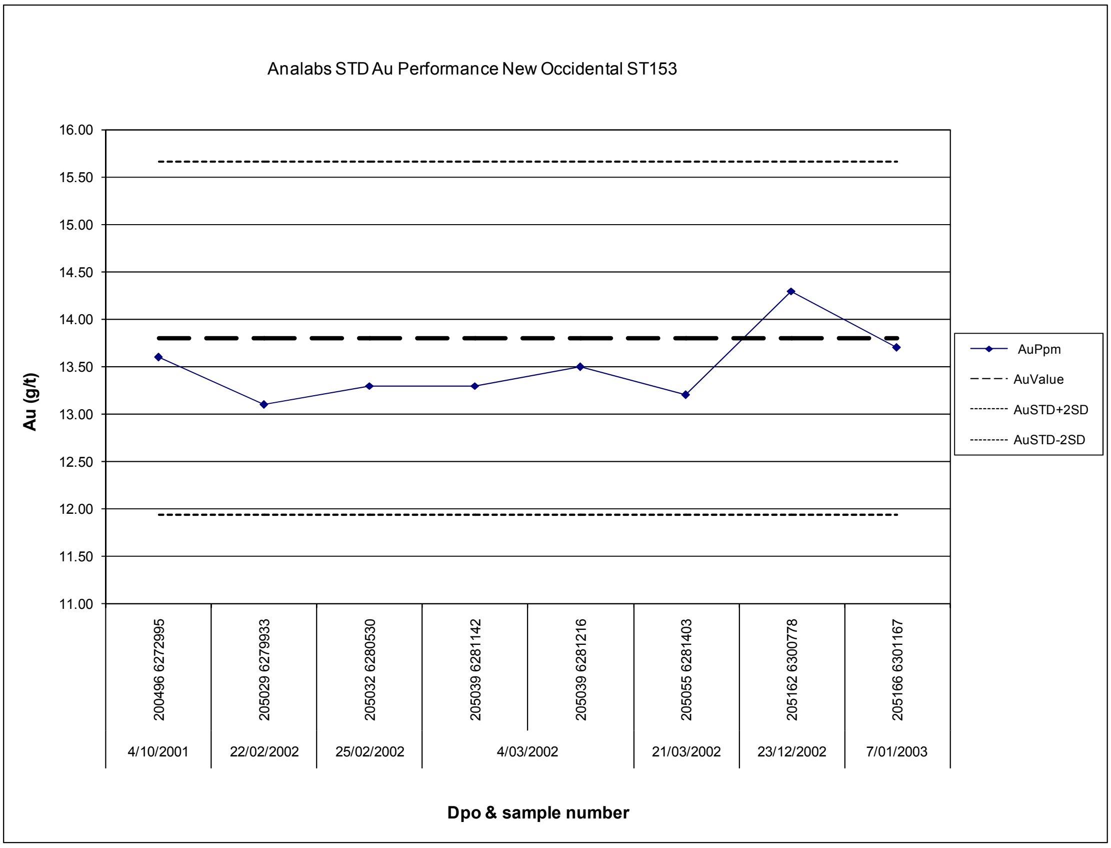

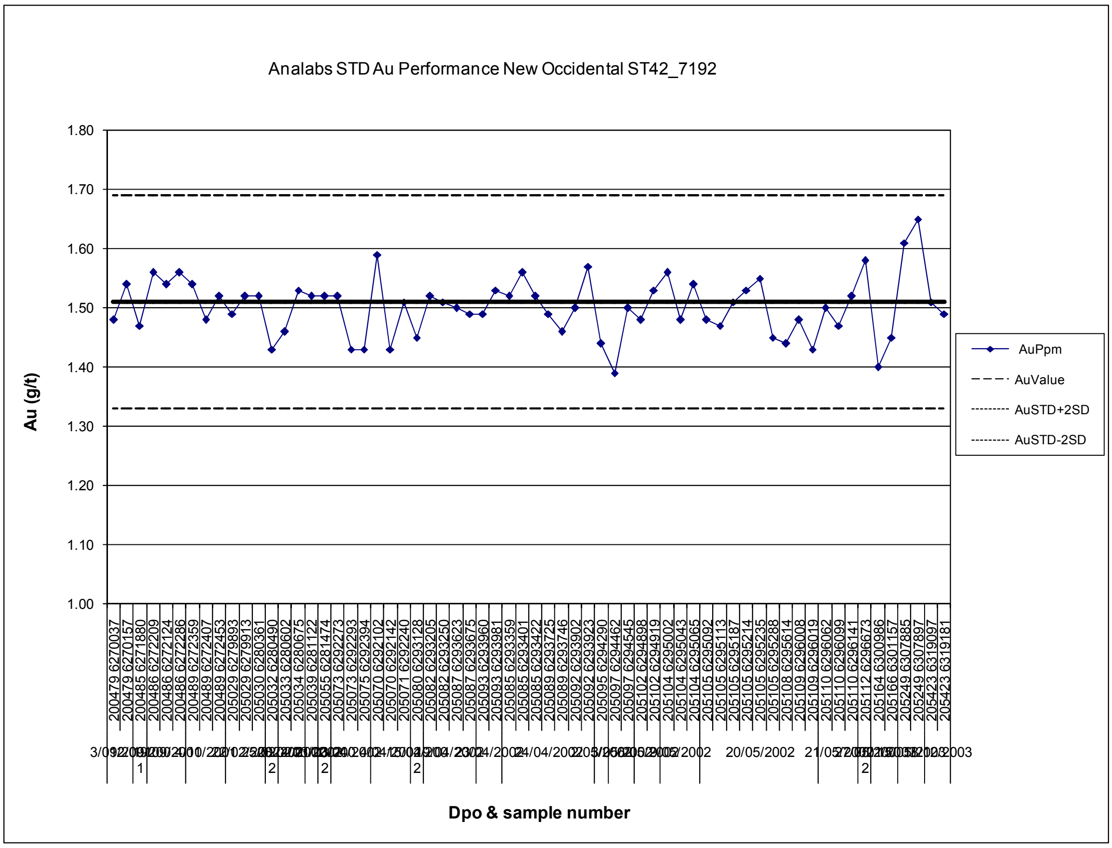

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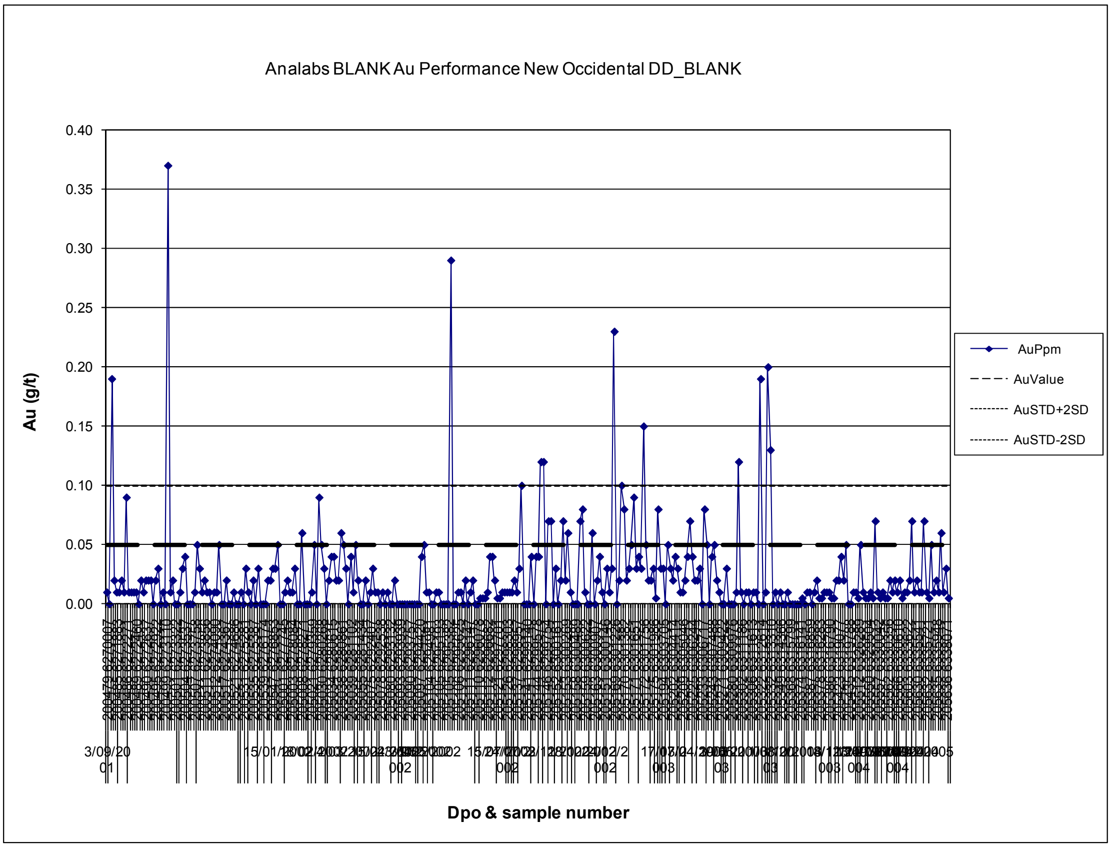

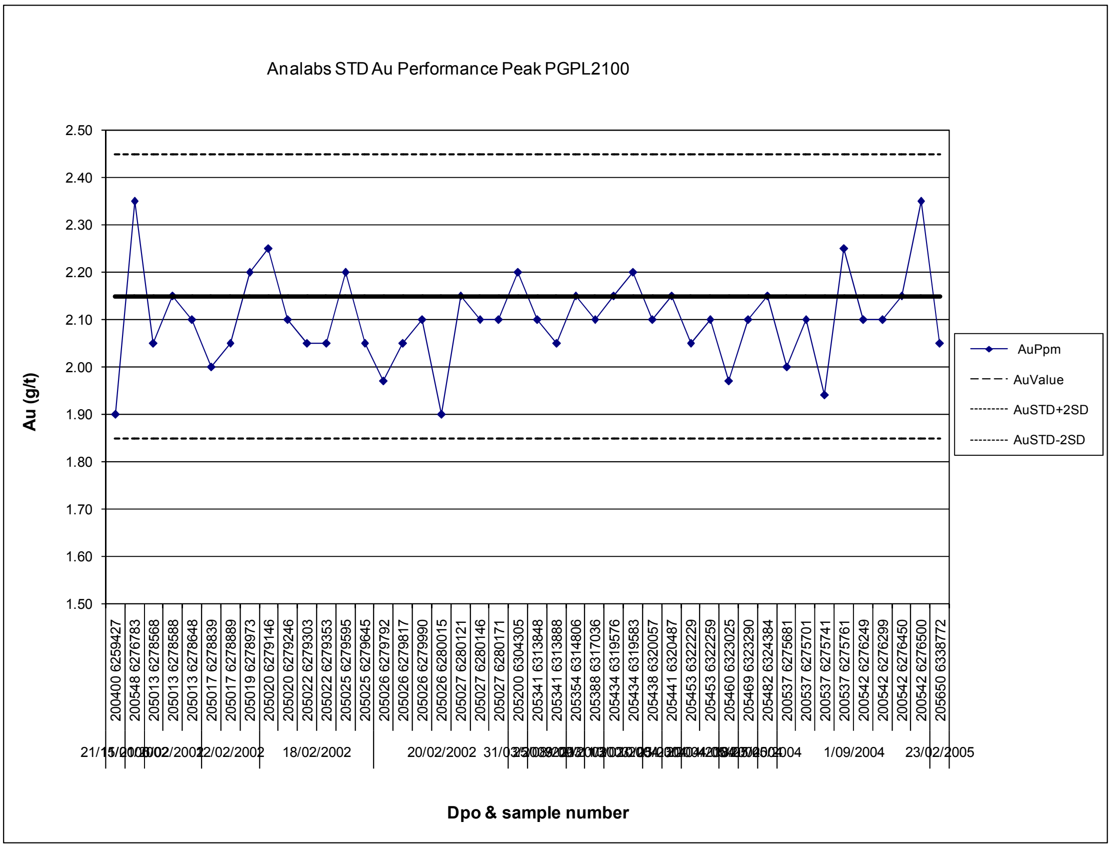

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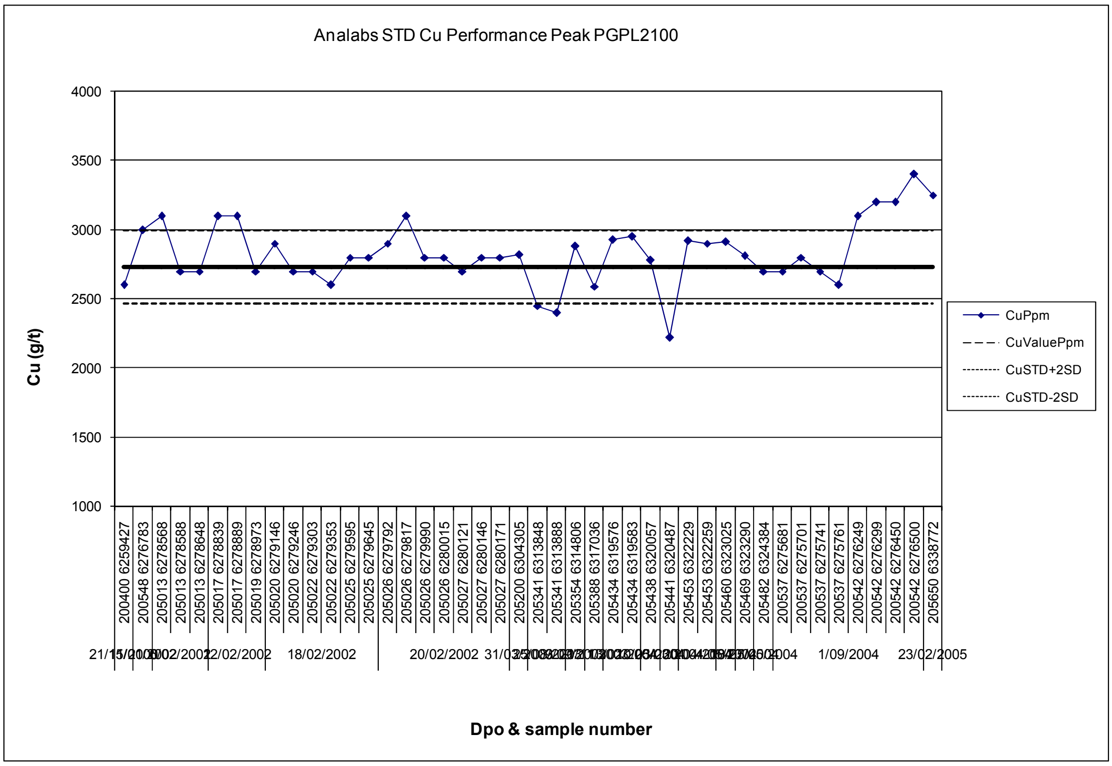

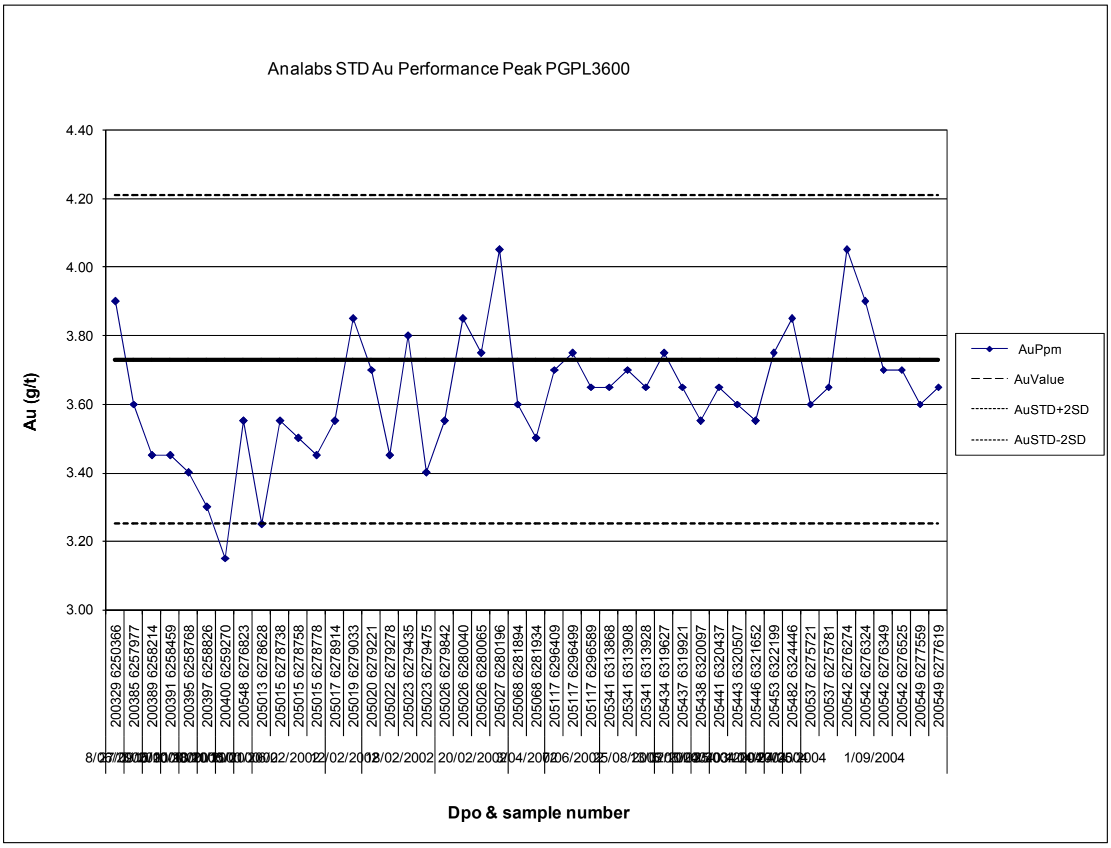

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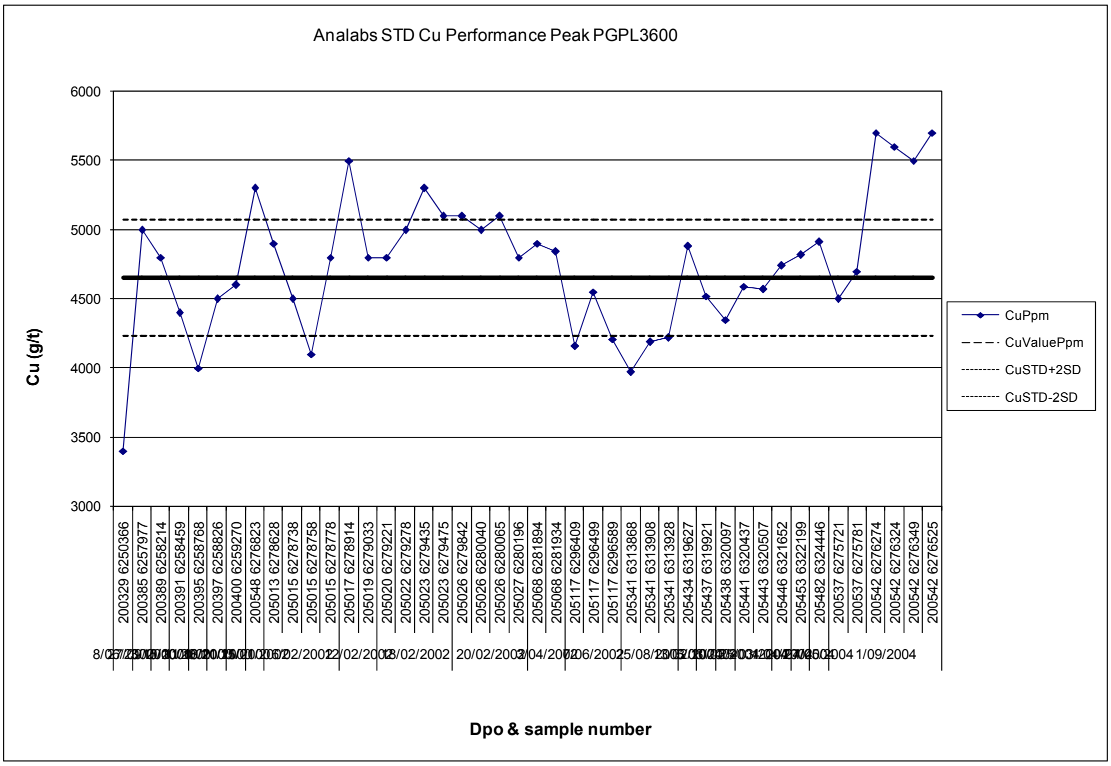

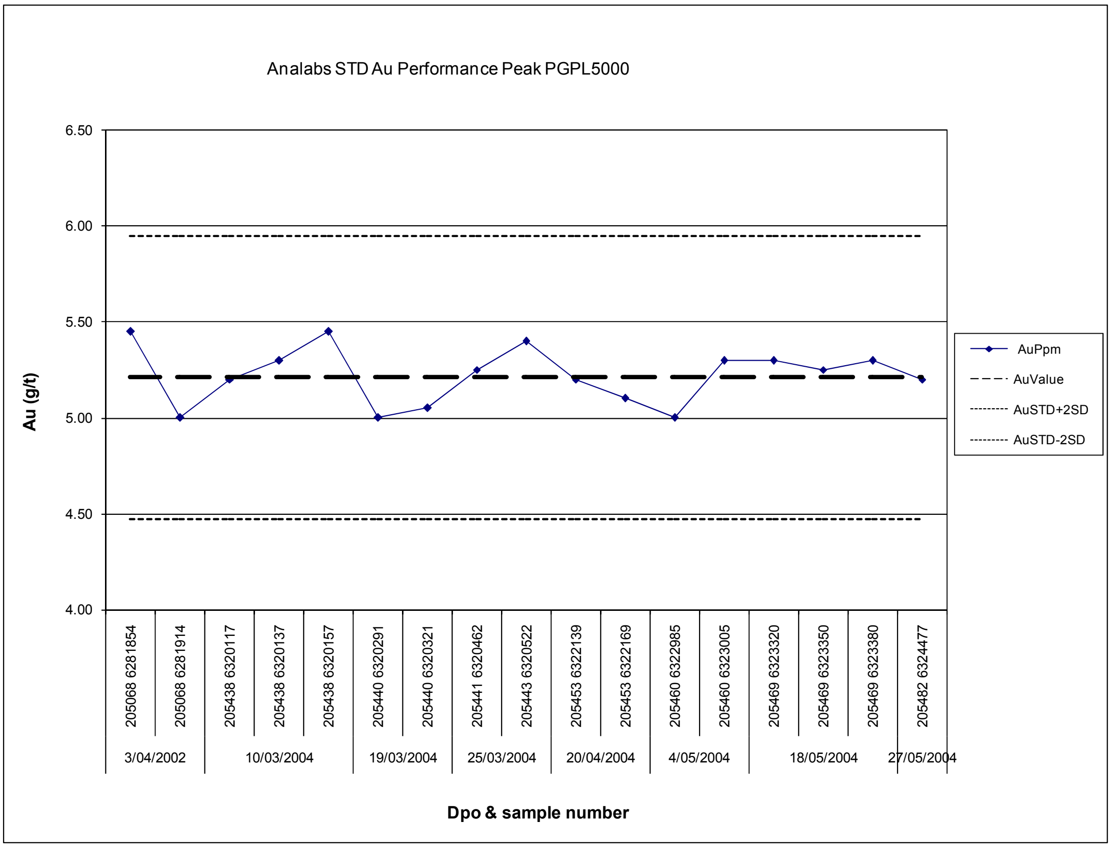

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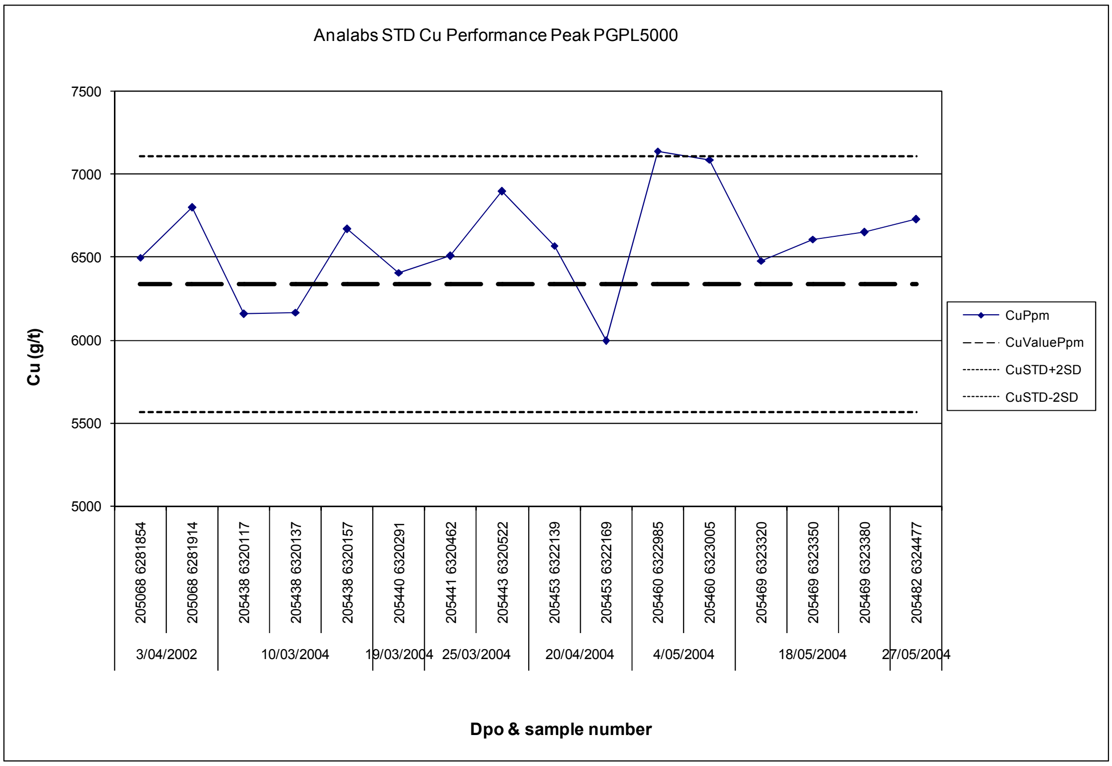

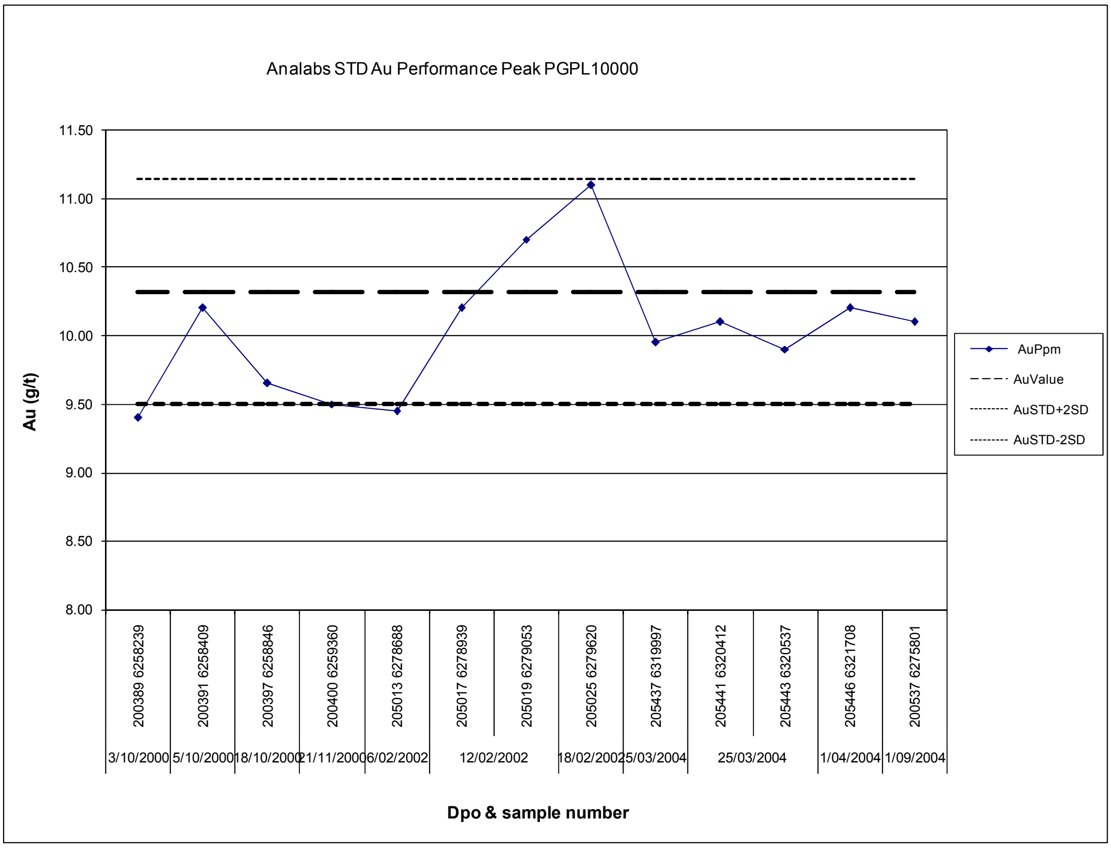

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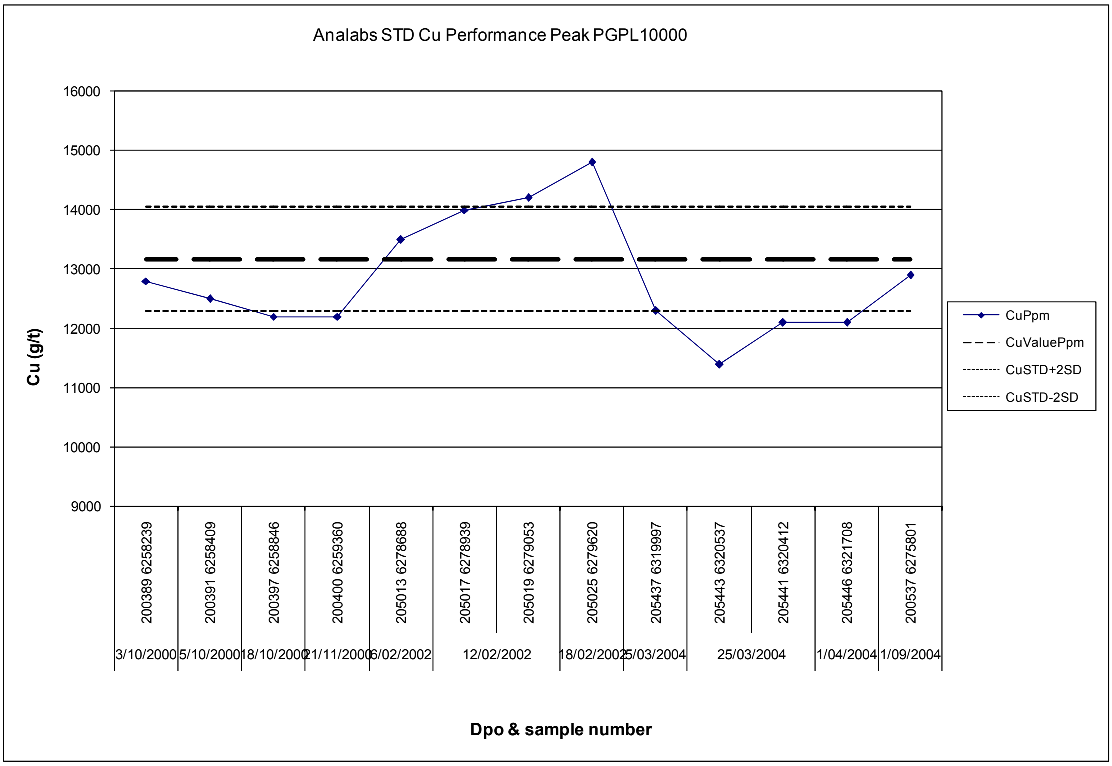

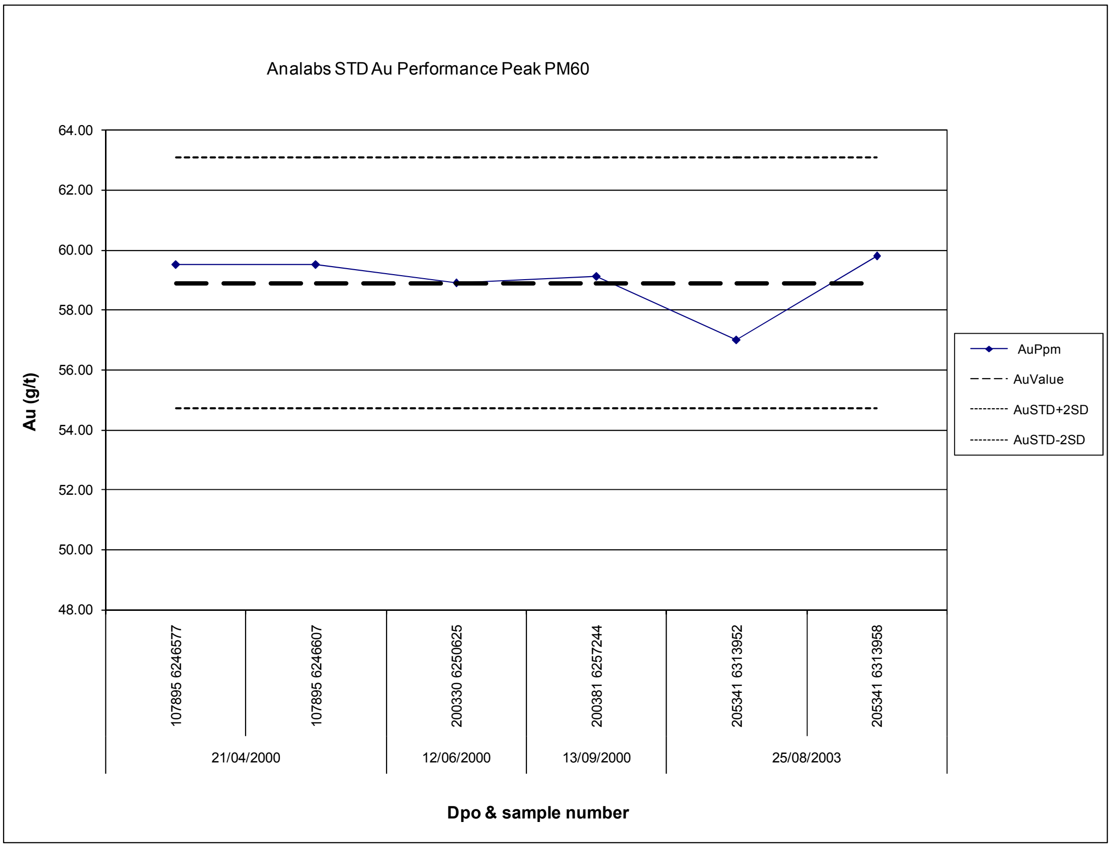

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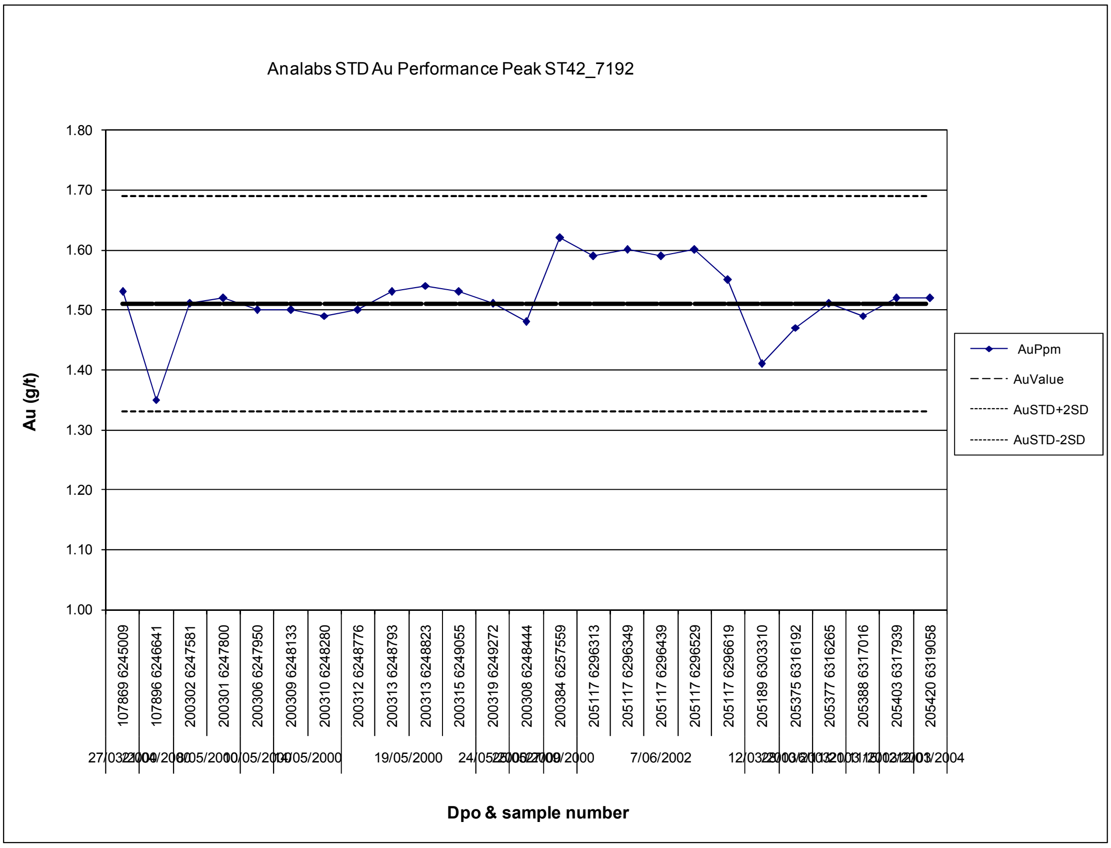

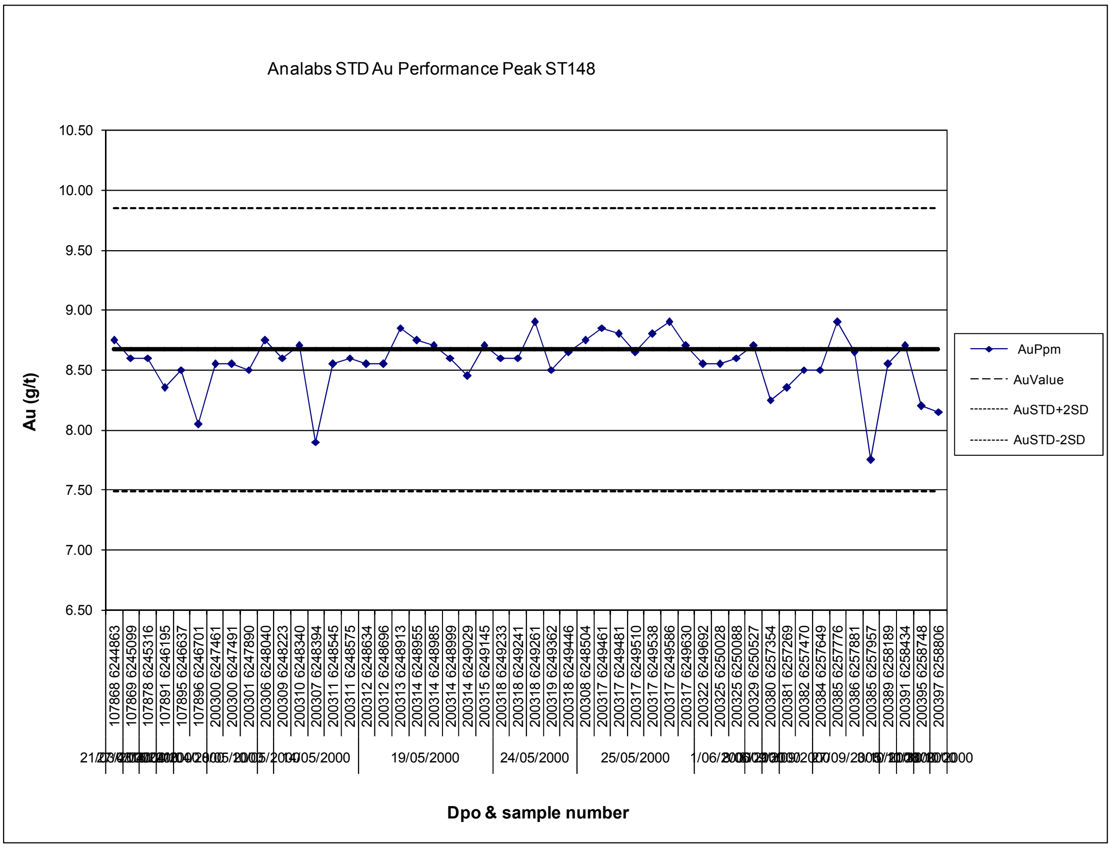

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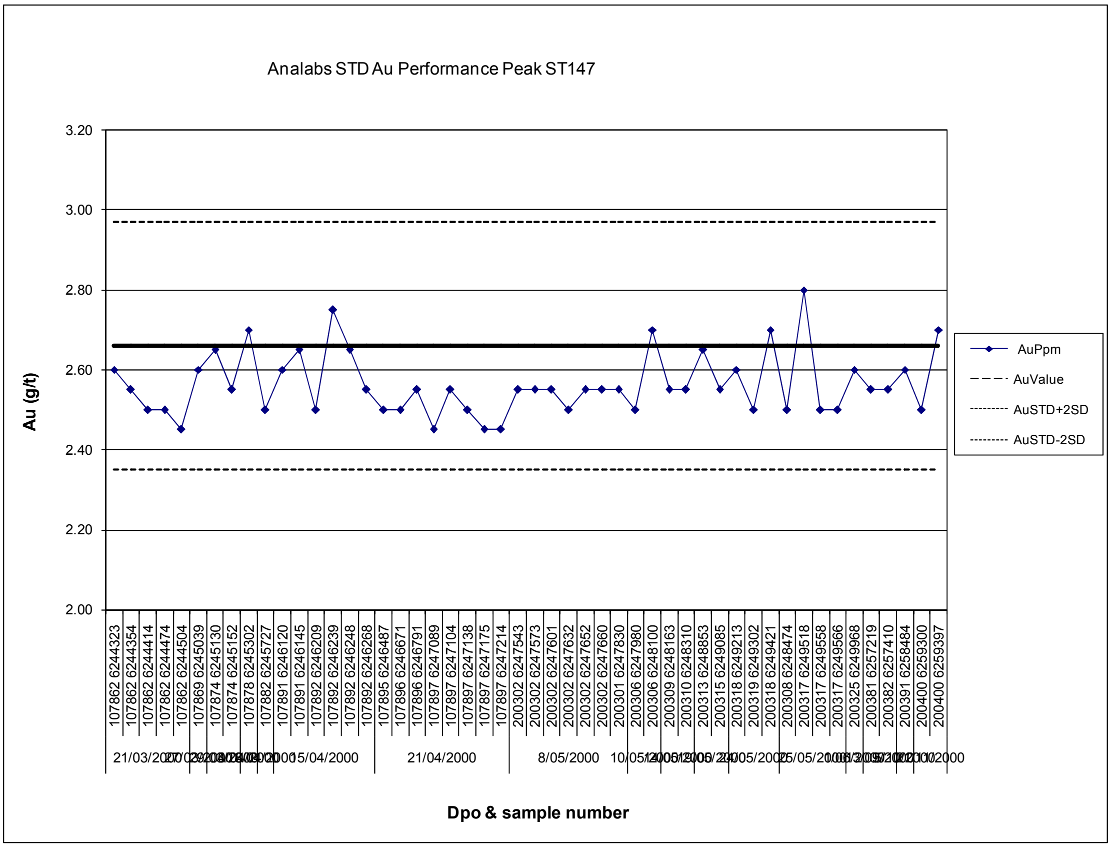

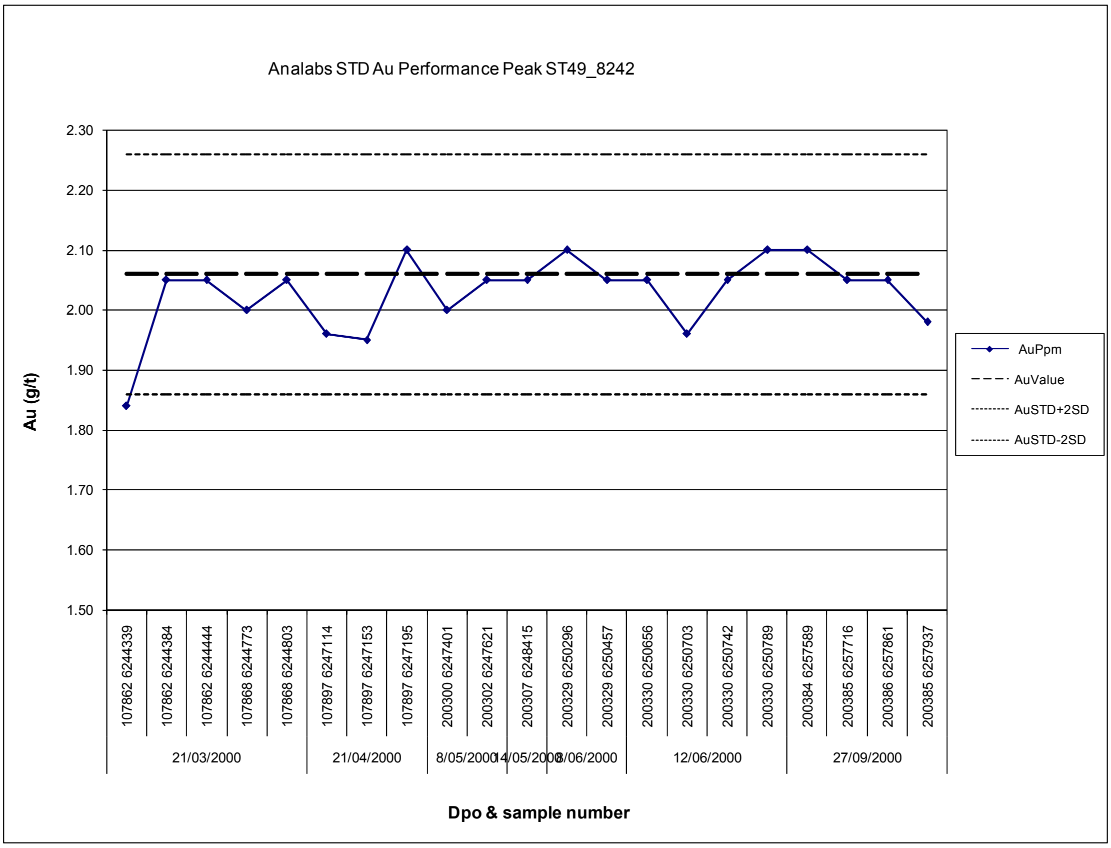

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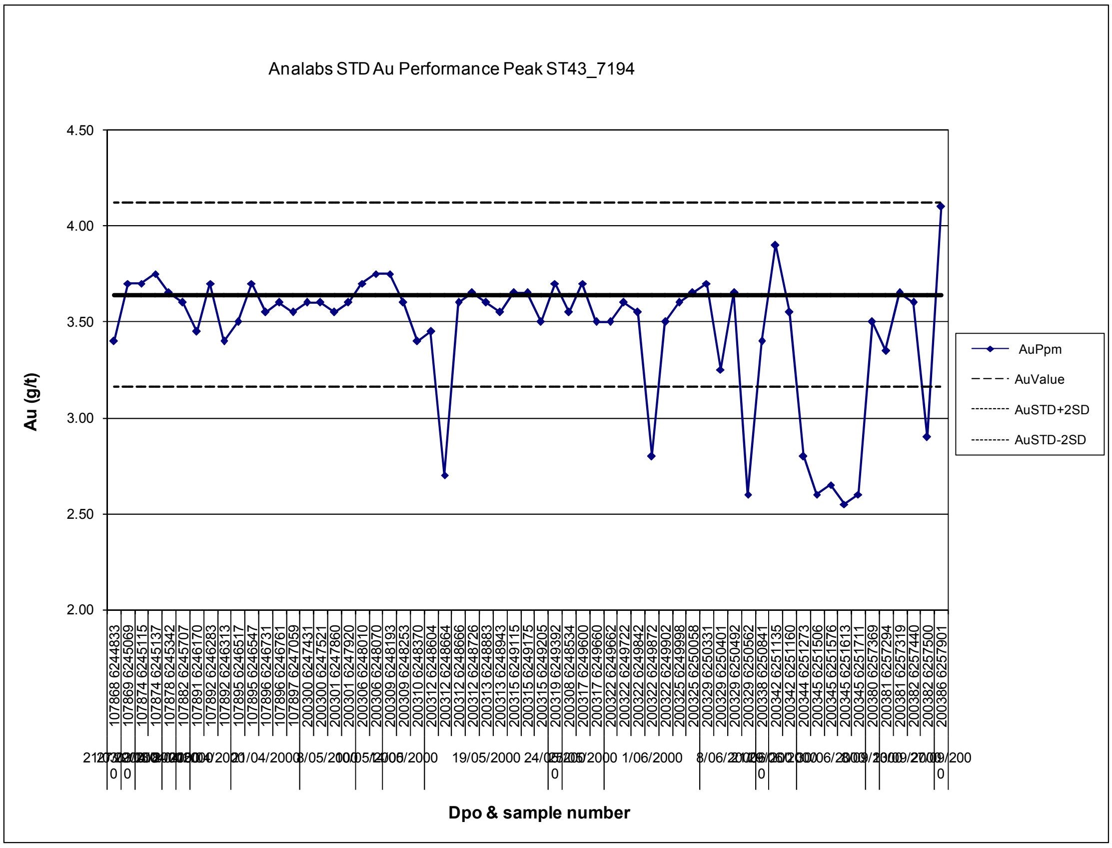

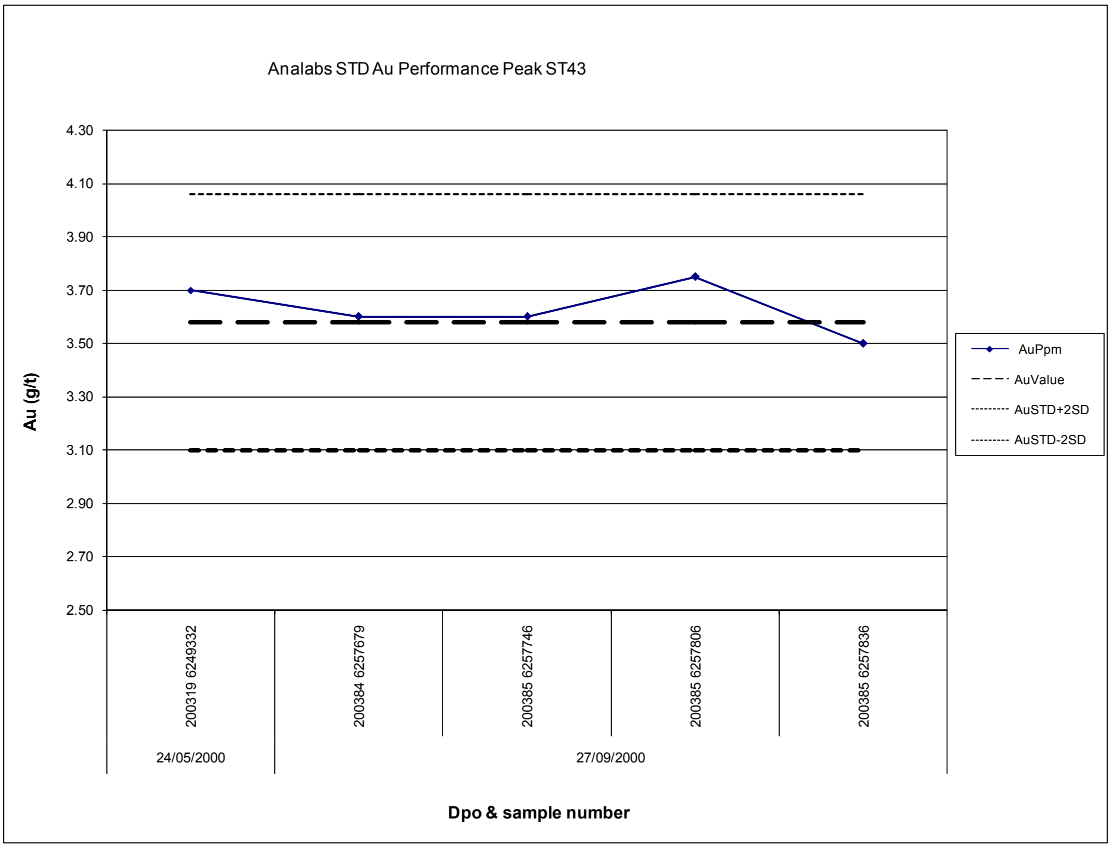

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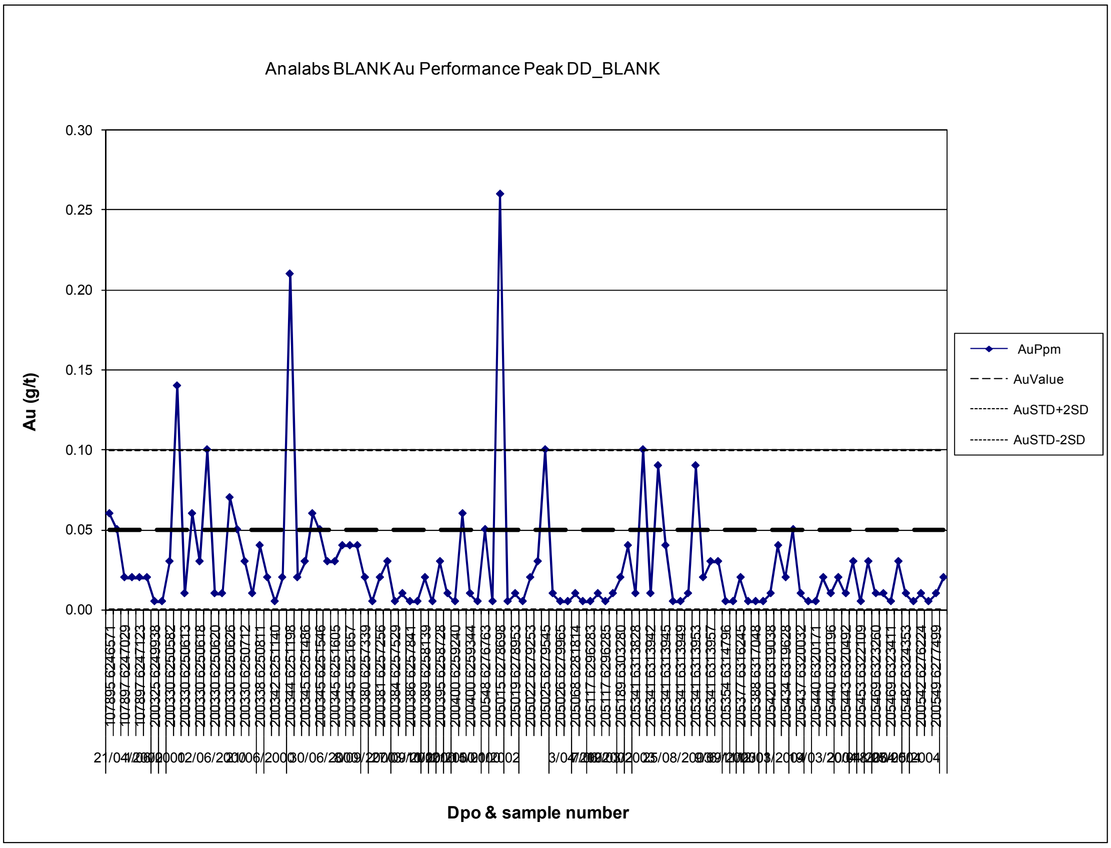

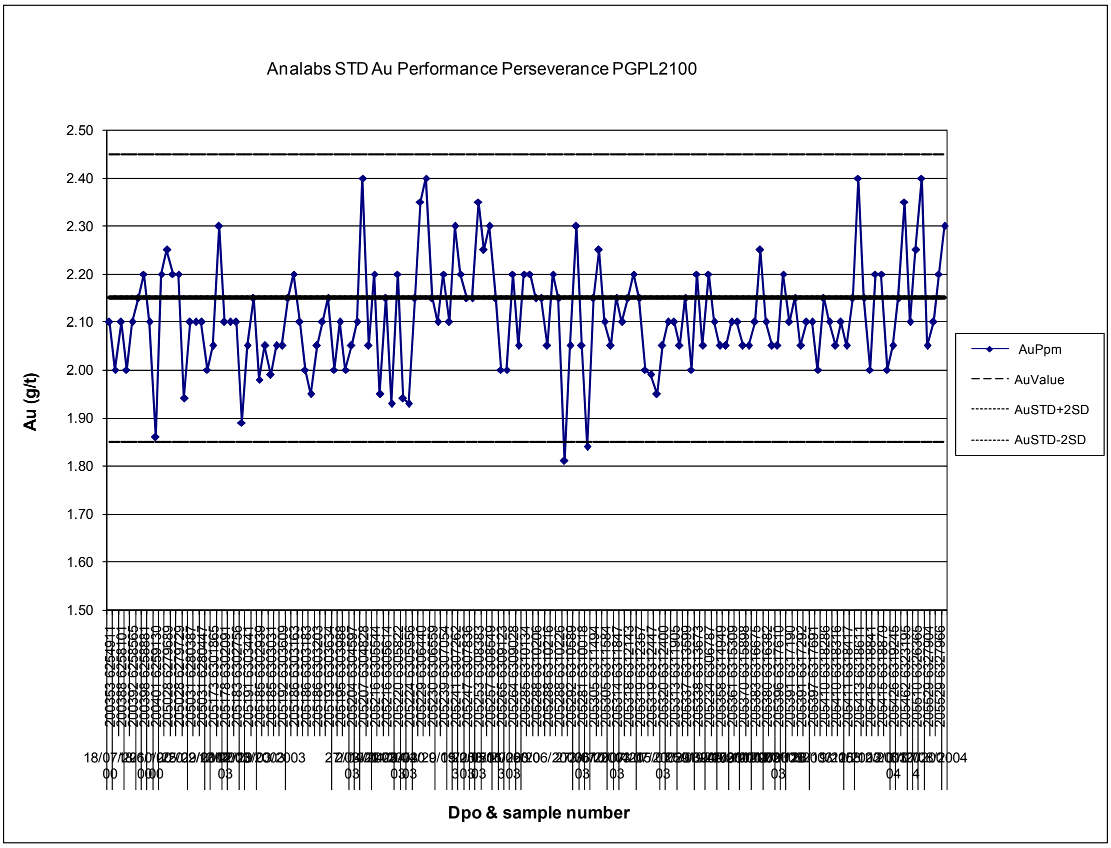

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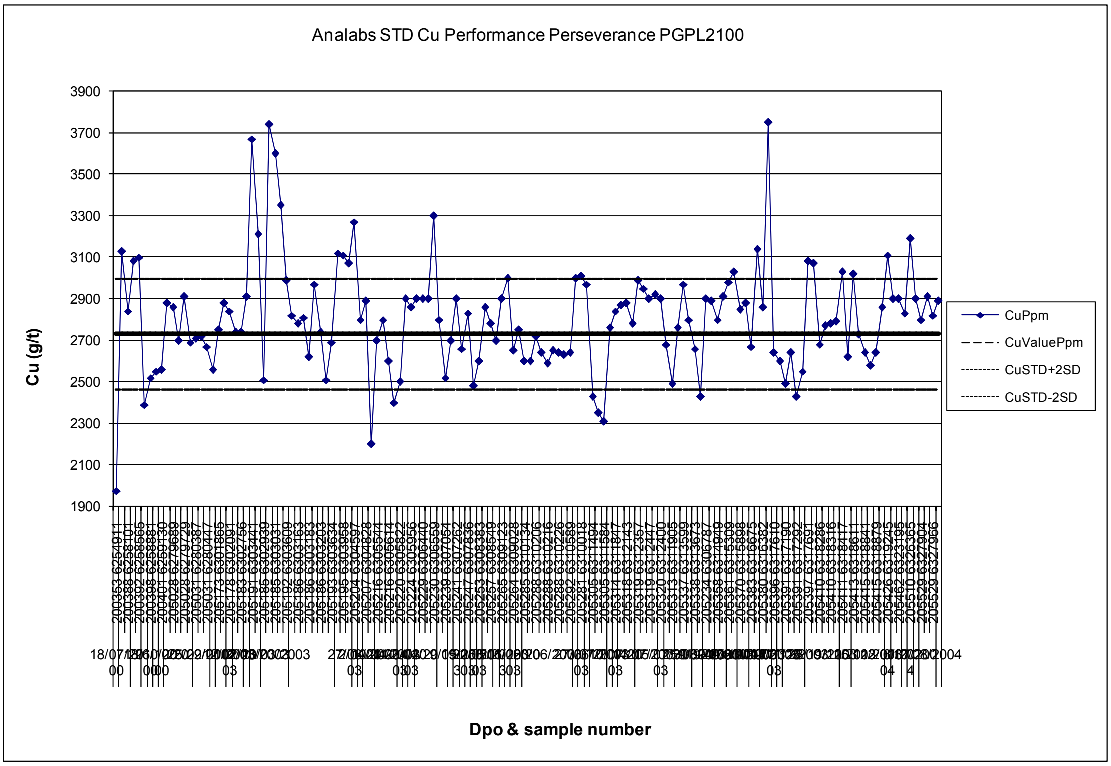

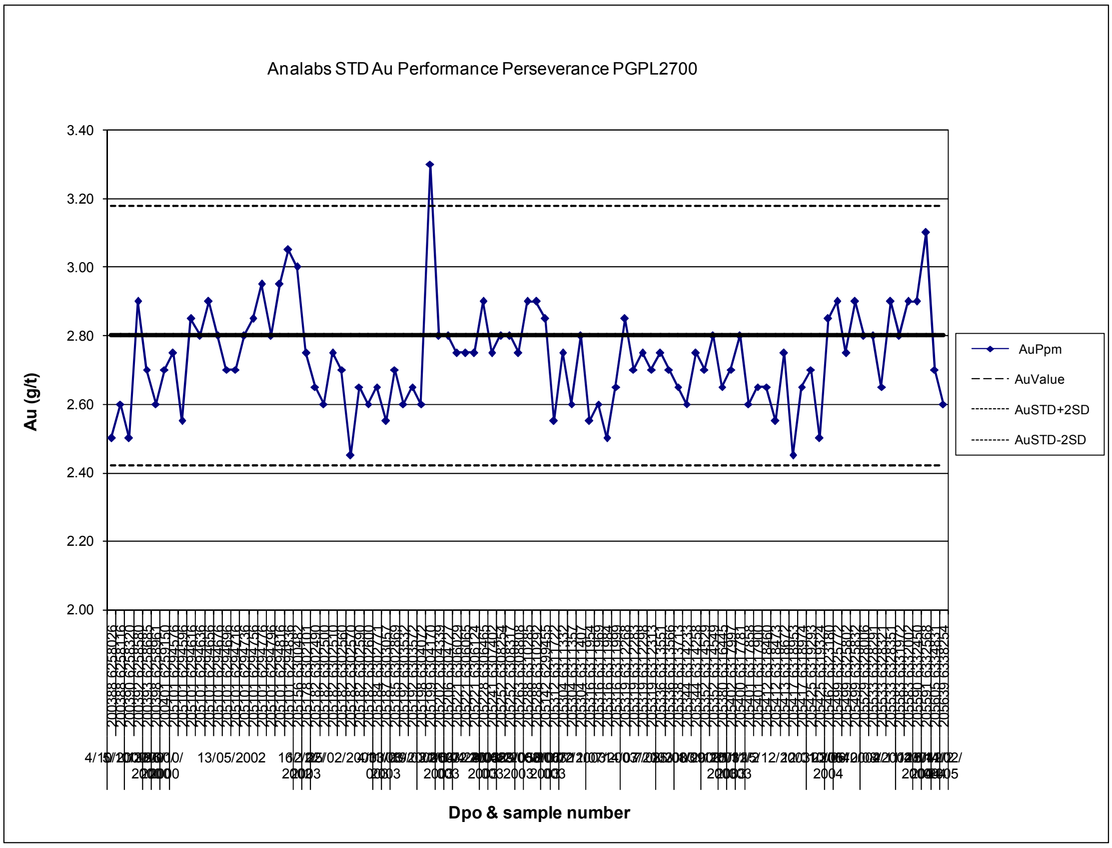

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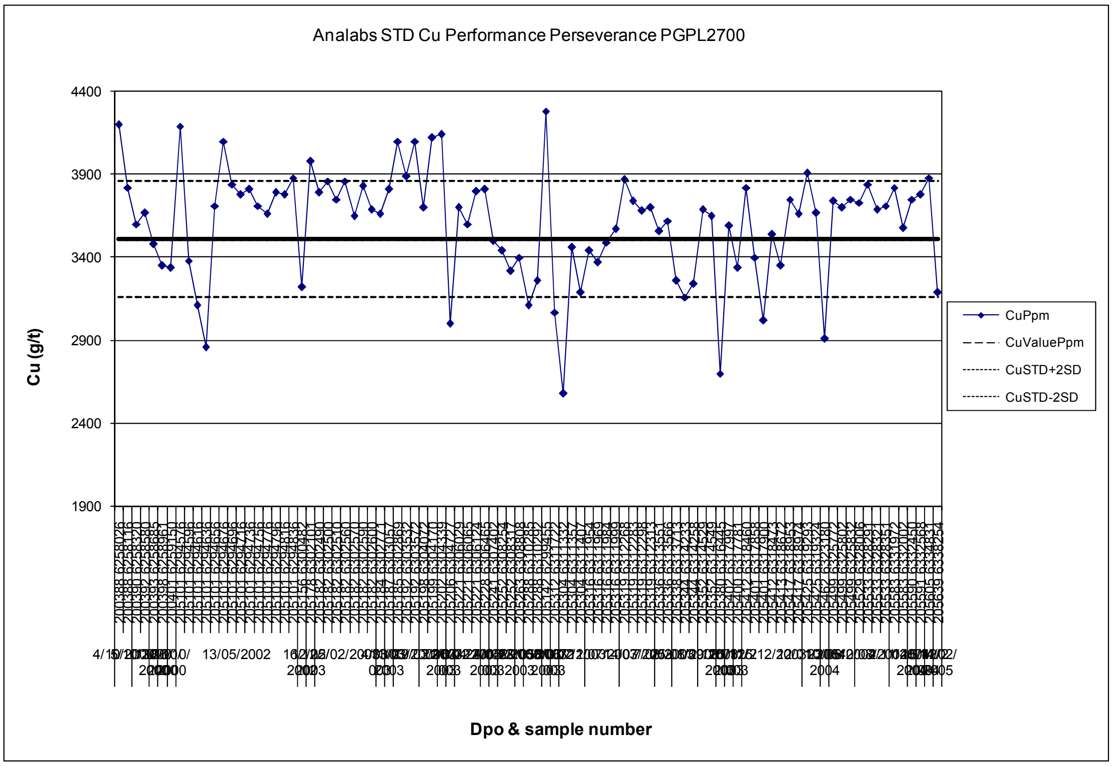

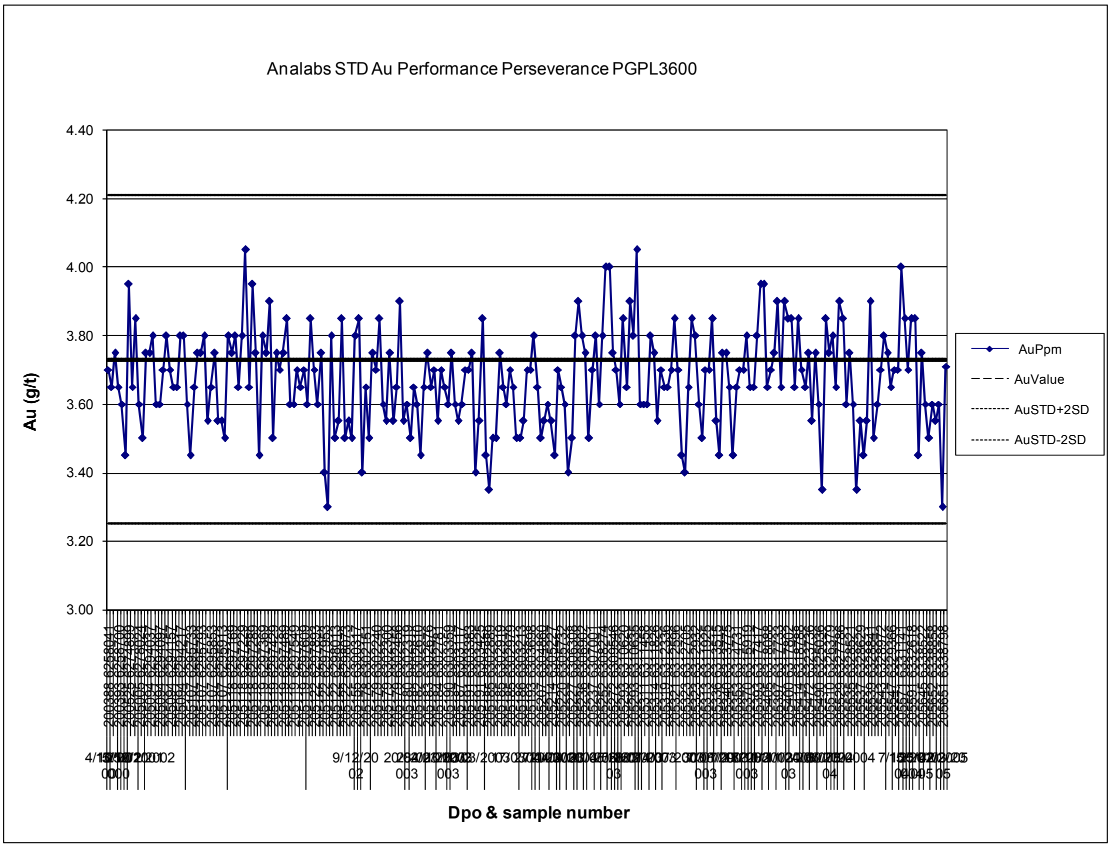

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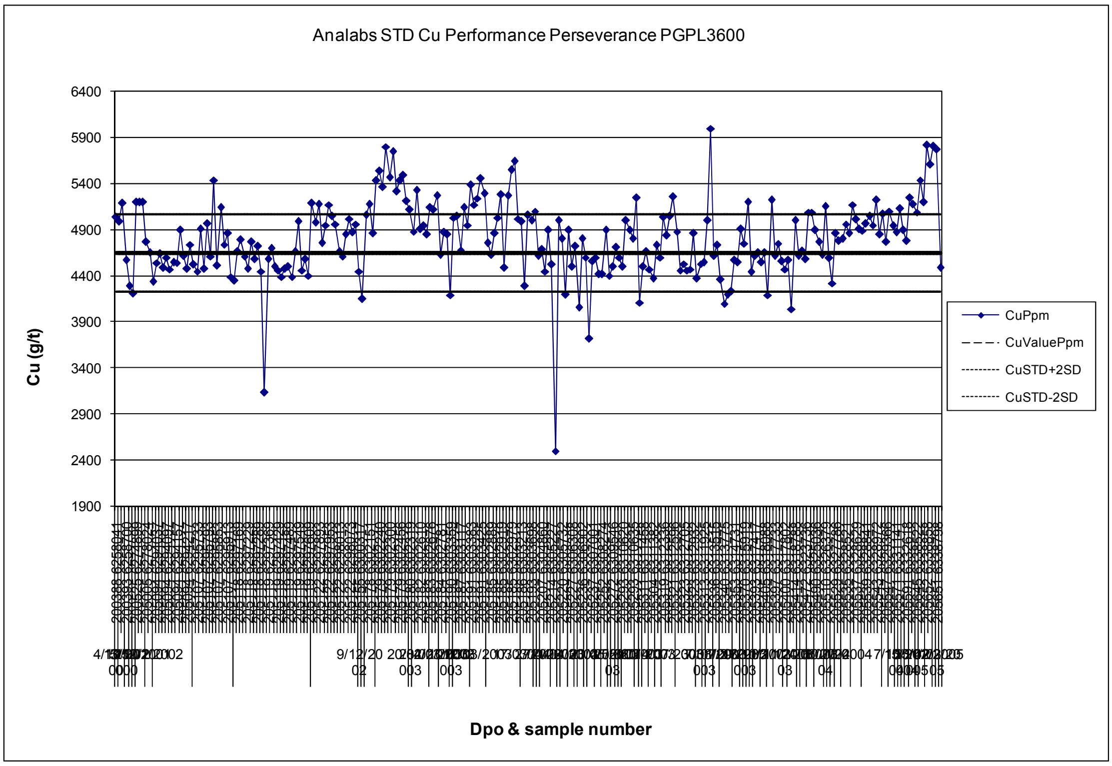

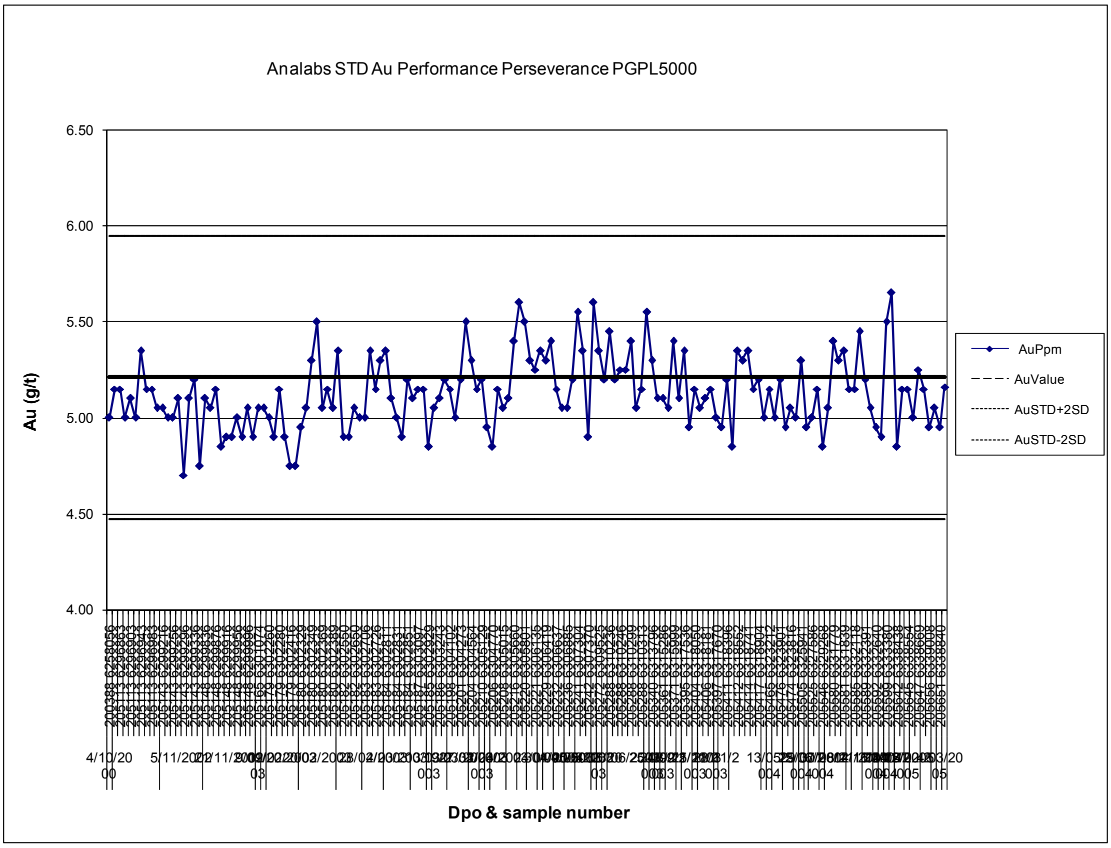

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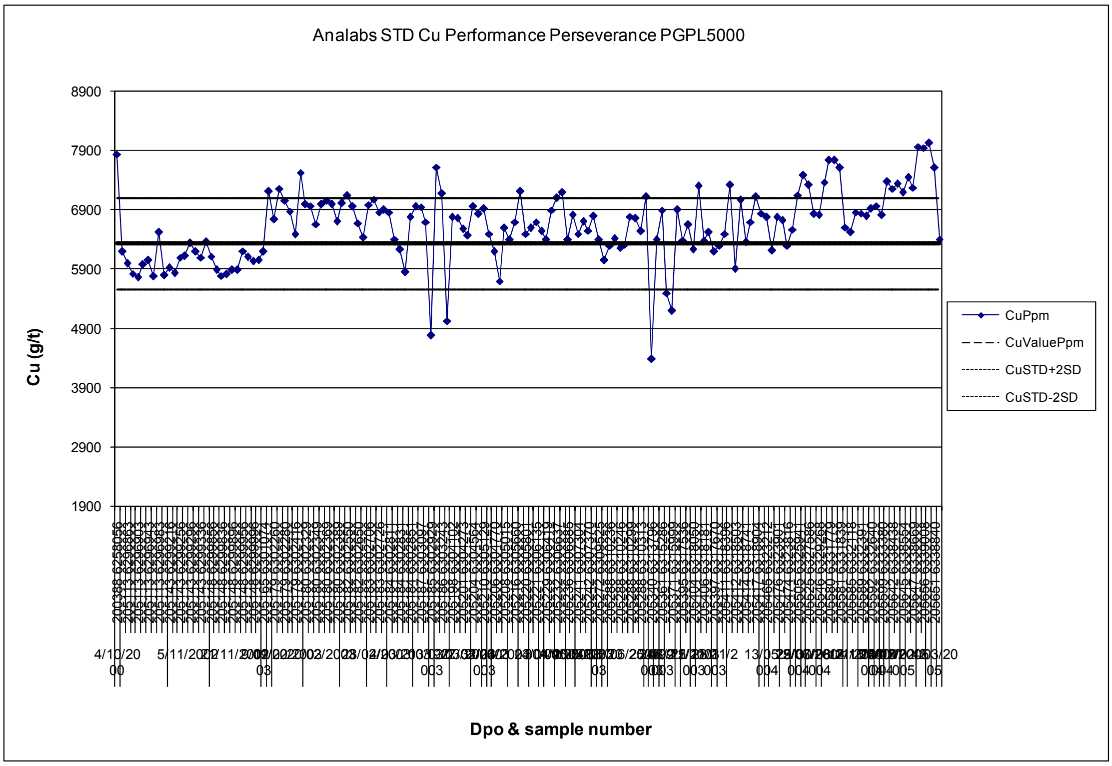

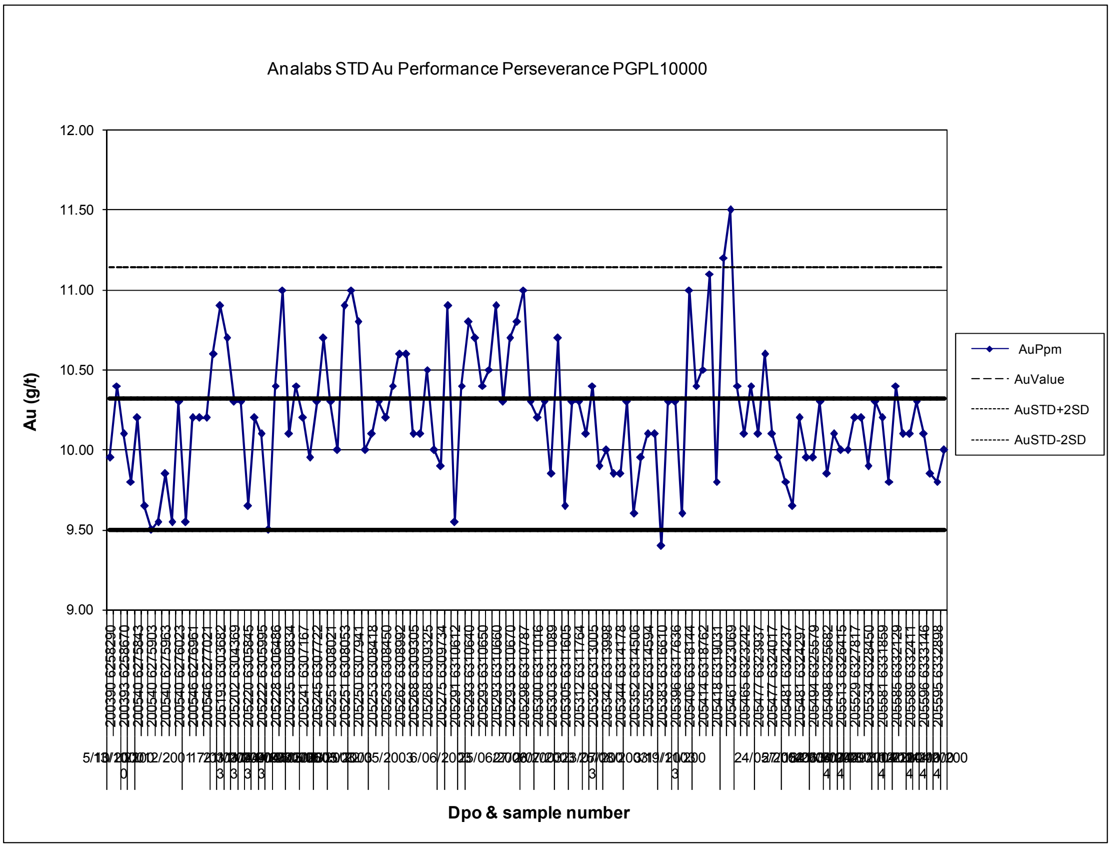

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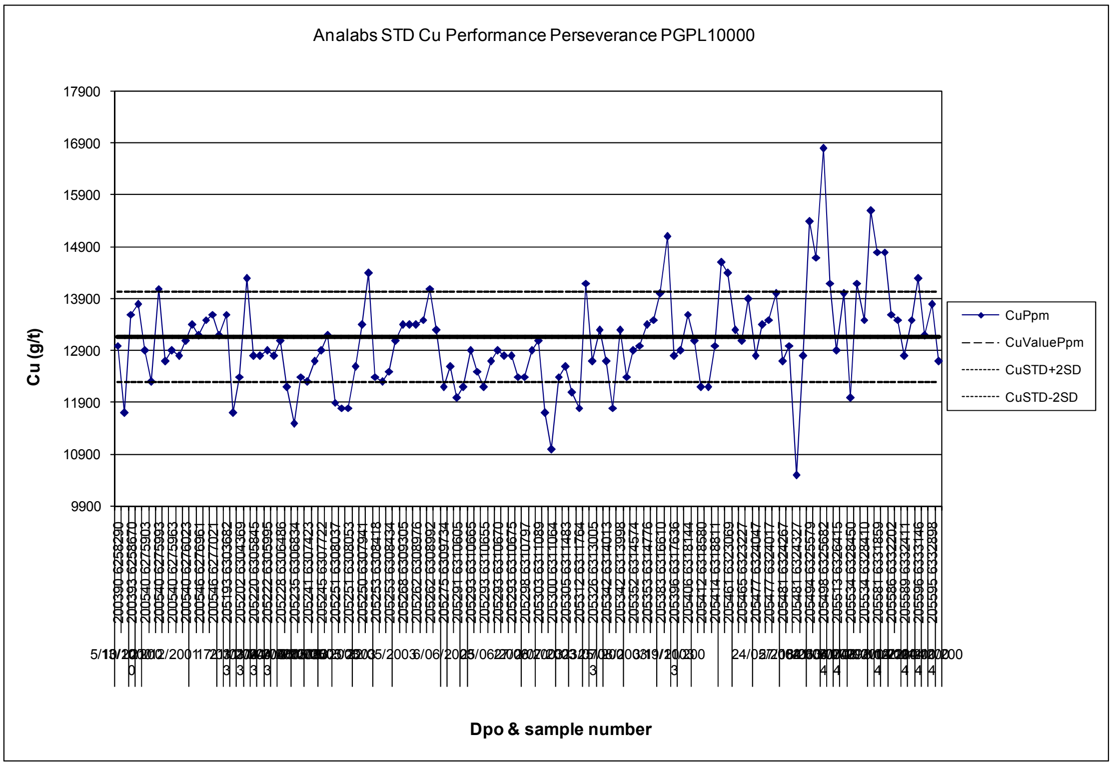

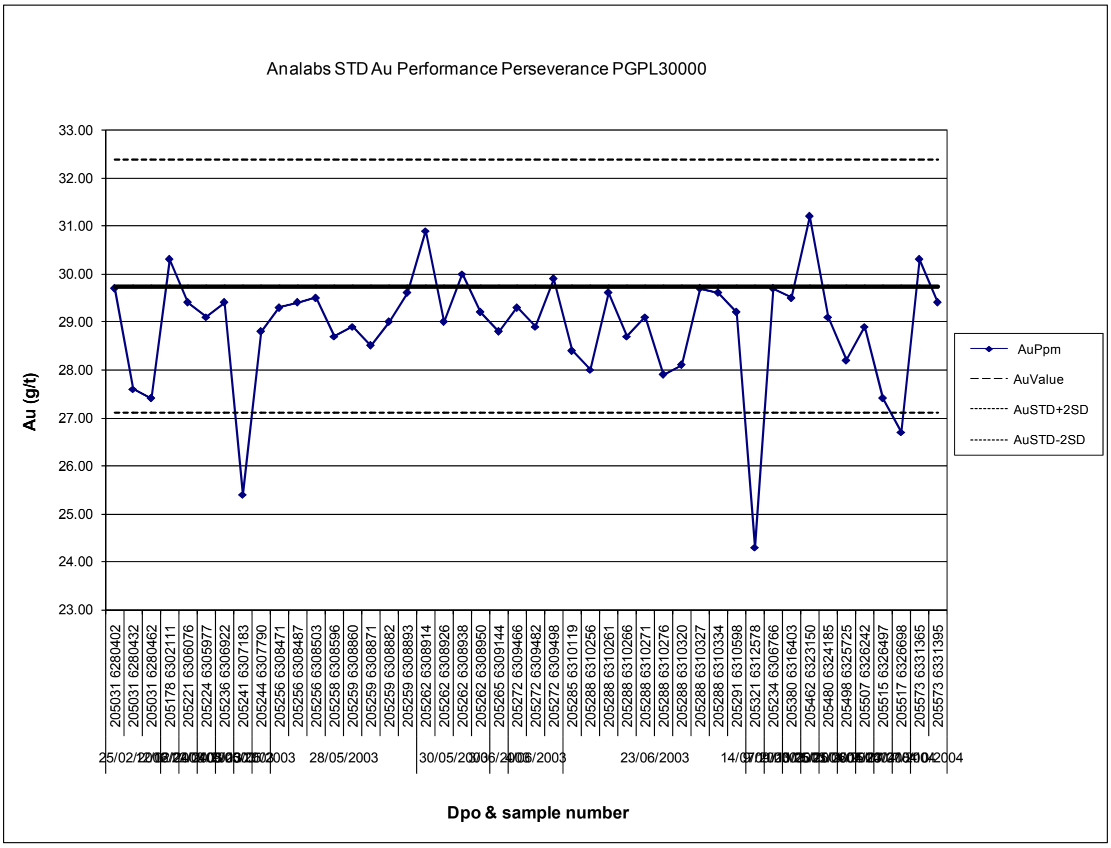

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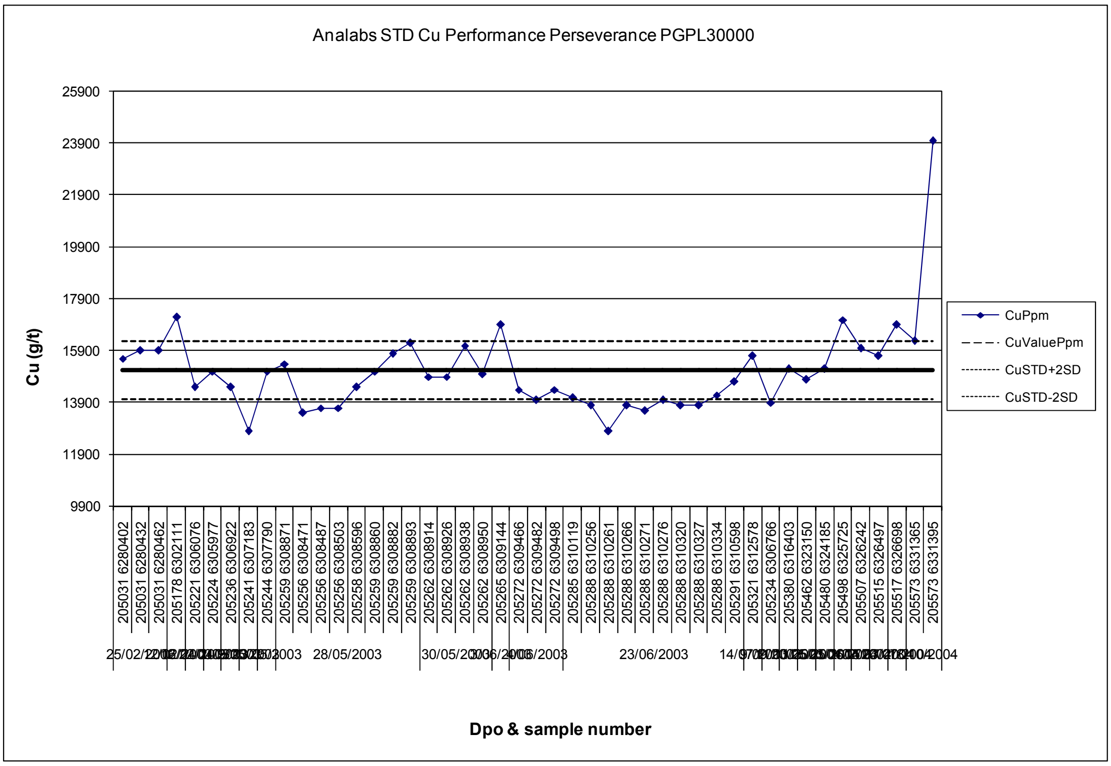

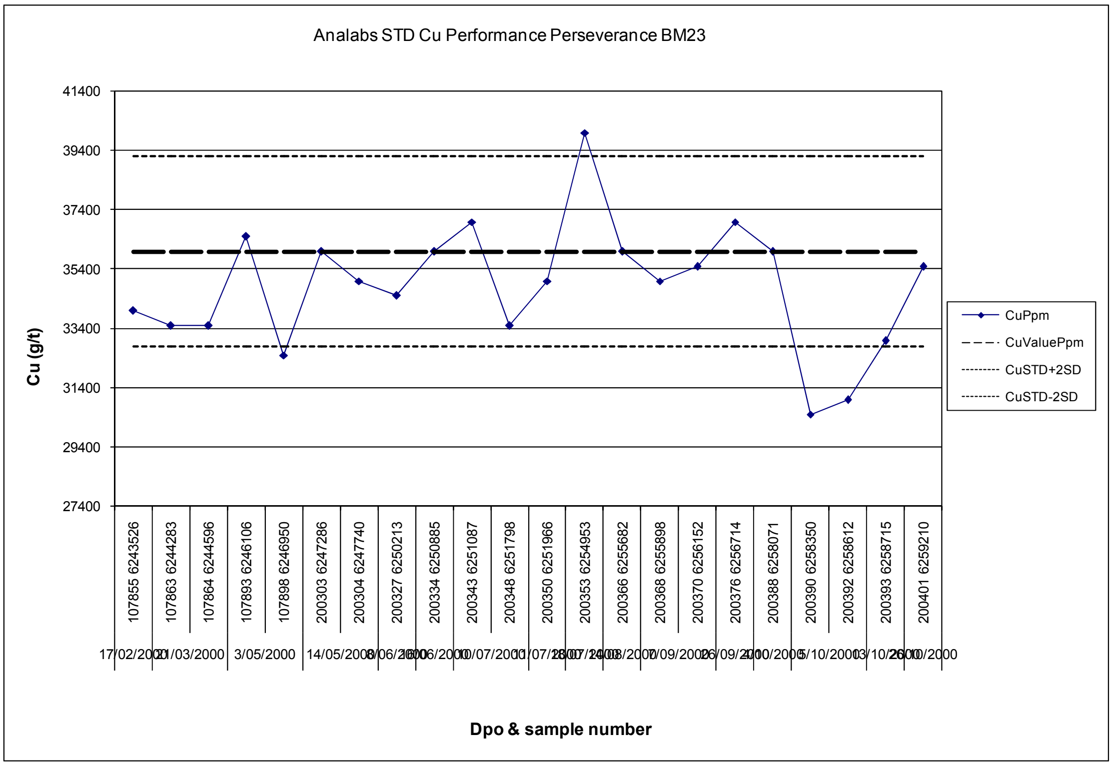

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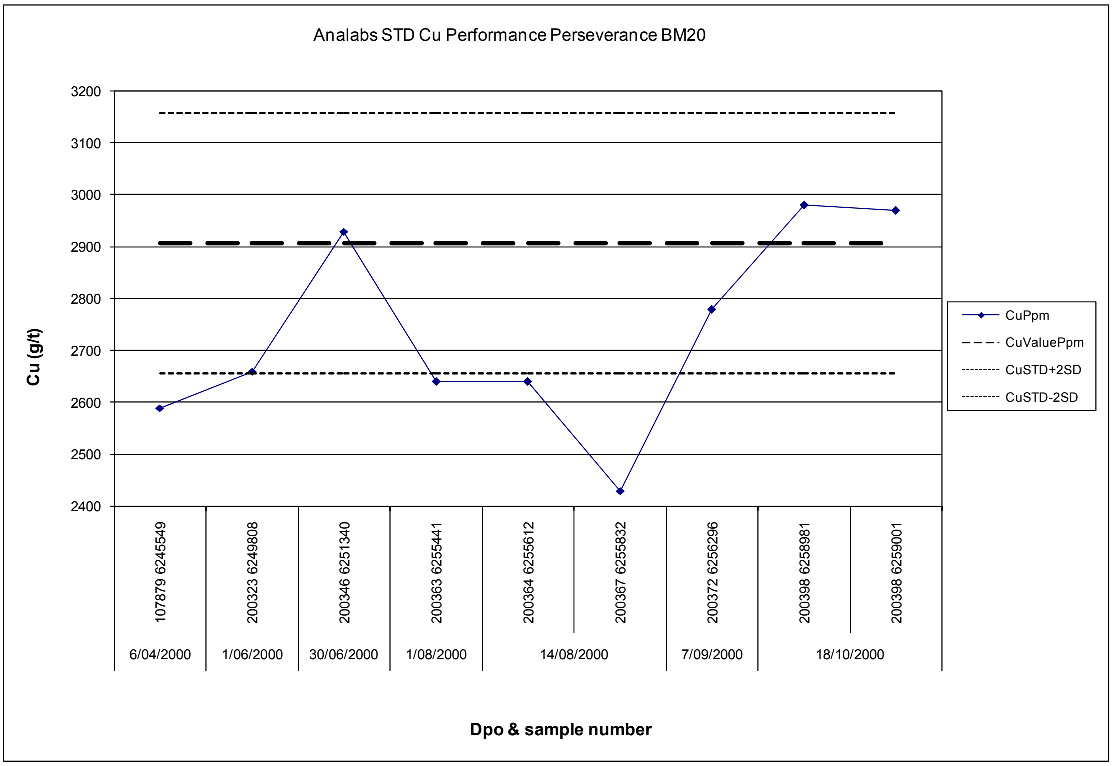
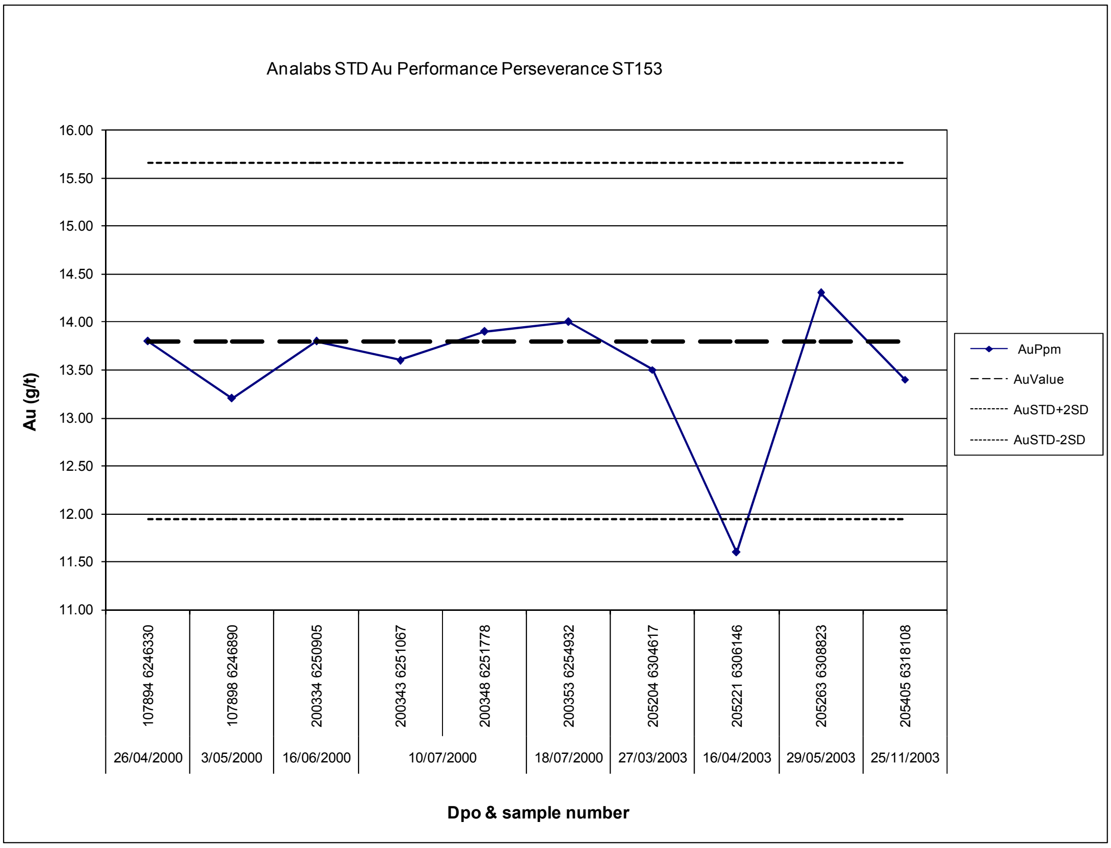
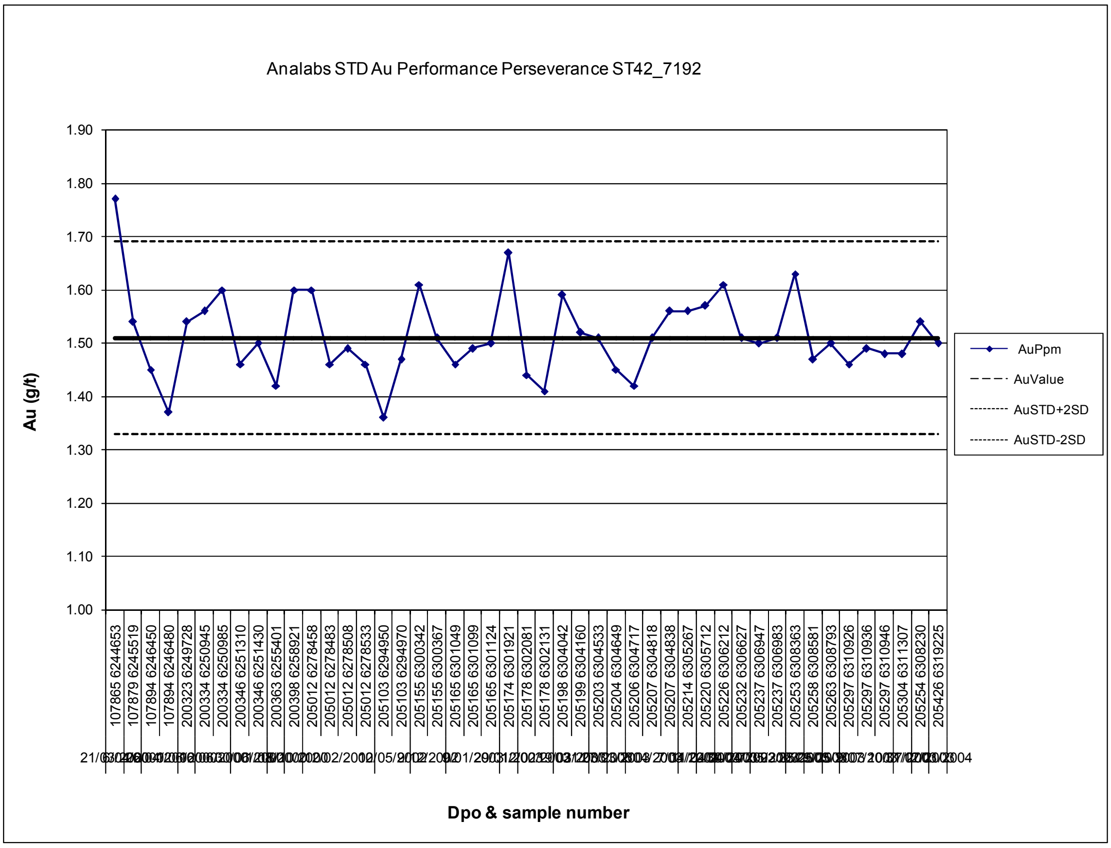
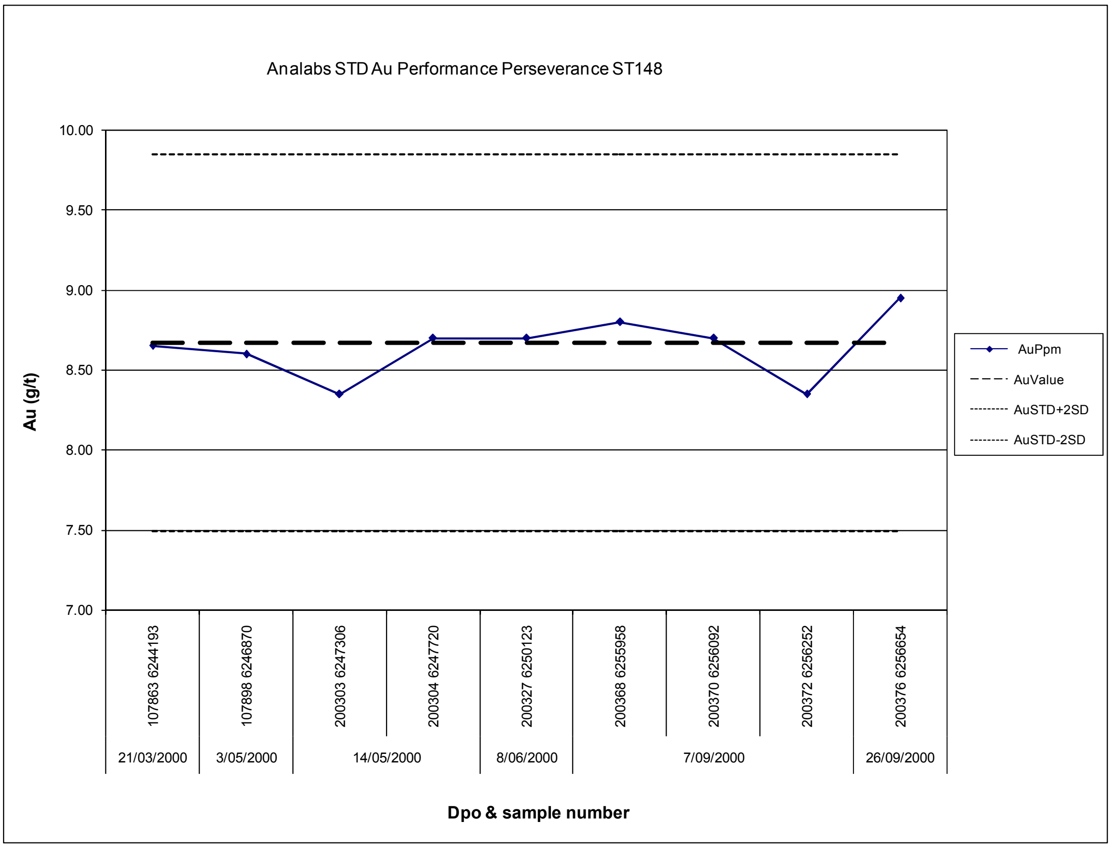


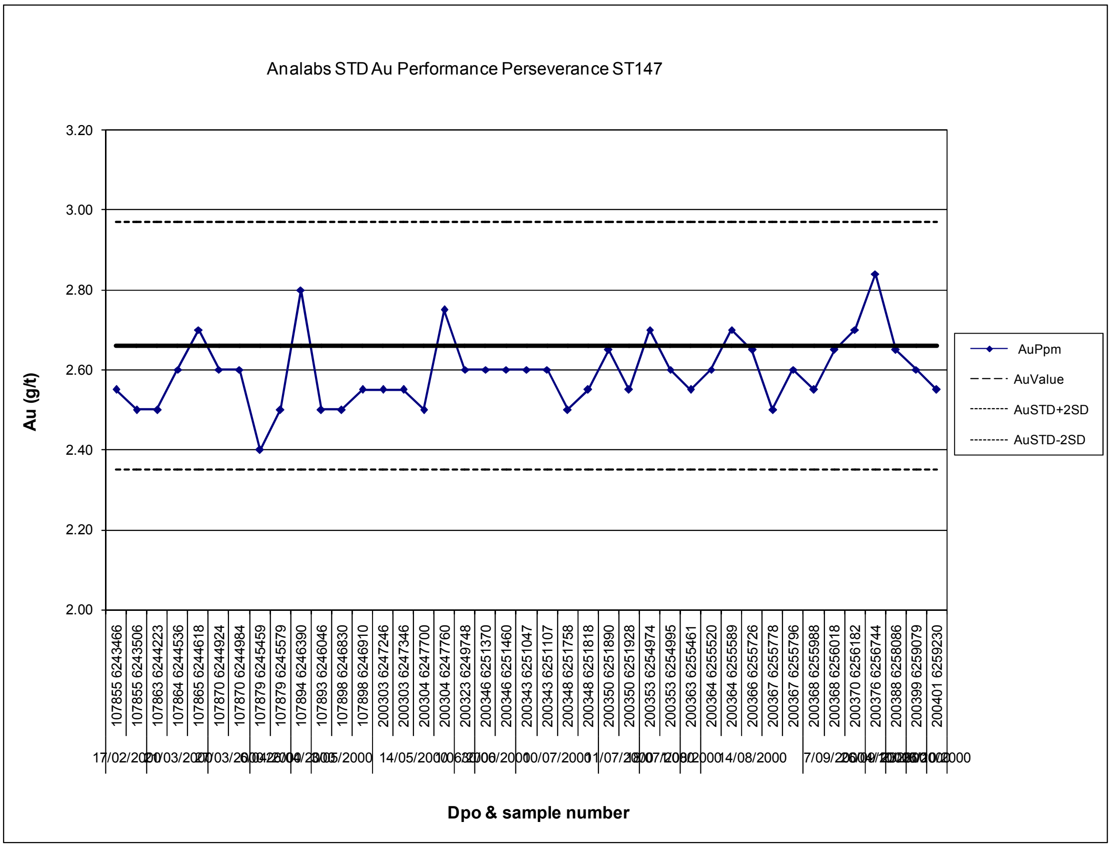


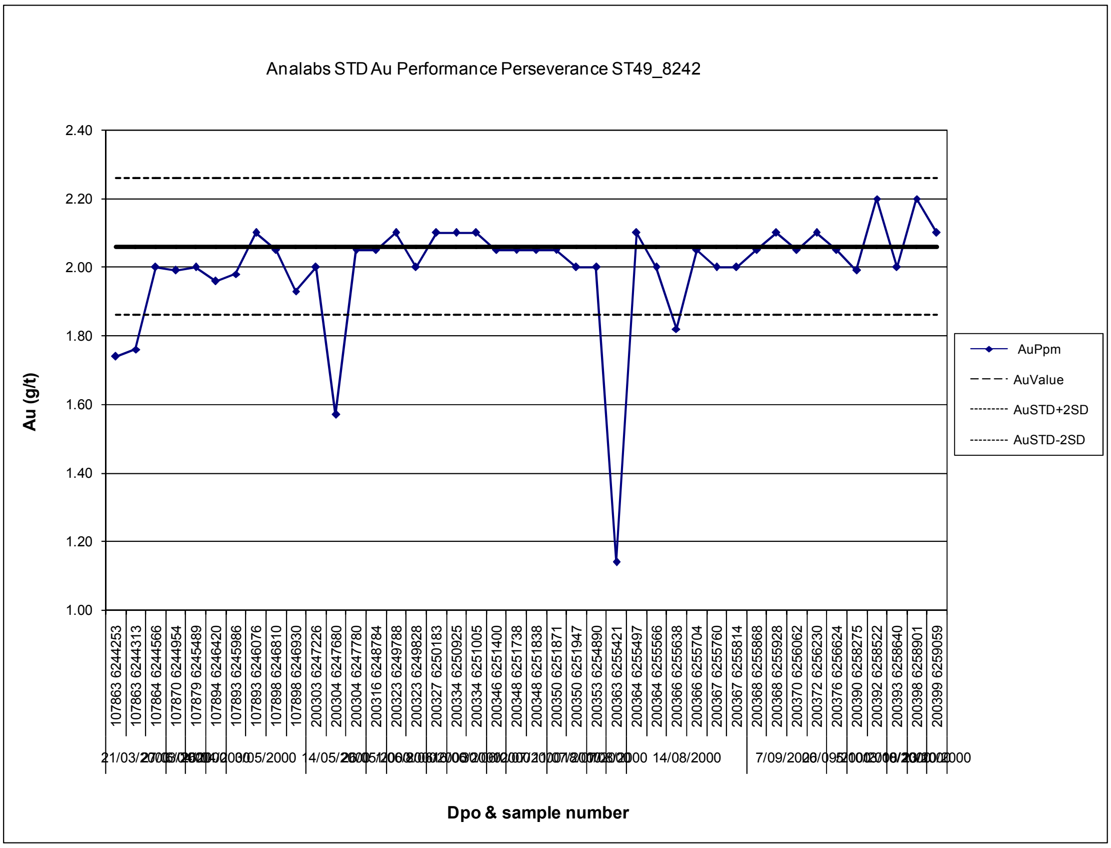
Appendix A - Standards



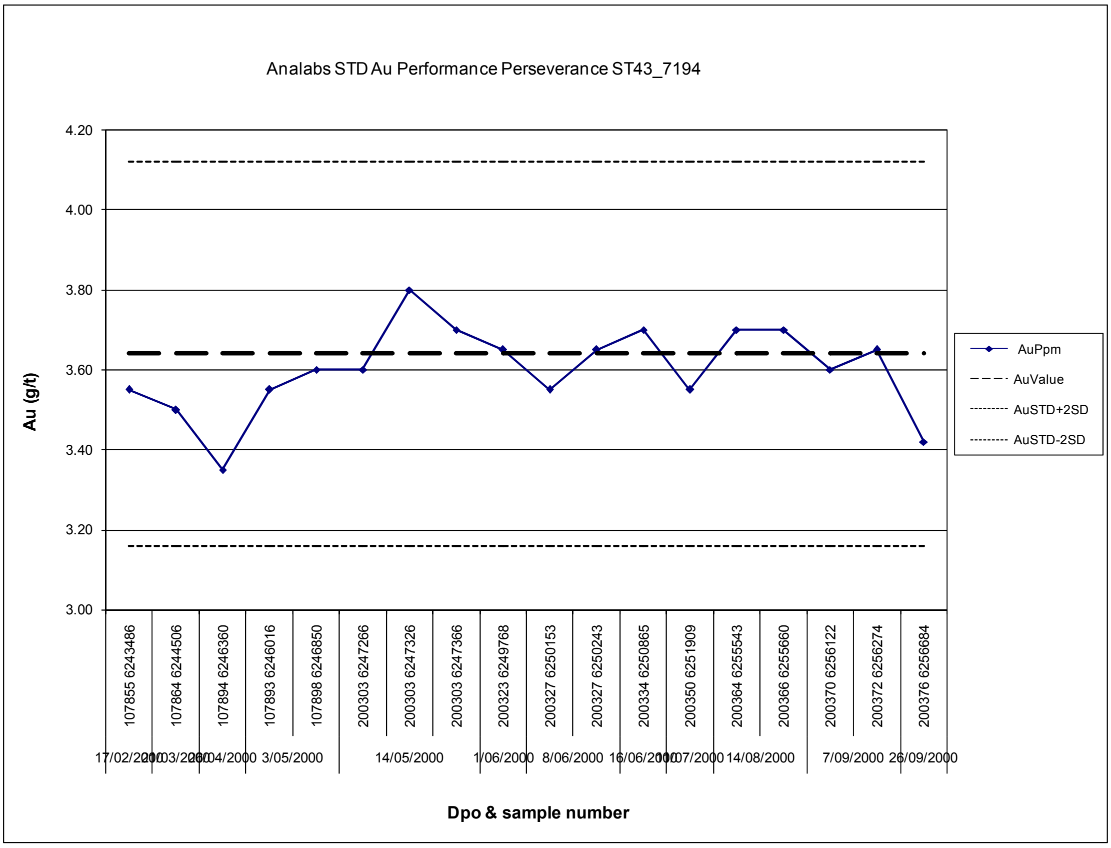


Appendix A - Standards





Appendix A - Standards



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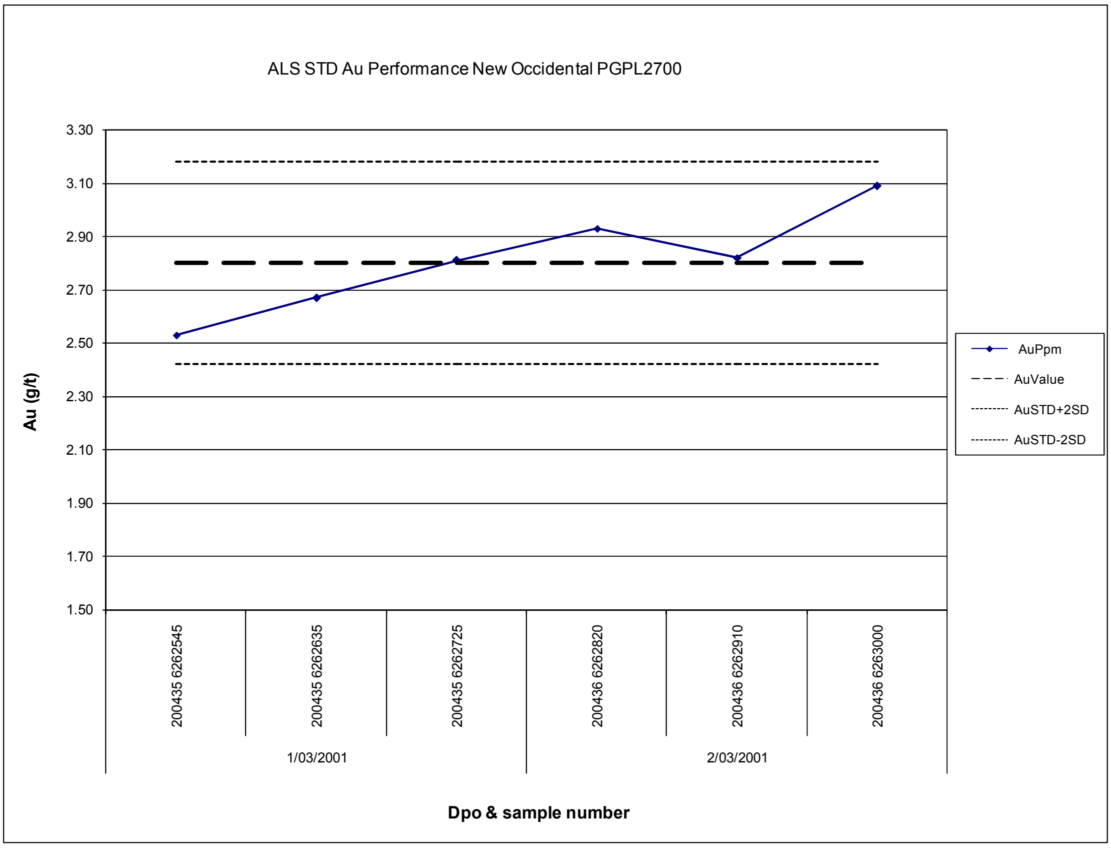

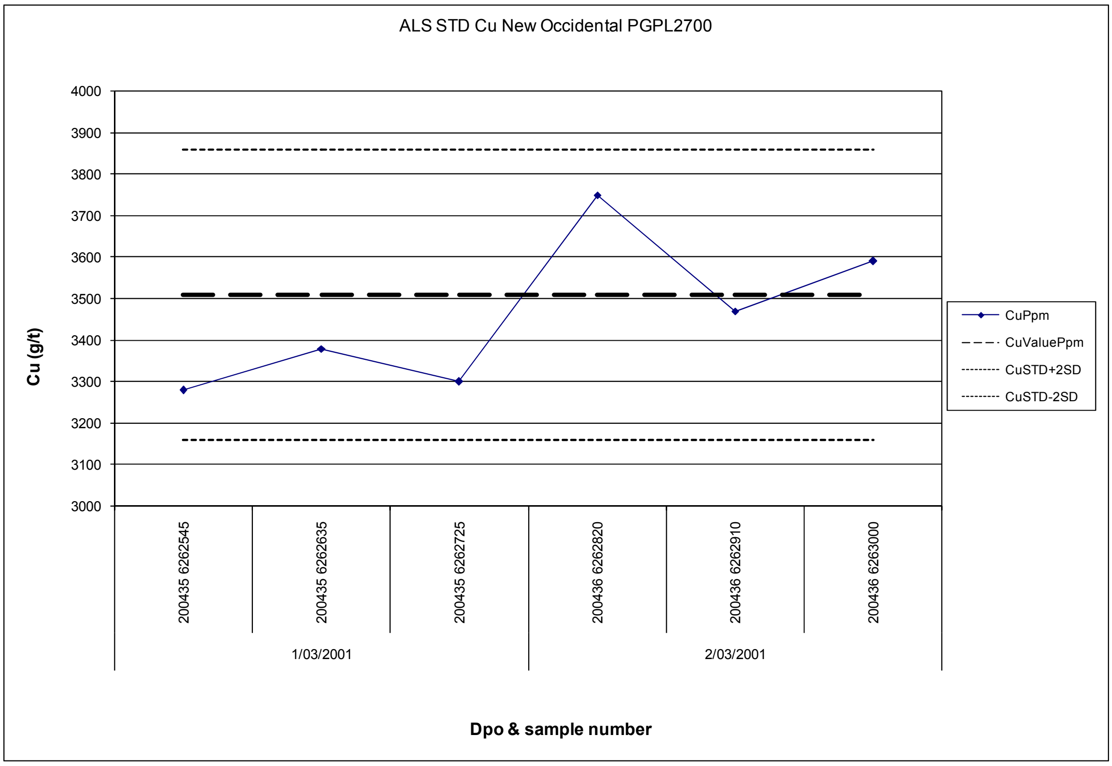

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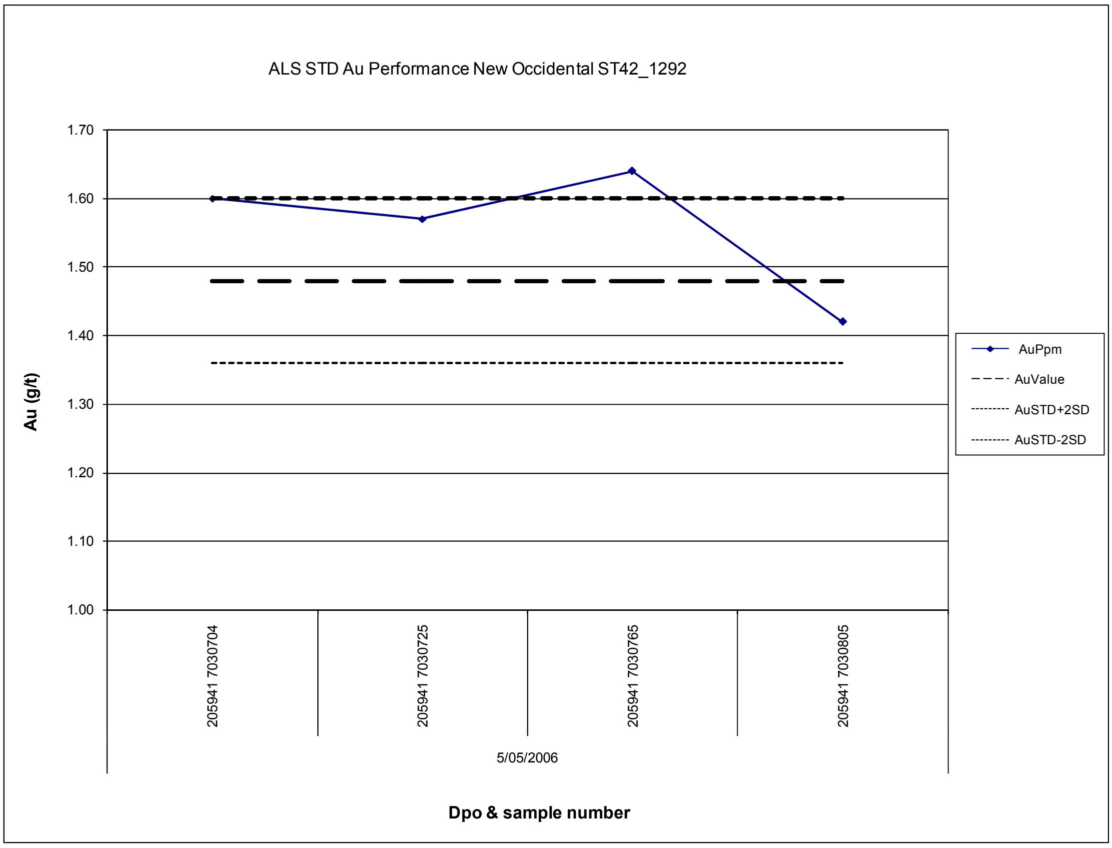

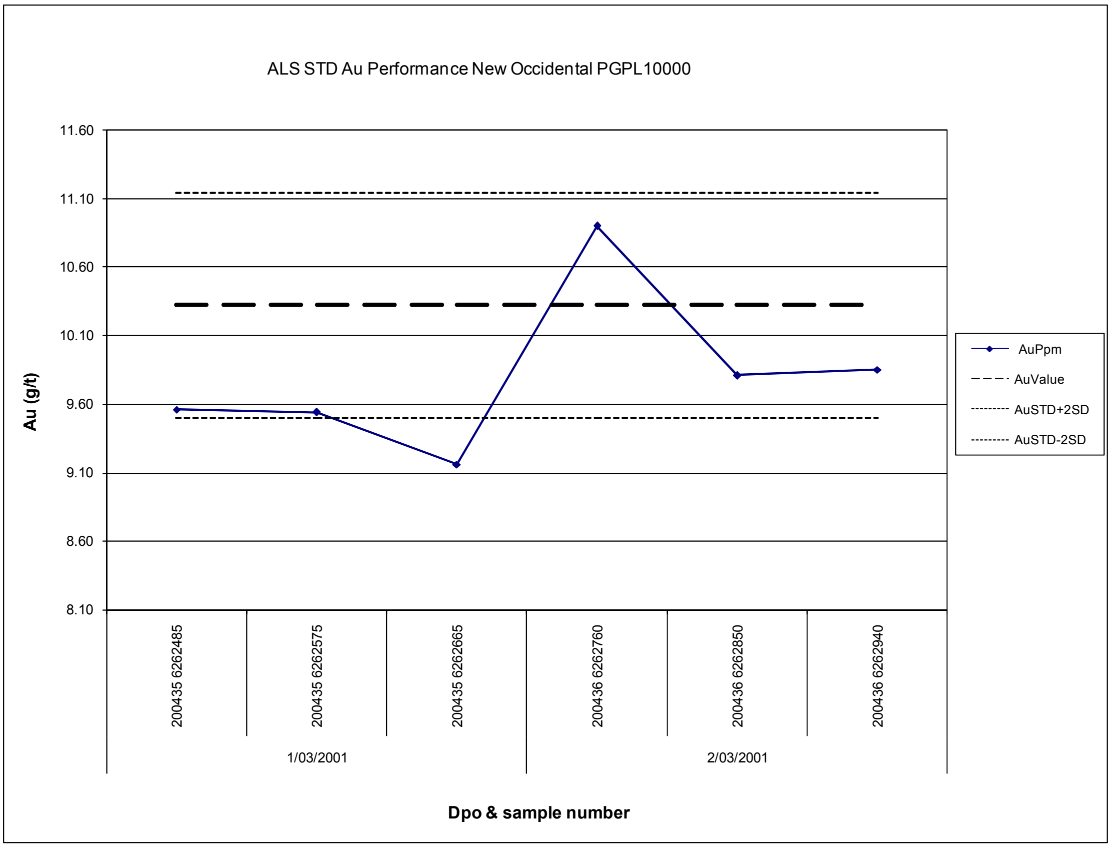

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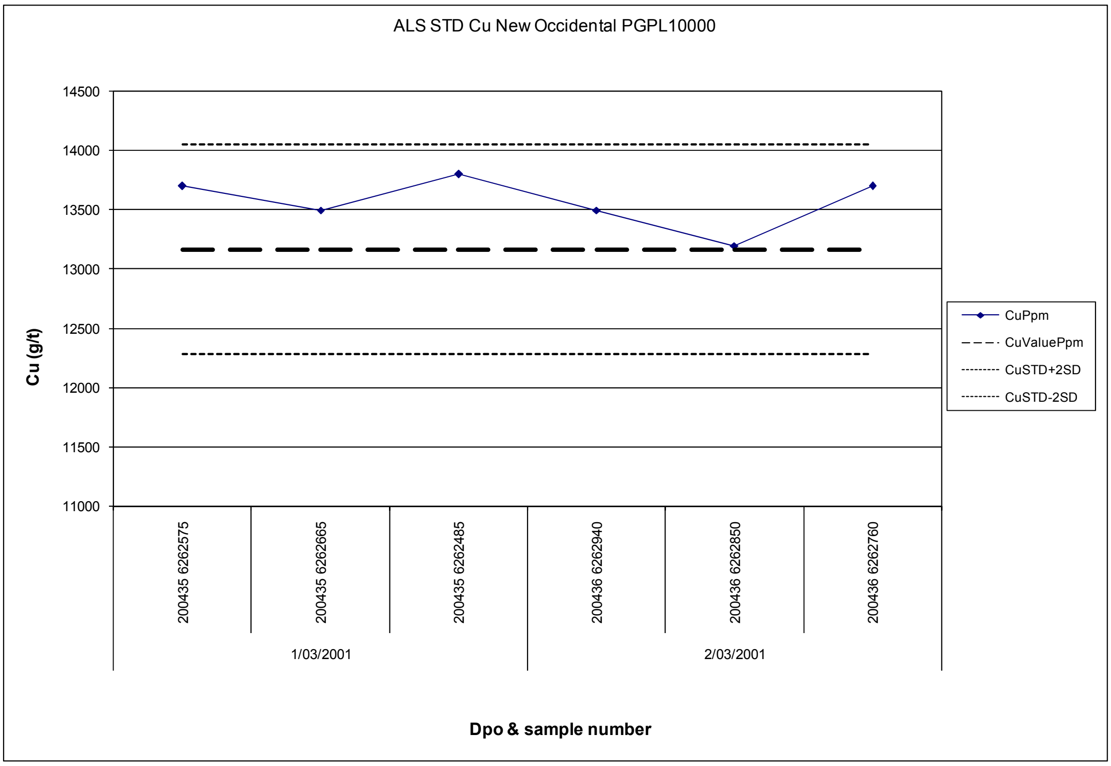

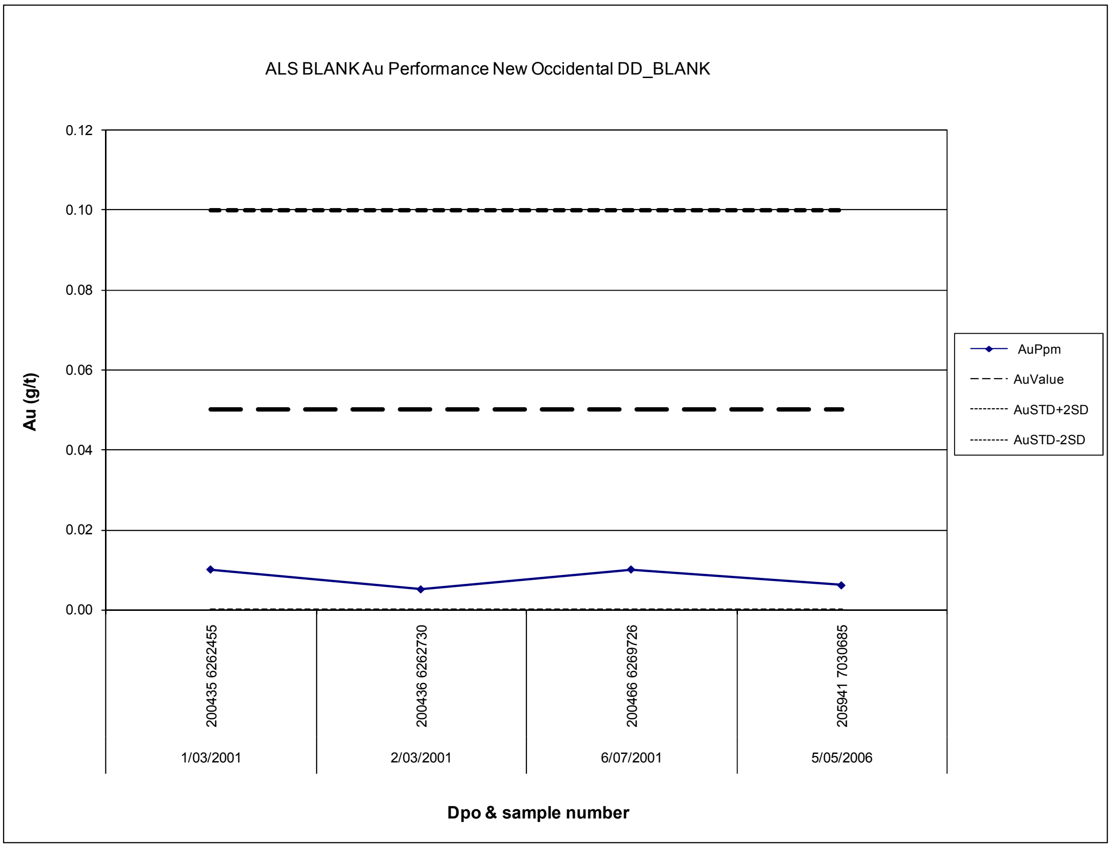

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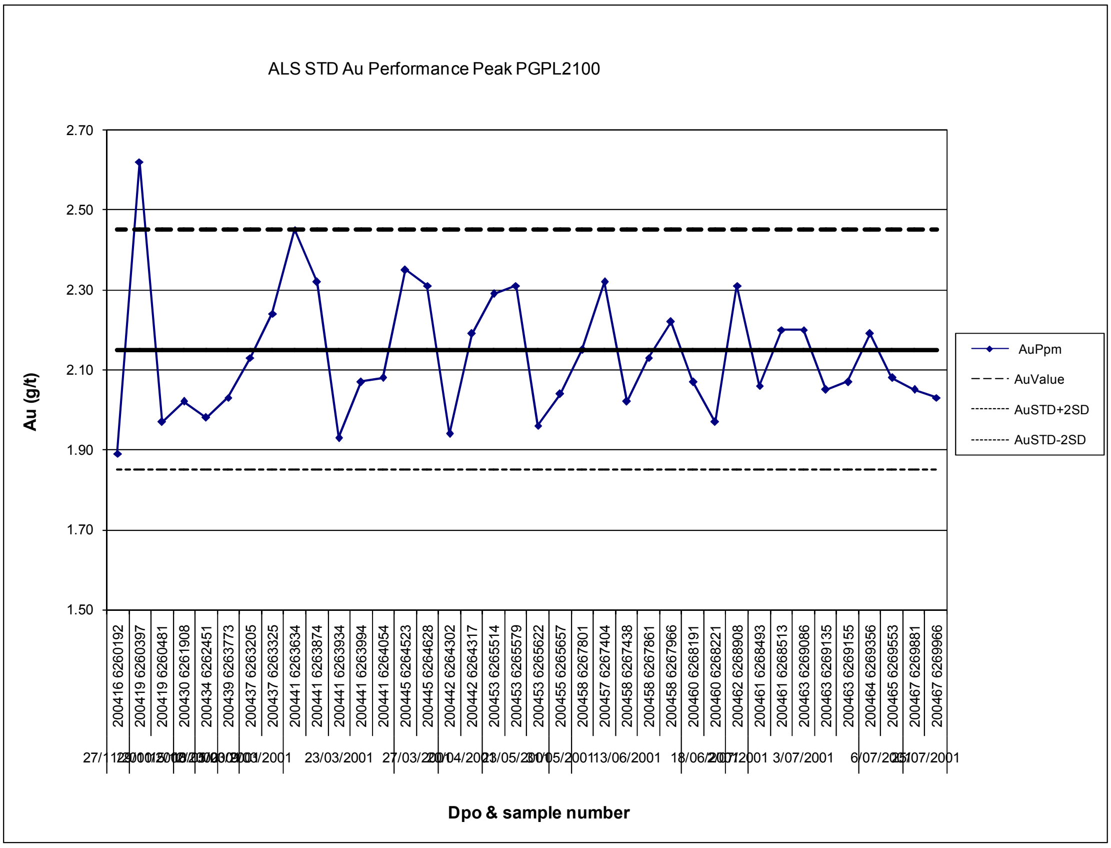
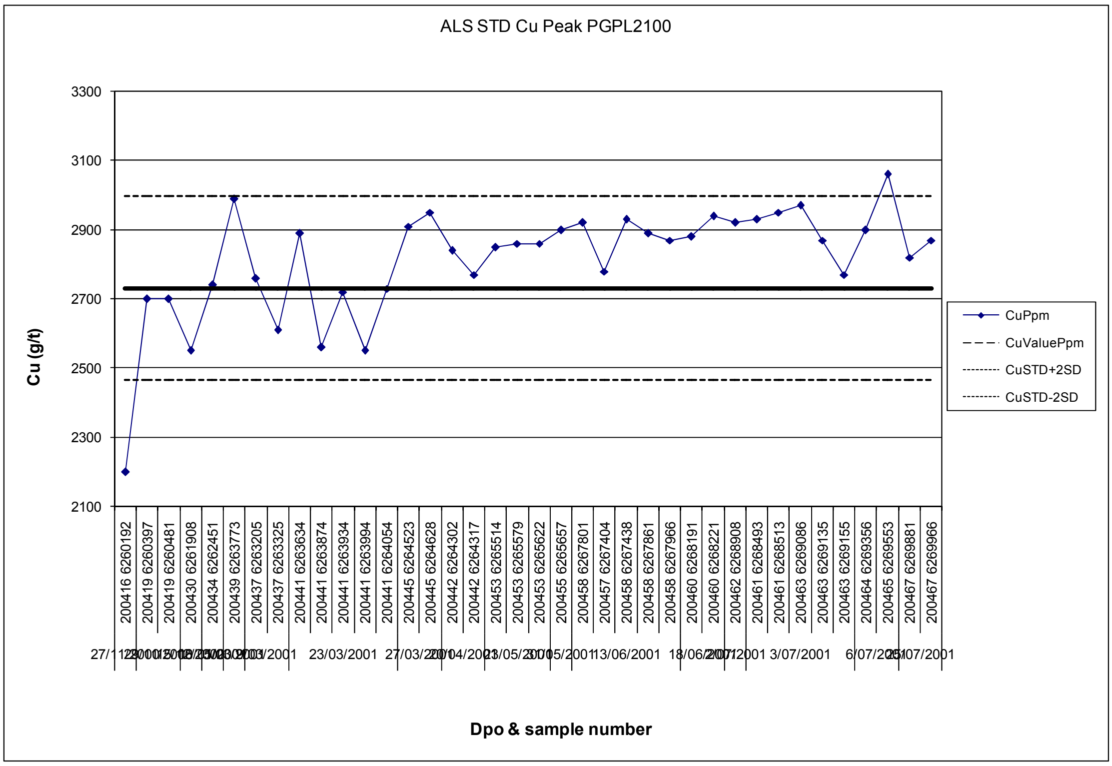
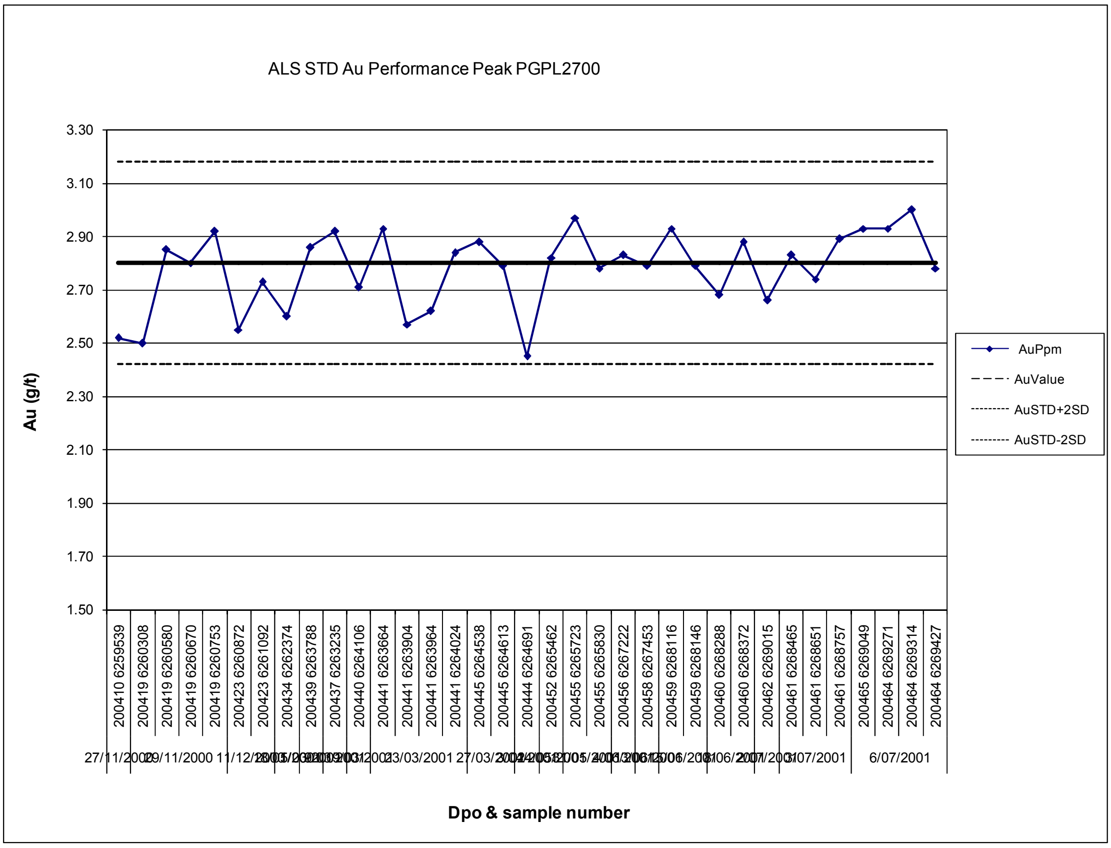
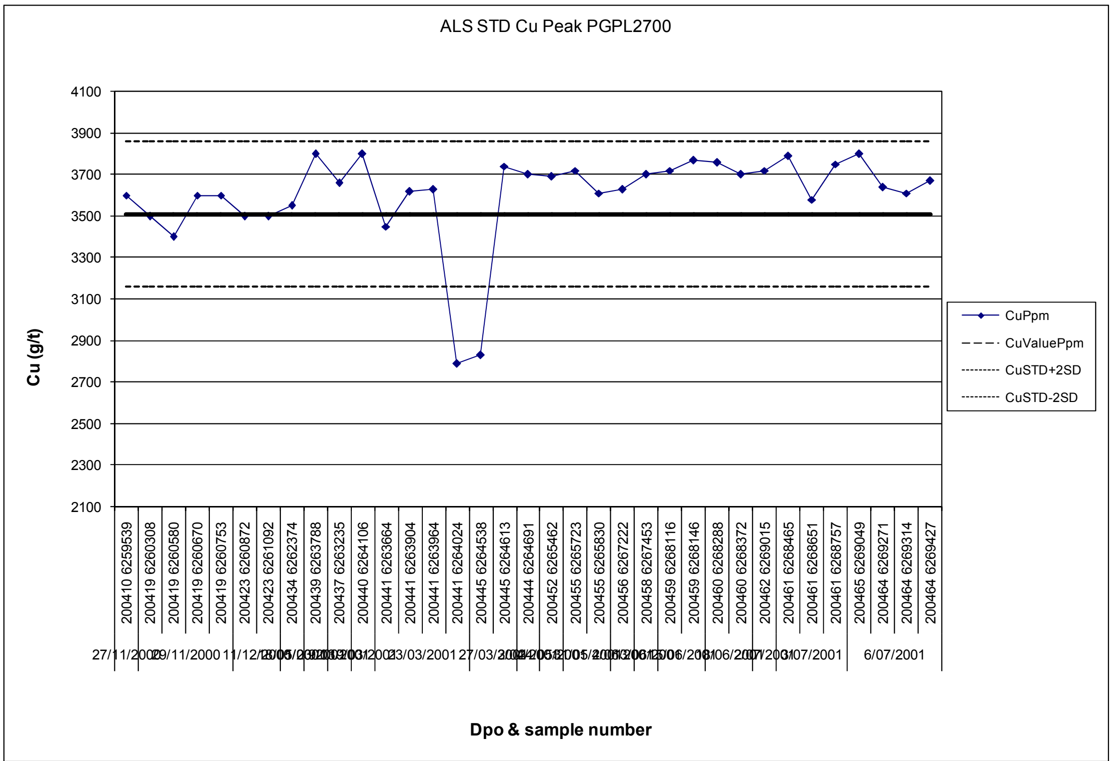


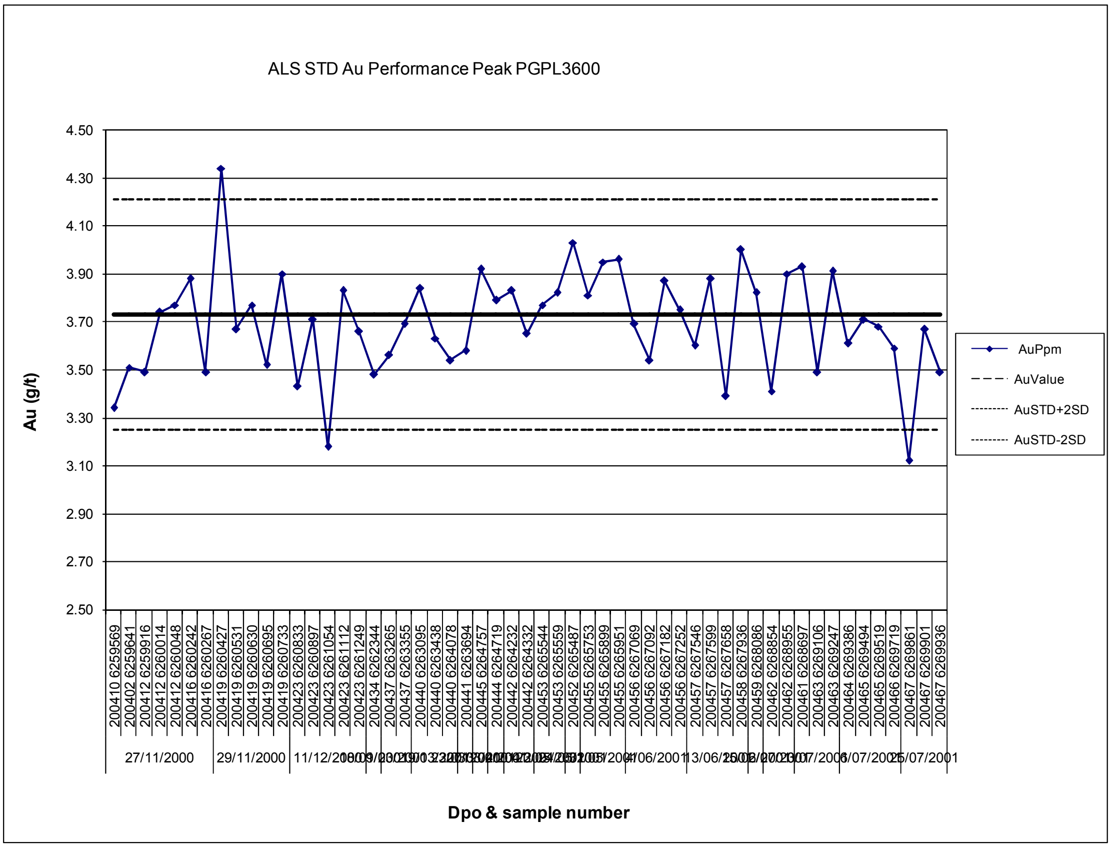


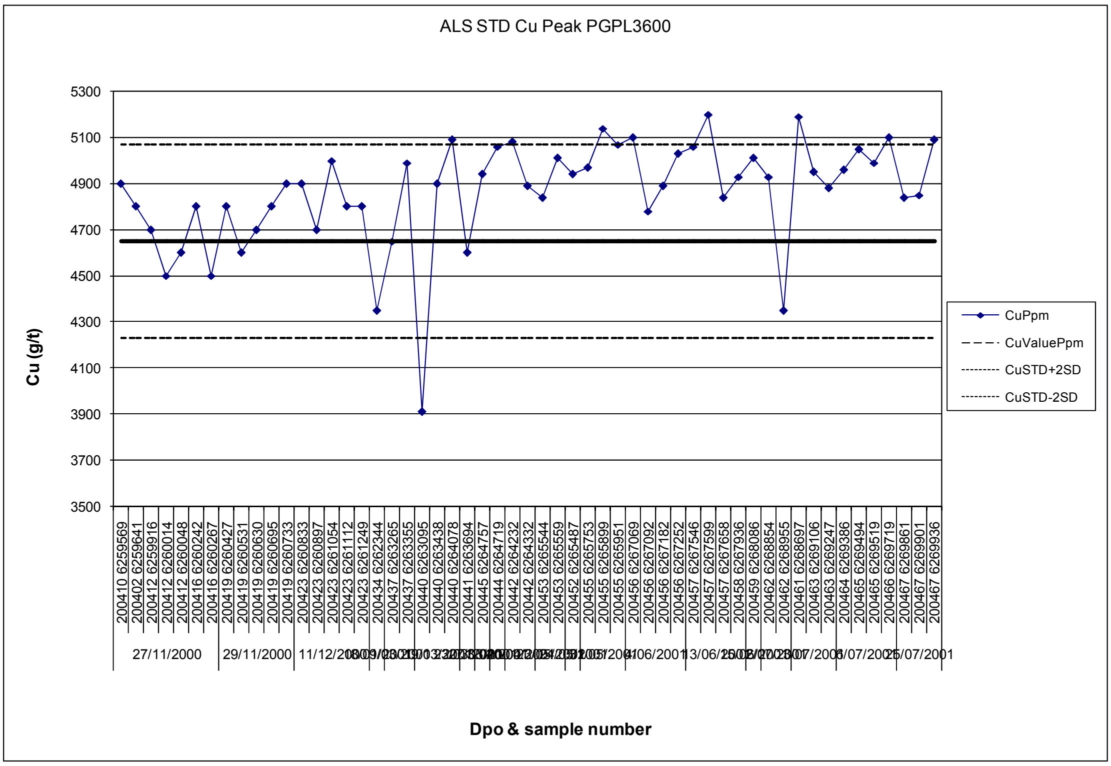
Appendix A - Standards



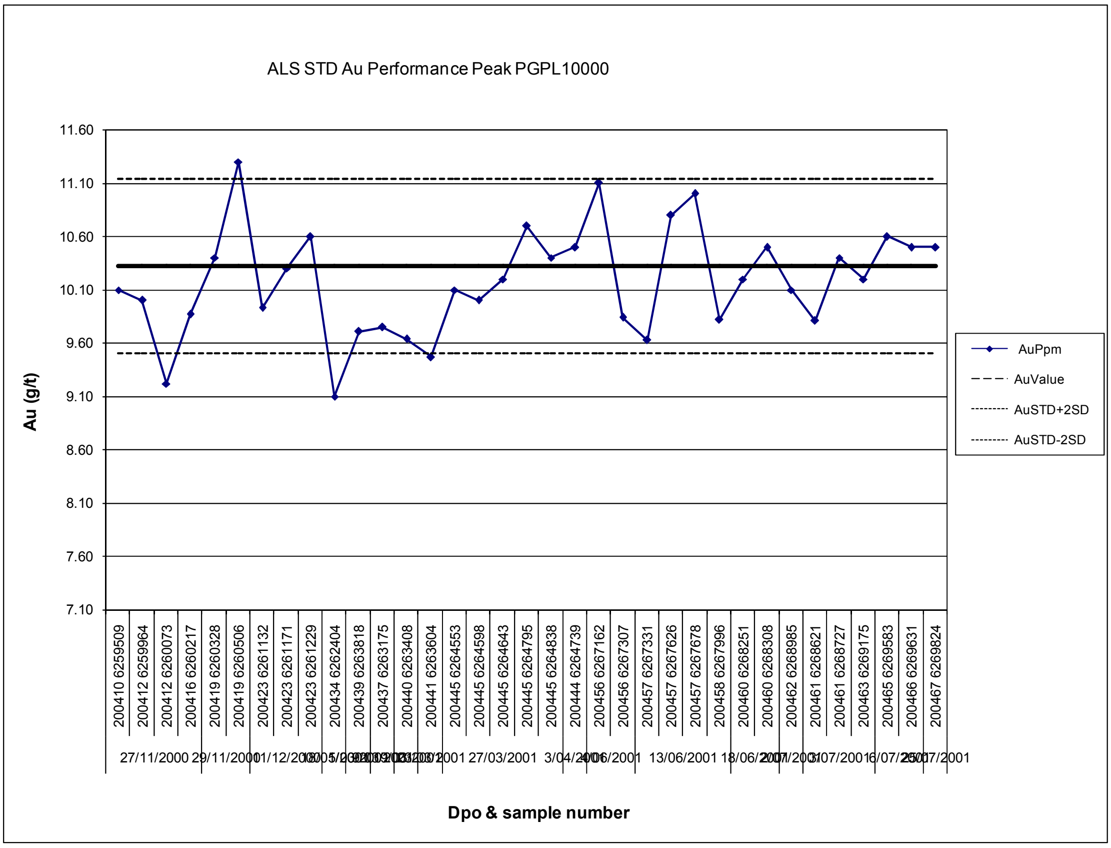


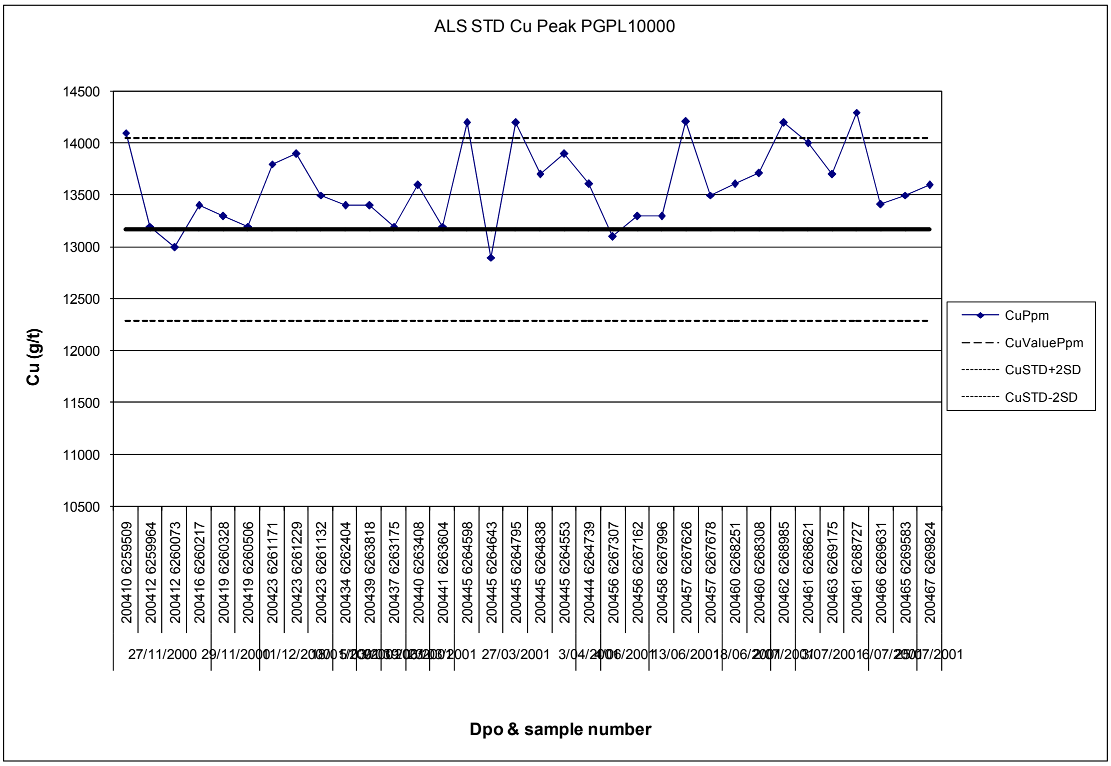
Appendix A - Standards



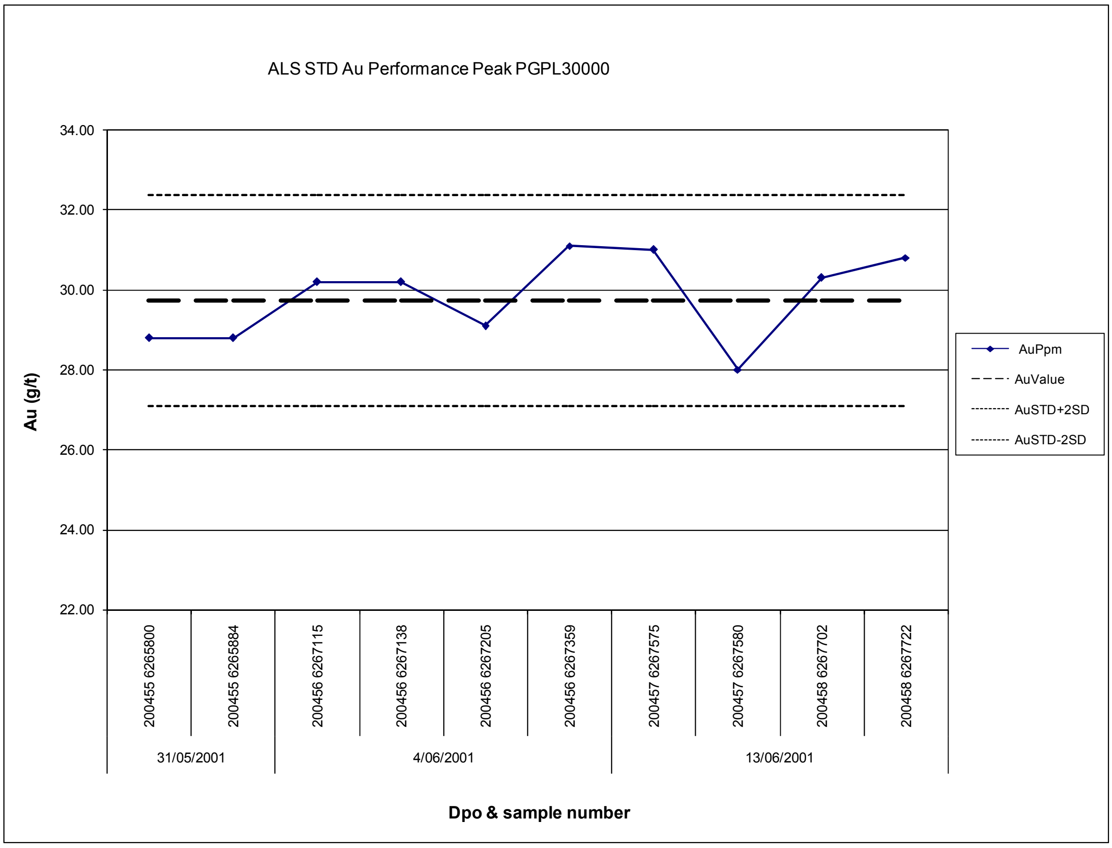


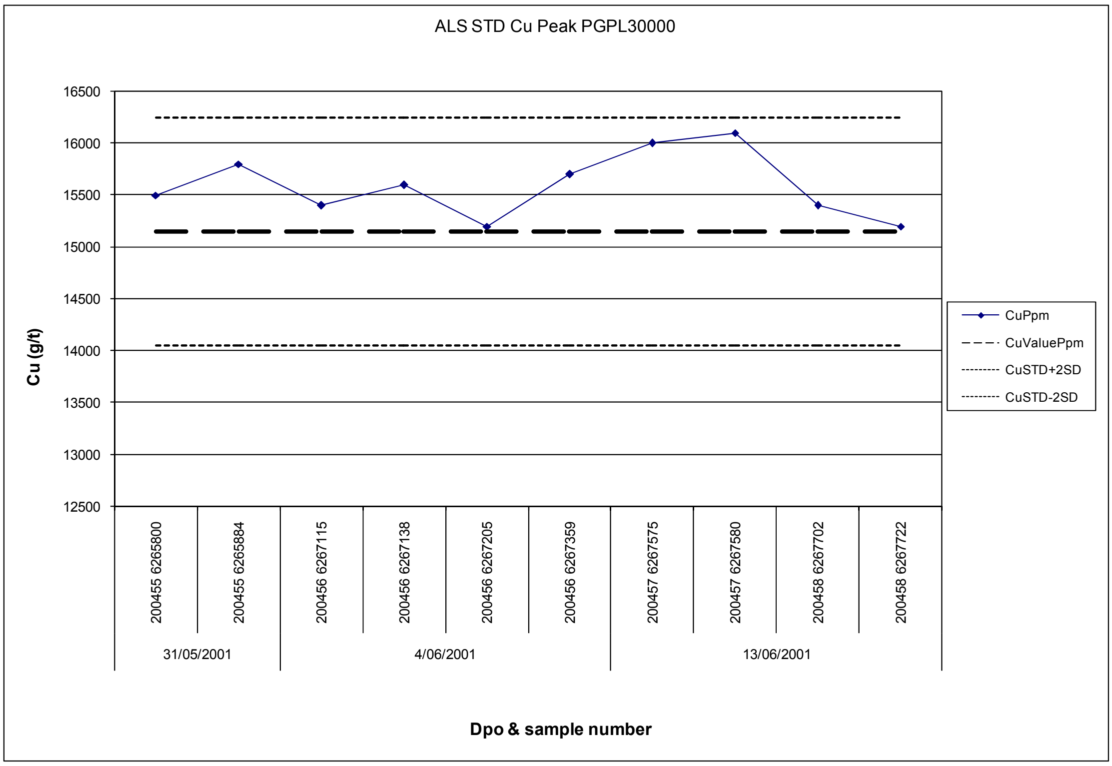
Appendix A - Standards



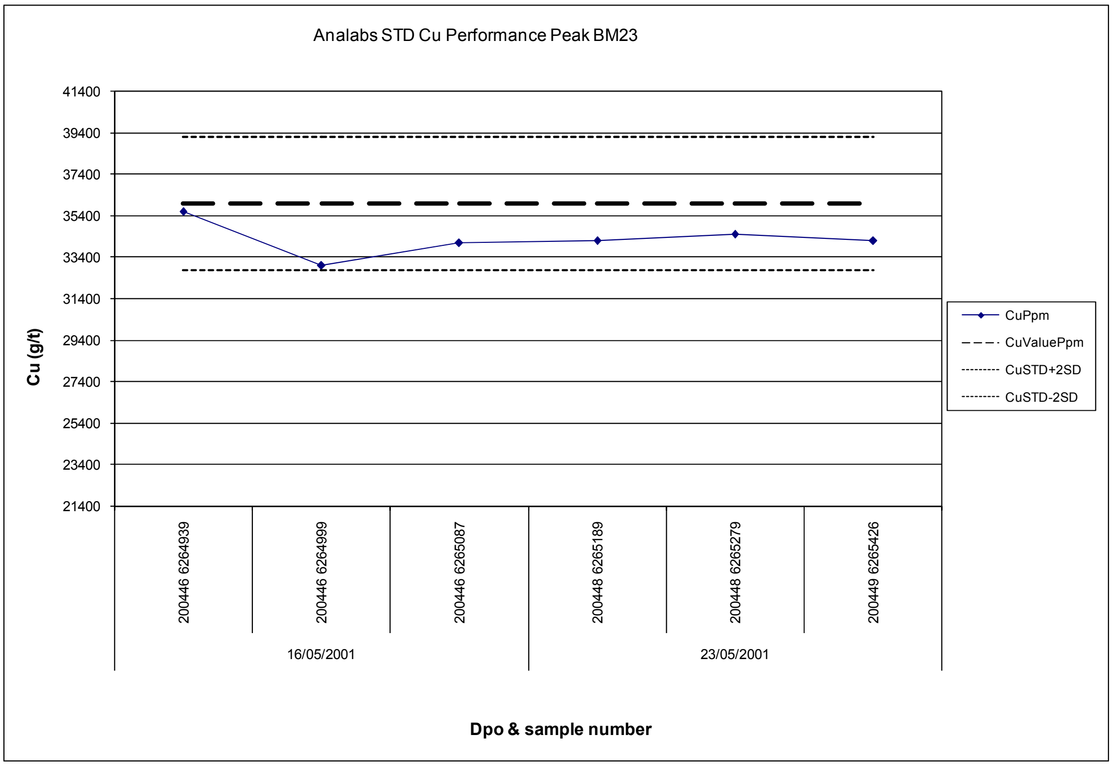


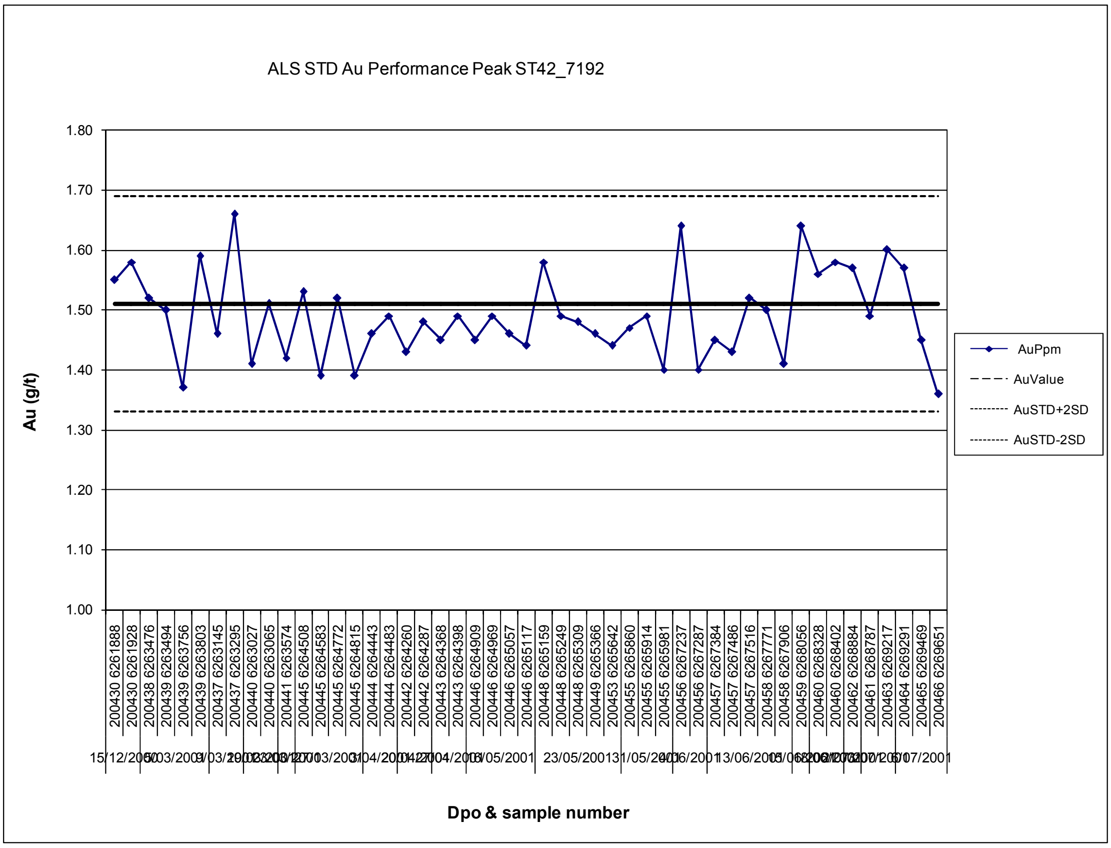
Appendix A - Standards



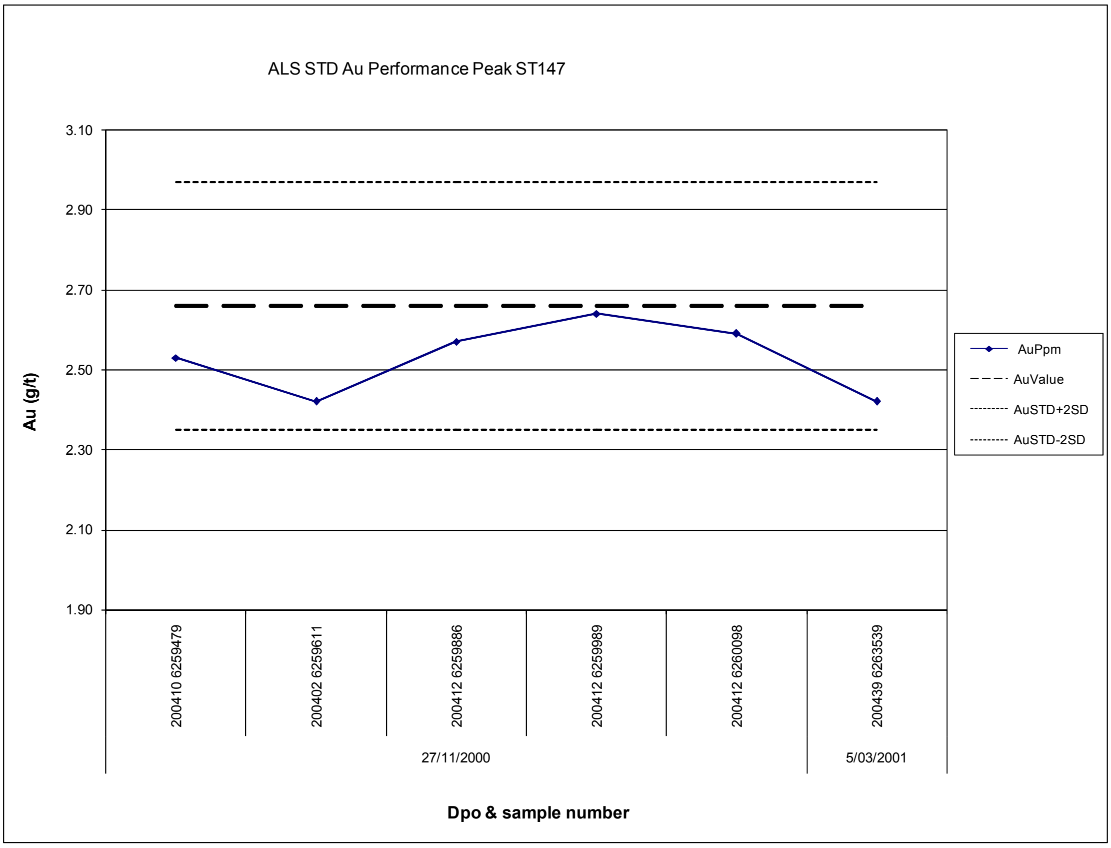


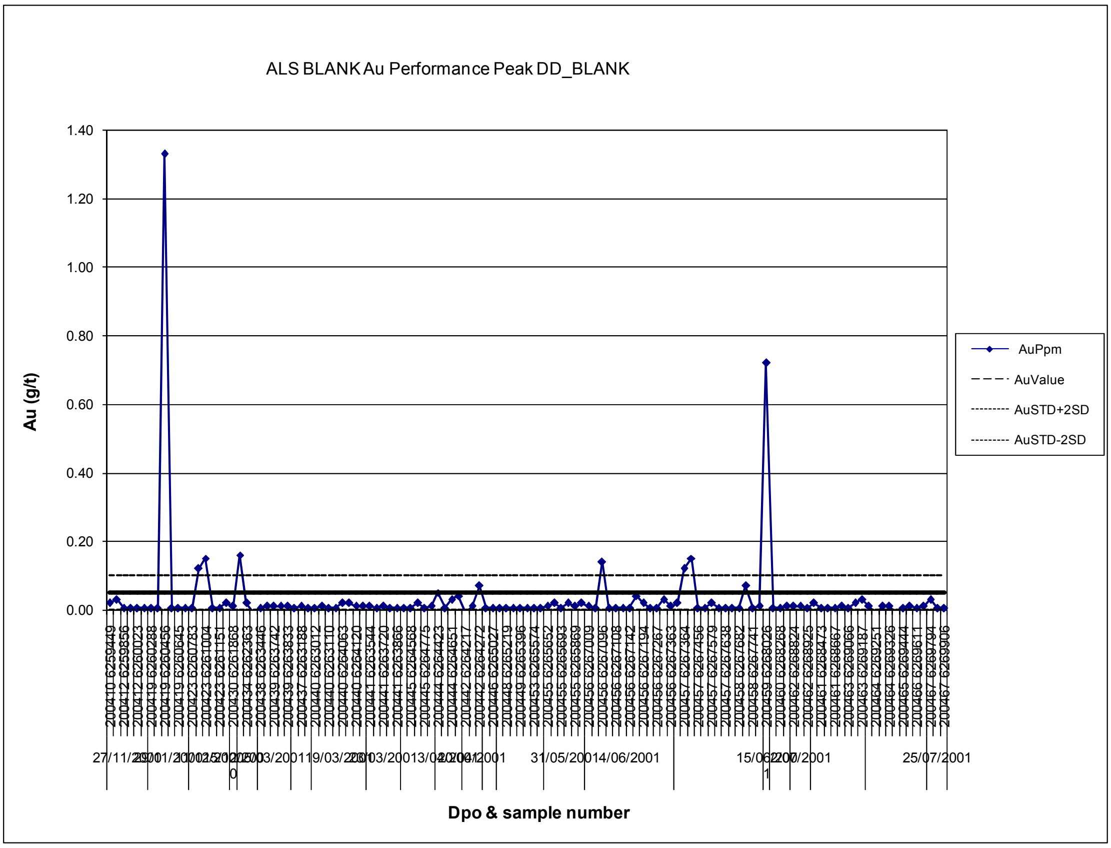
Appendix A - Standards



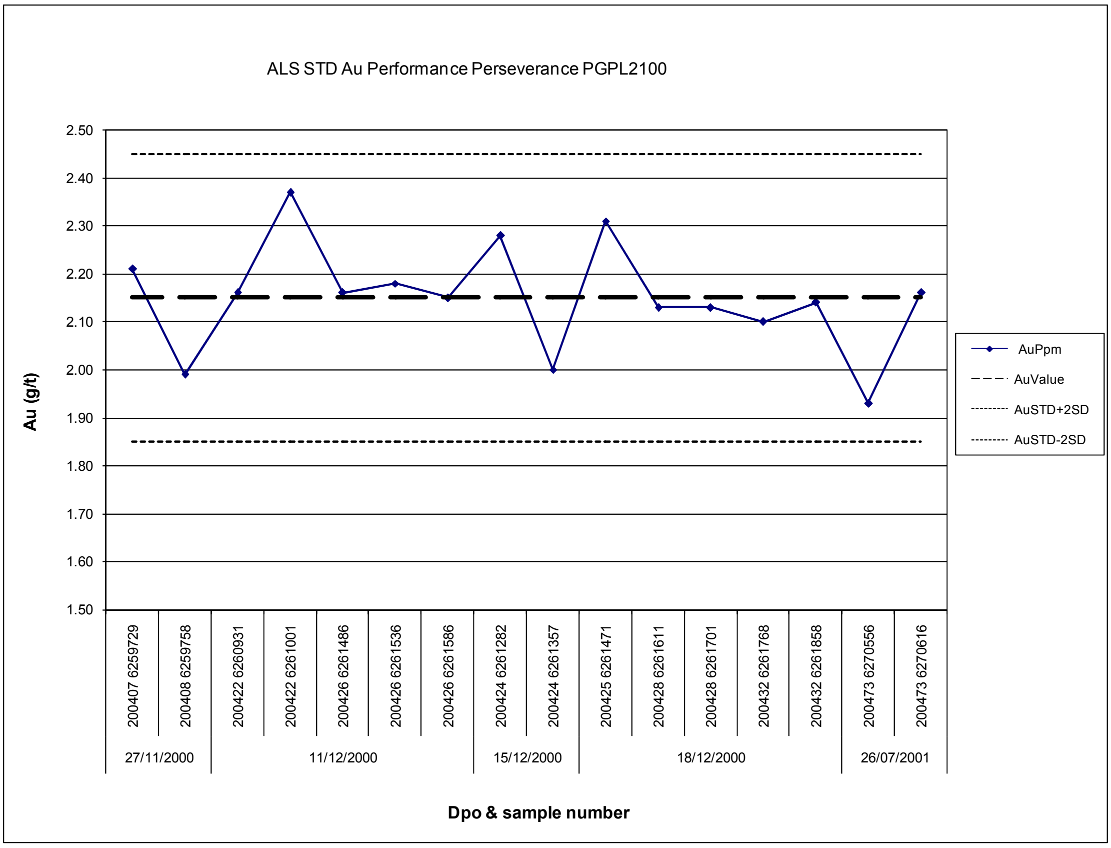


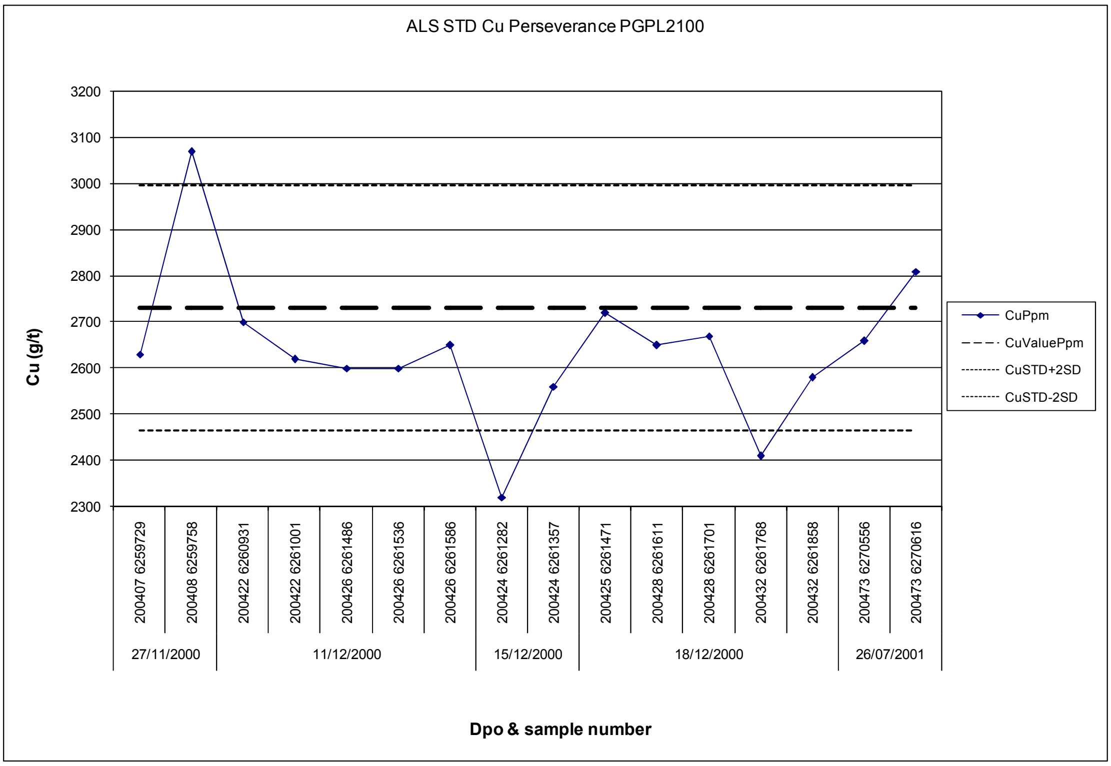
Appendix A - Standards



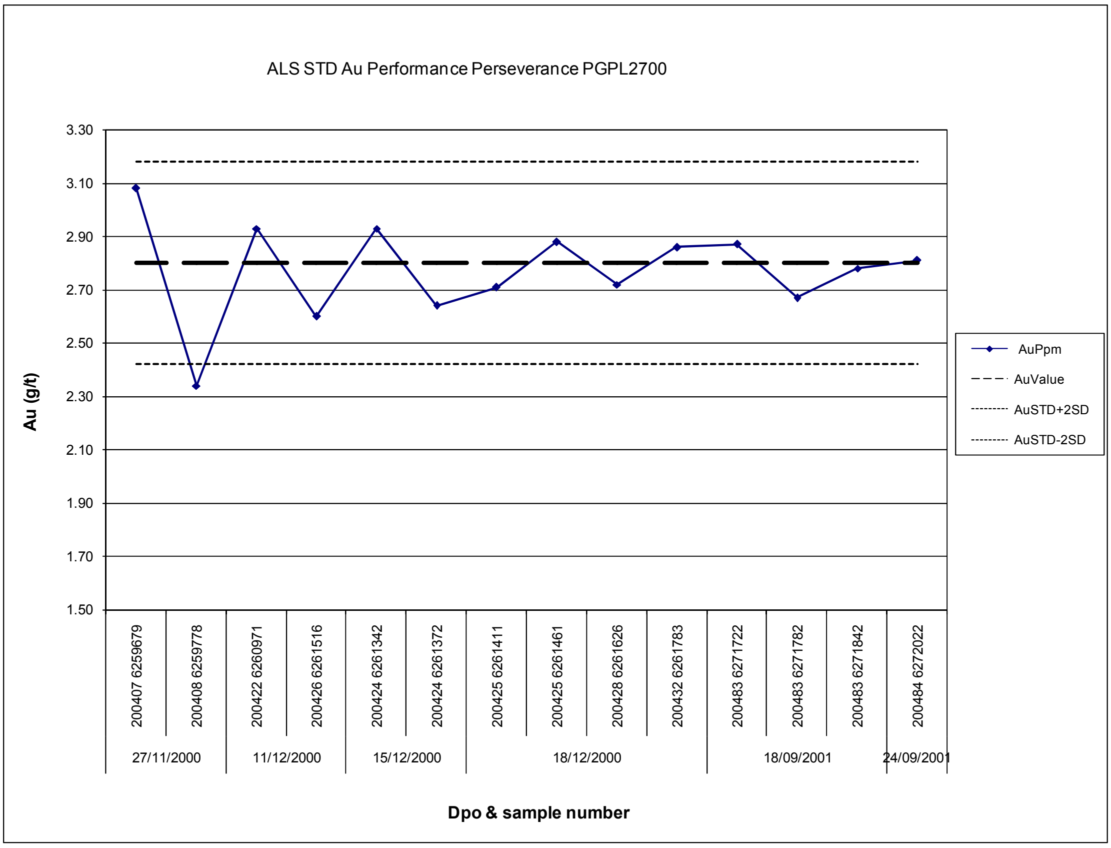


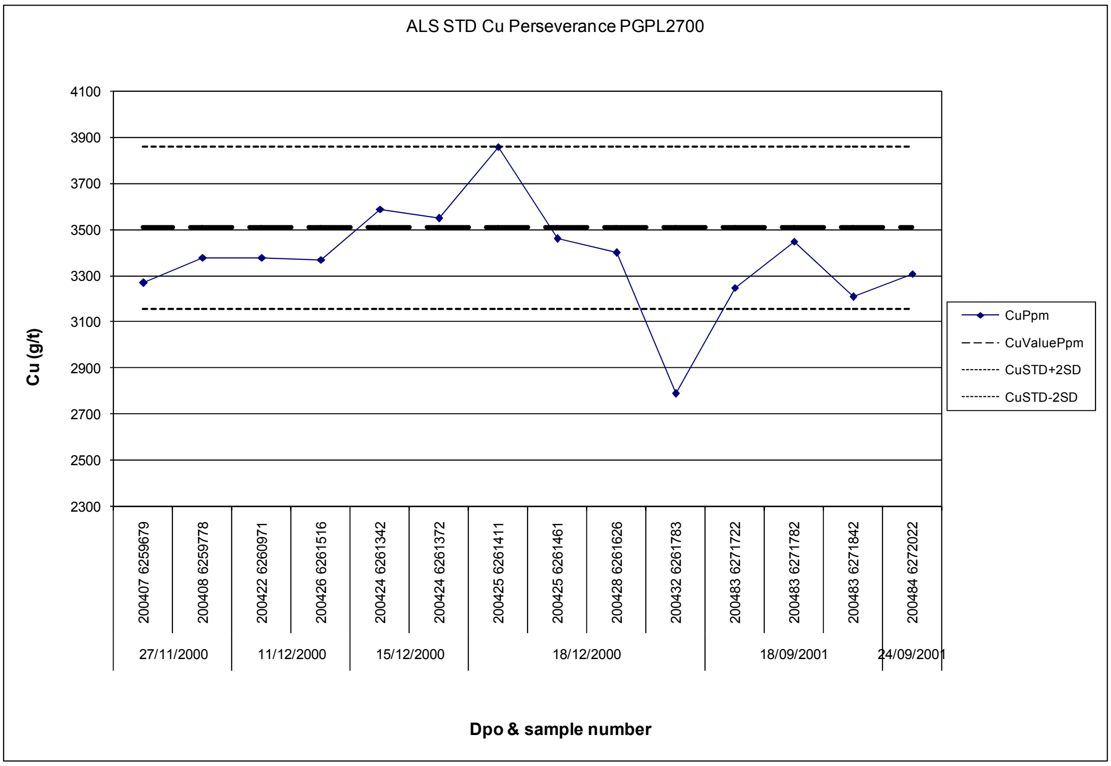
Appendix A - Standards



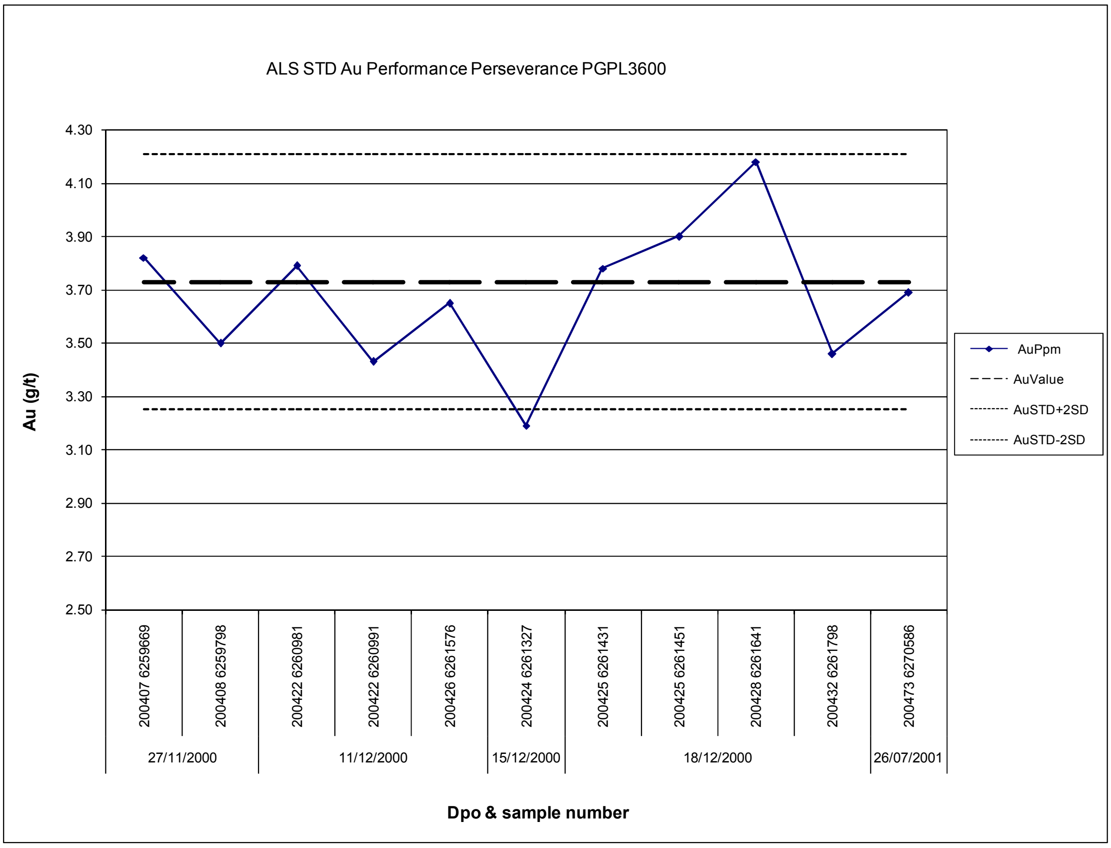


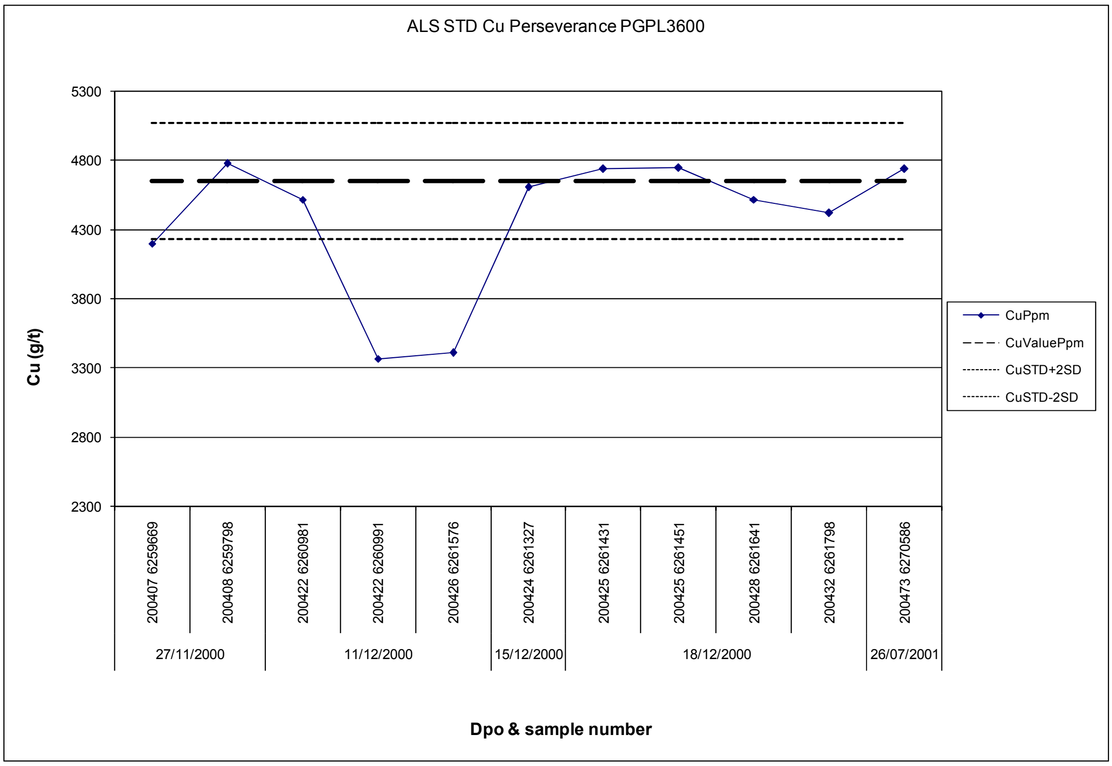
Appendix A - Standards



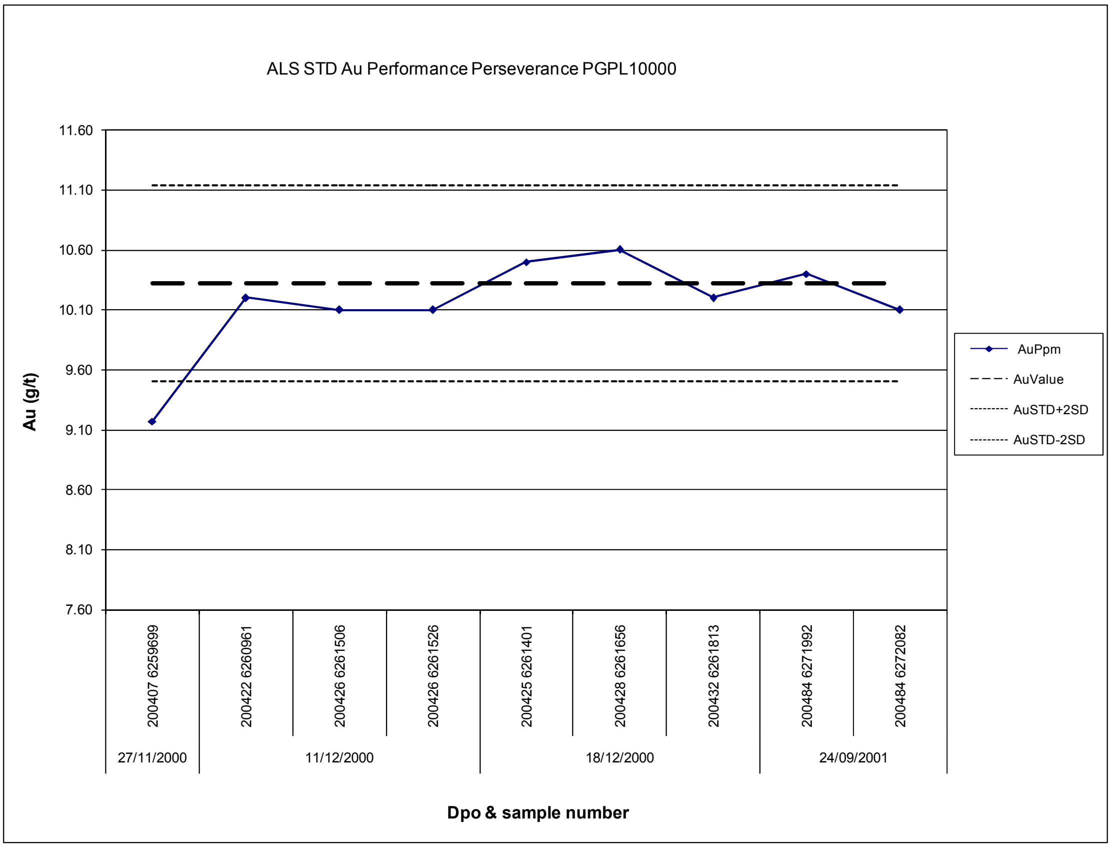


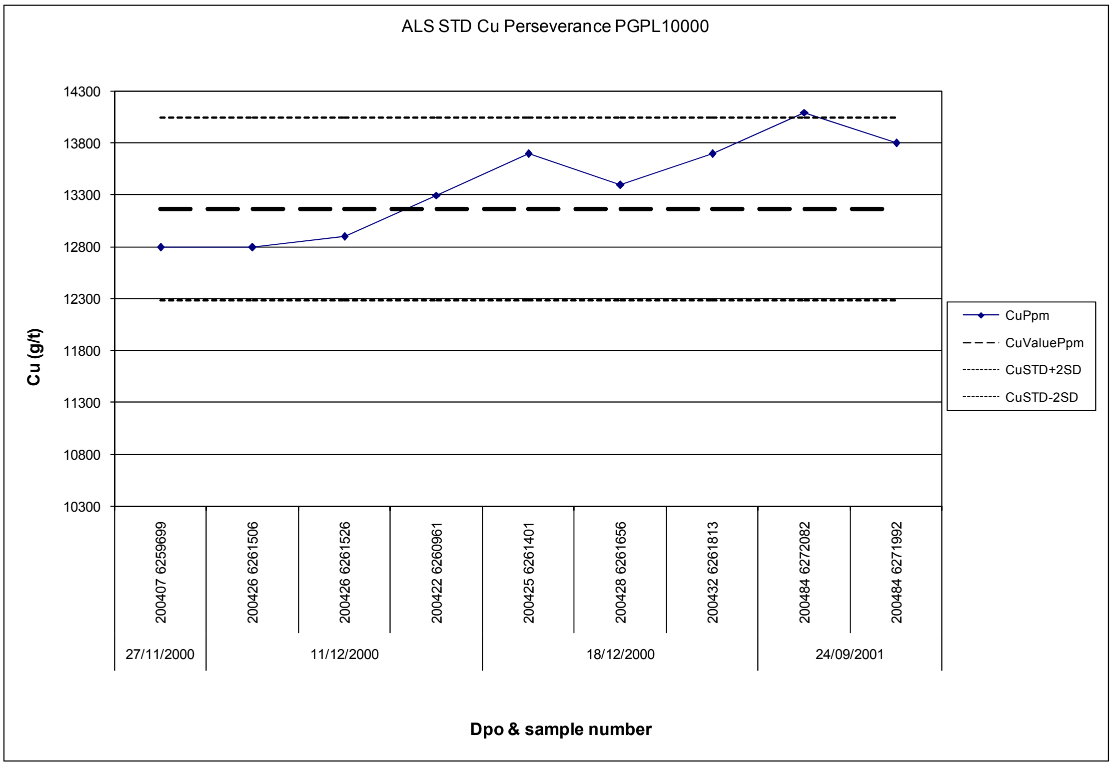
Appendix A - Standards



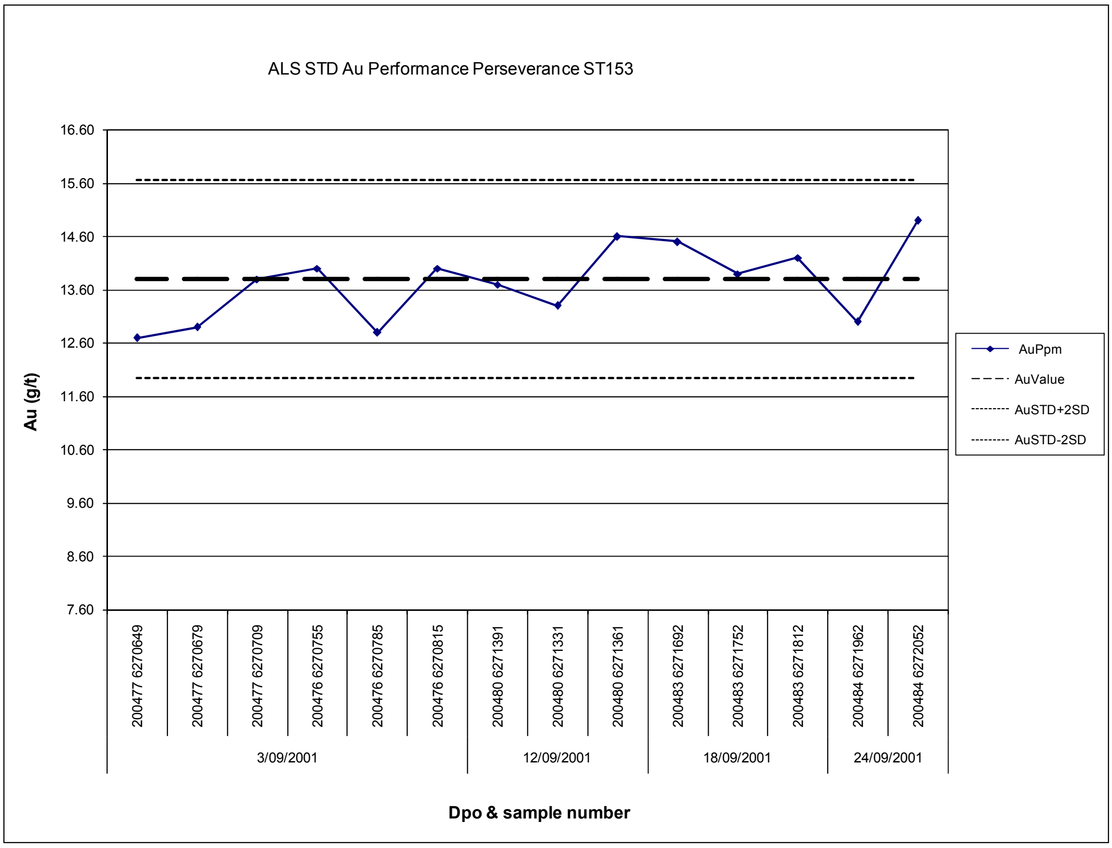


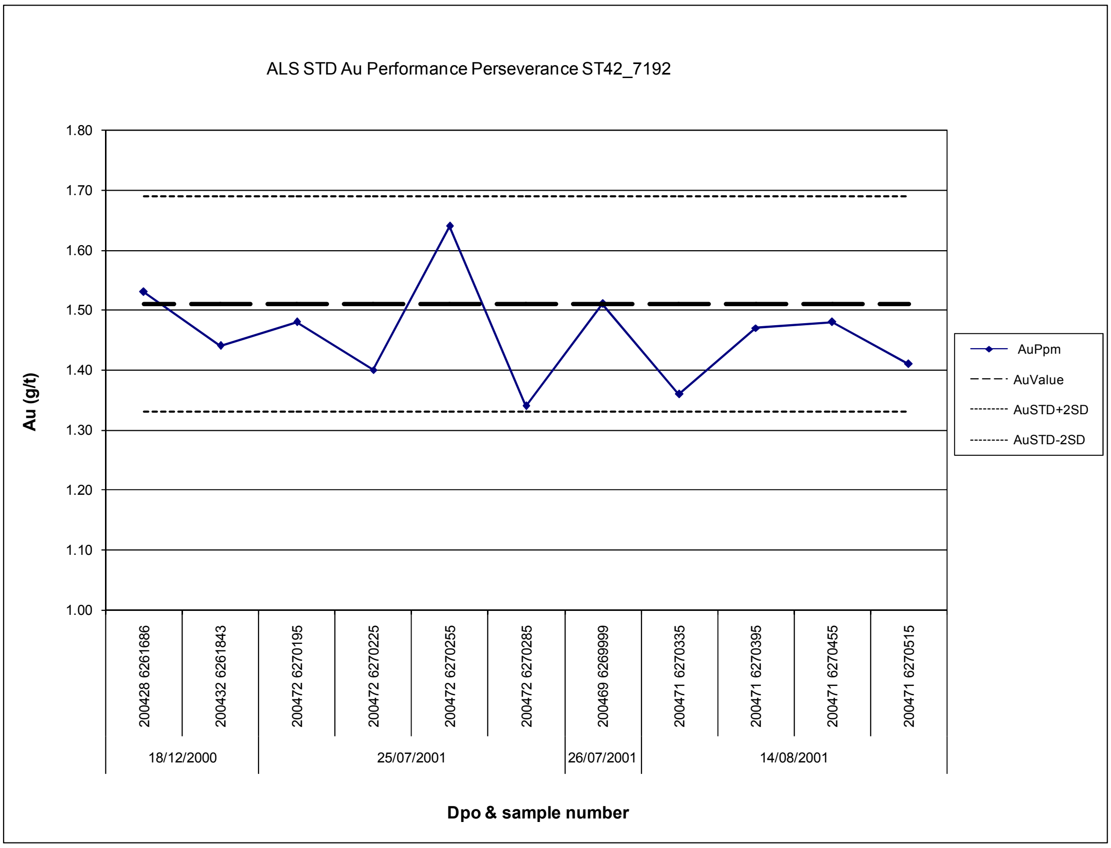
Appendix A - Standards



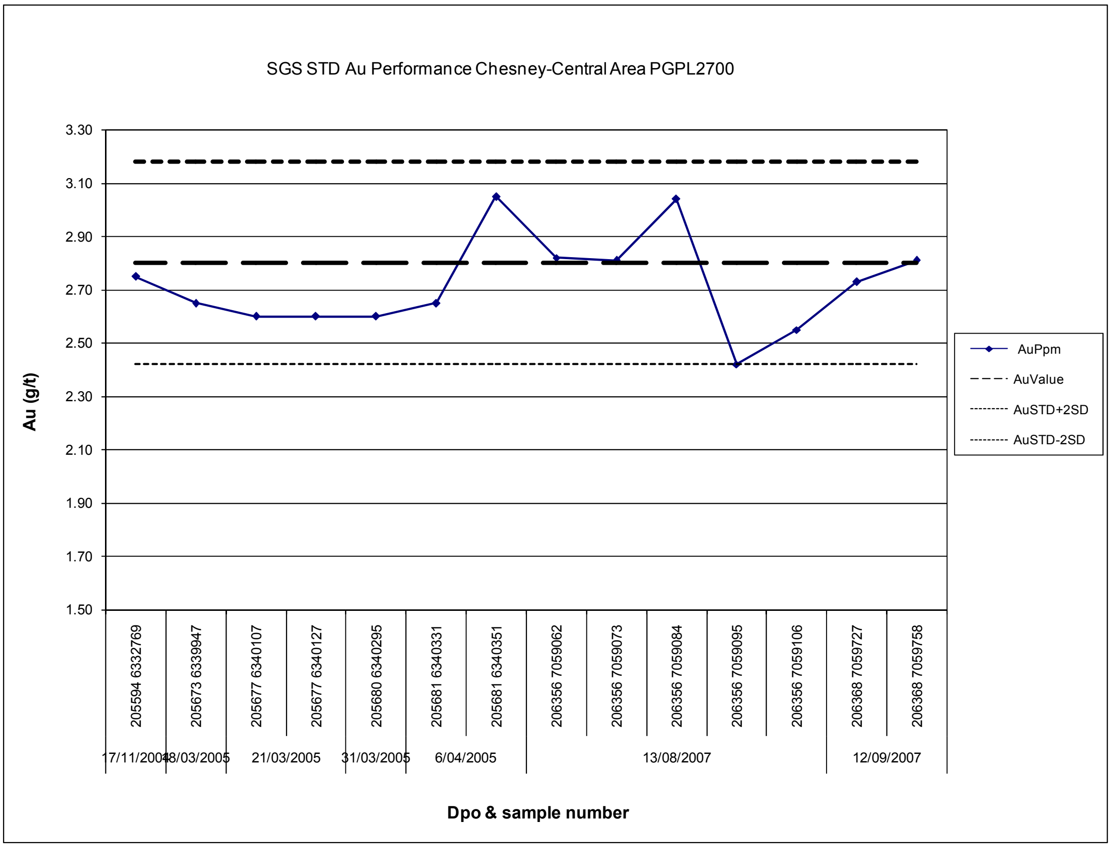


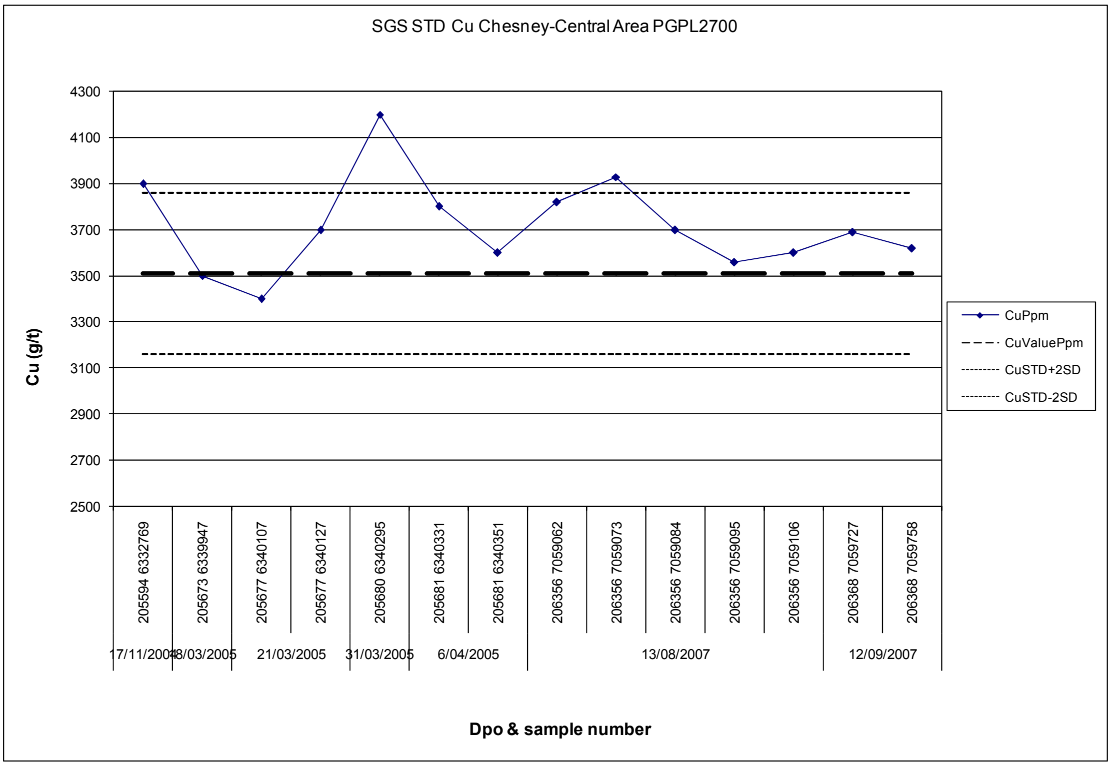
Appendix A - Standards



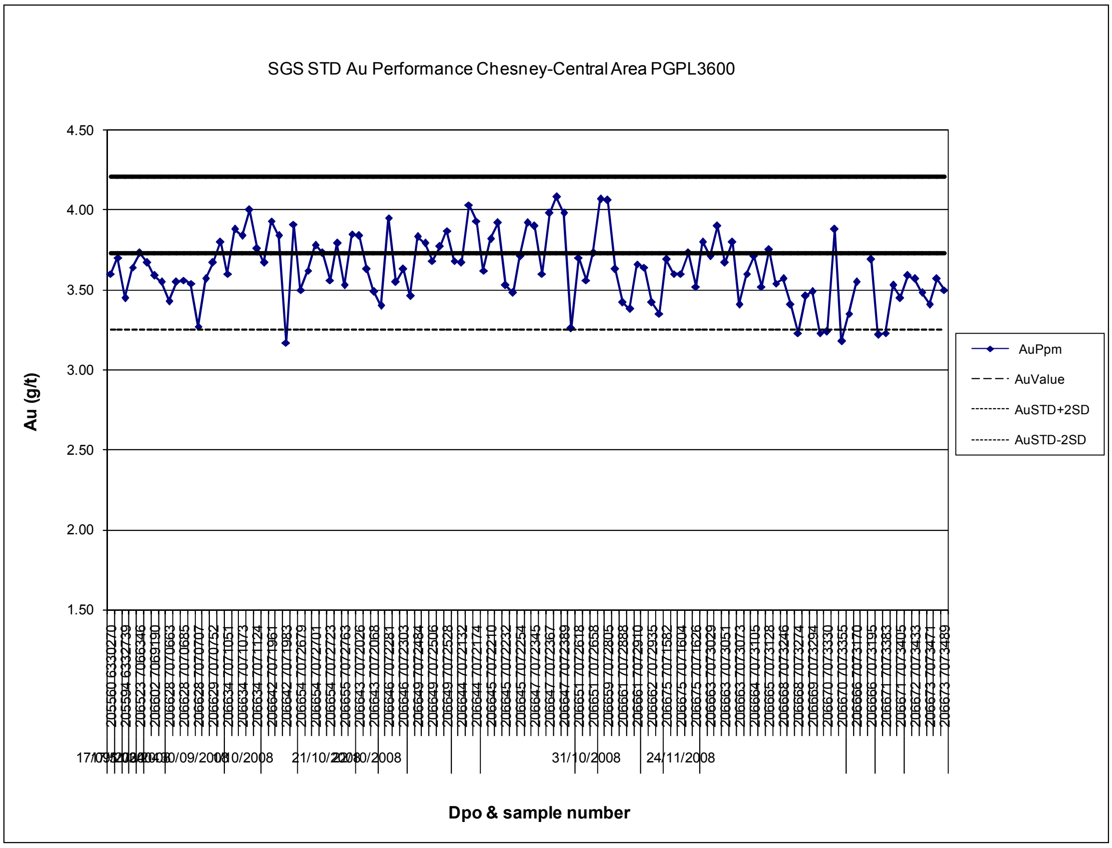


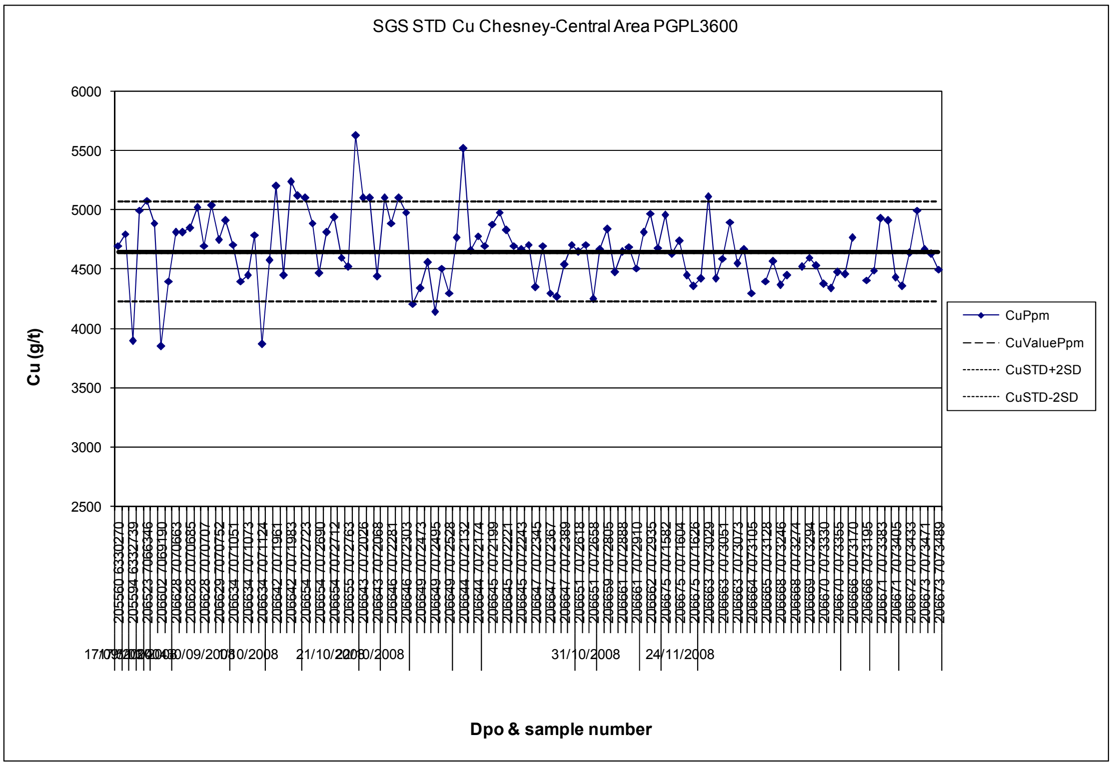
Appendix A - Standards



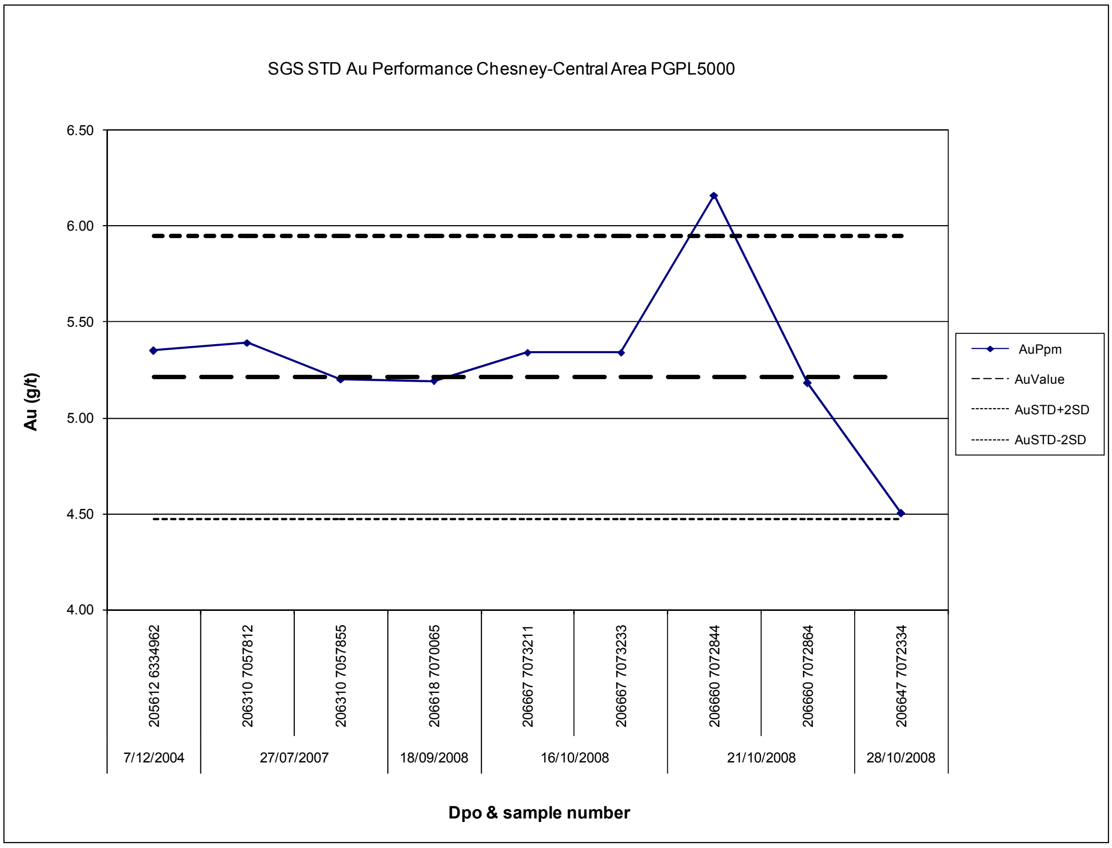


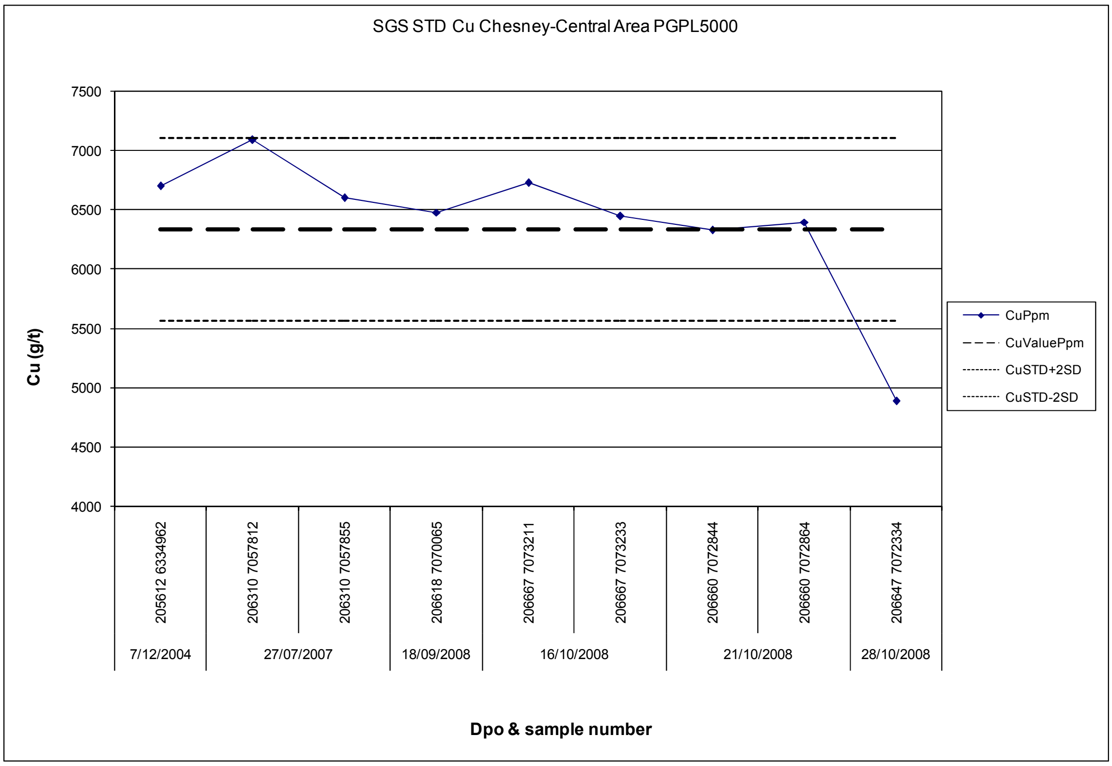
Appendix A - Standards



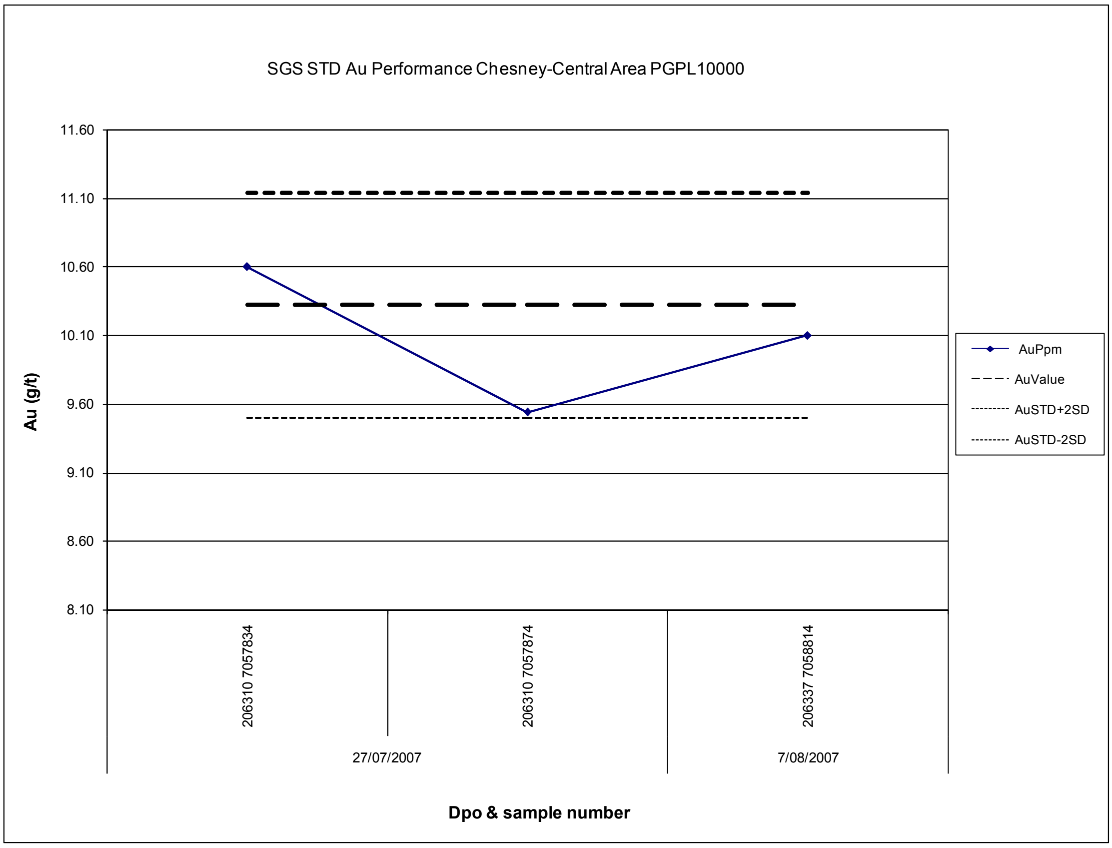


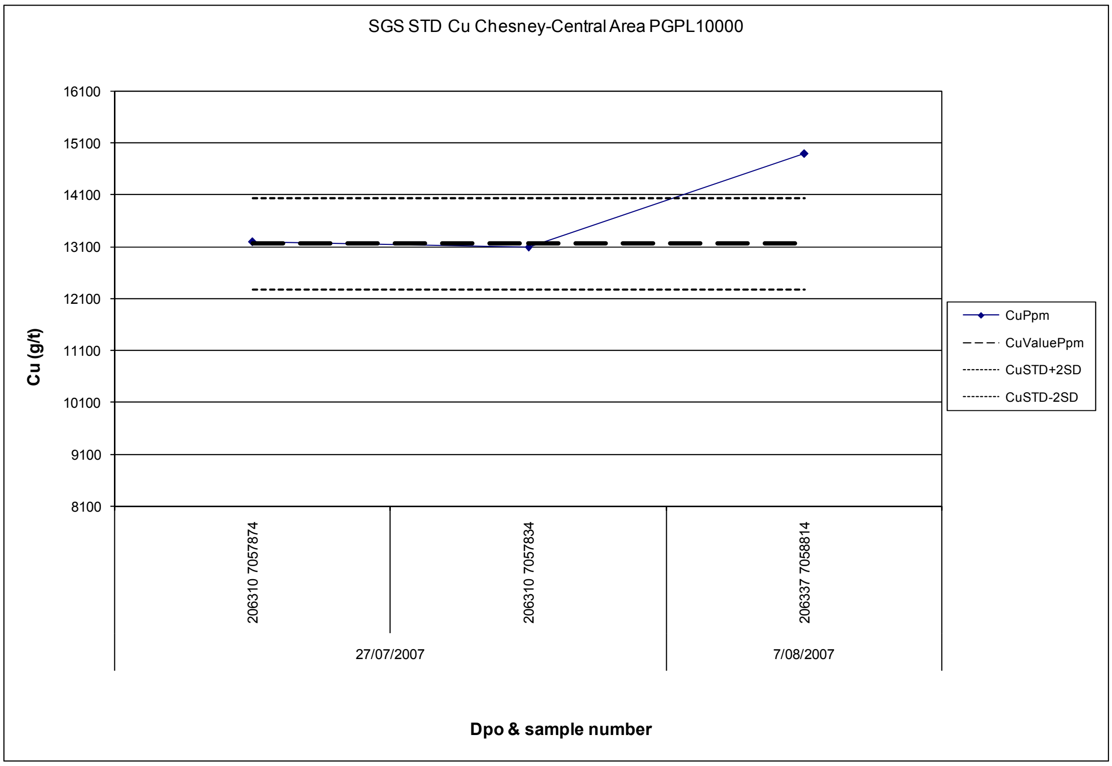
Appendix A - Standards



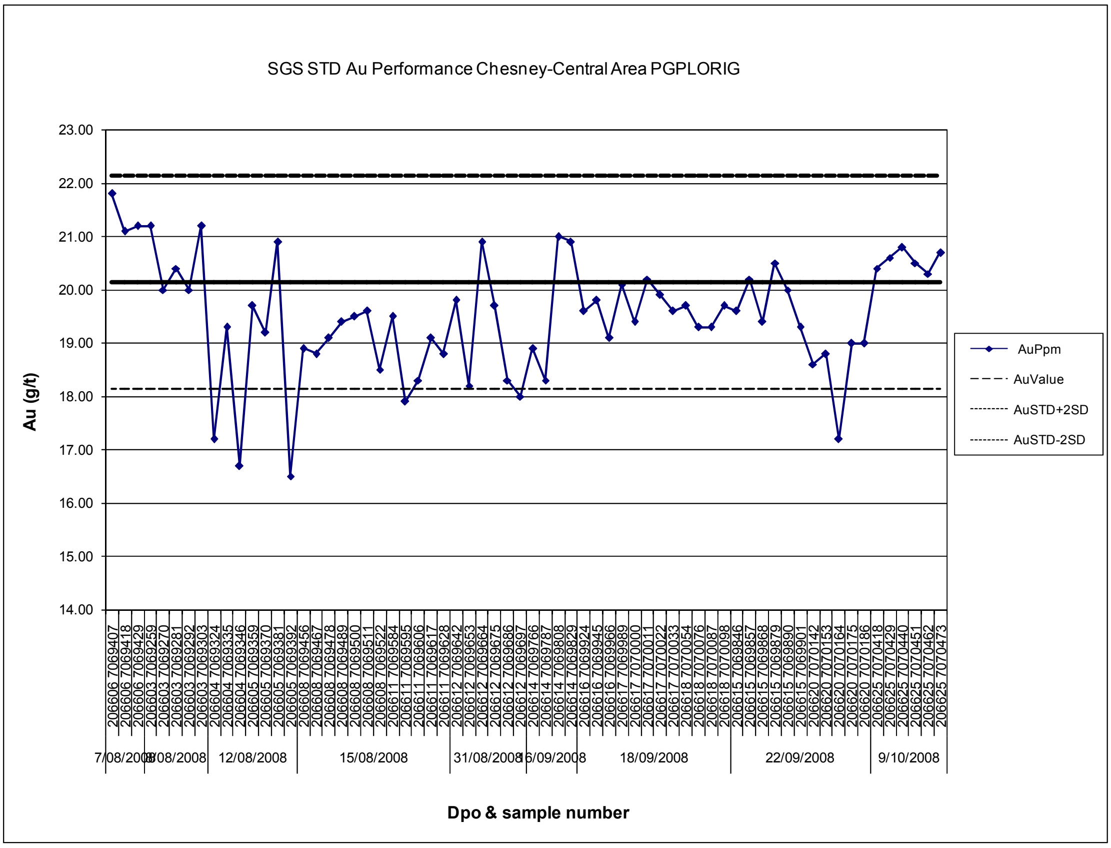


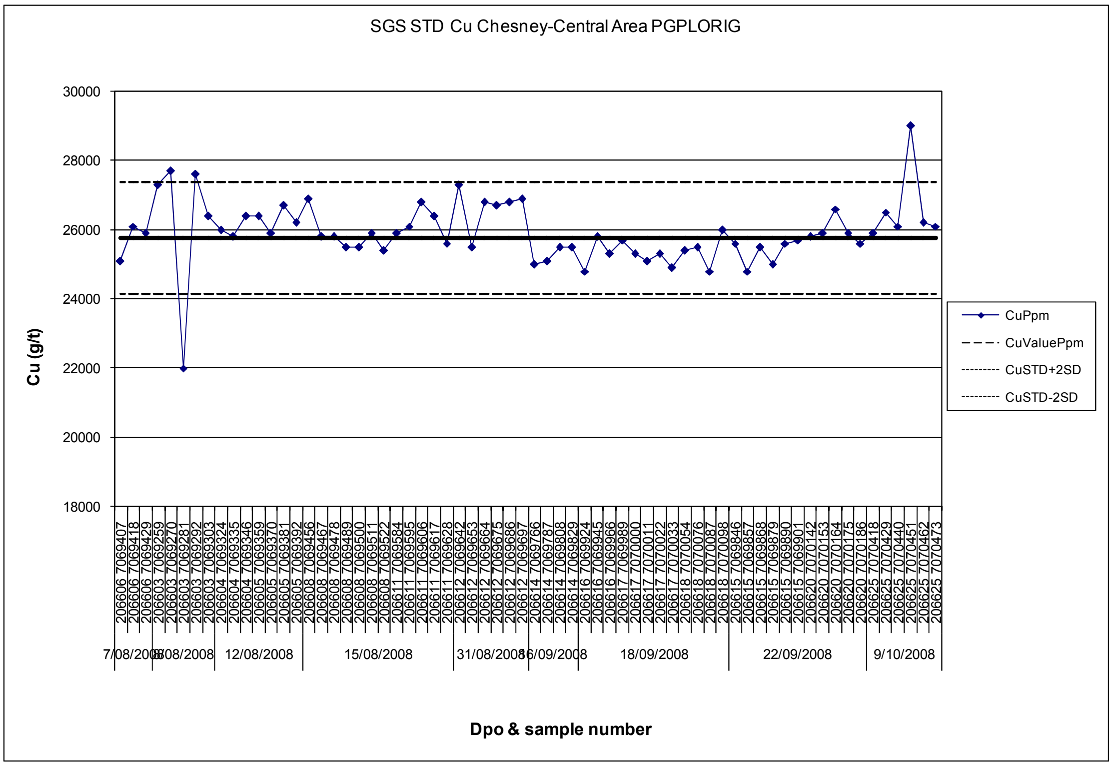
Appendix A - Standards



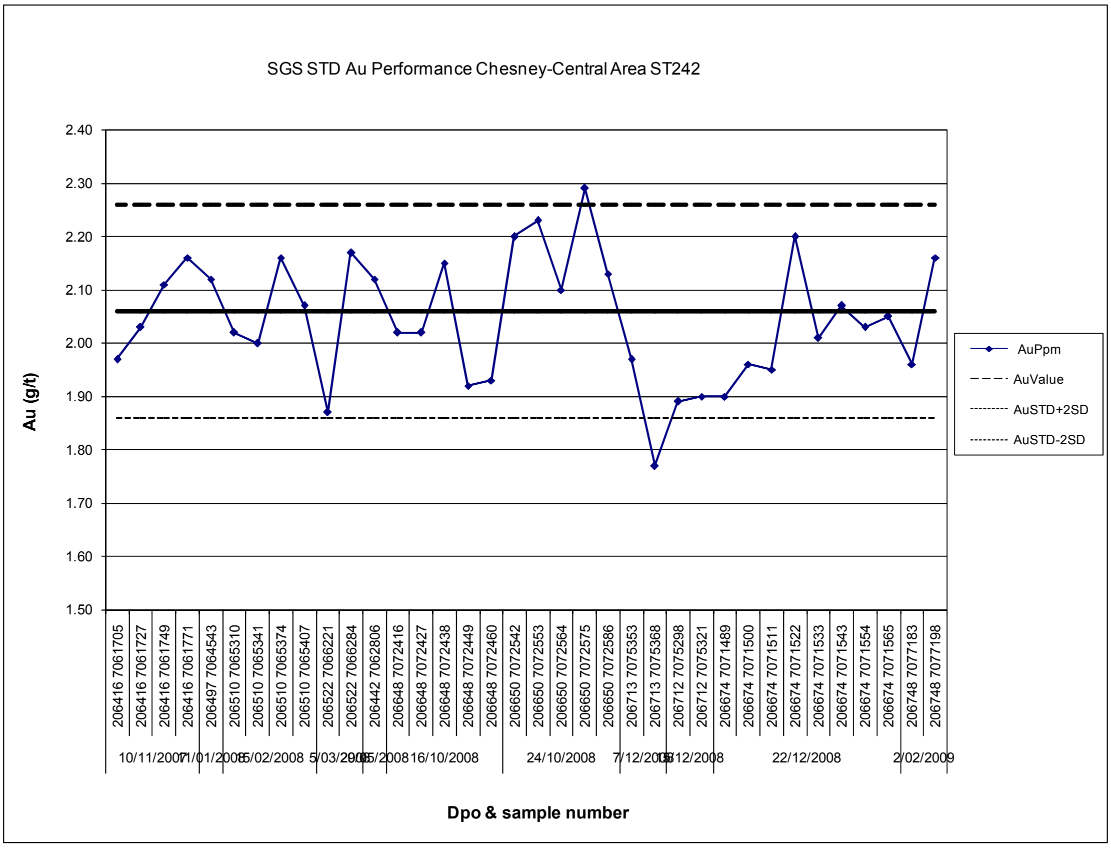


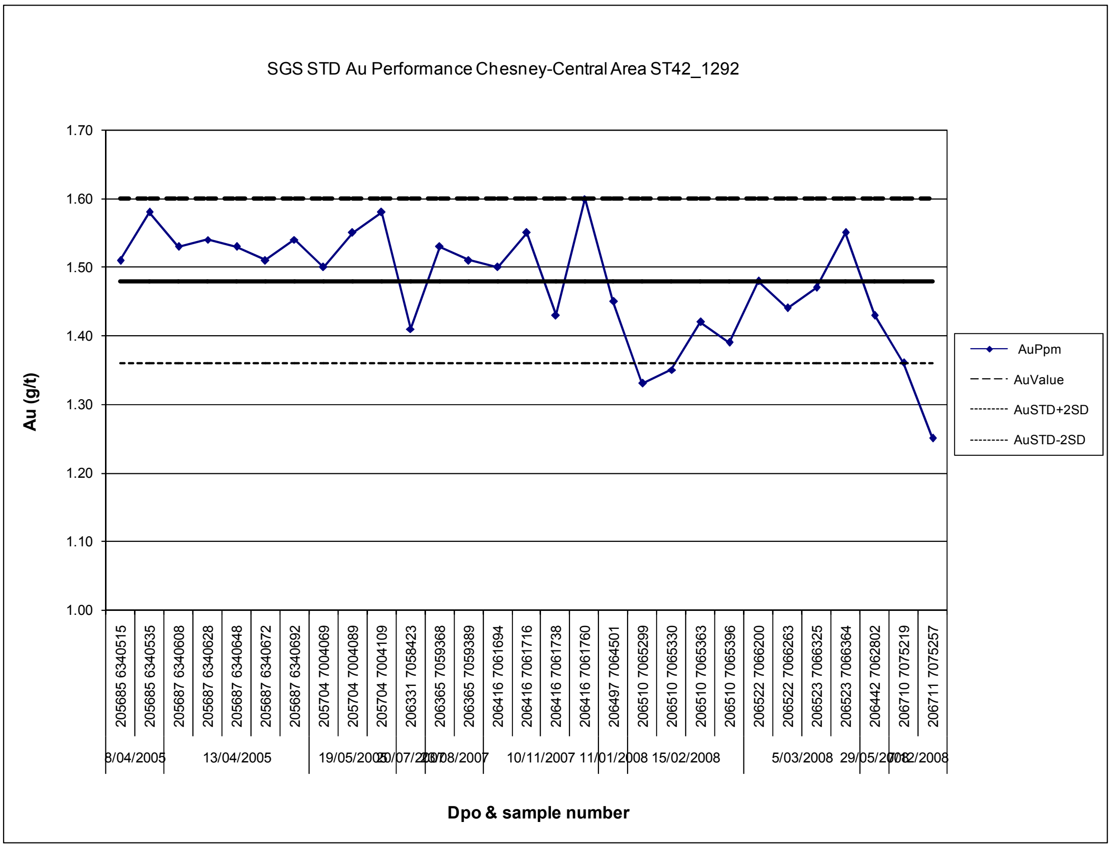
Appendix A - Standards



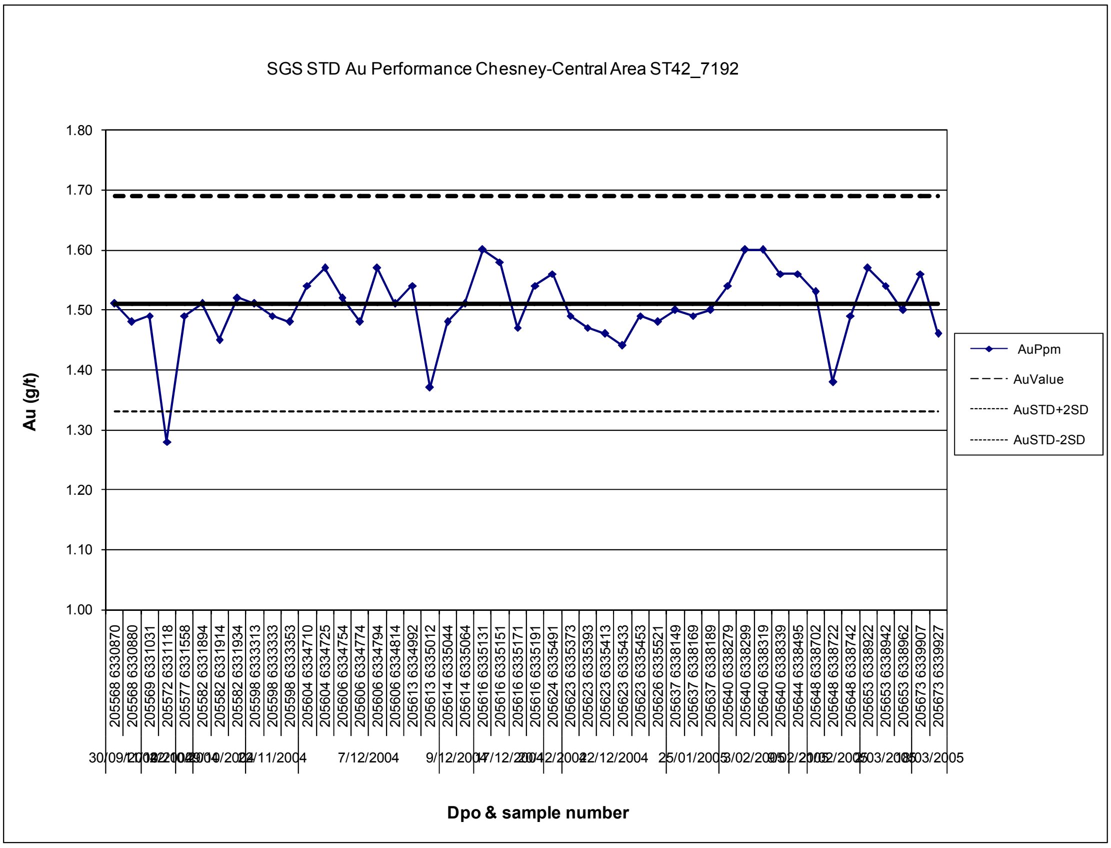


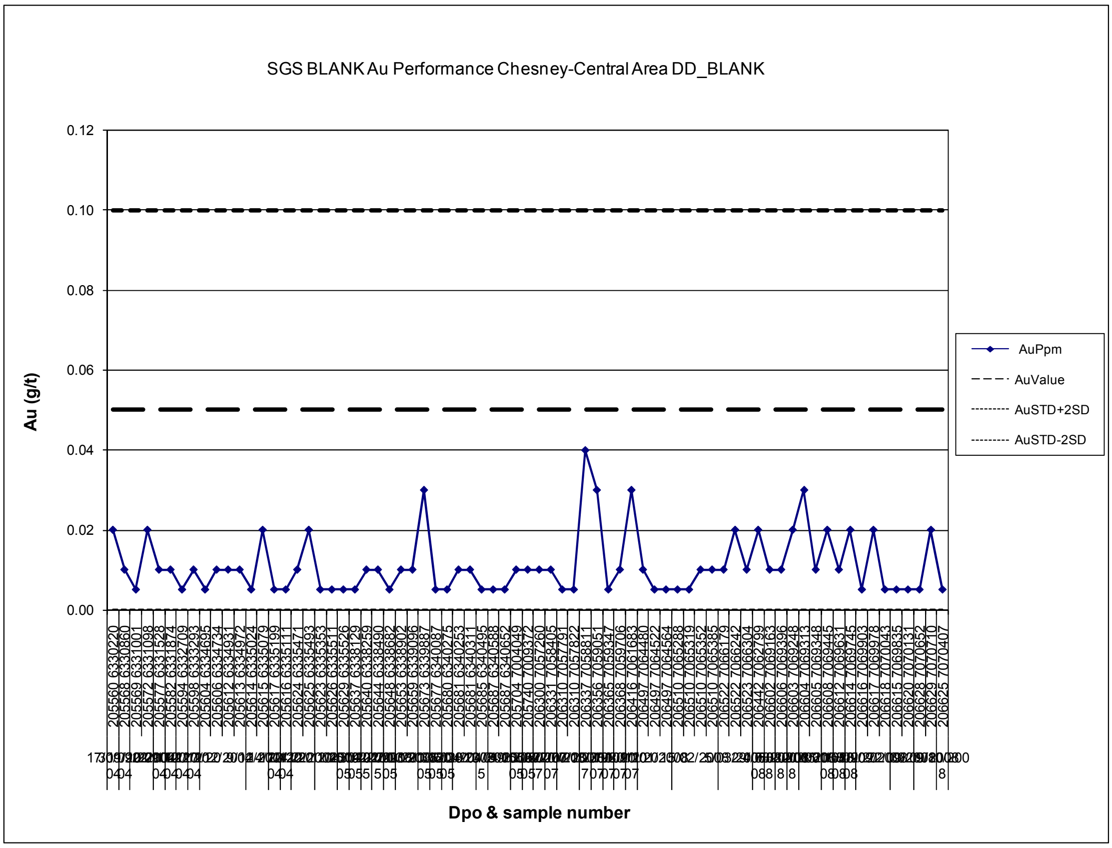
Appendix A - Standards



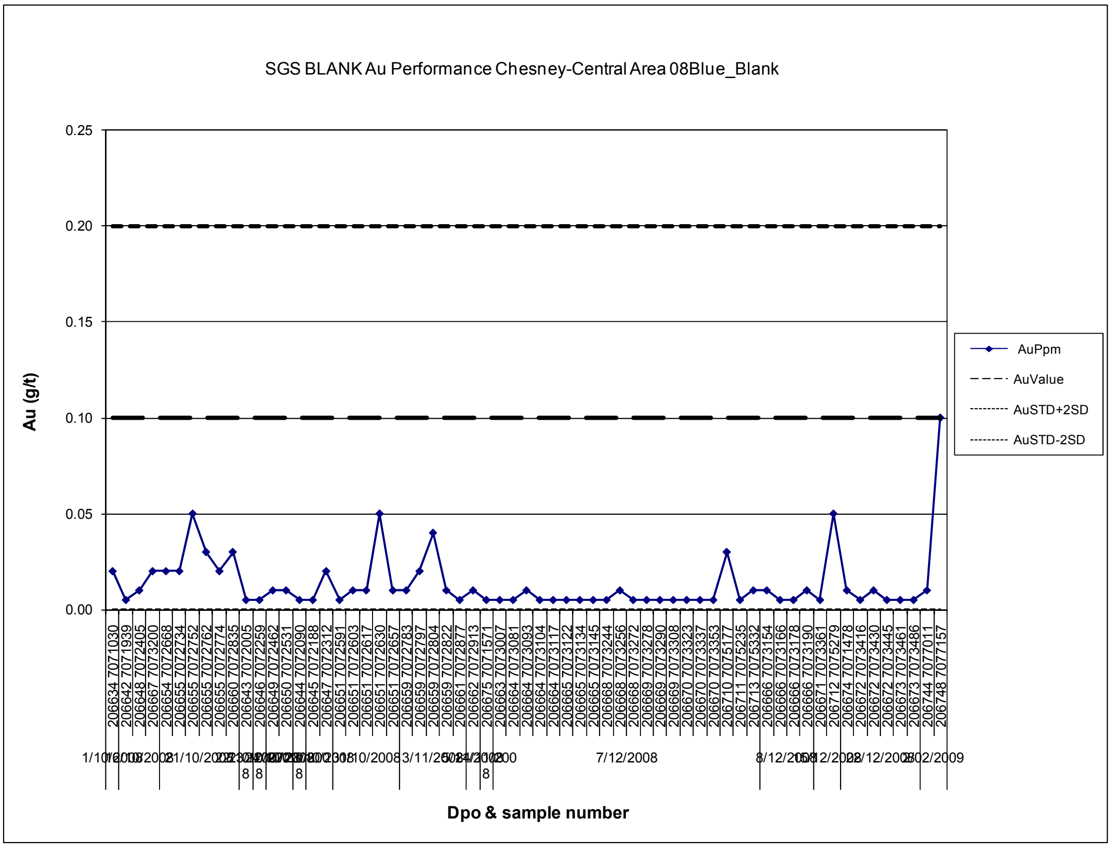


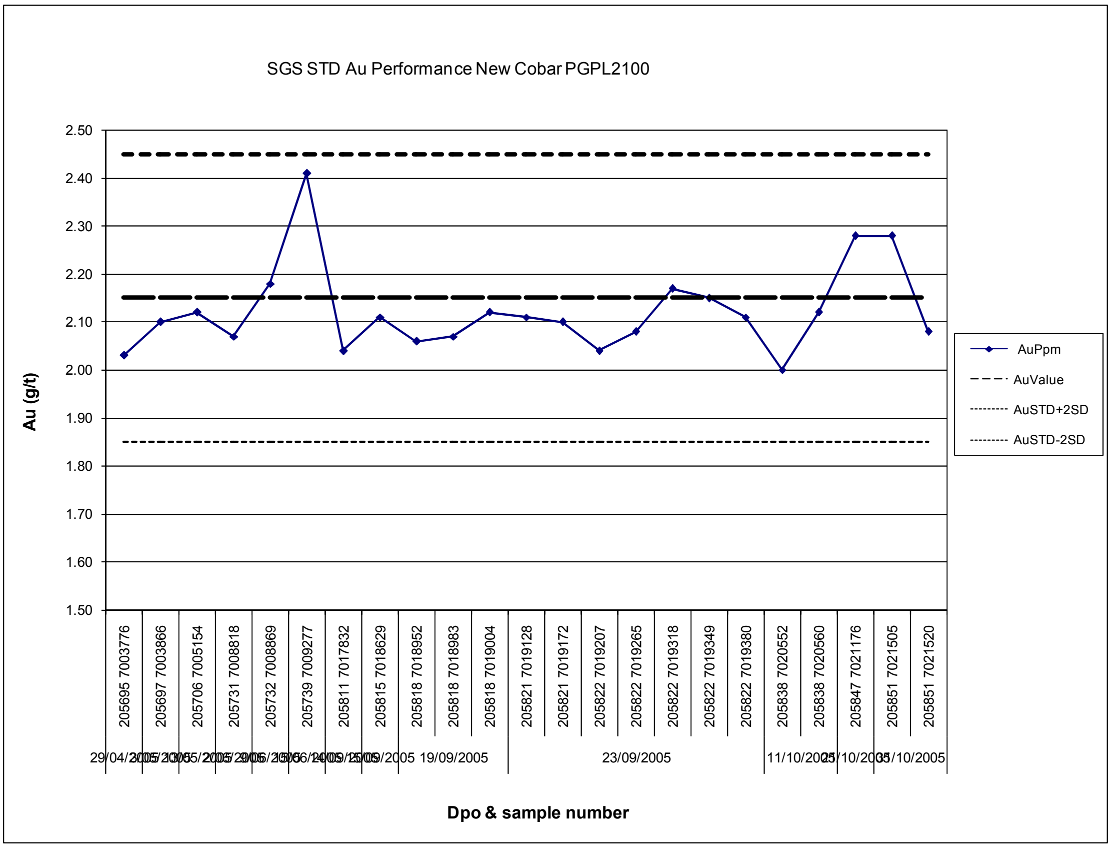
Appendix A - Standards



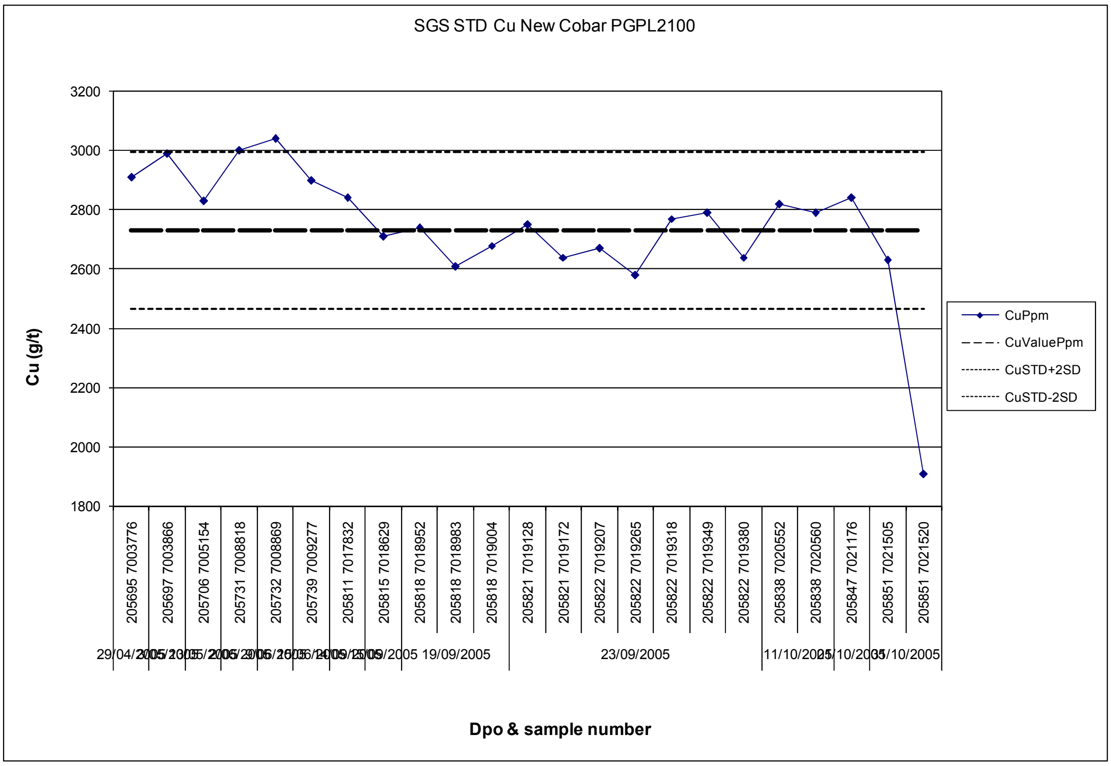


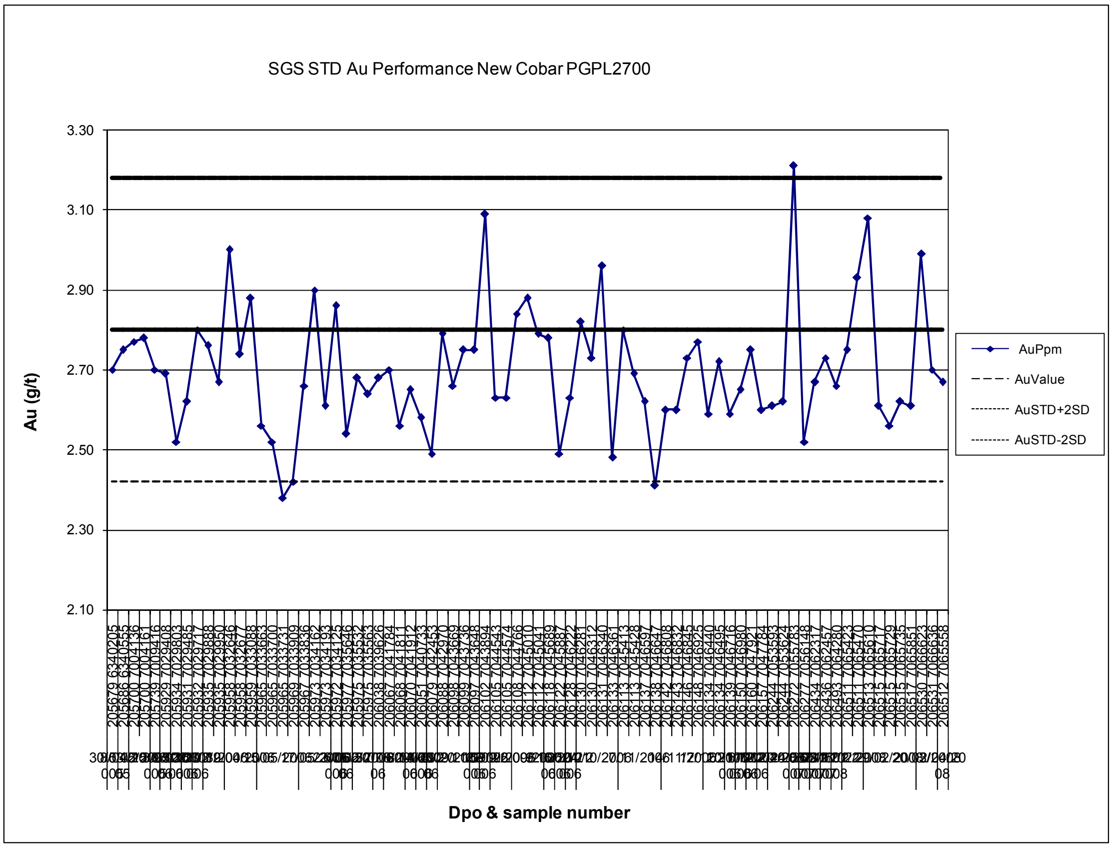
Appendix A - Standards



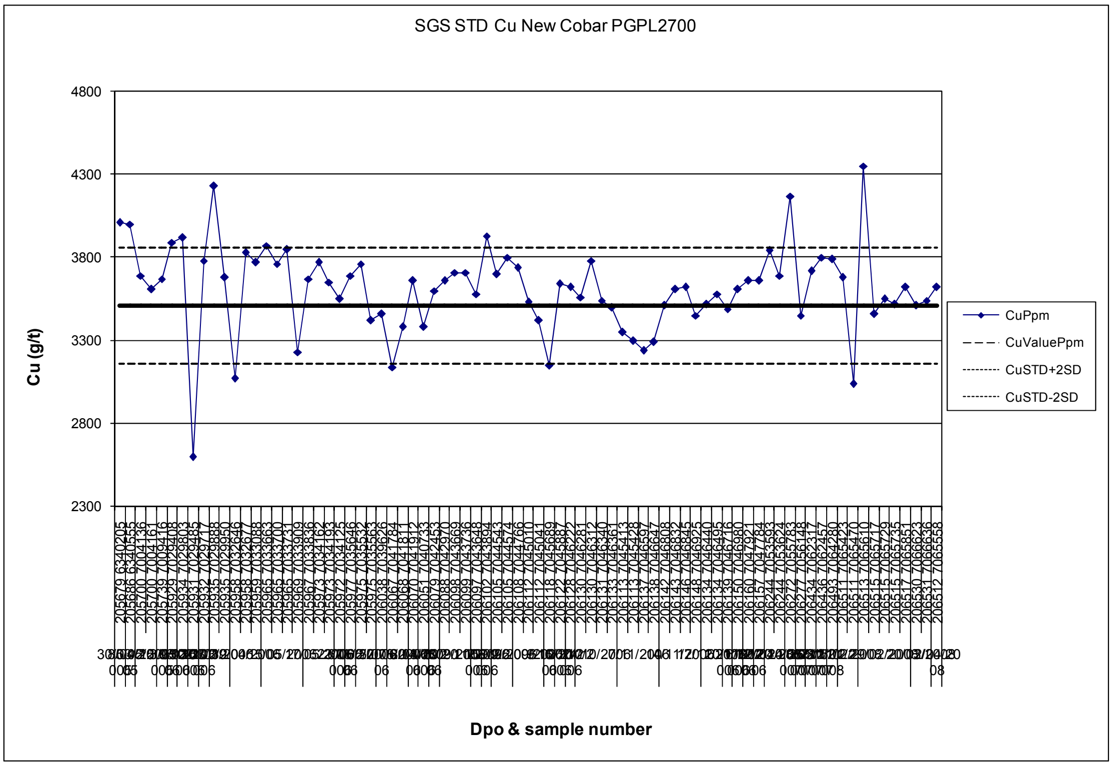


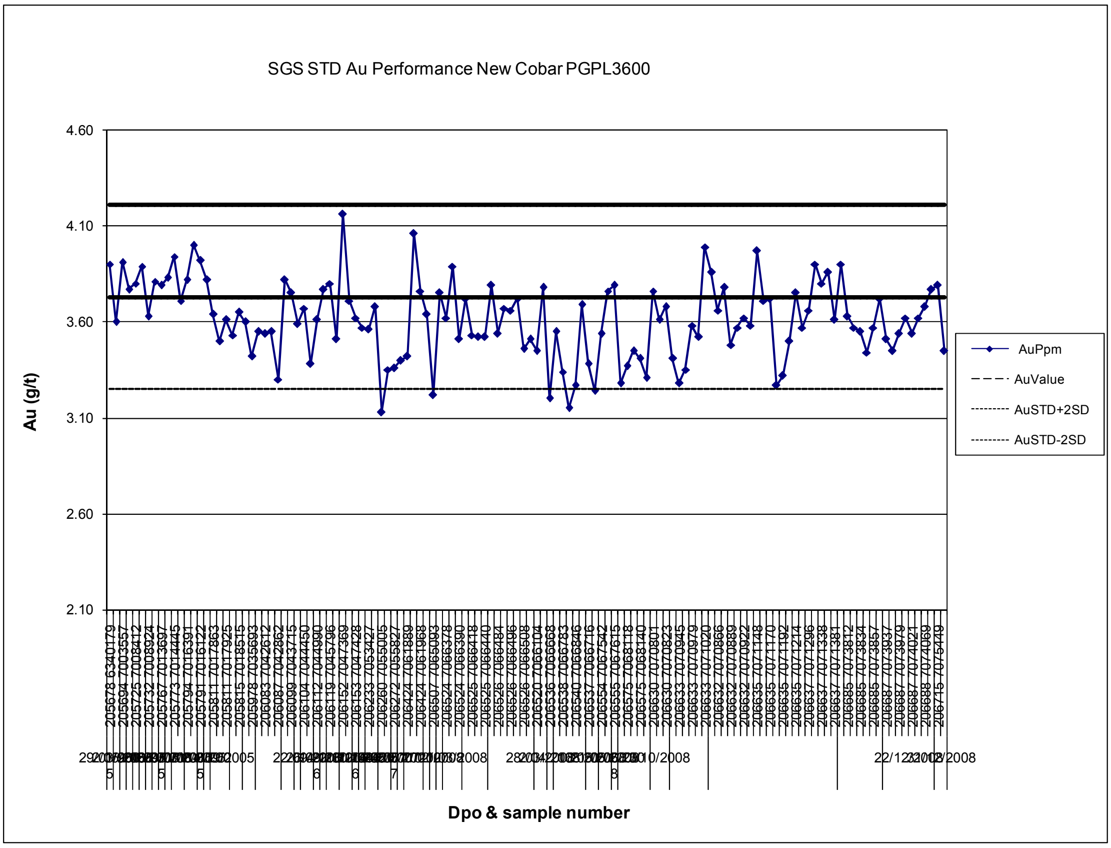
Appendix A - Standards



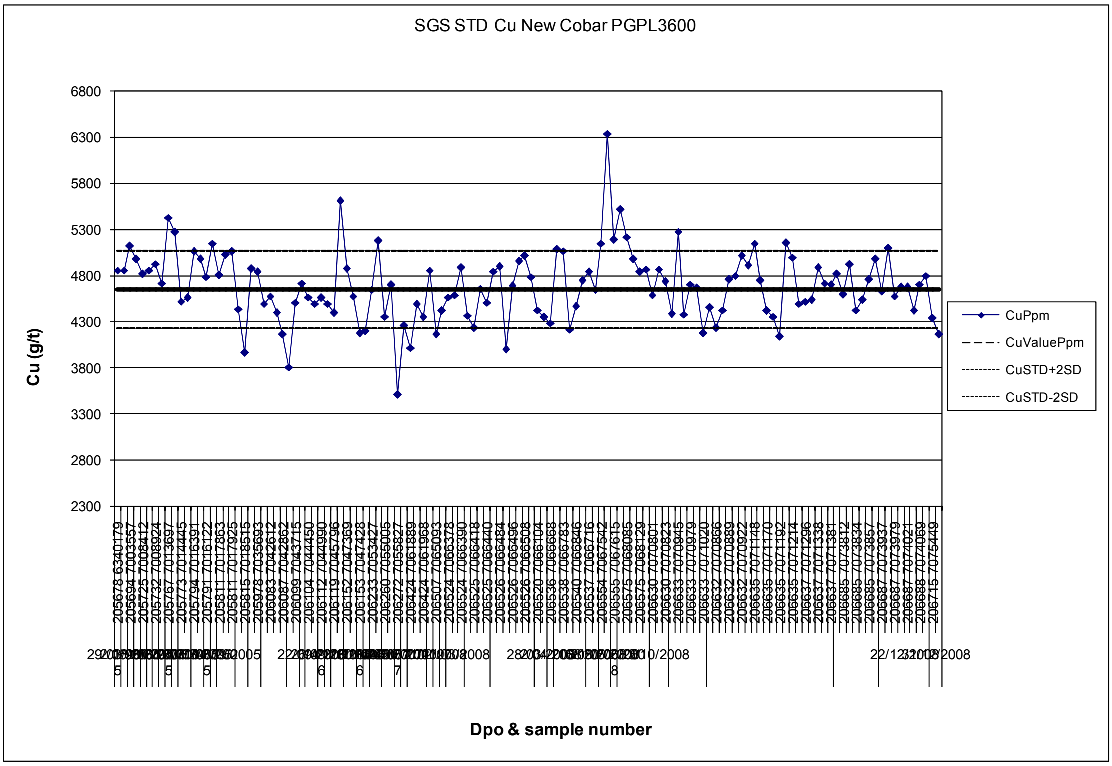


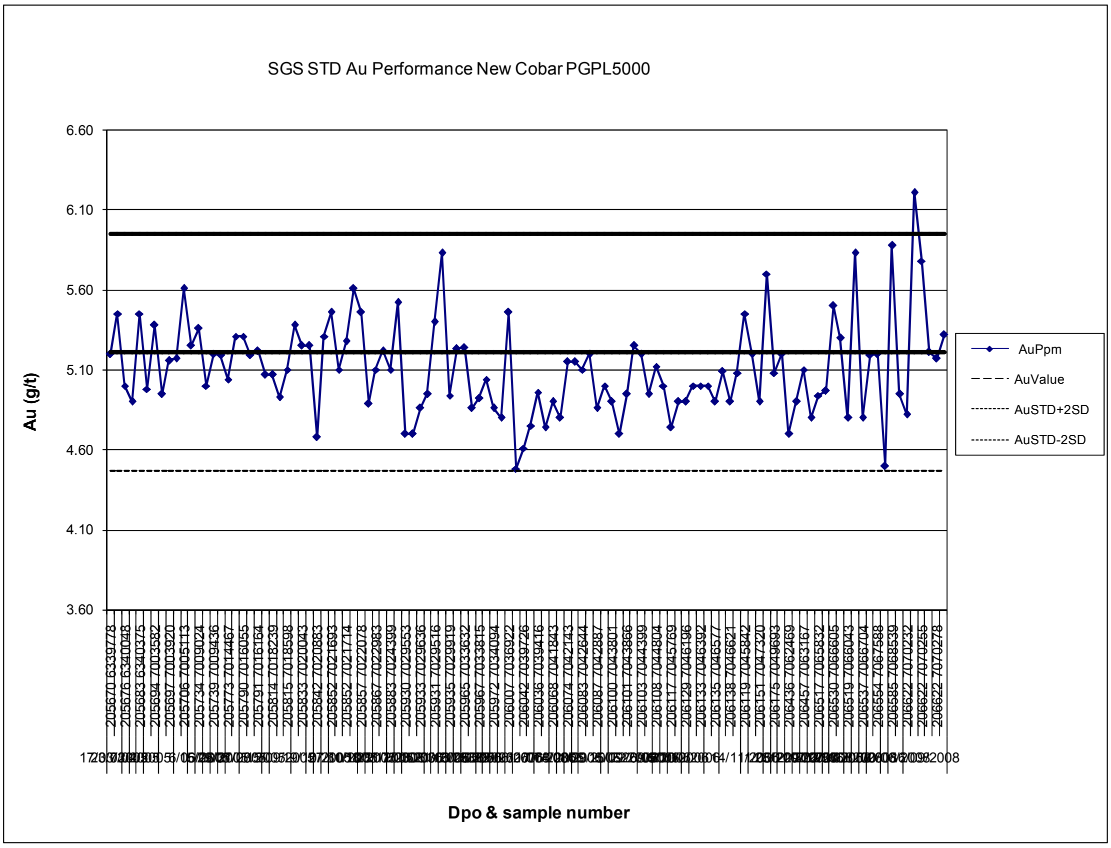
Appendix A - Standards



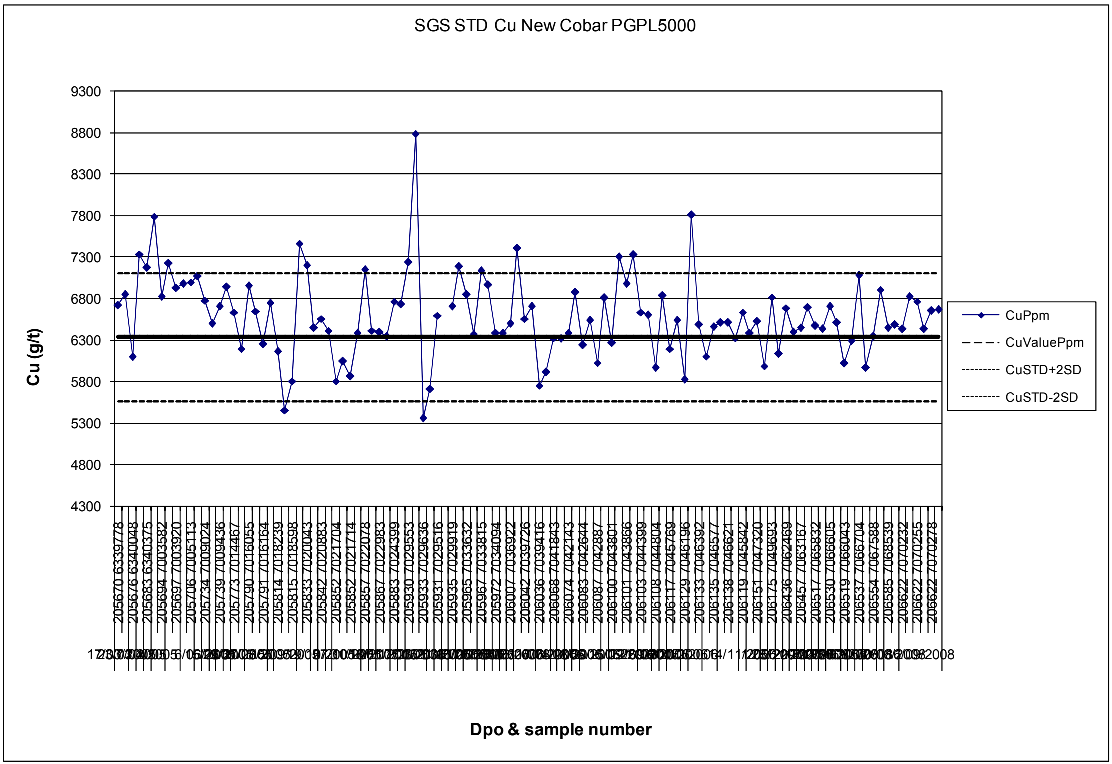


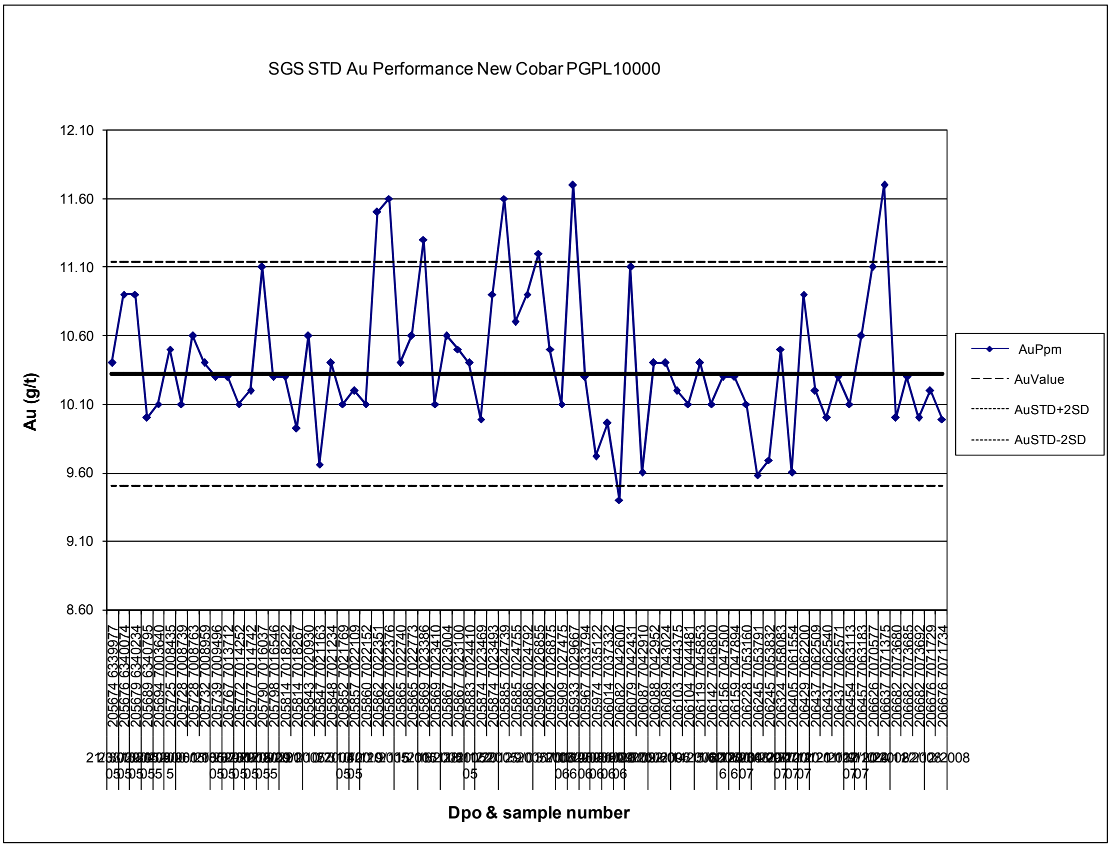
Appendix A - Standards



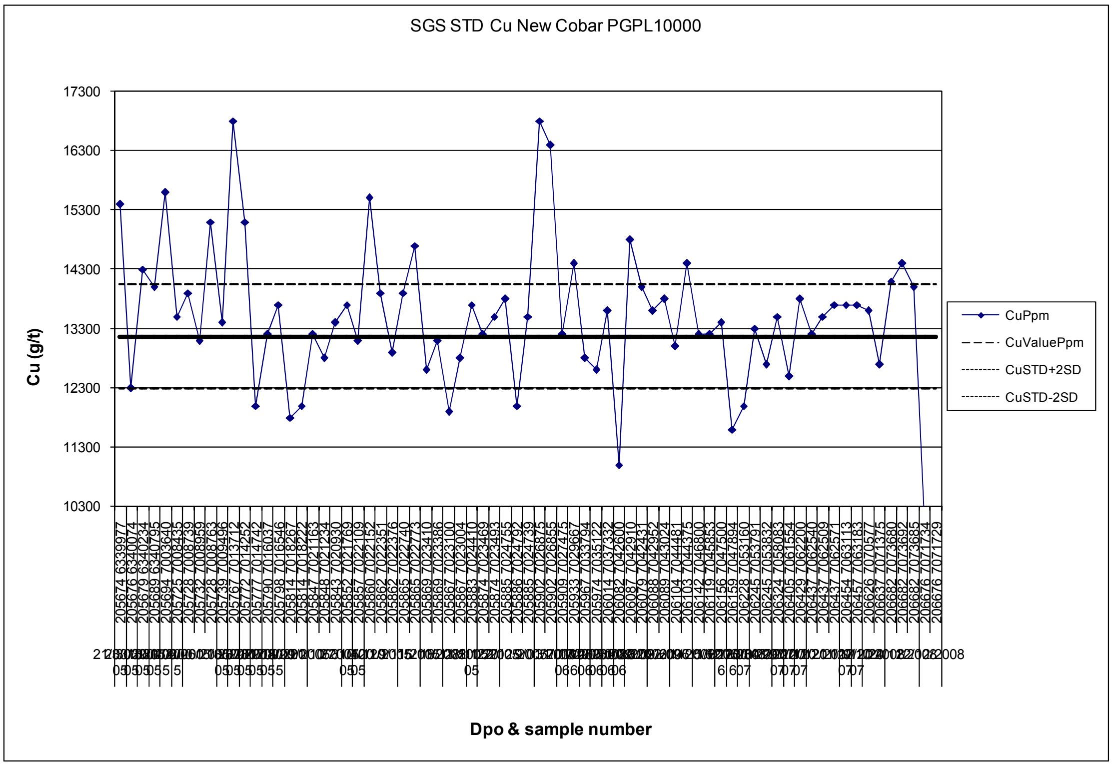


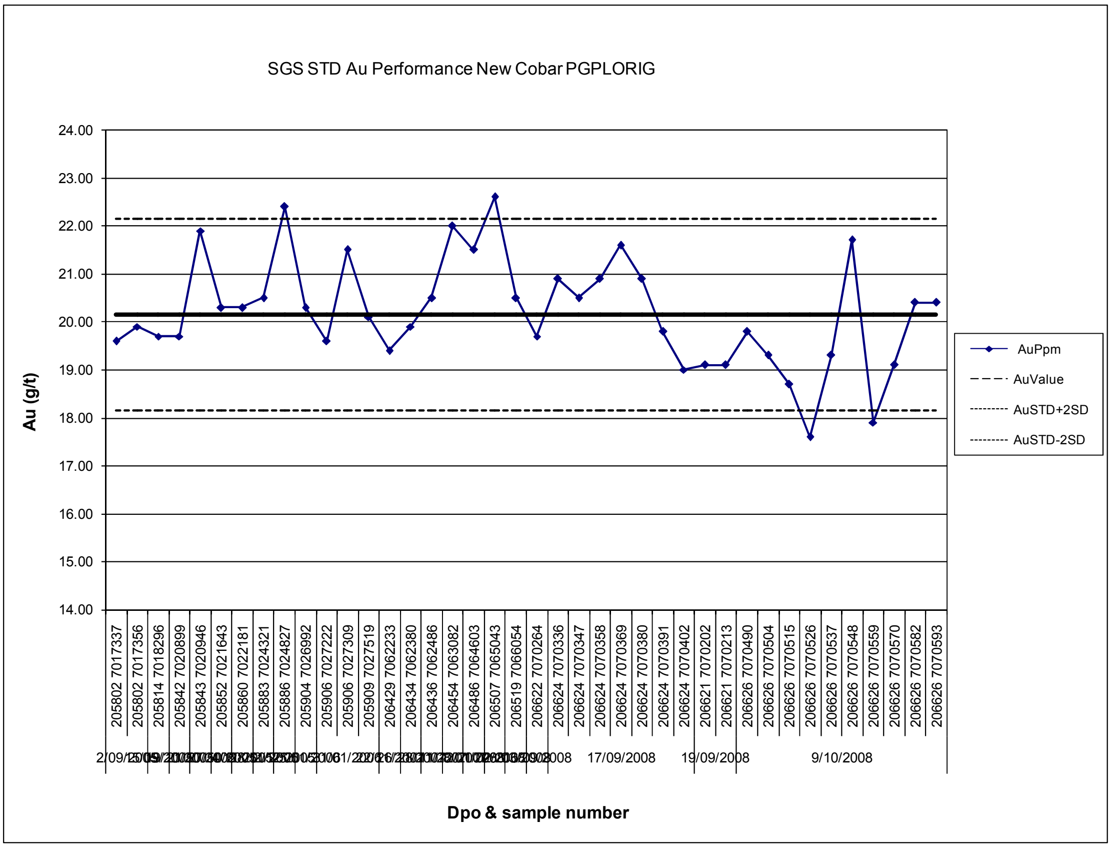
Appendix A - Standards



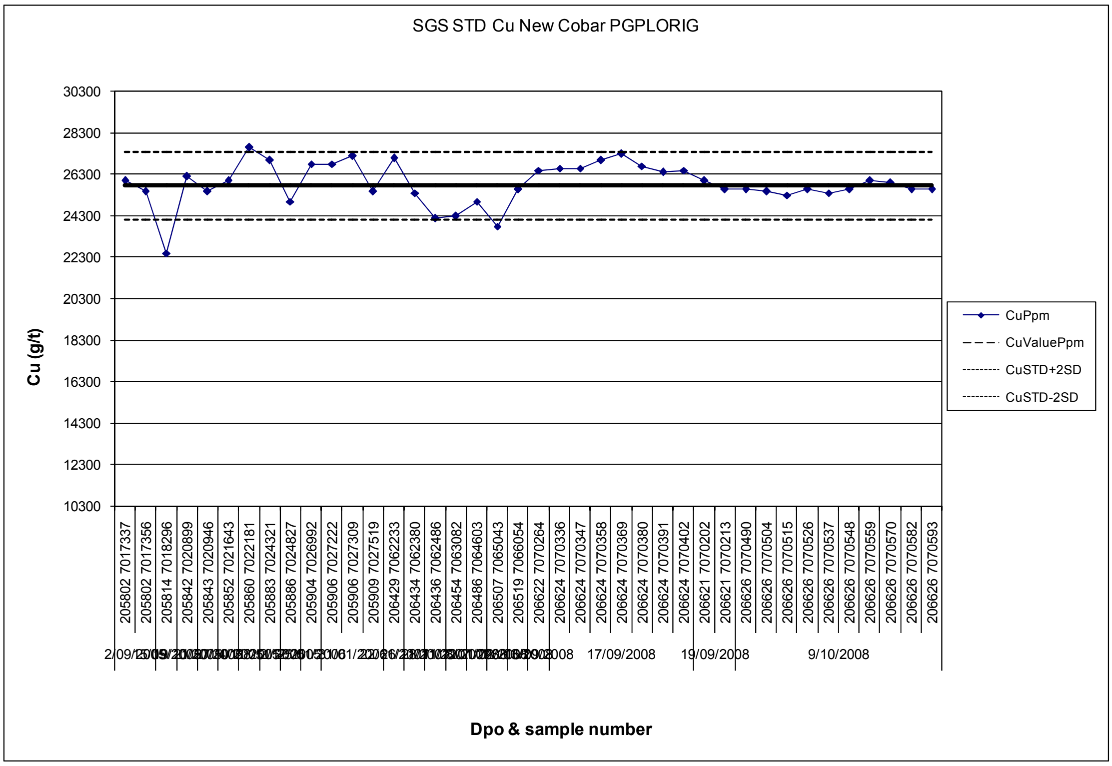


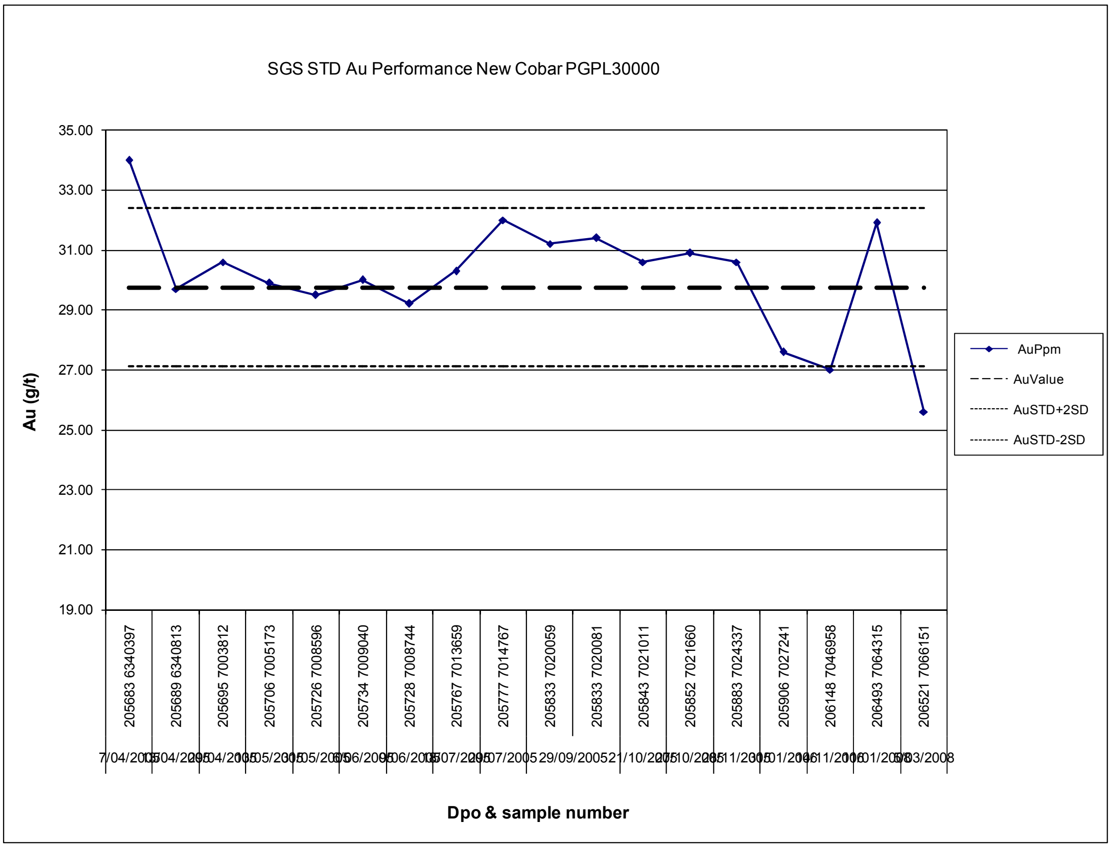
Appendix A - Standards



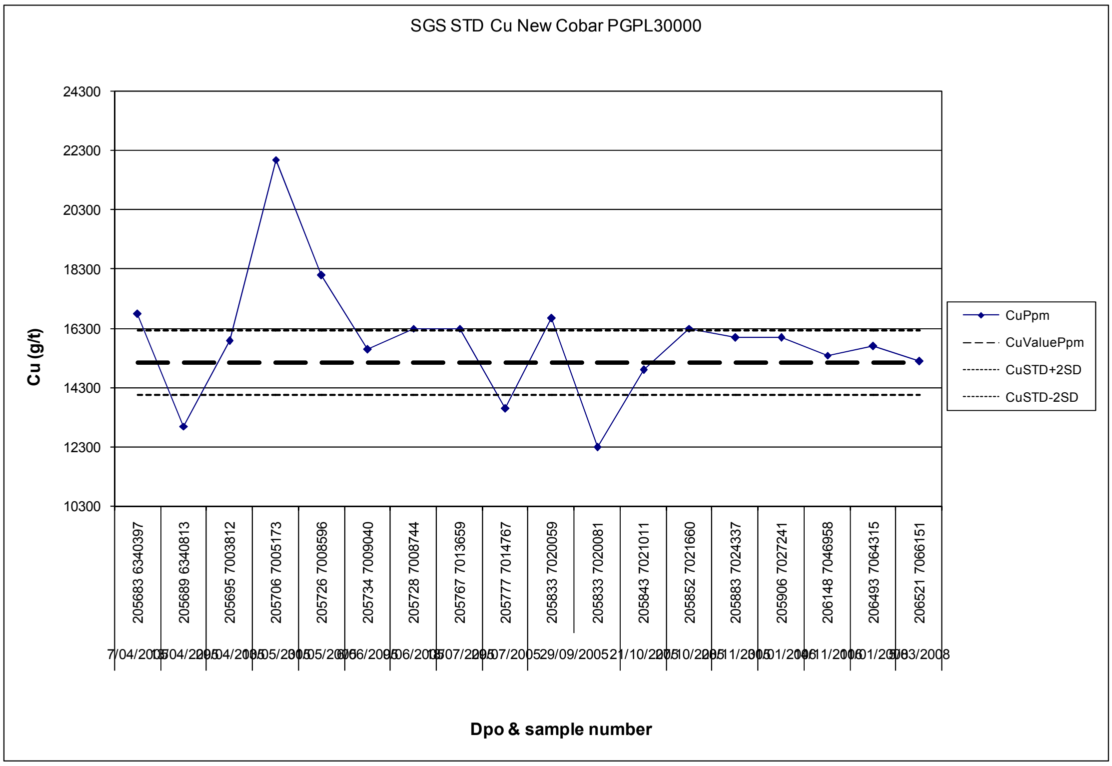


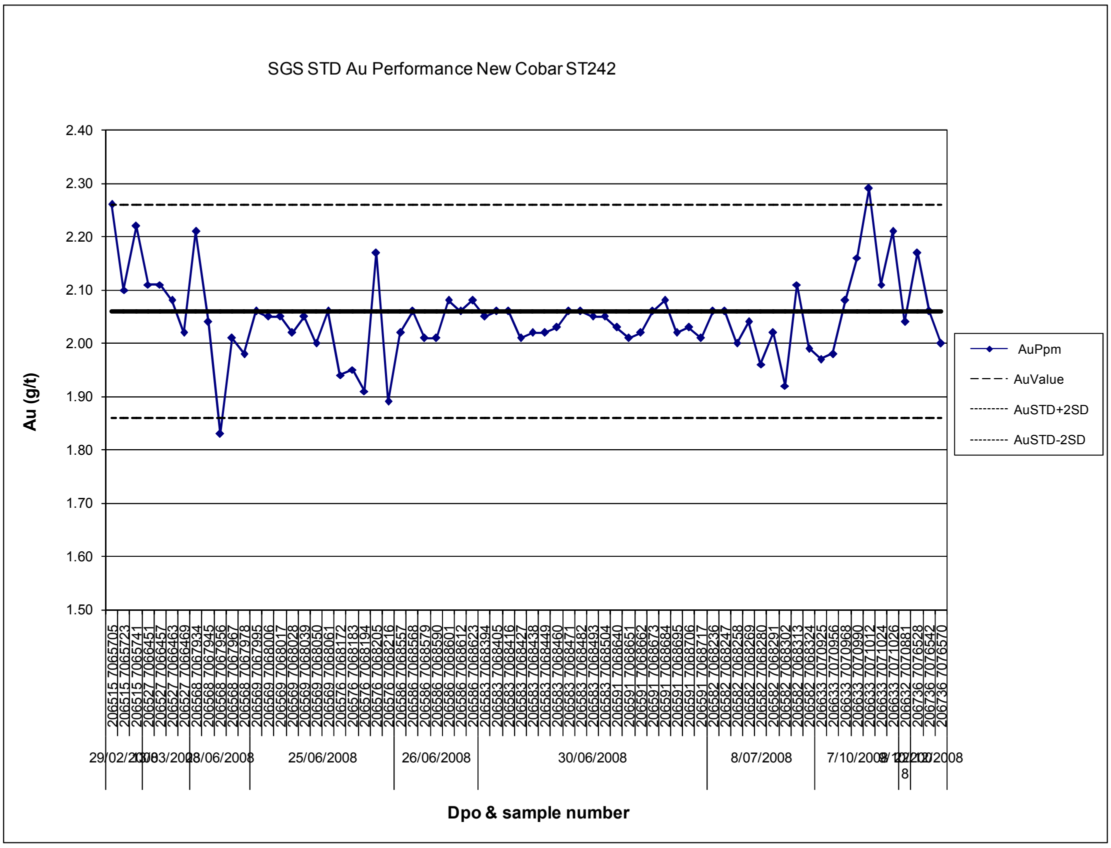
Appendix A - Standards



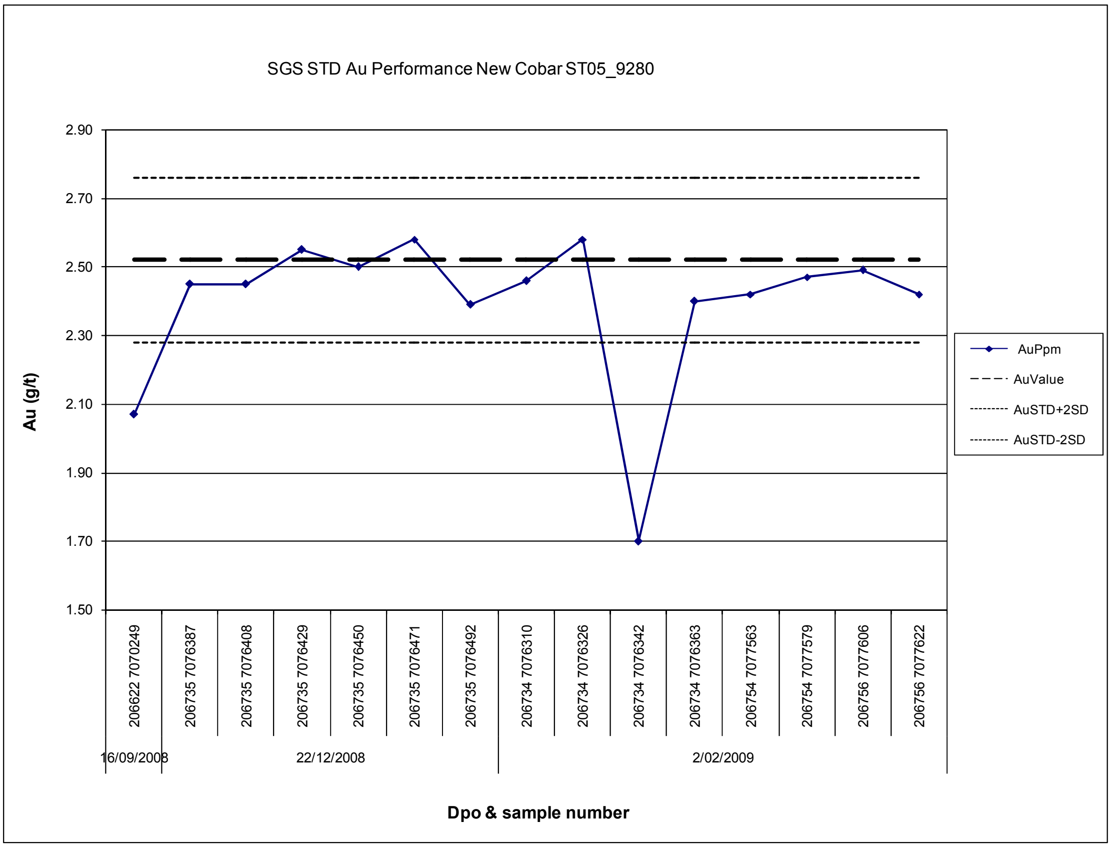


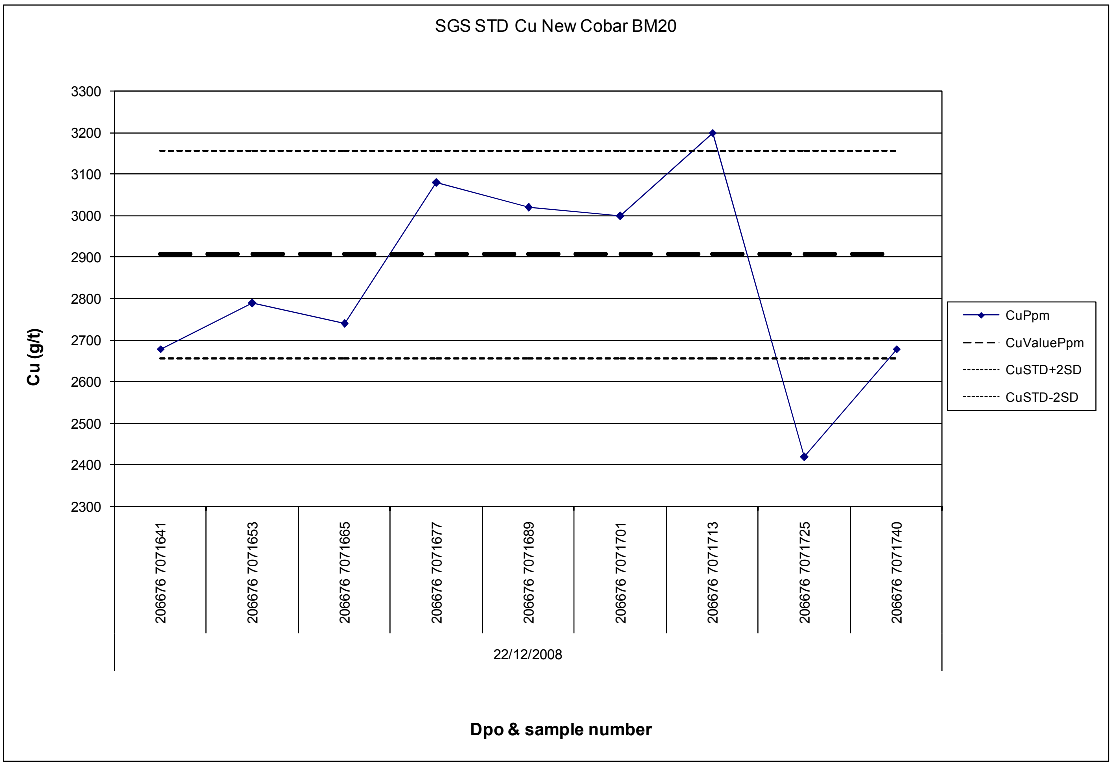
Appendix A - Standards



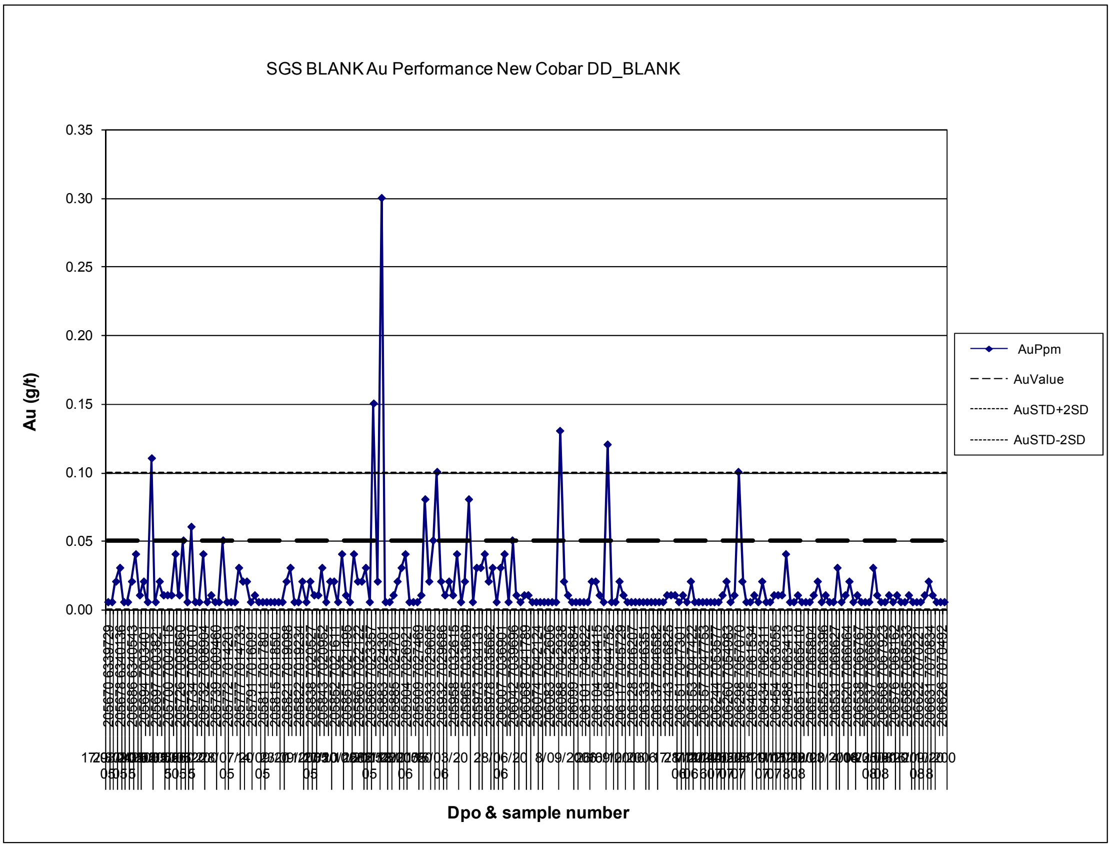


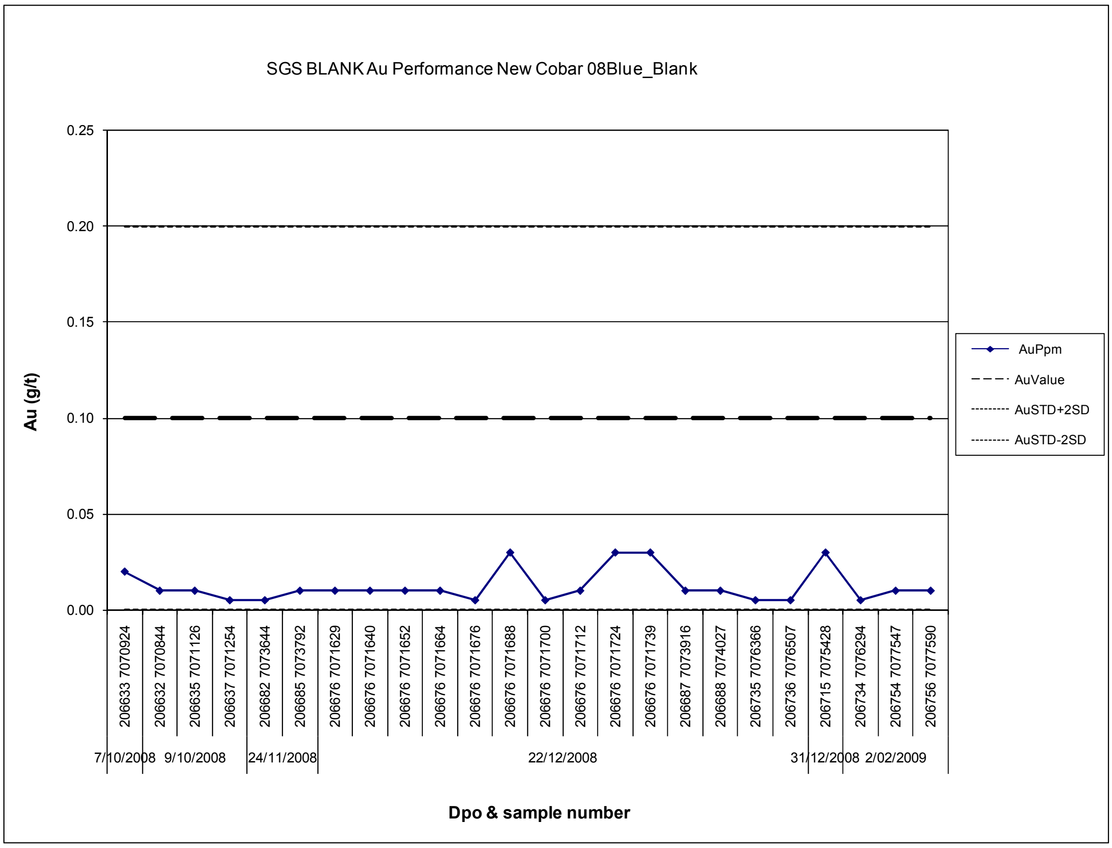
Appendix A - Standards



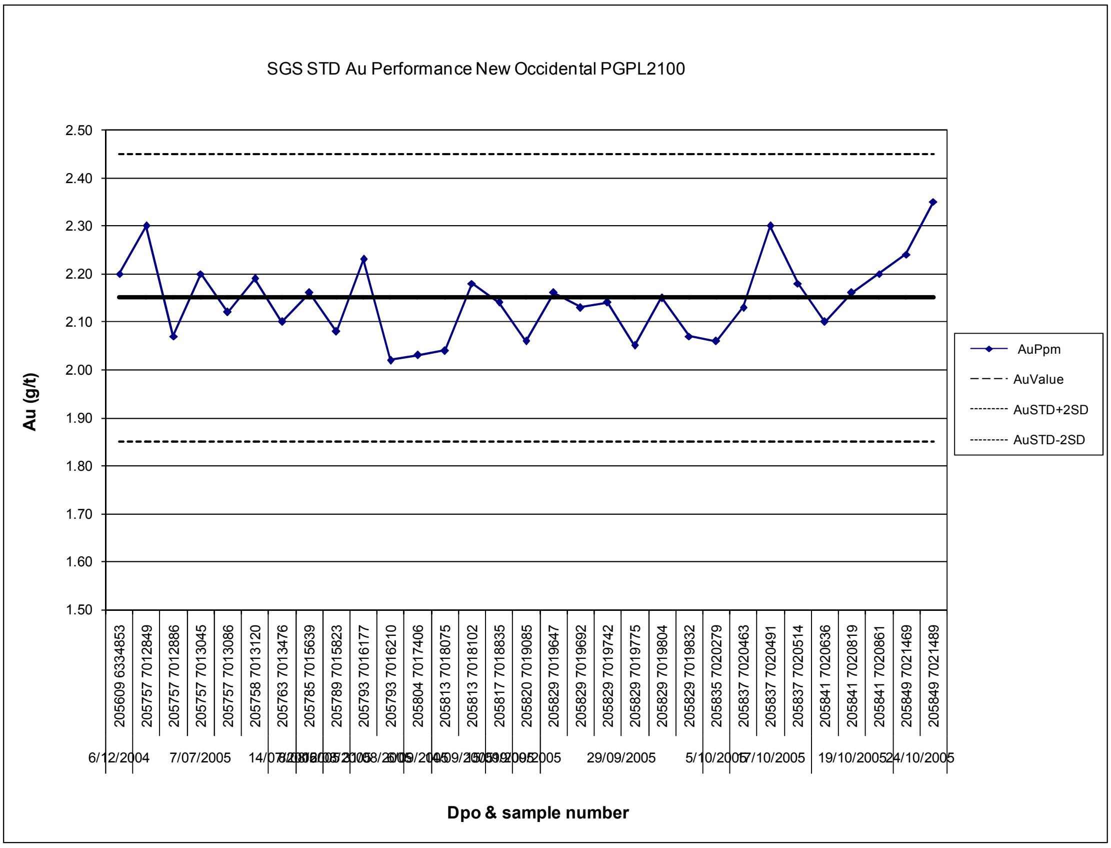


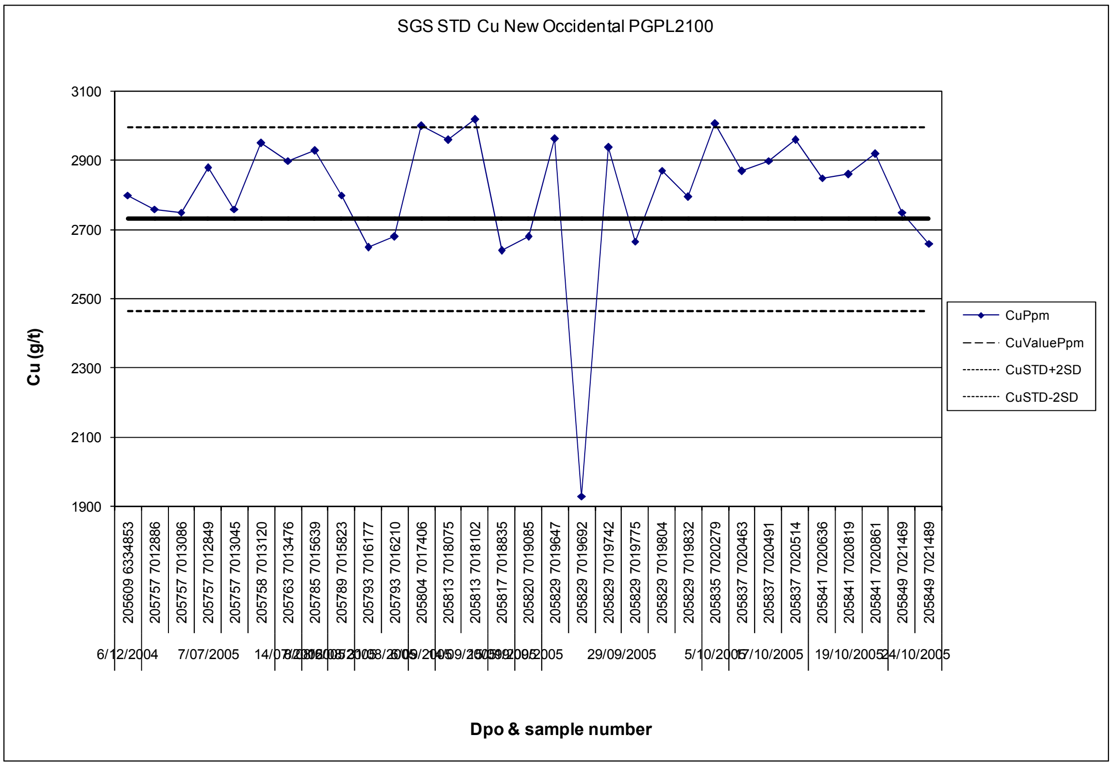
Appendix A - Standards



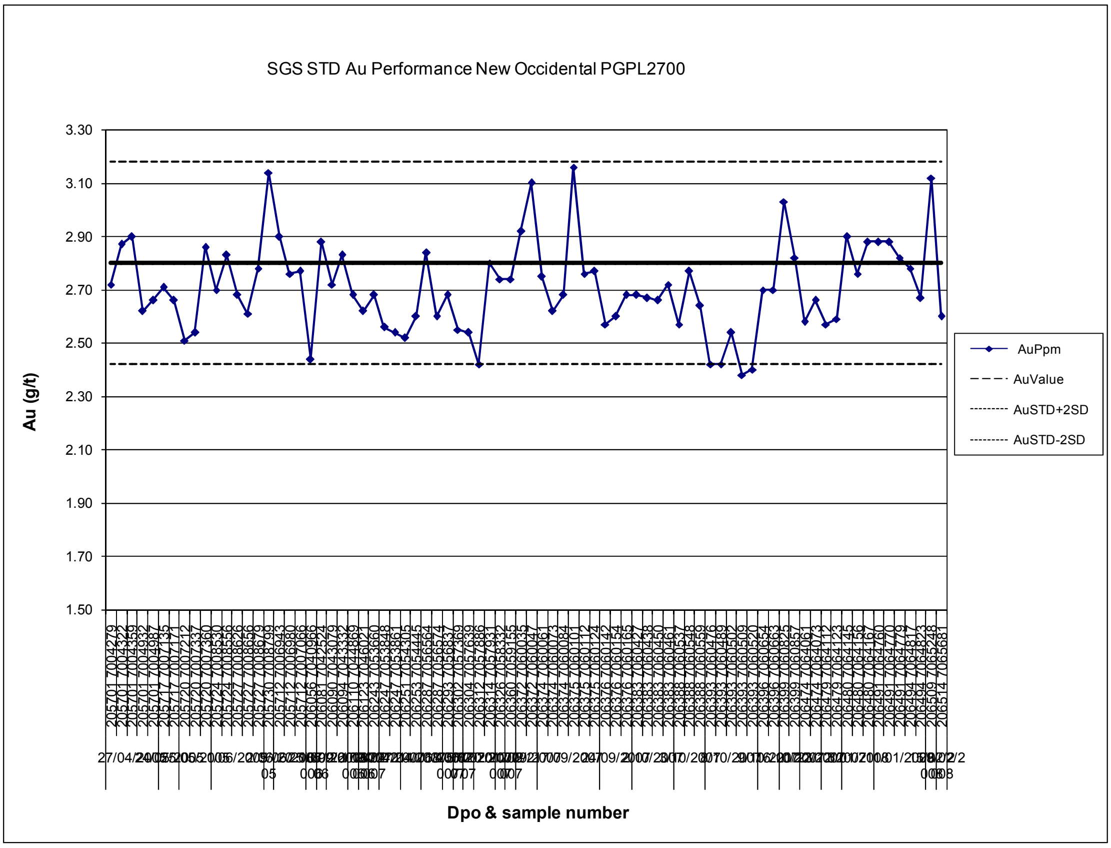


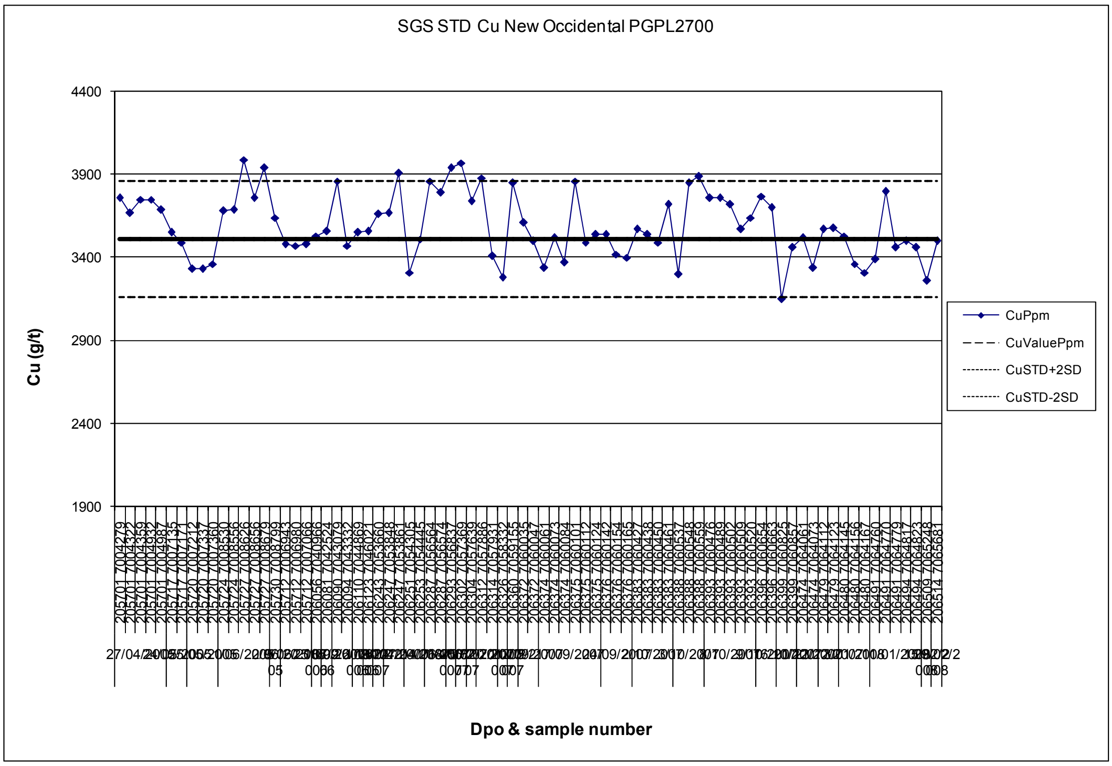
Appendix A - Standards



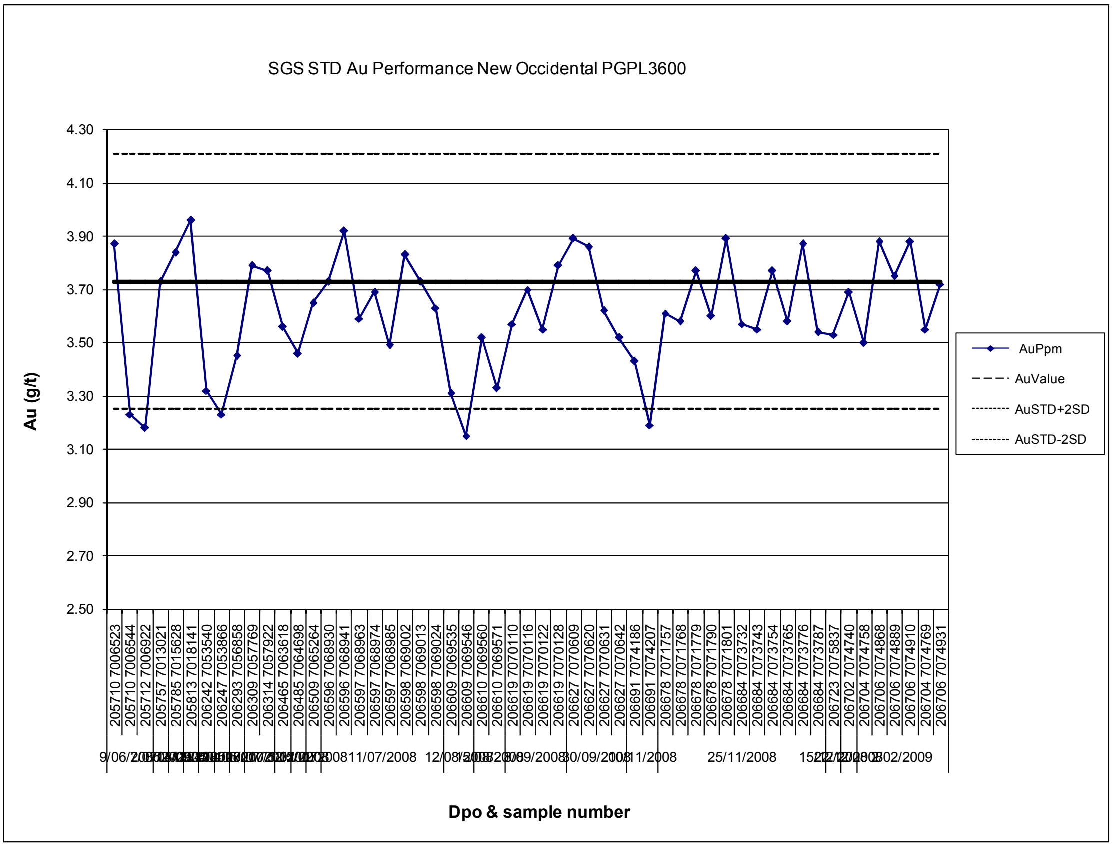


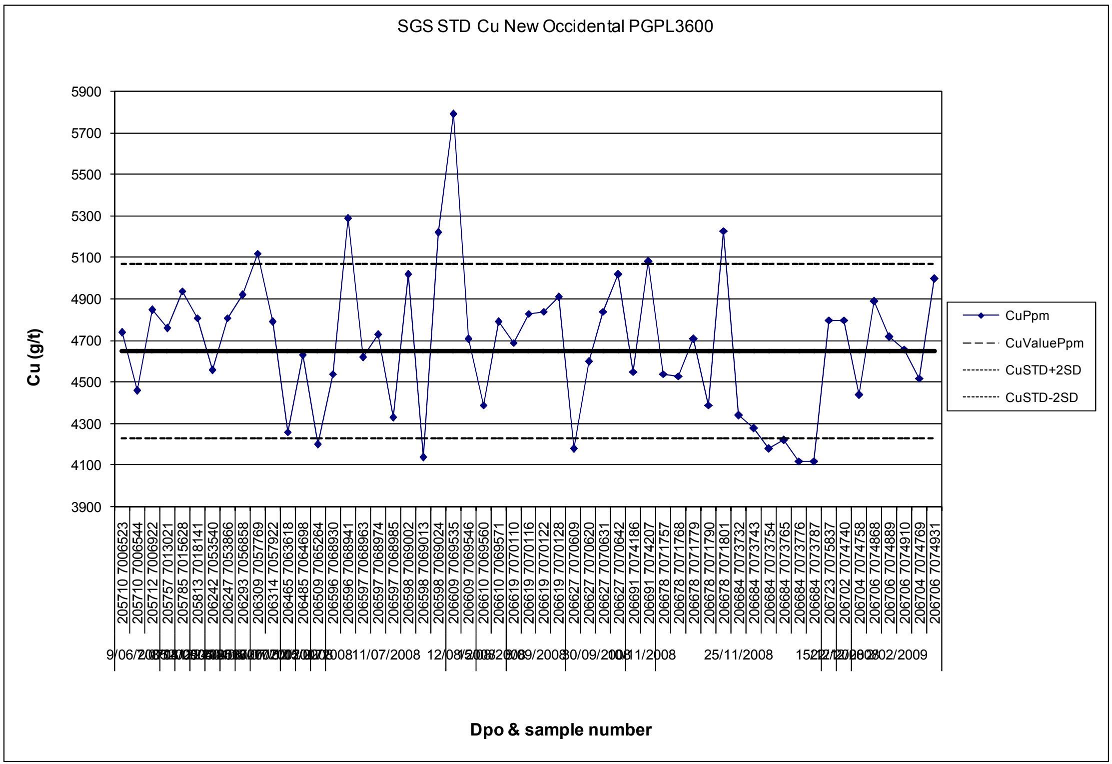
Appendix A - Standards



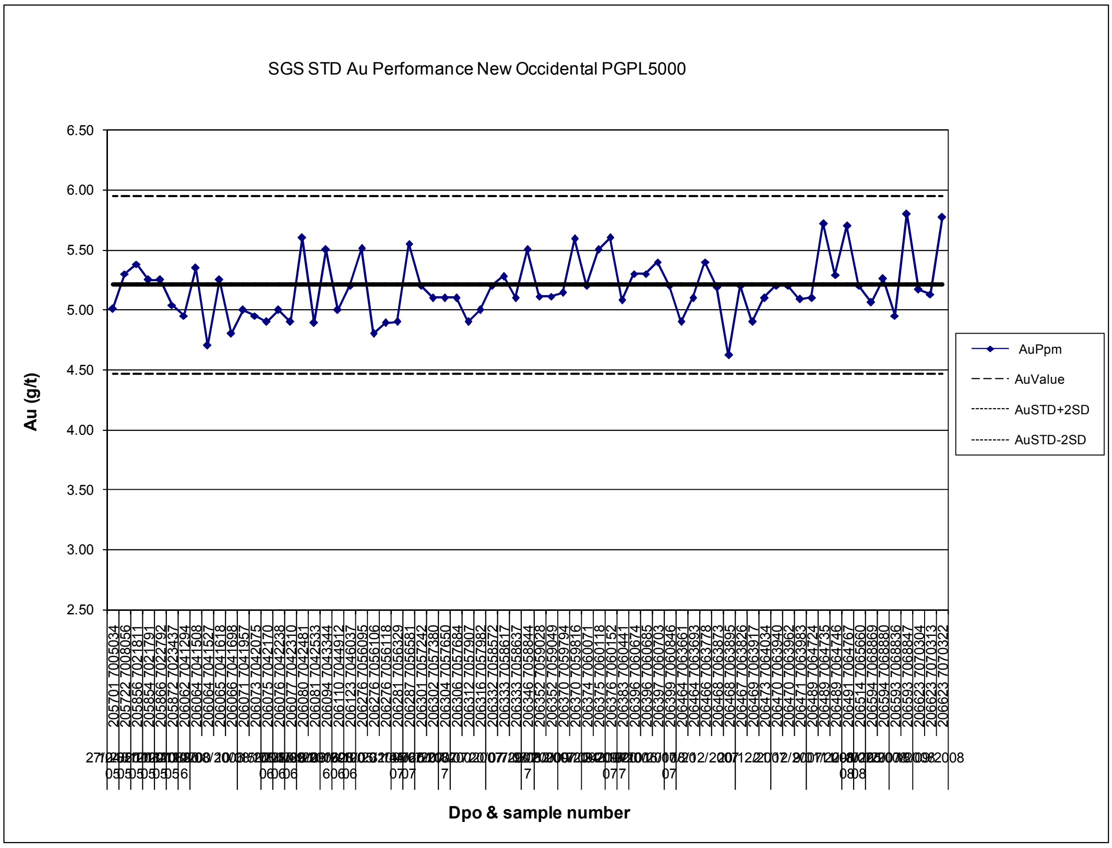


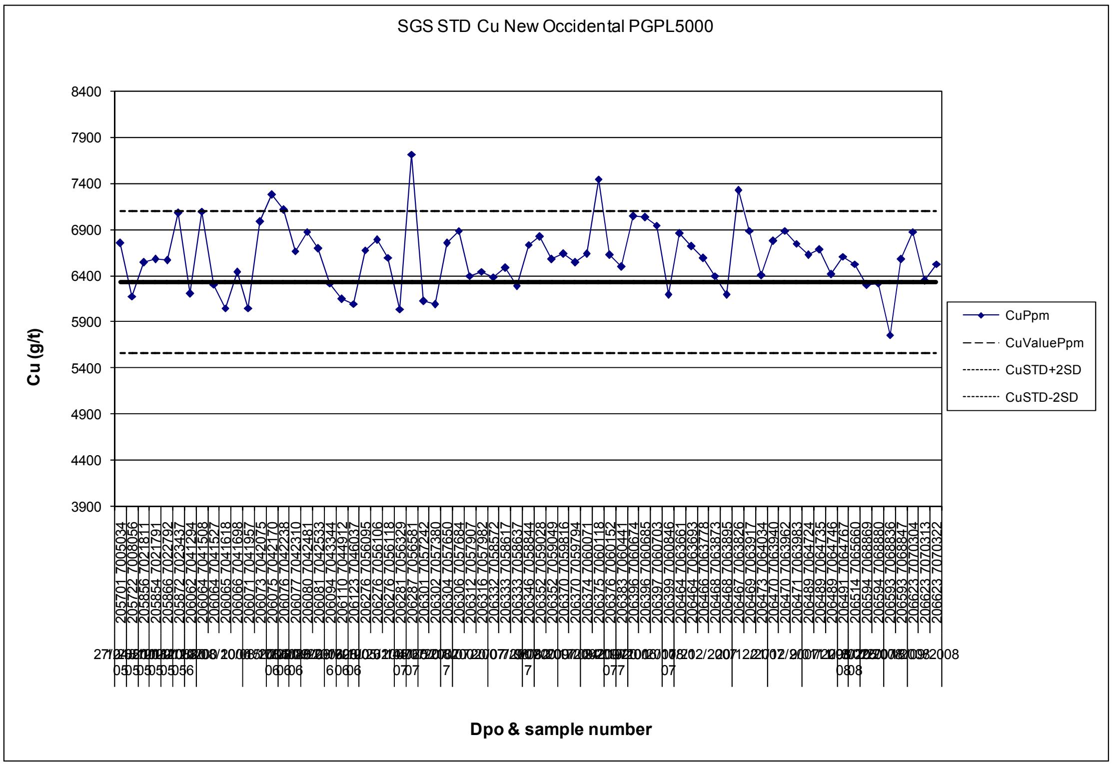
Appendix A - Standards



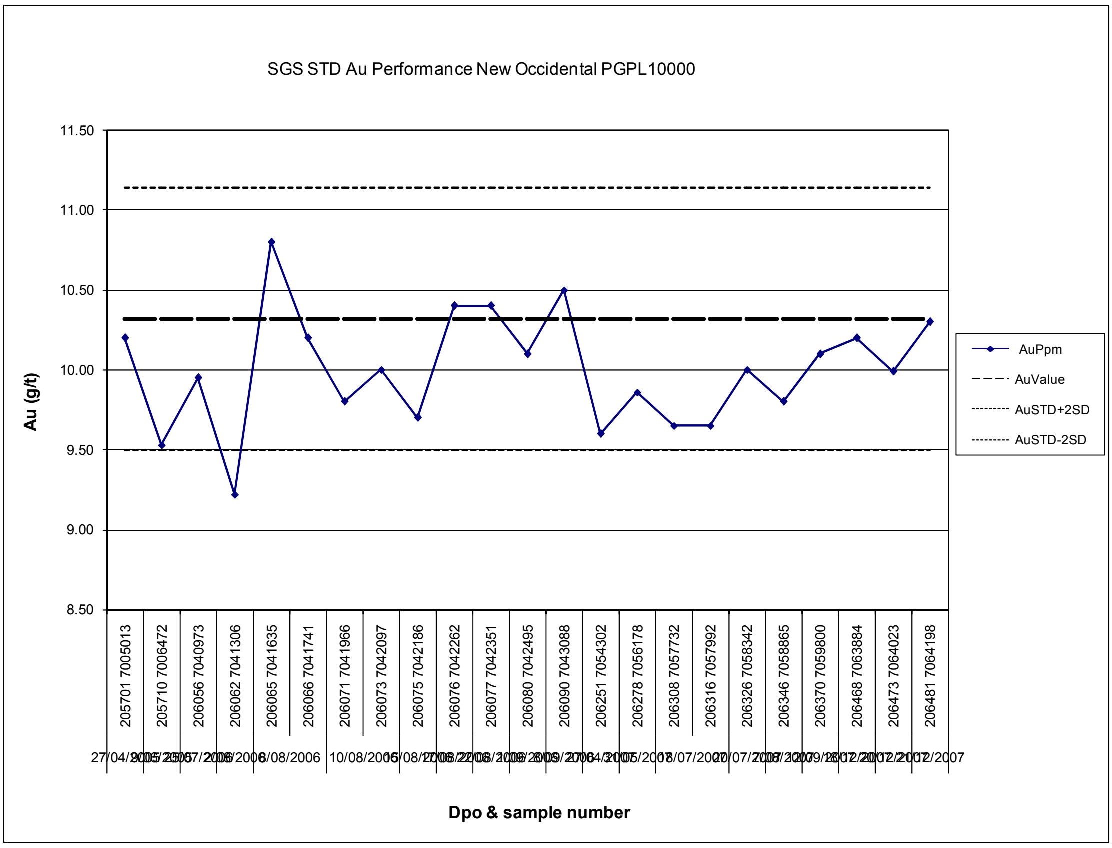


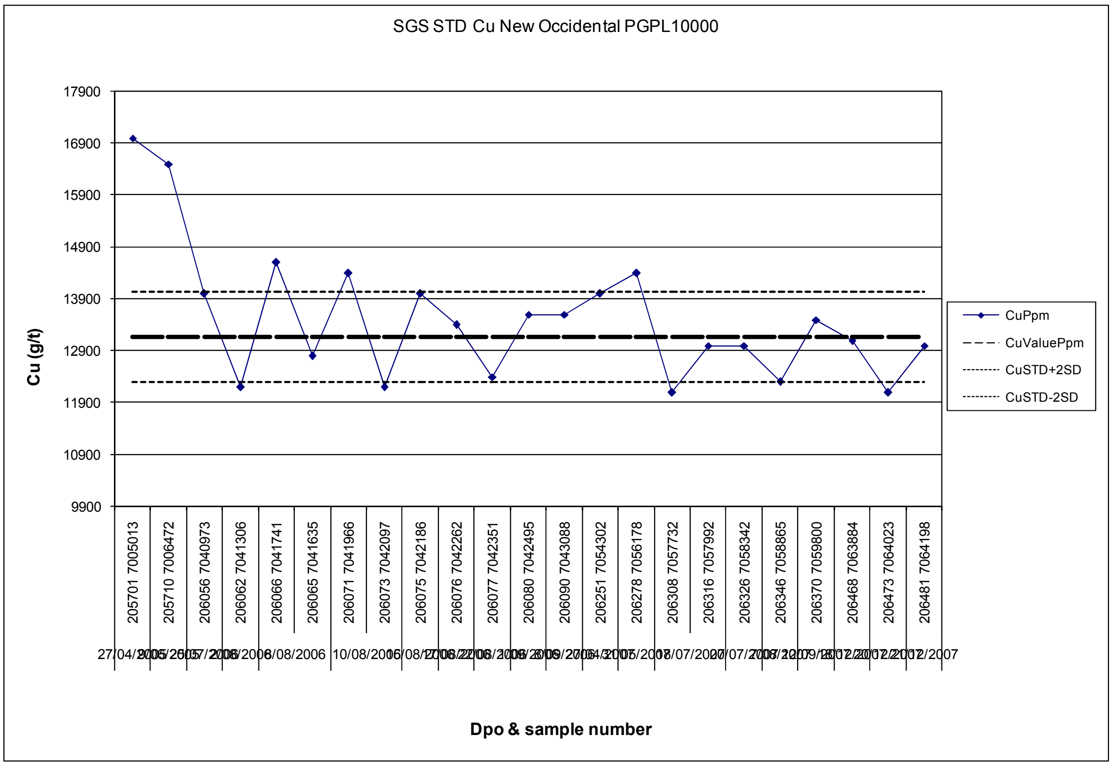
Appendix A - Standards



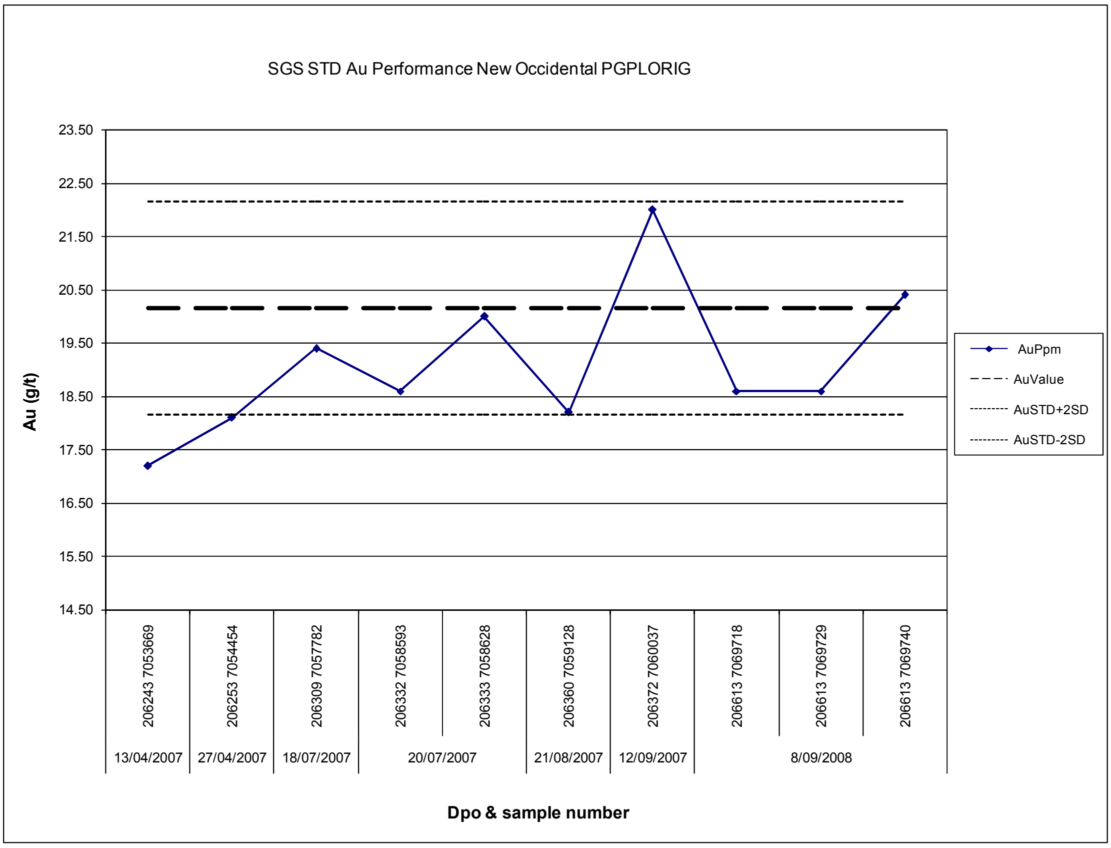


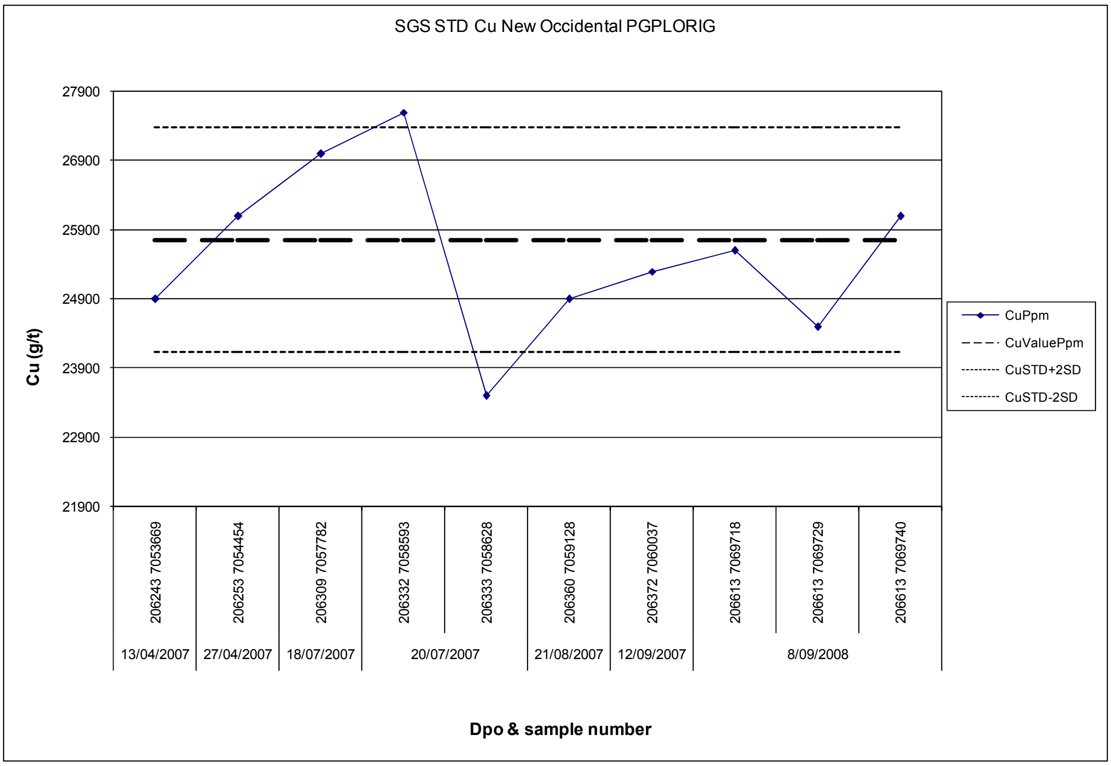
Appendix A - Standards





Appendix A - Standards





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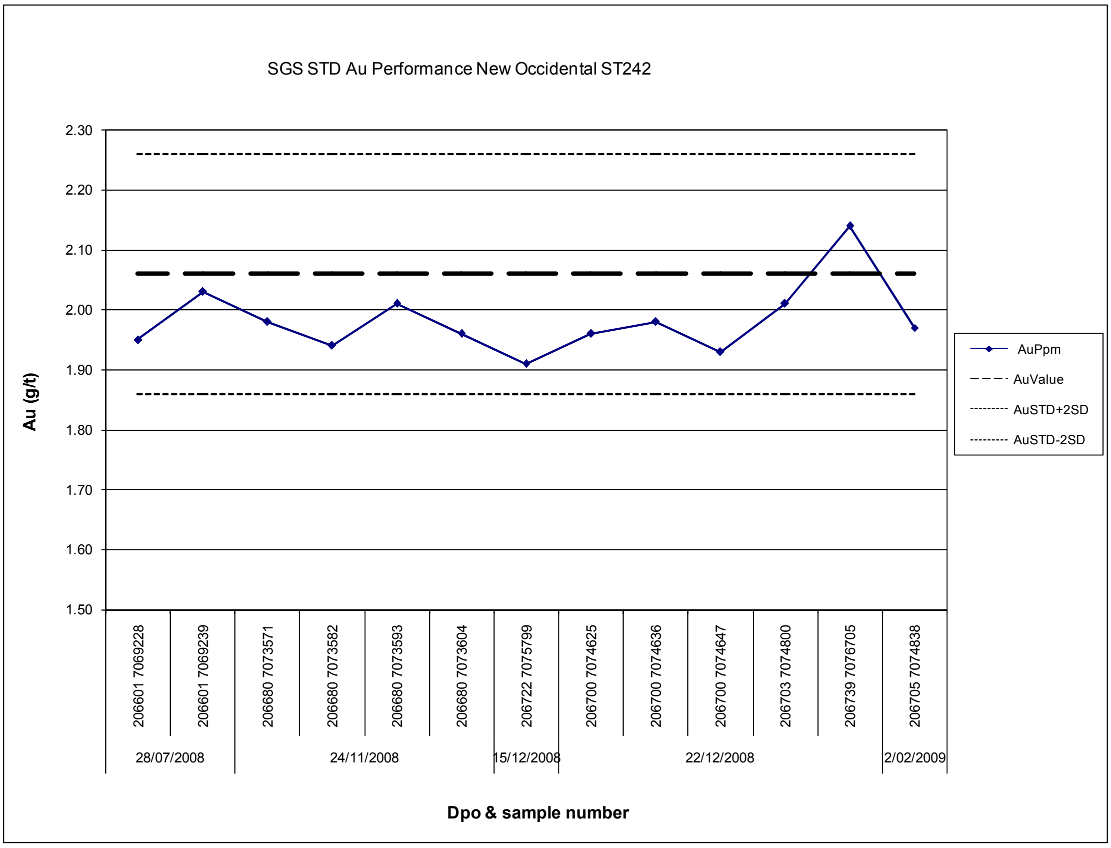

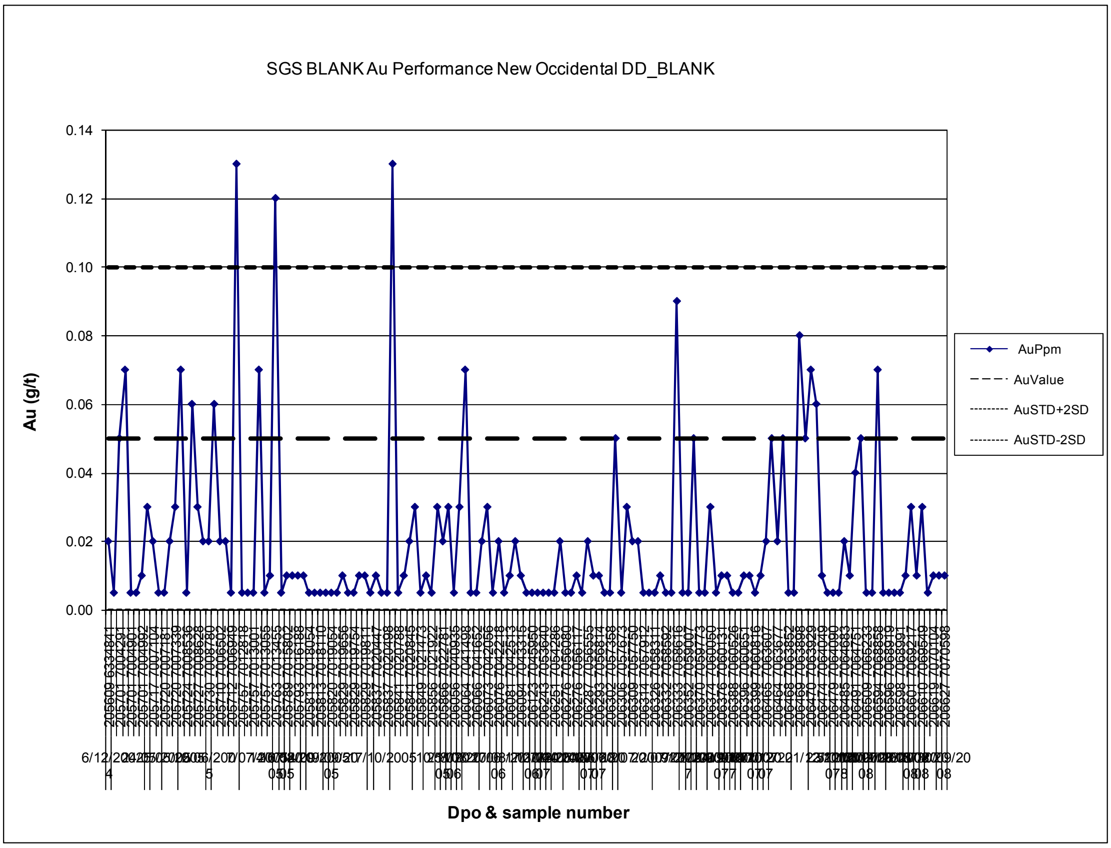

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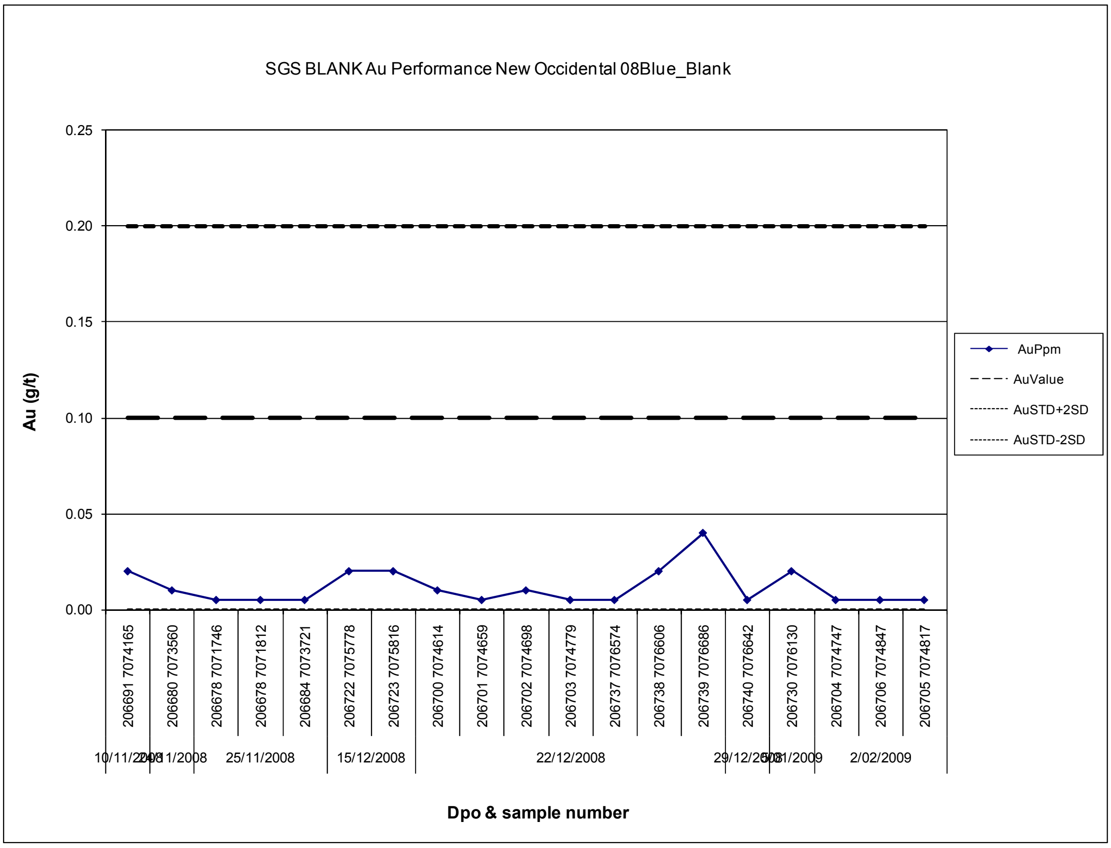
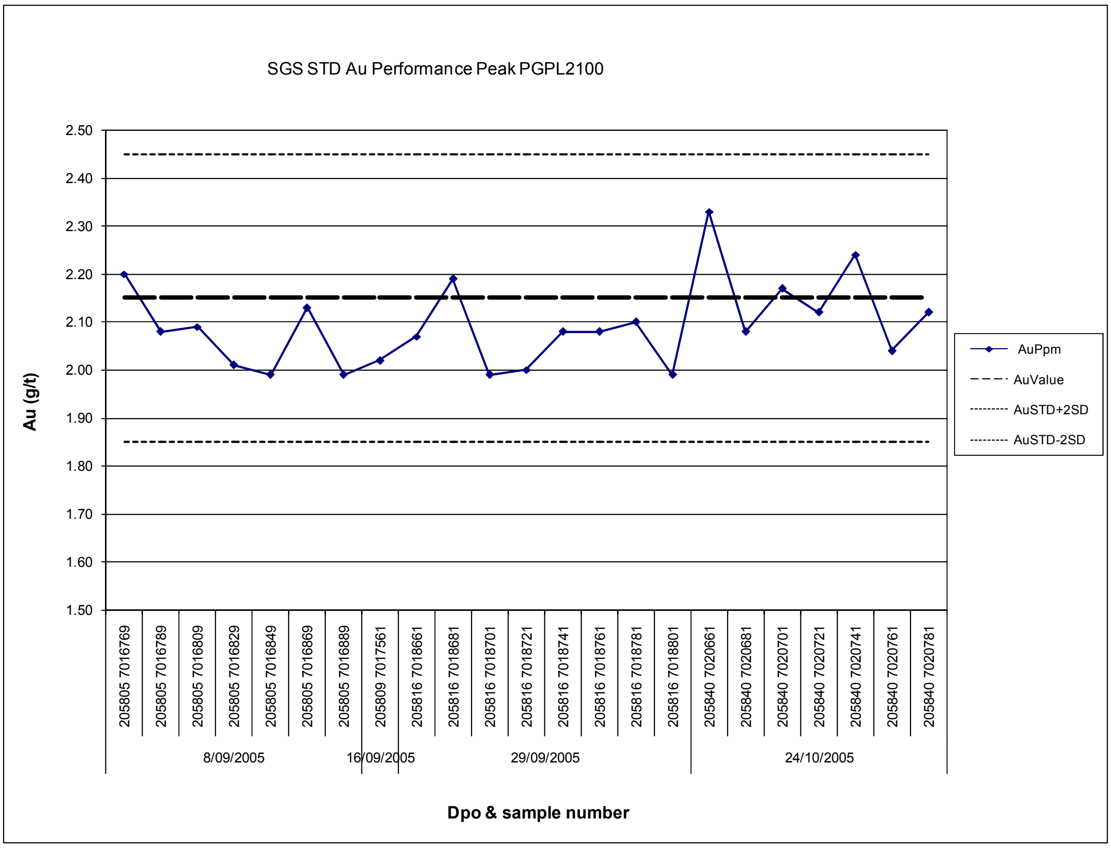
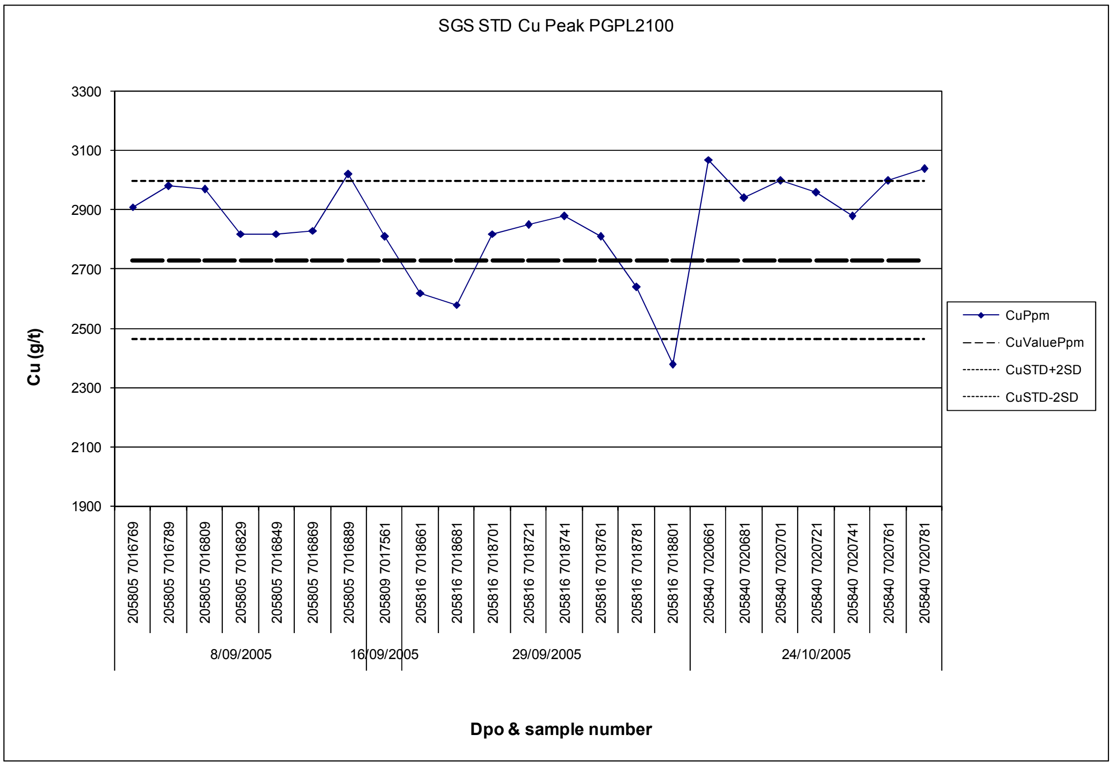
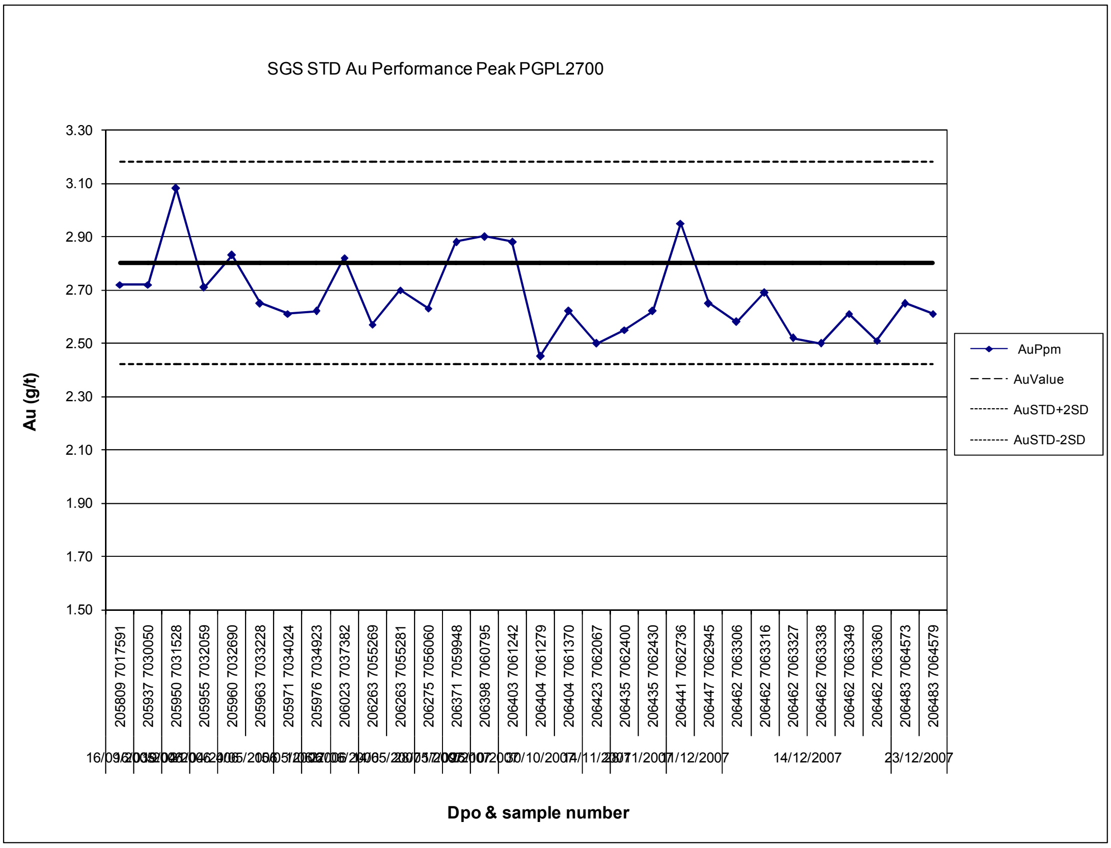


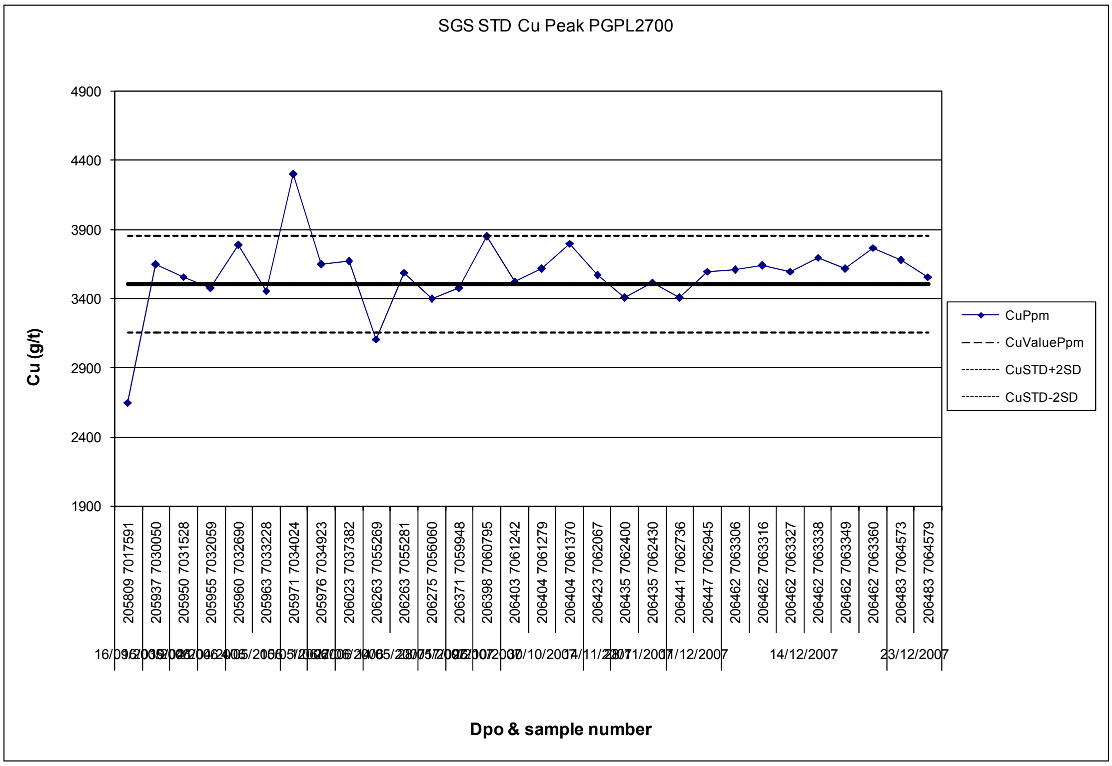


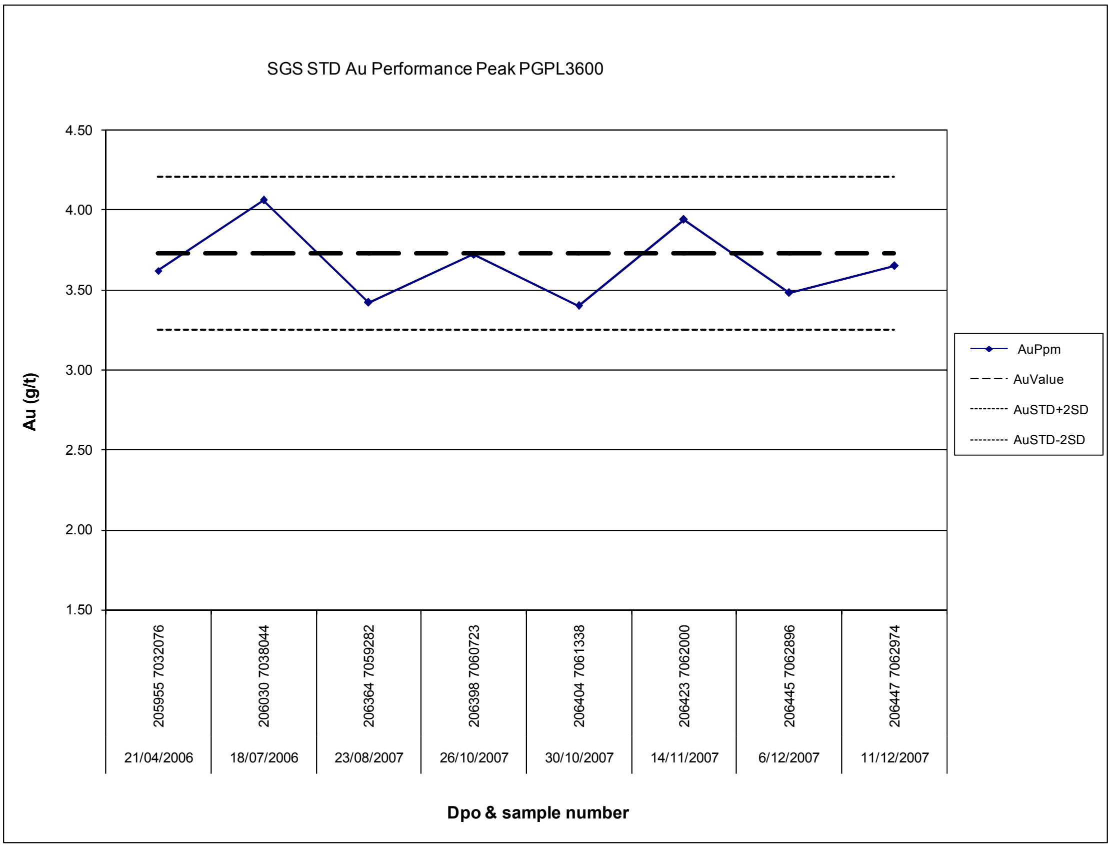
Appendix A - Standards



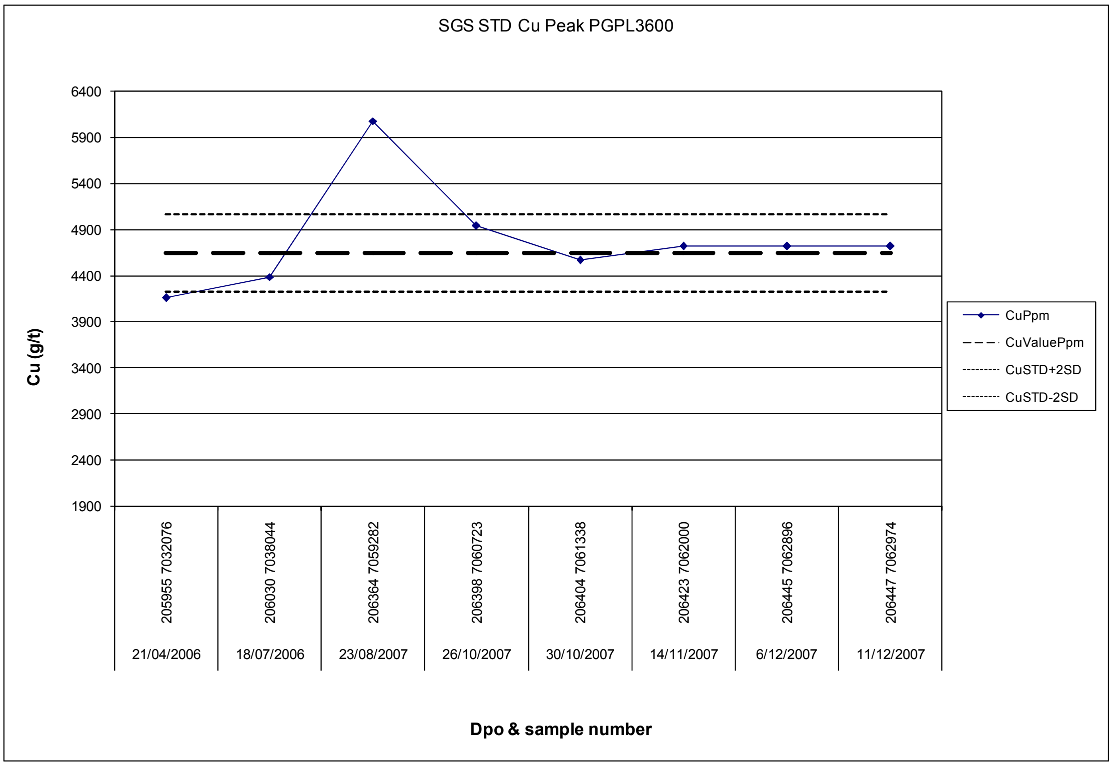


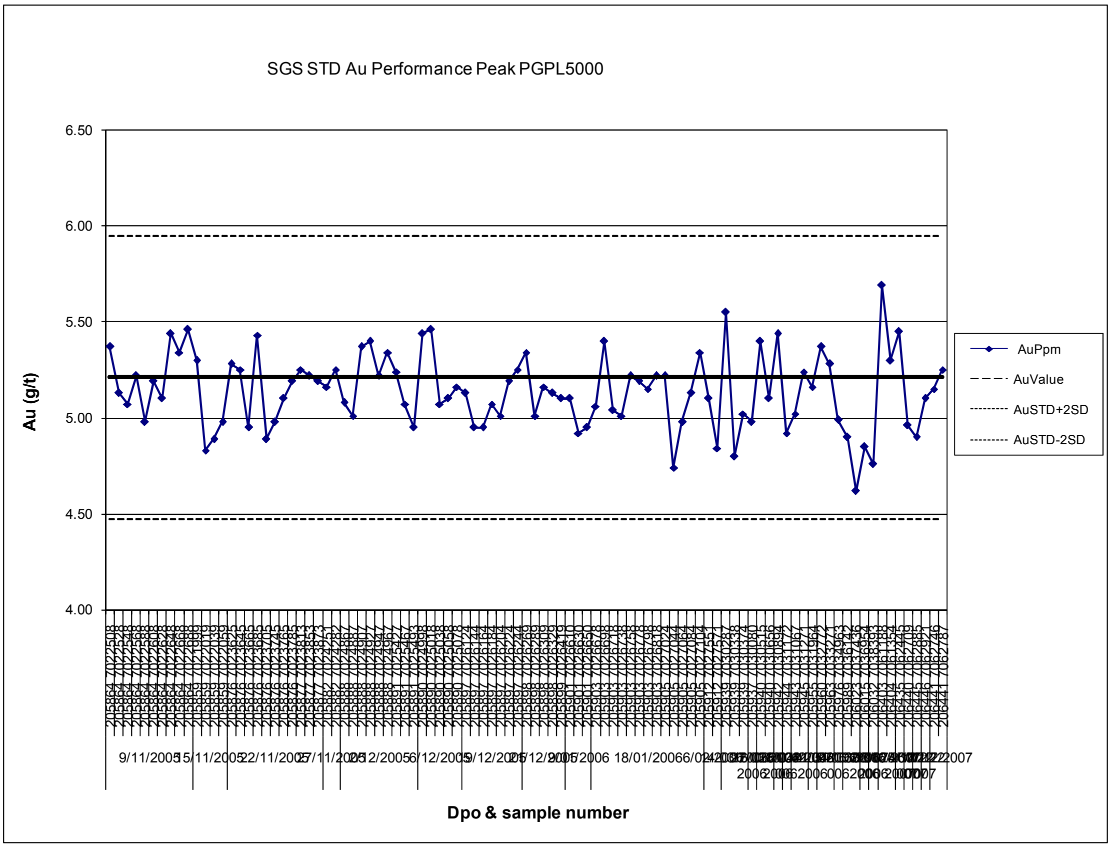
Appendix A - Standards



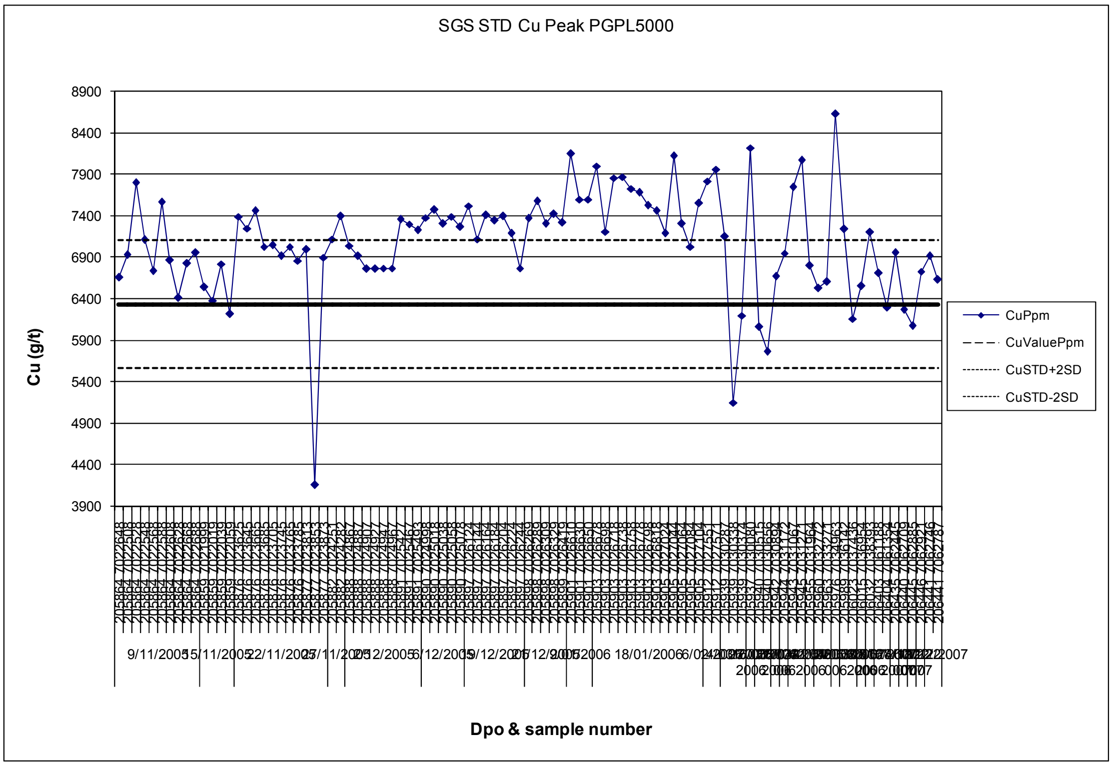


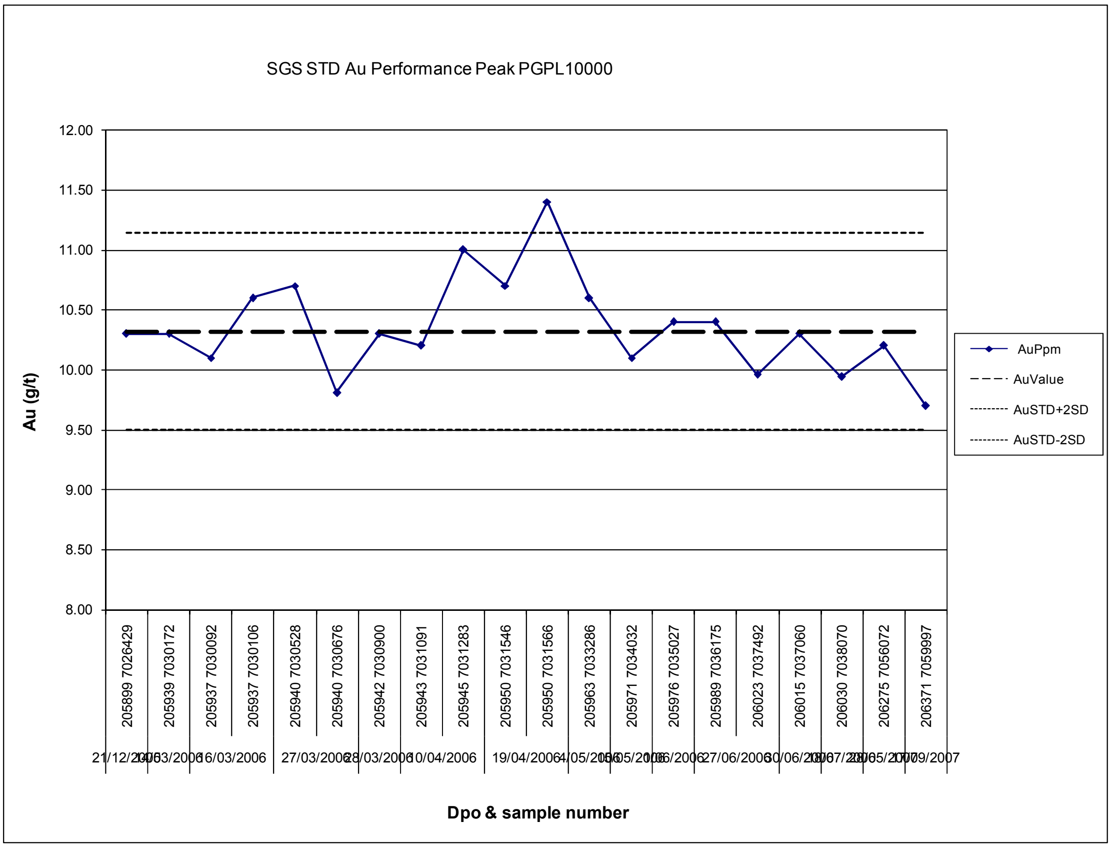
Appendix A - Standards



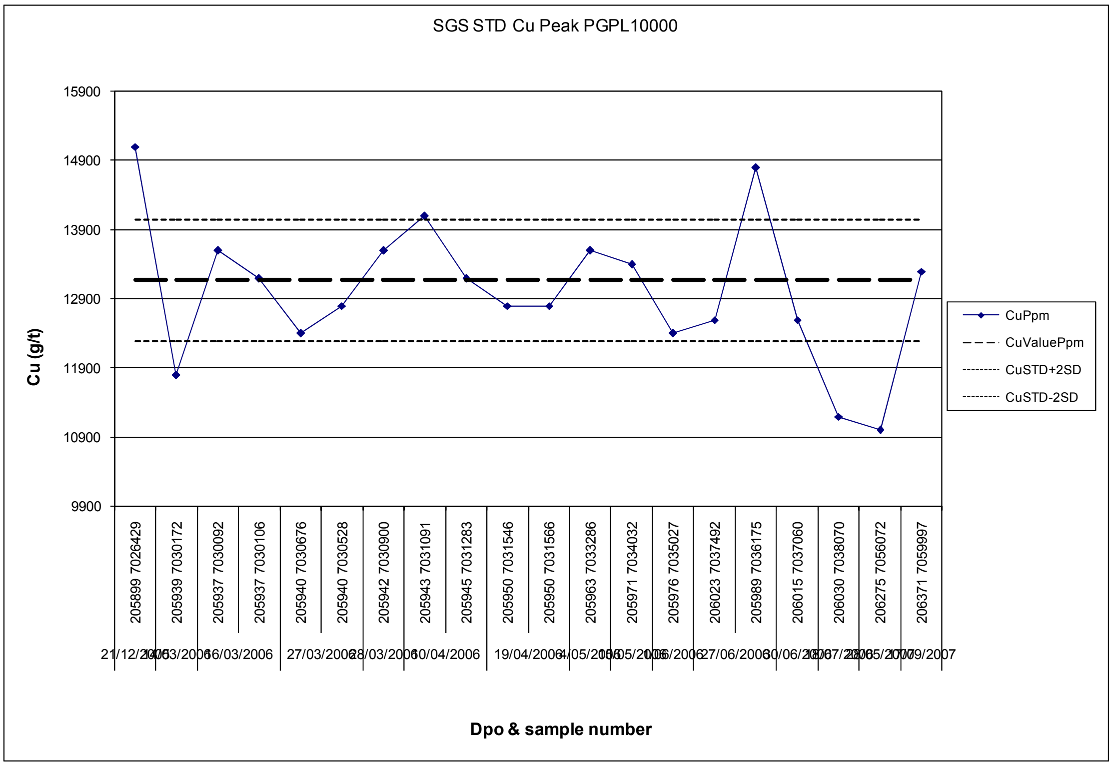


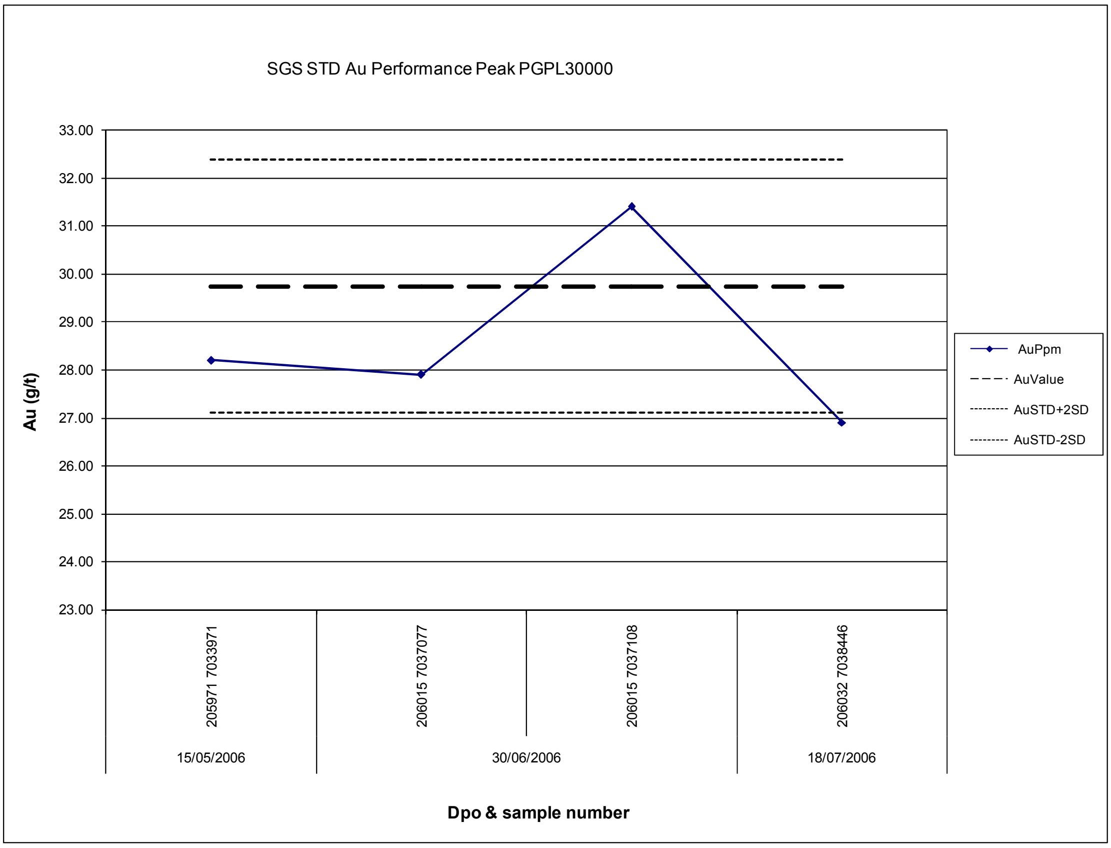
Appendix A - Standards



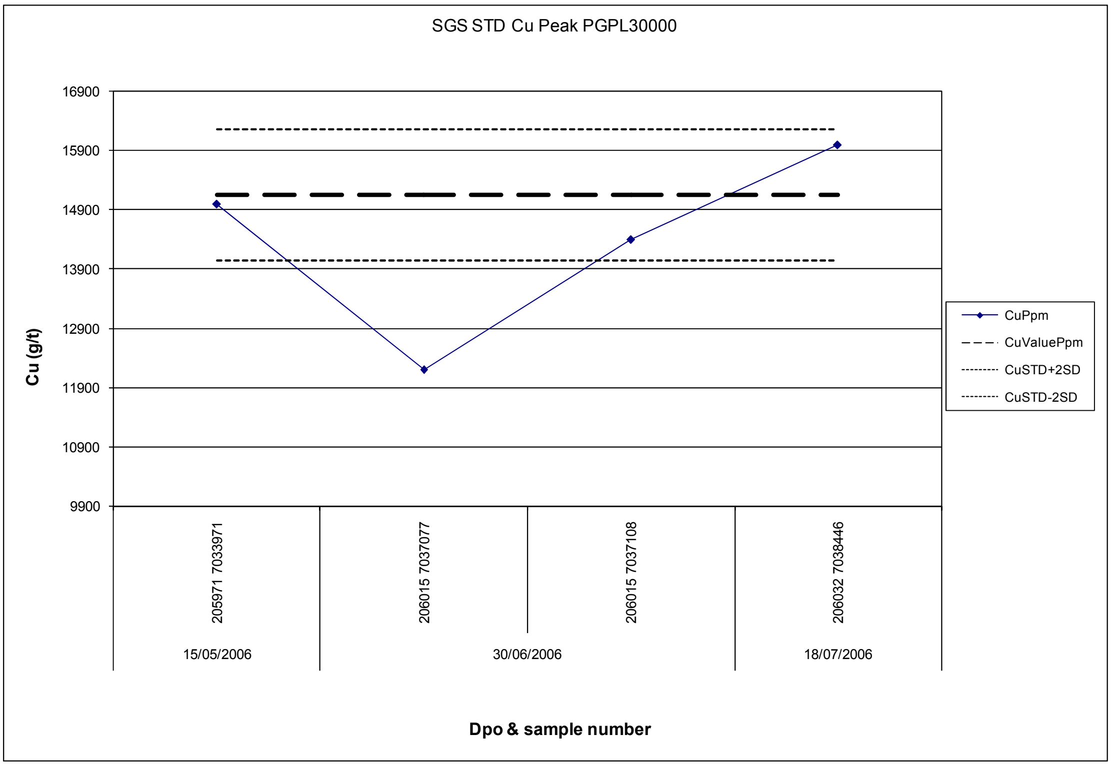


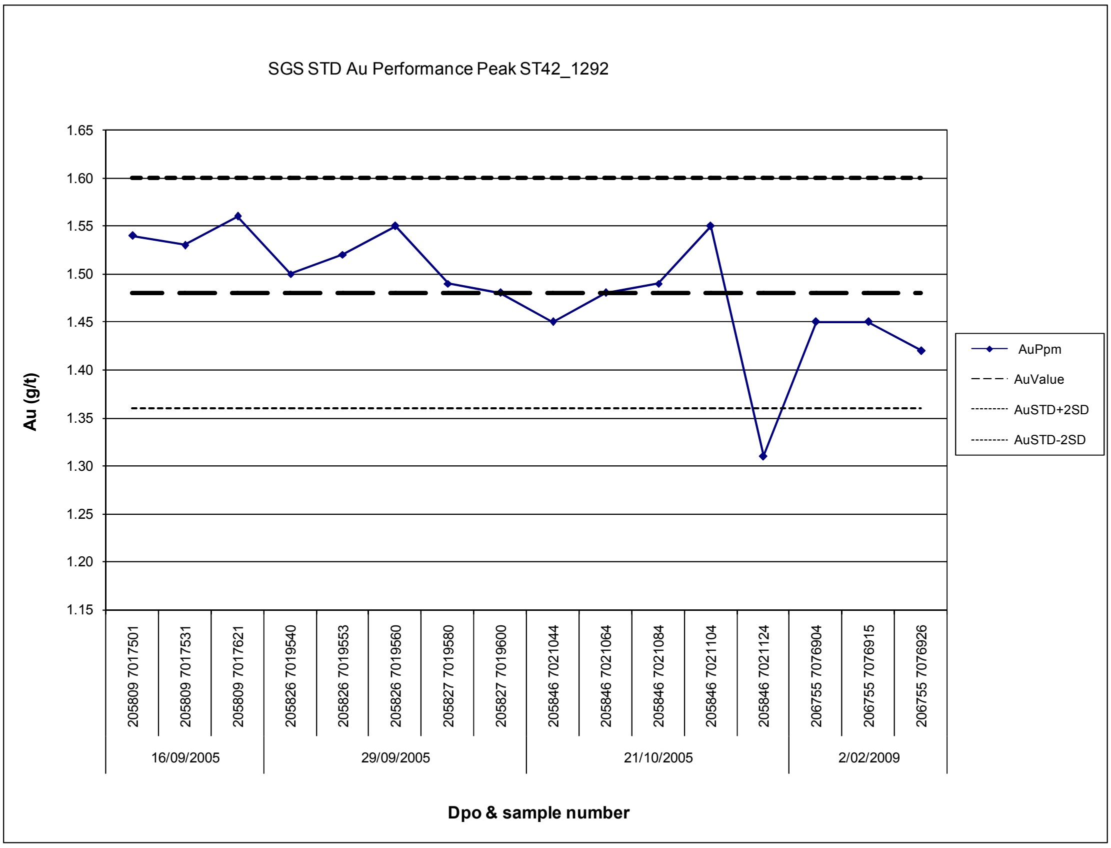
Appendix A - Standards



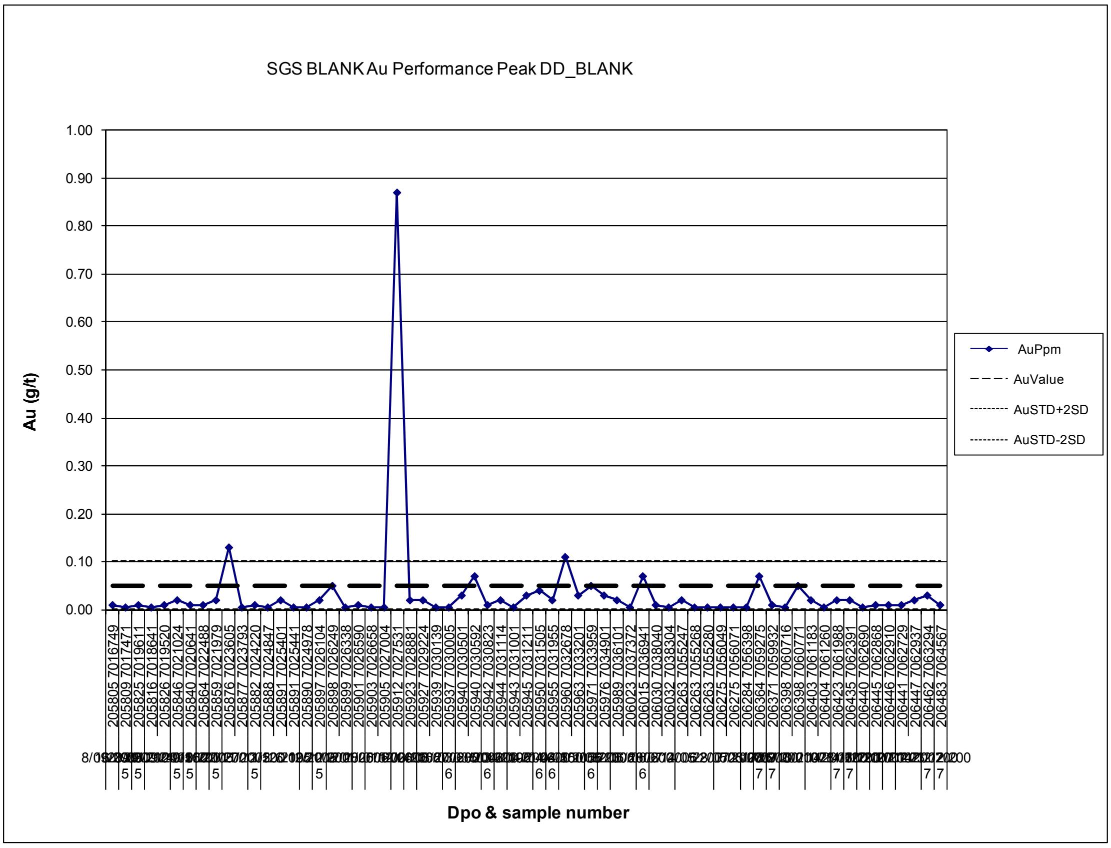


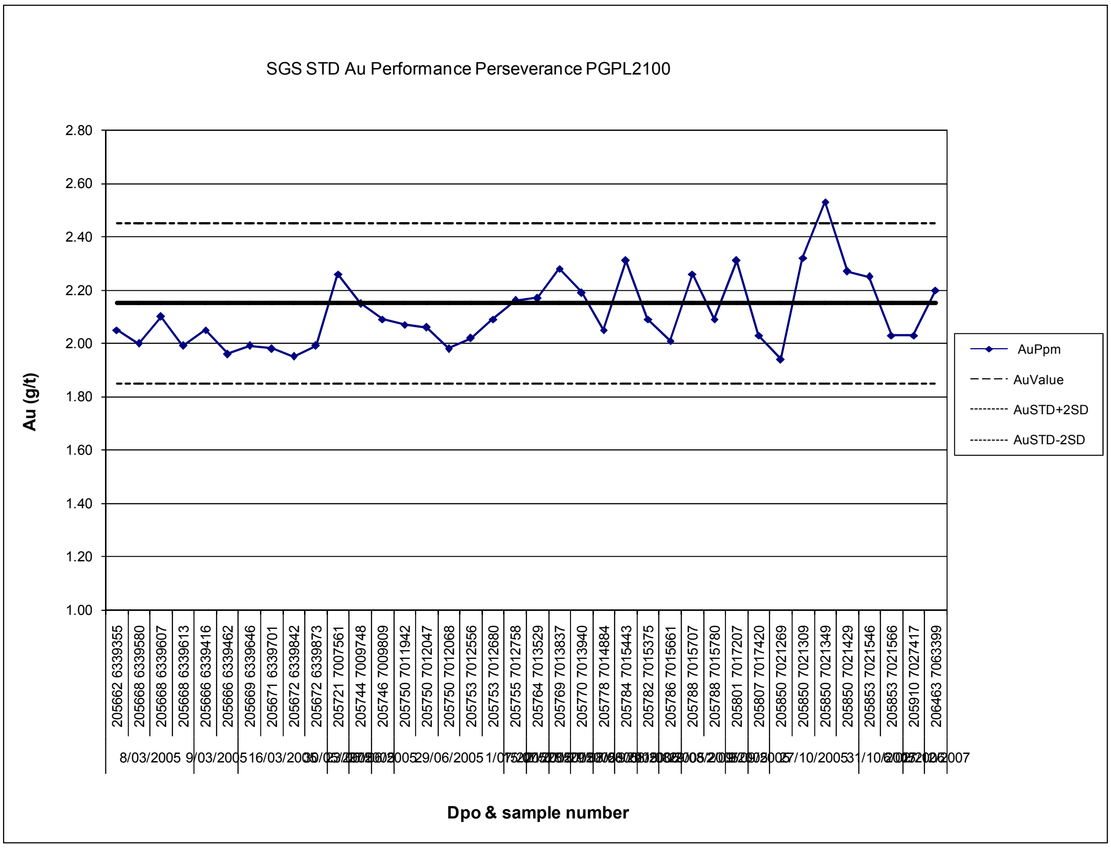
Appendix A - Standards



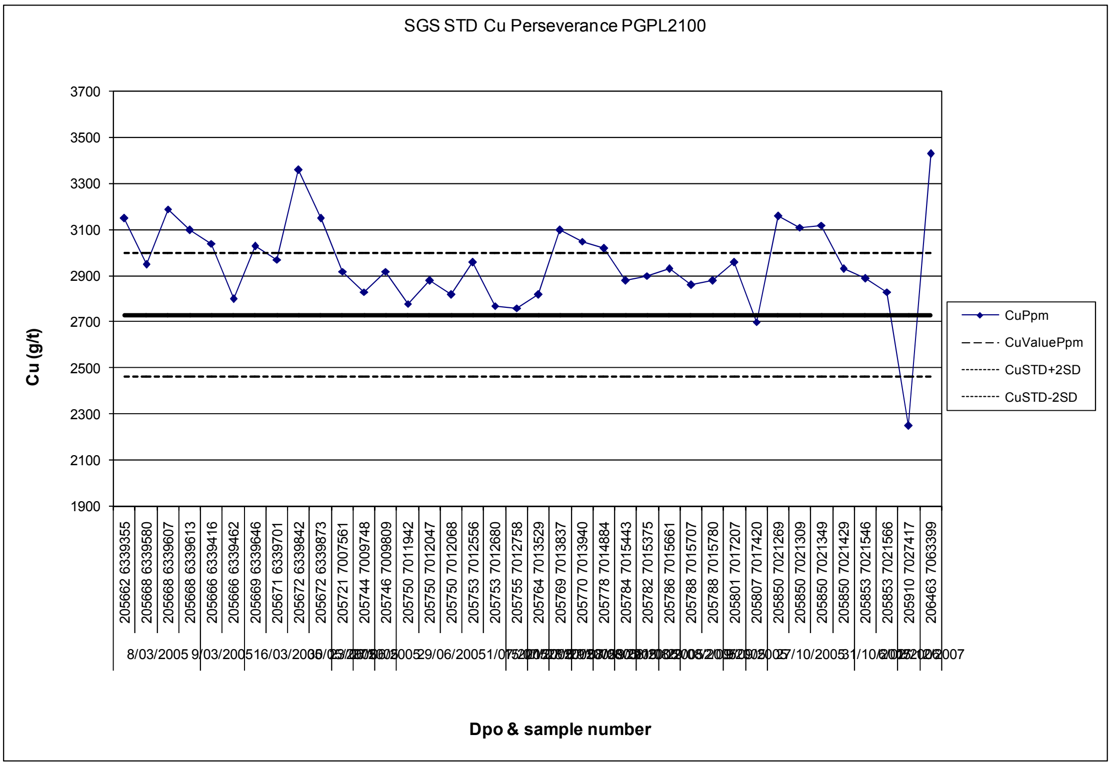


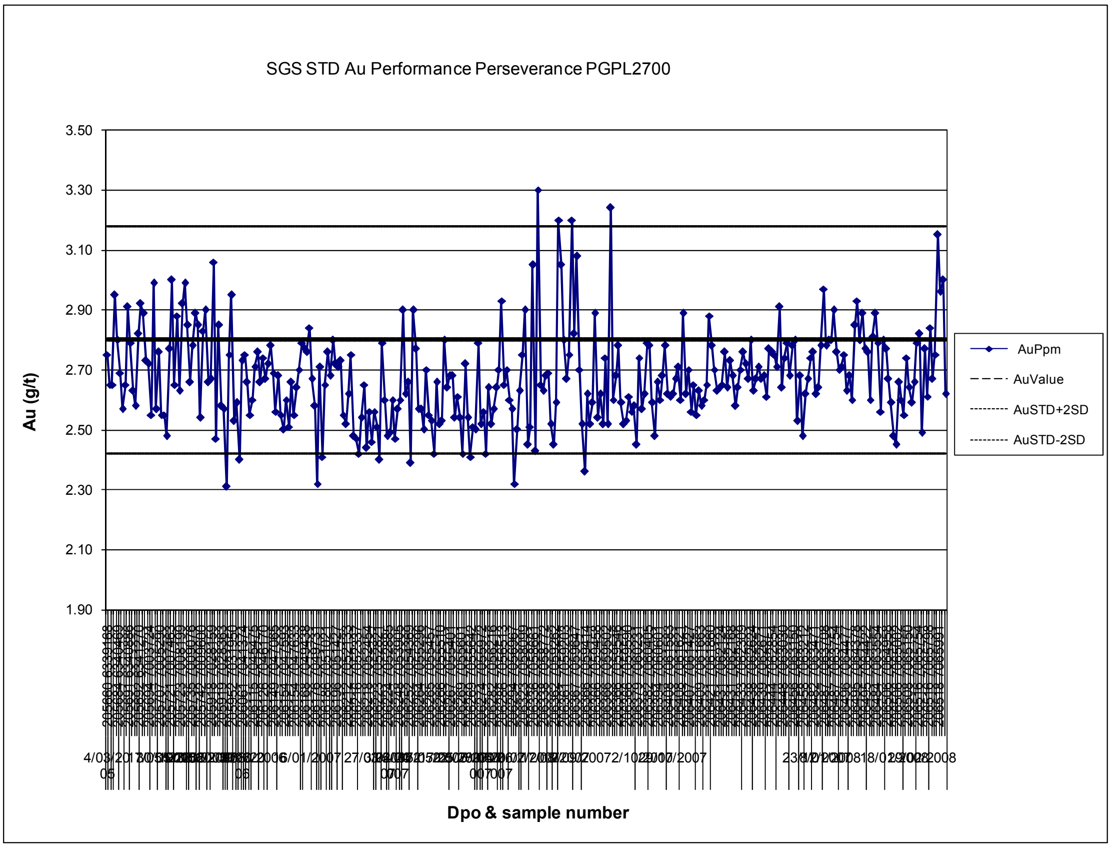
Appendix A - Standards



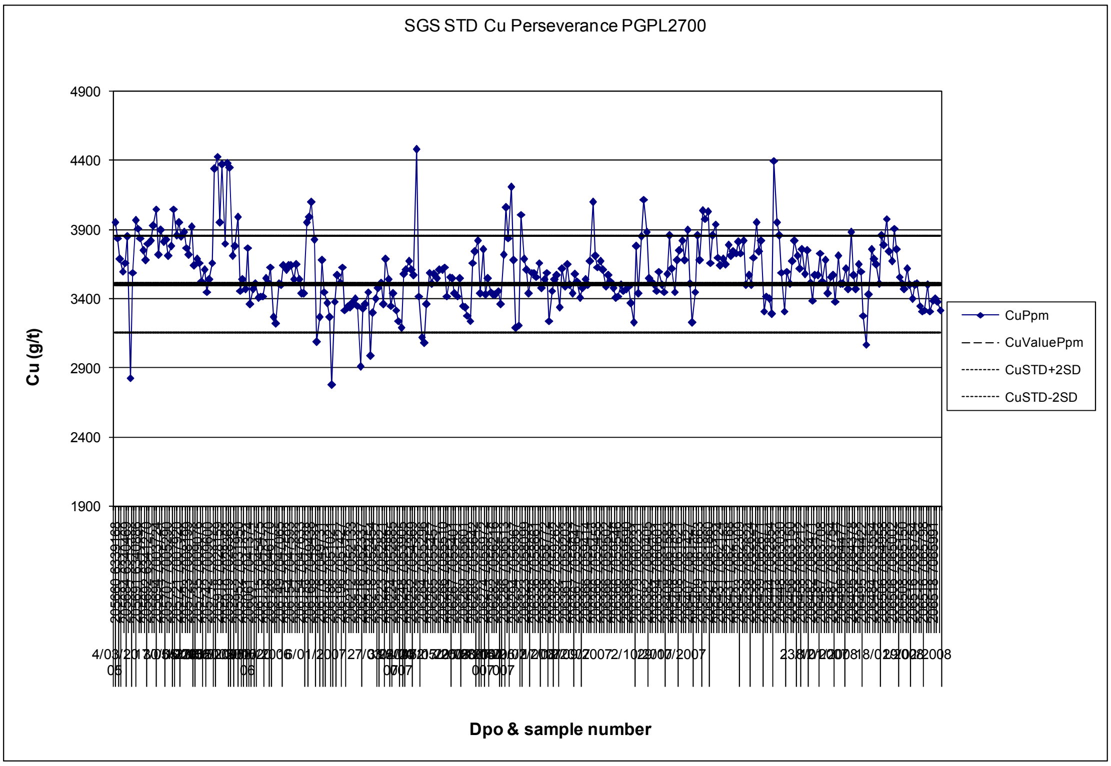


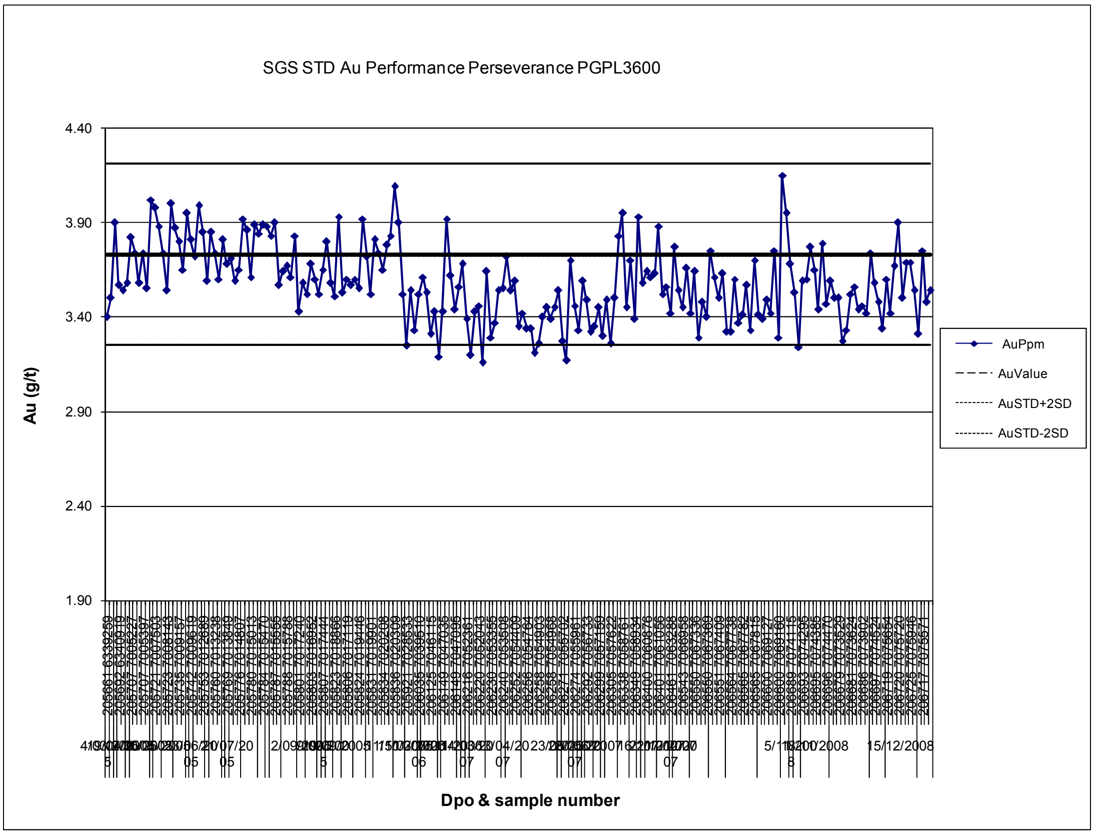
Appendix A - Standards



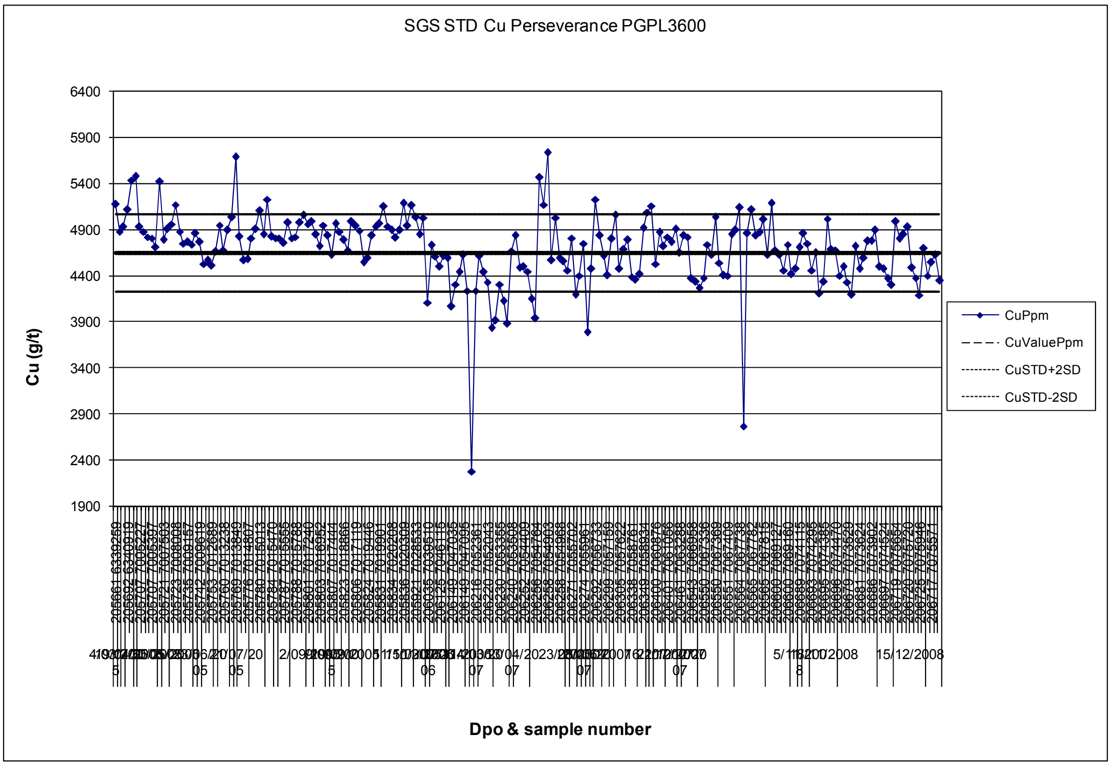


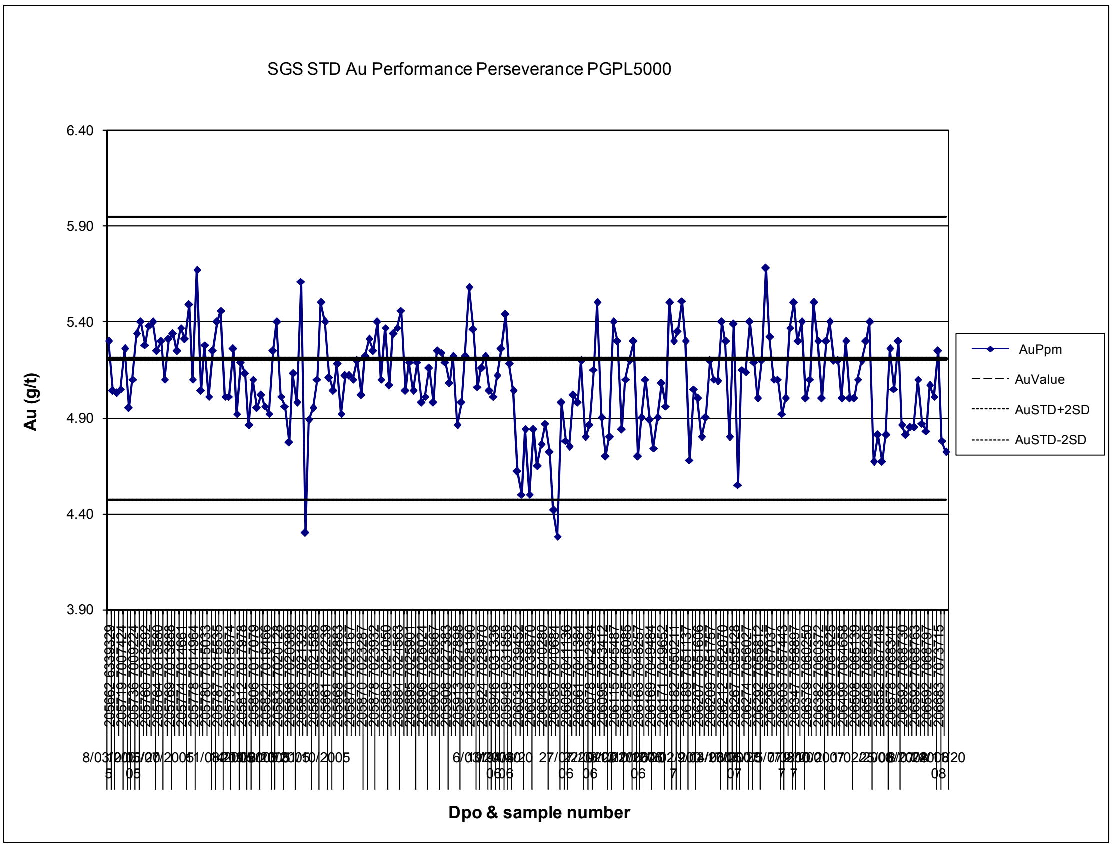
Appendix A - Standards



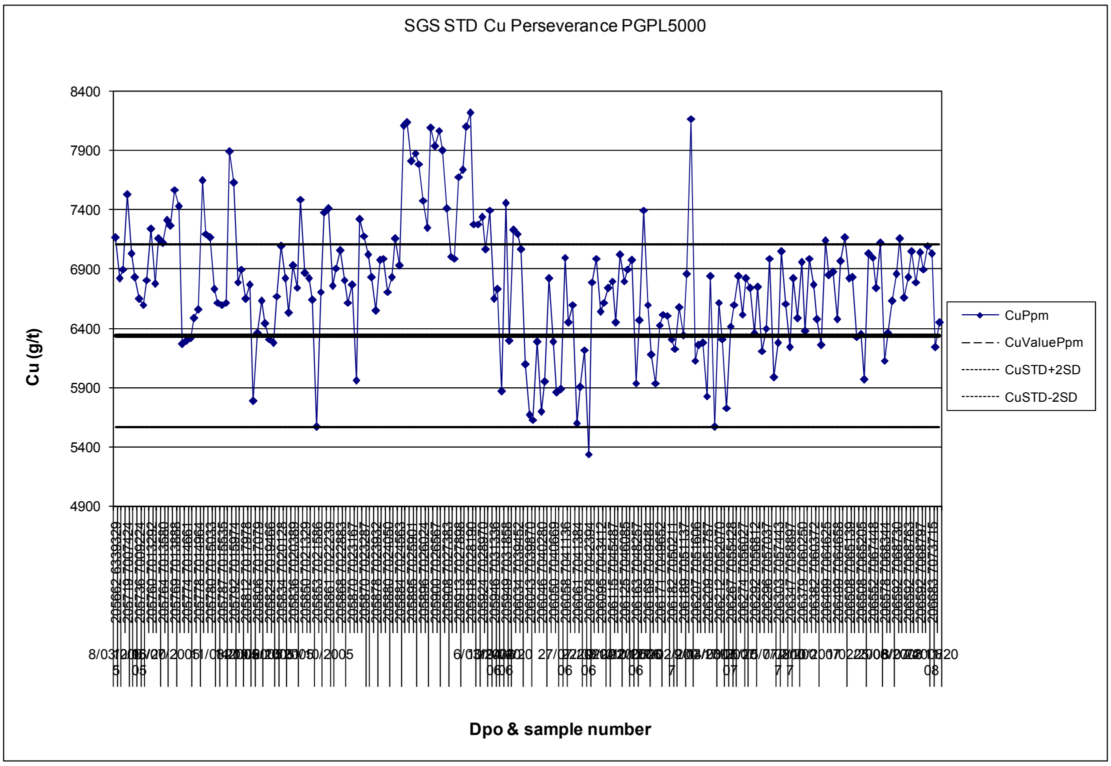


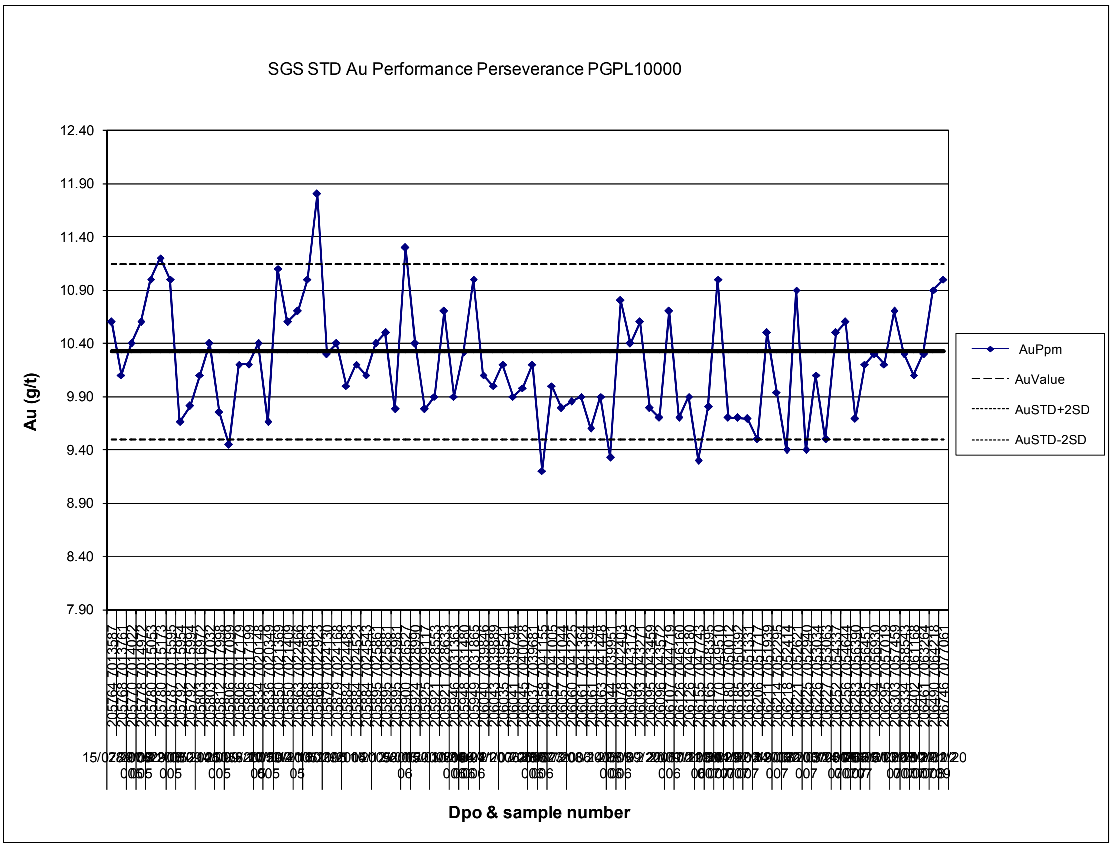
Appendix A - Standards



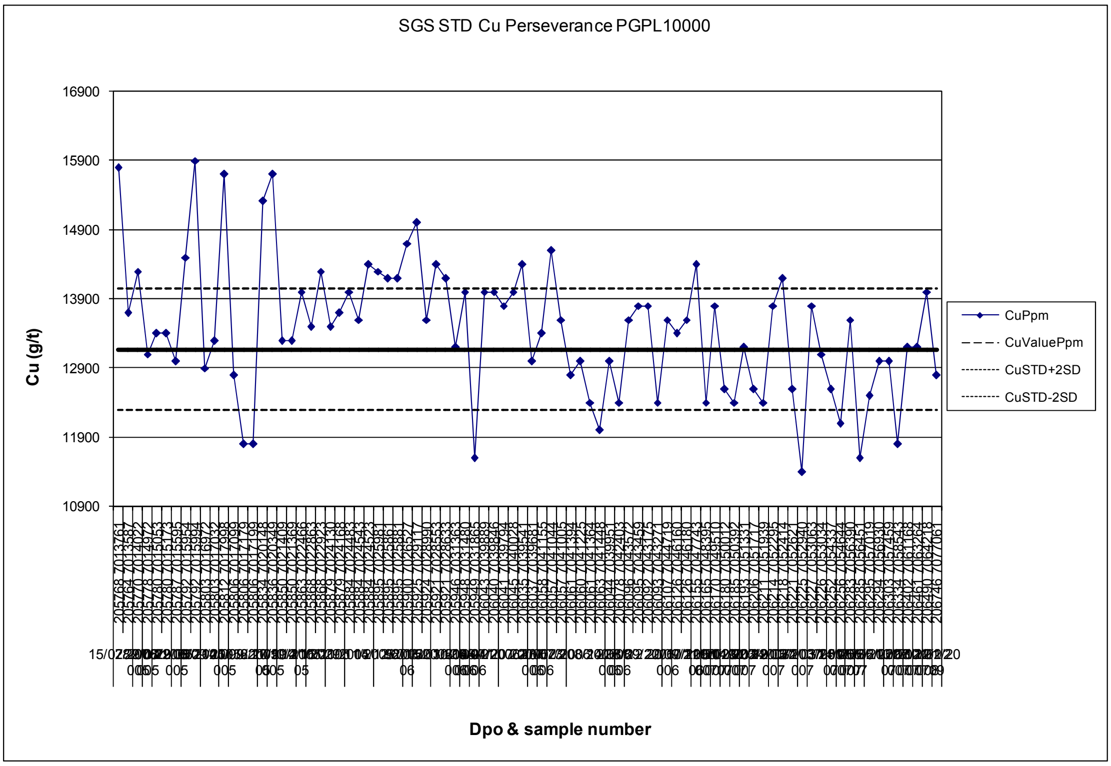


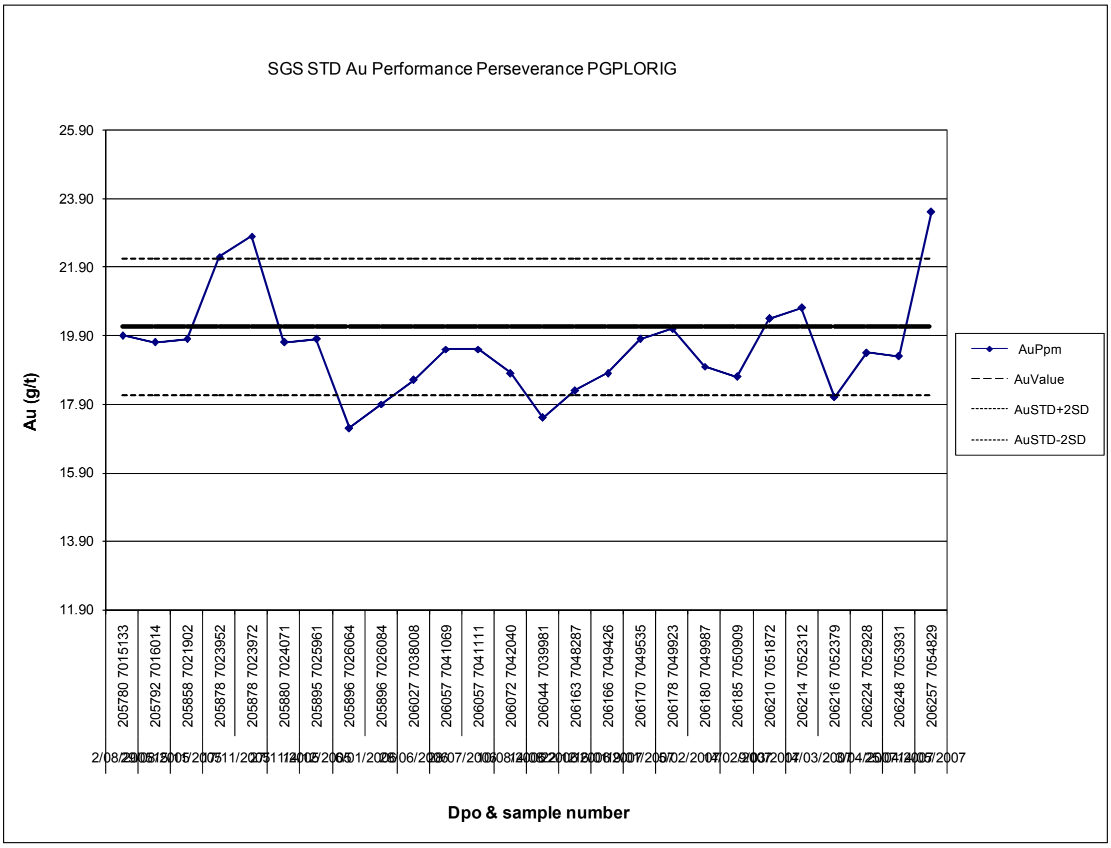
Appendix A - Standards



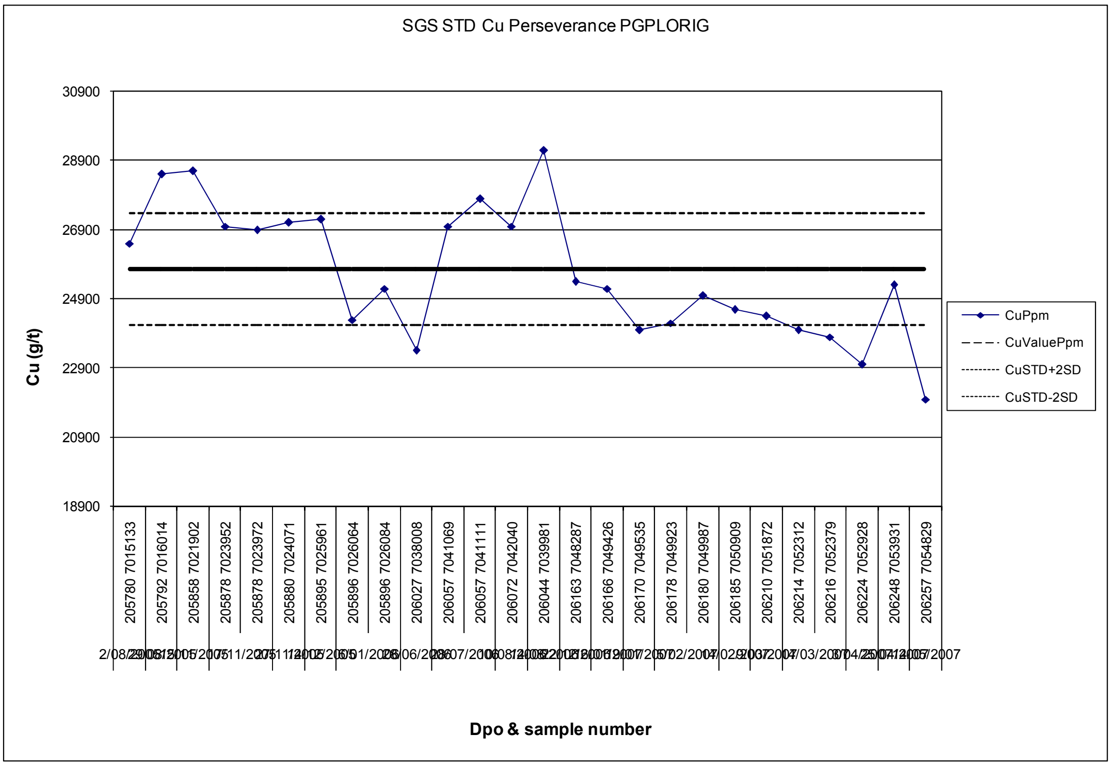


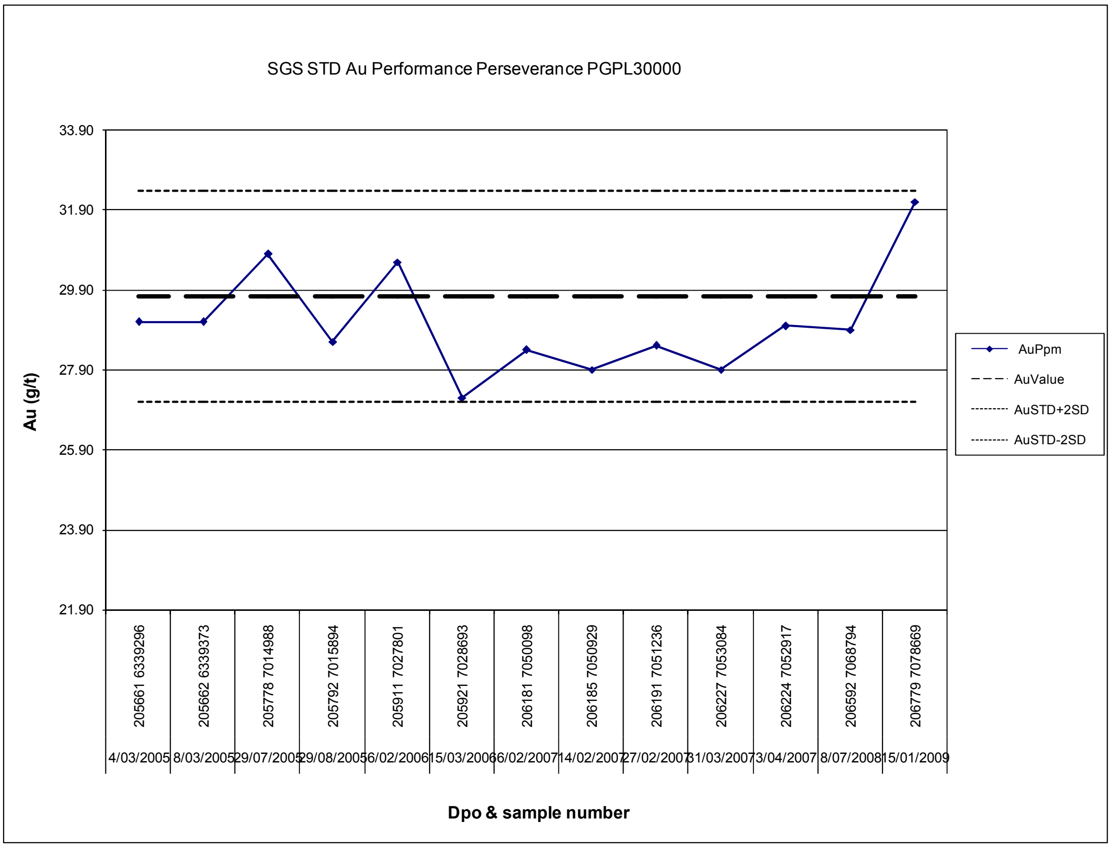
Appendix A - Standards



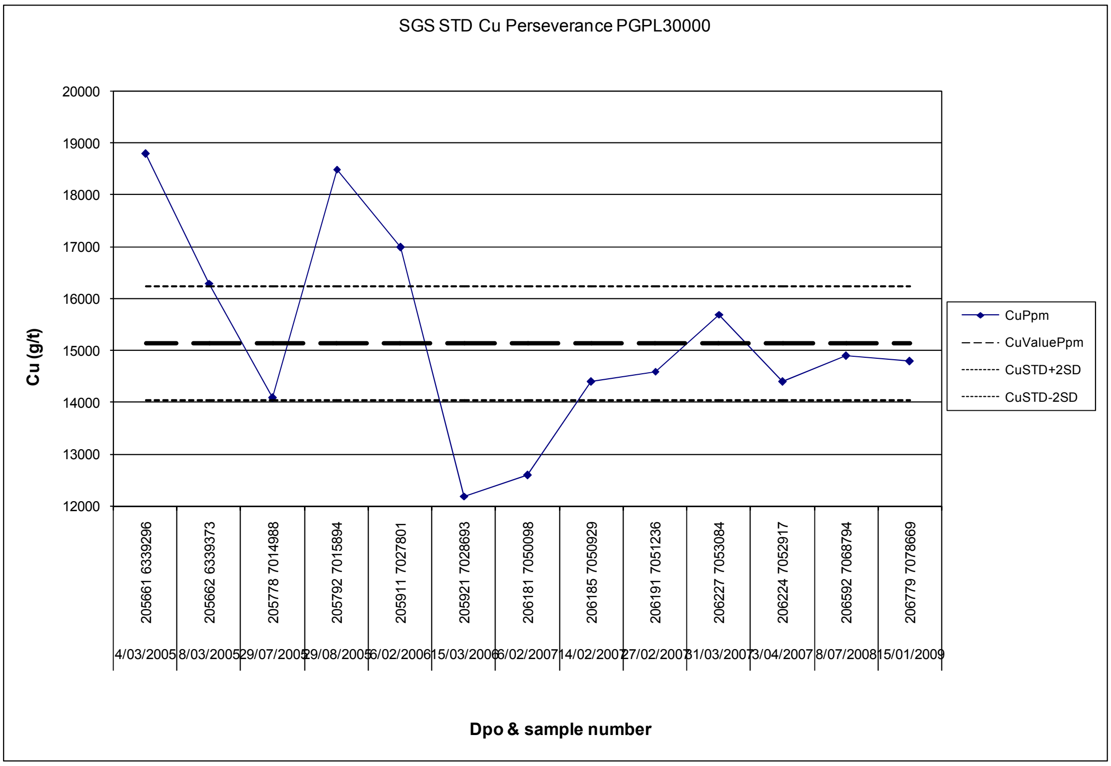


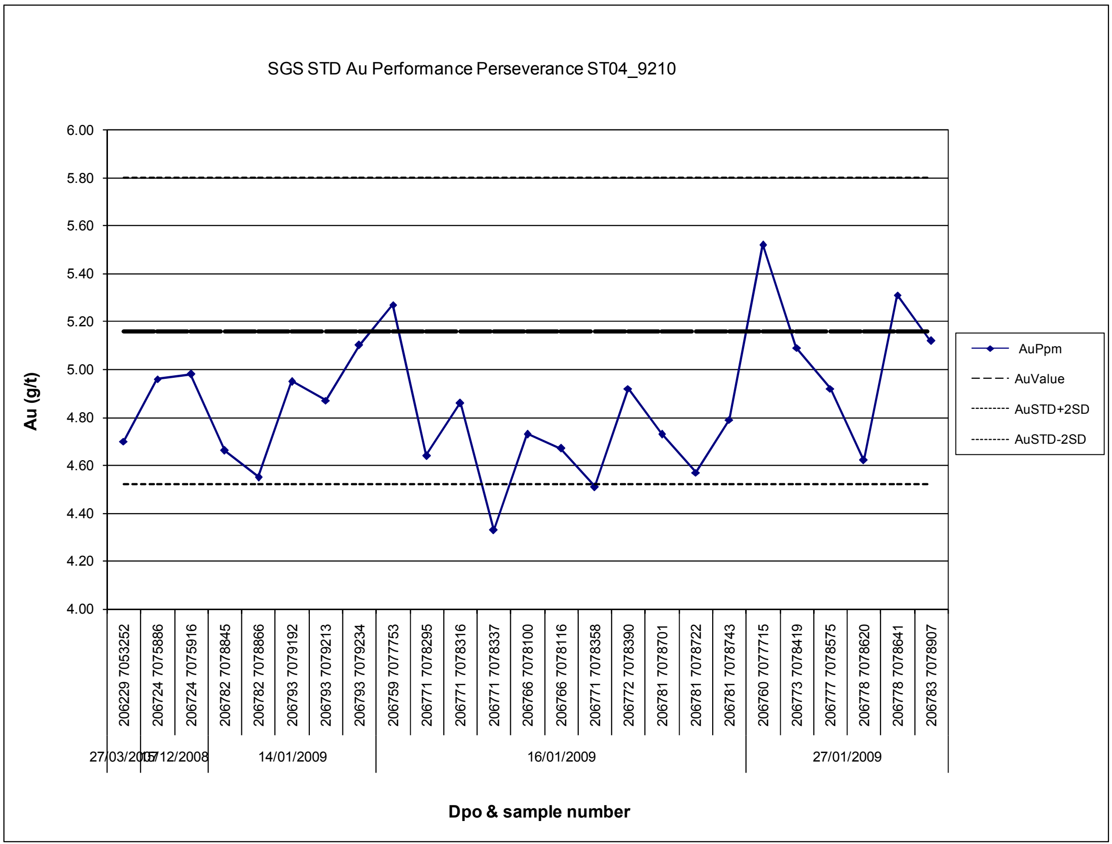
Appendix A - Standards



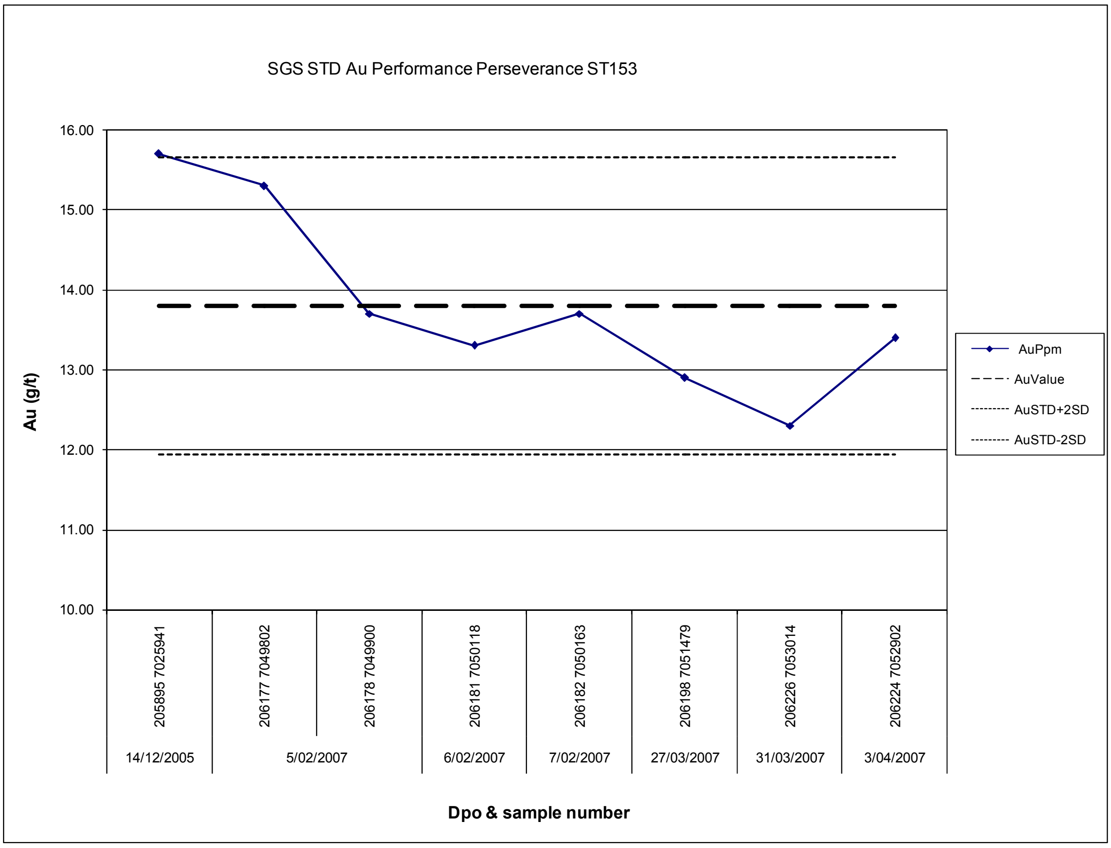


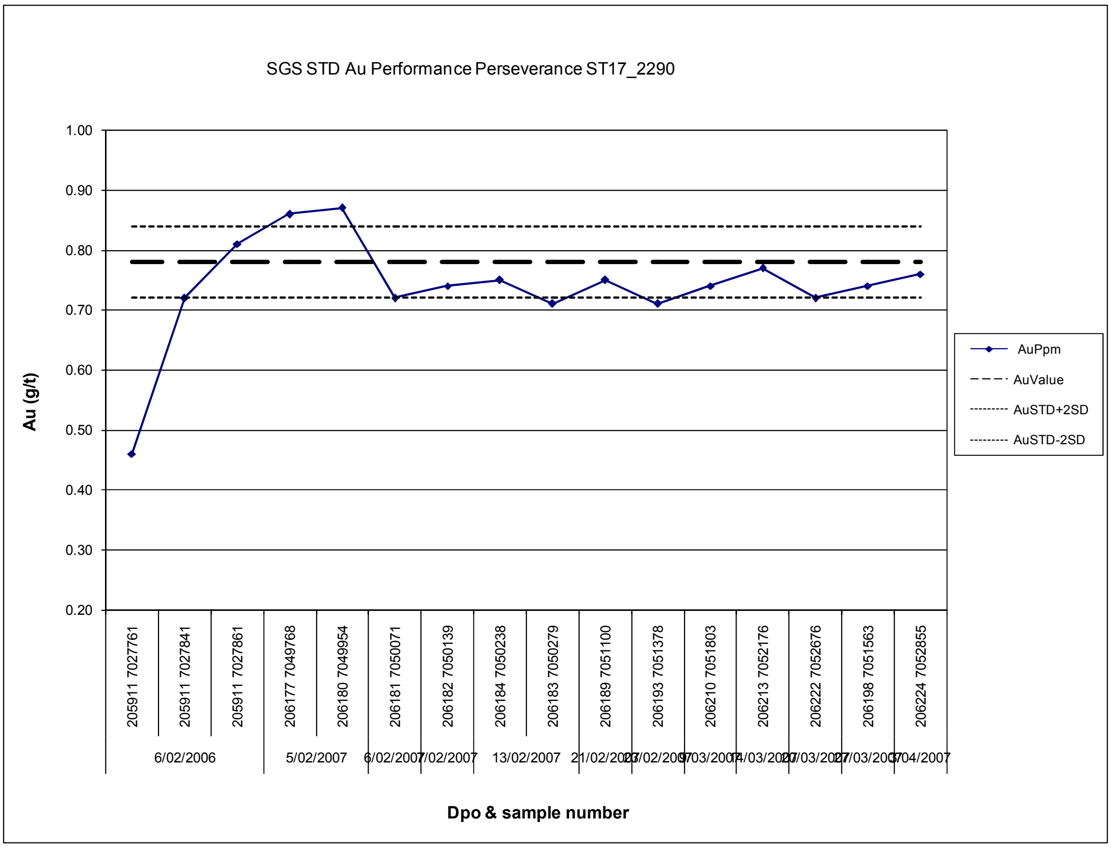
Appendix A - Standards



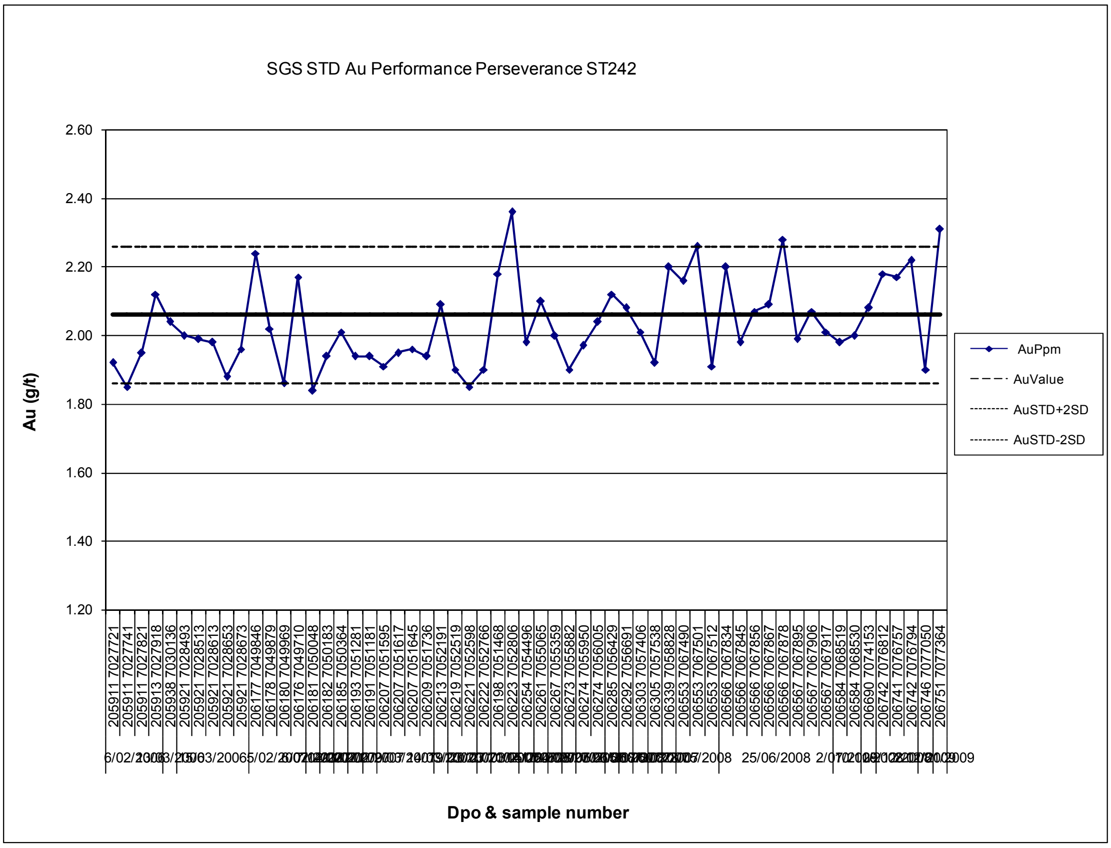


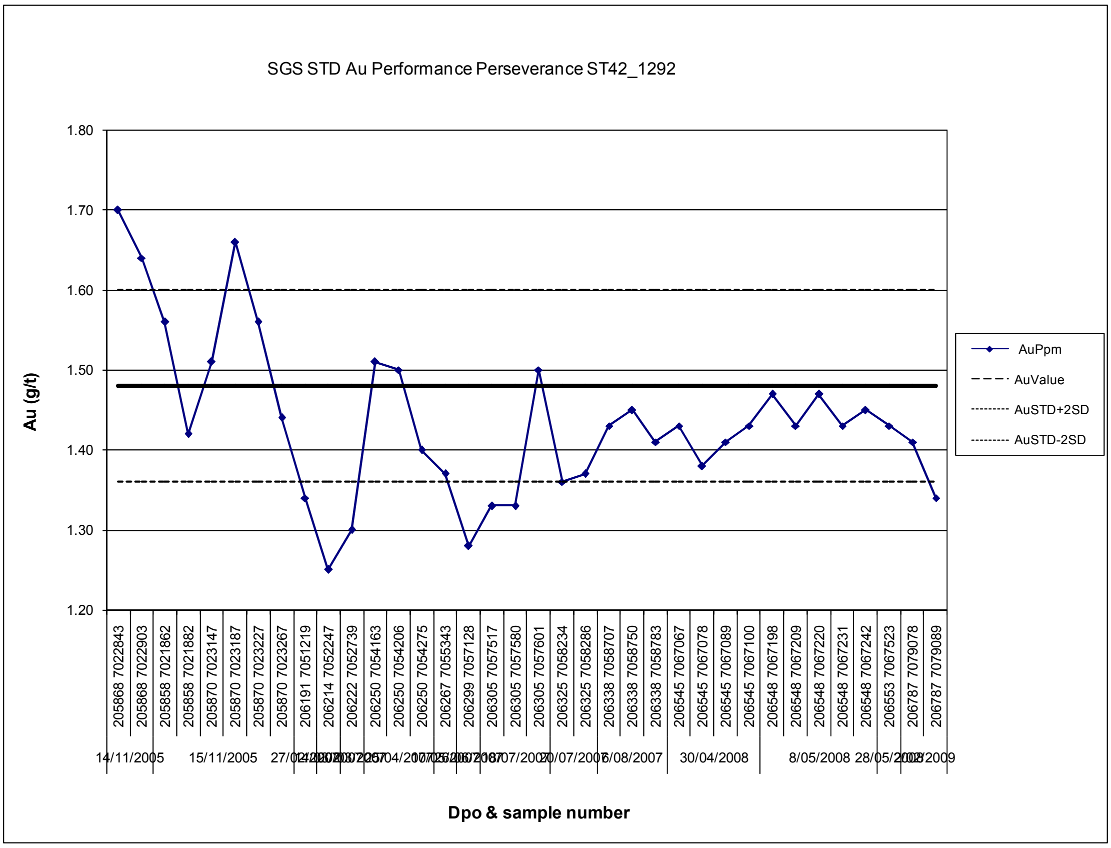
Appendix A - Standards



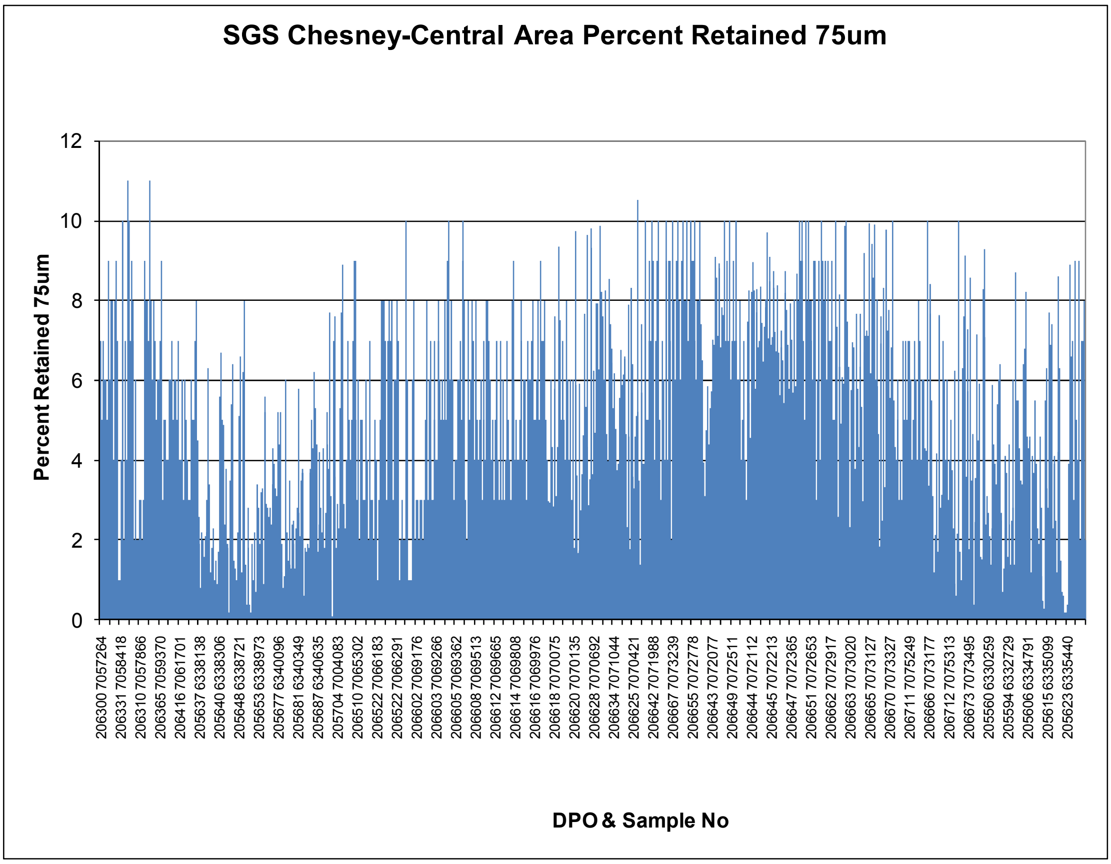


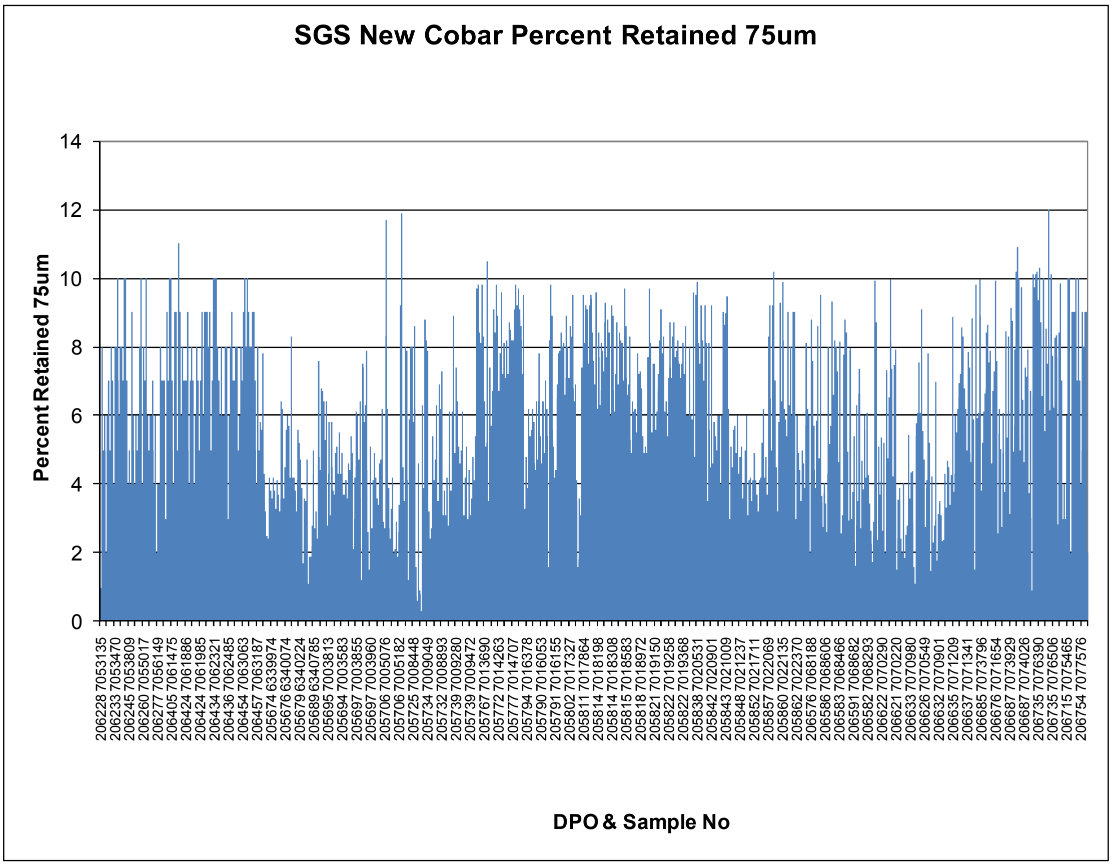
Appendix A - Standards



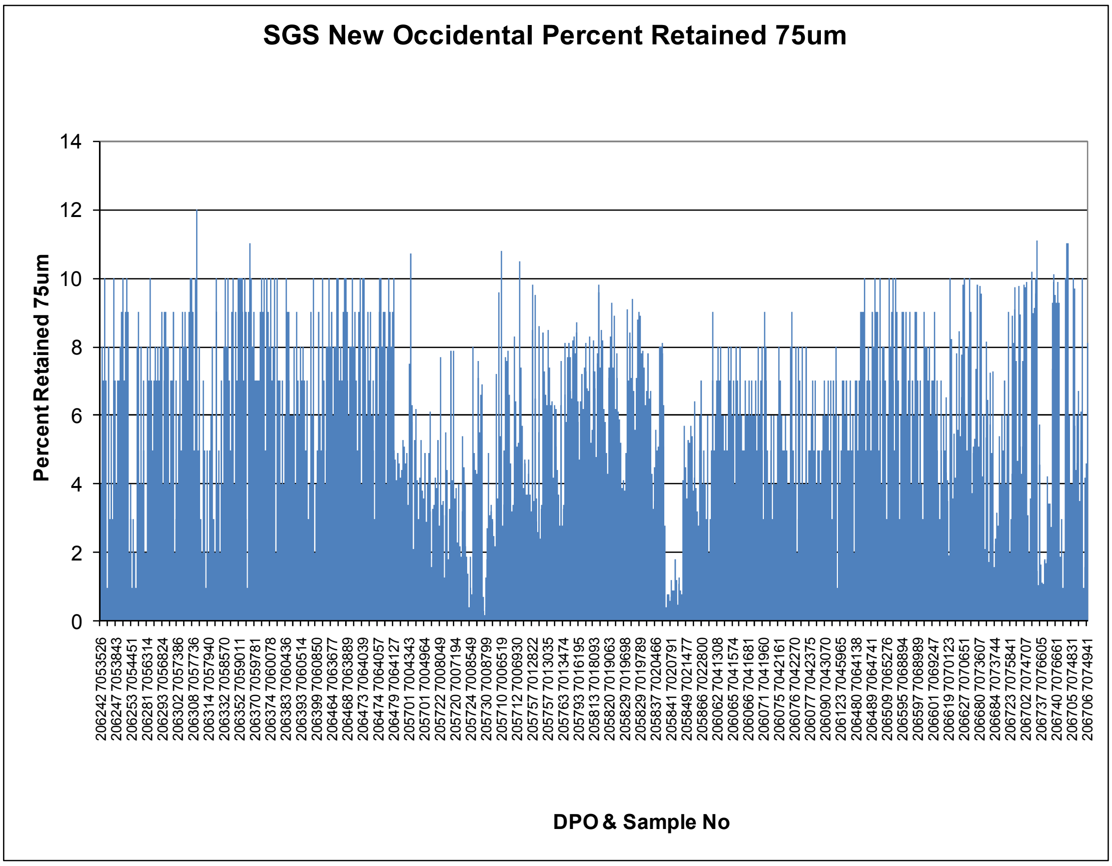


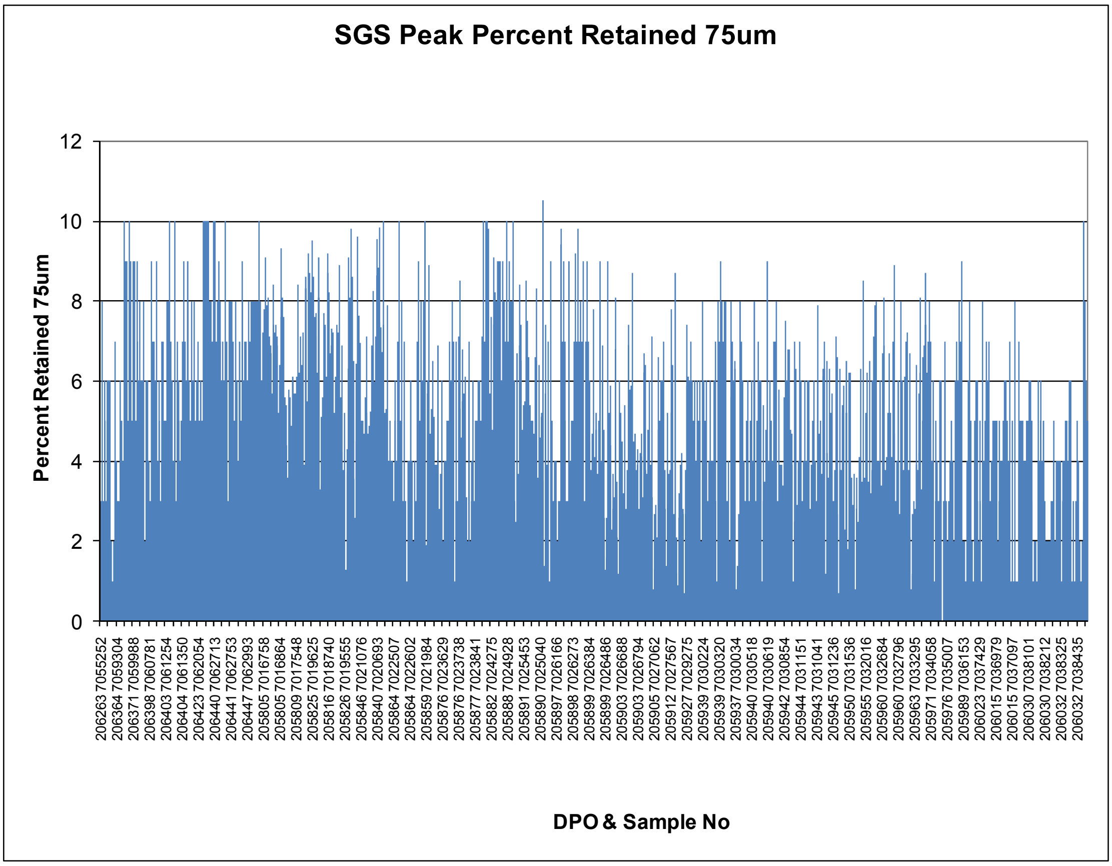
Appendix A - Standards



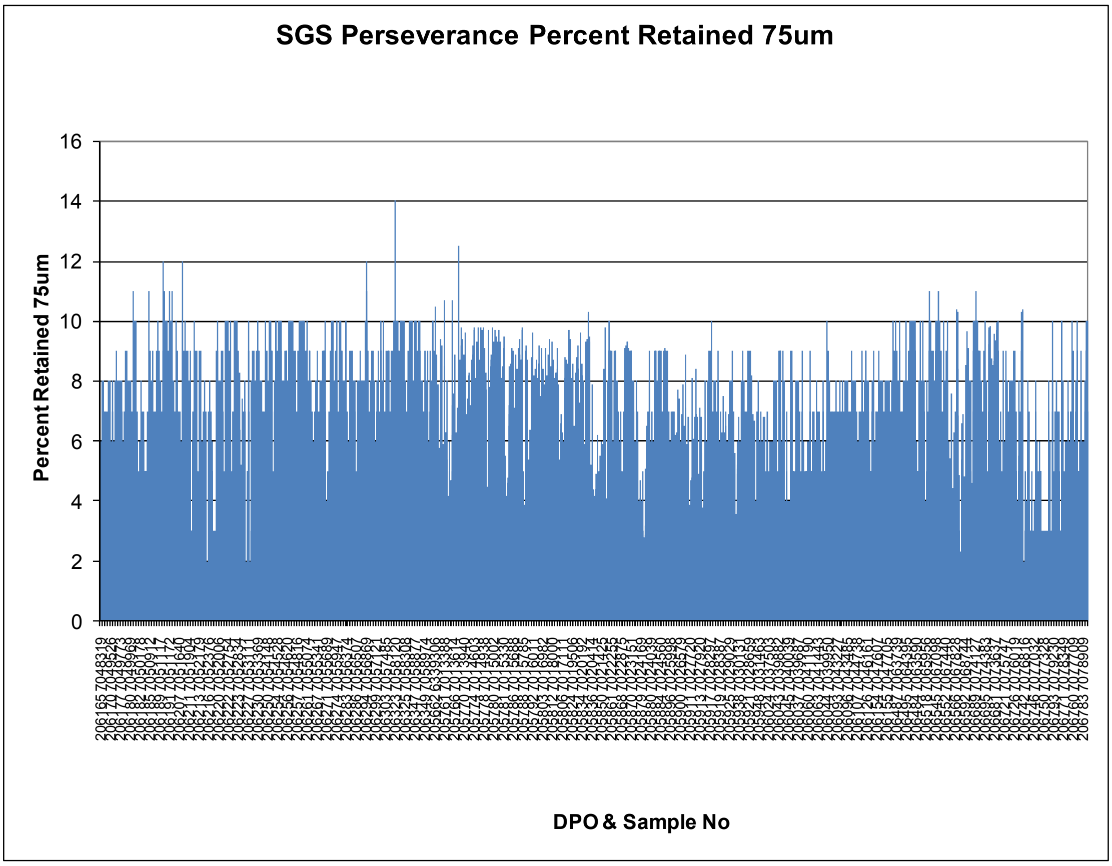


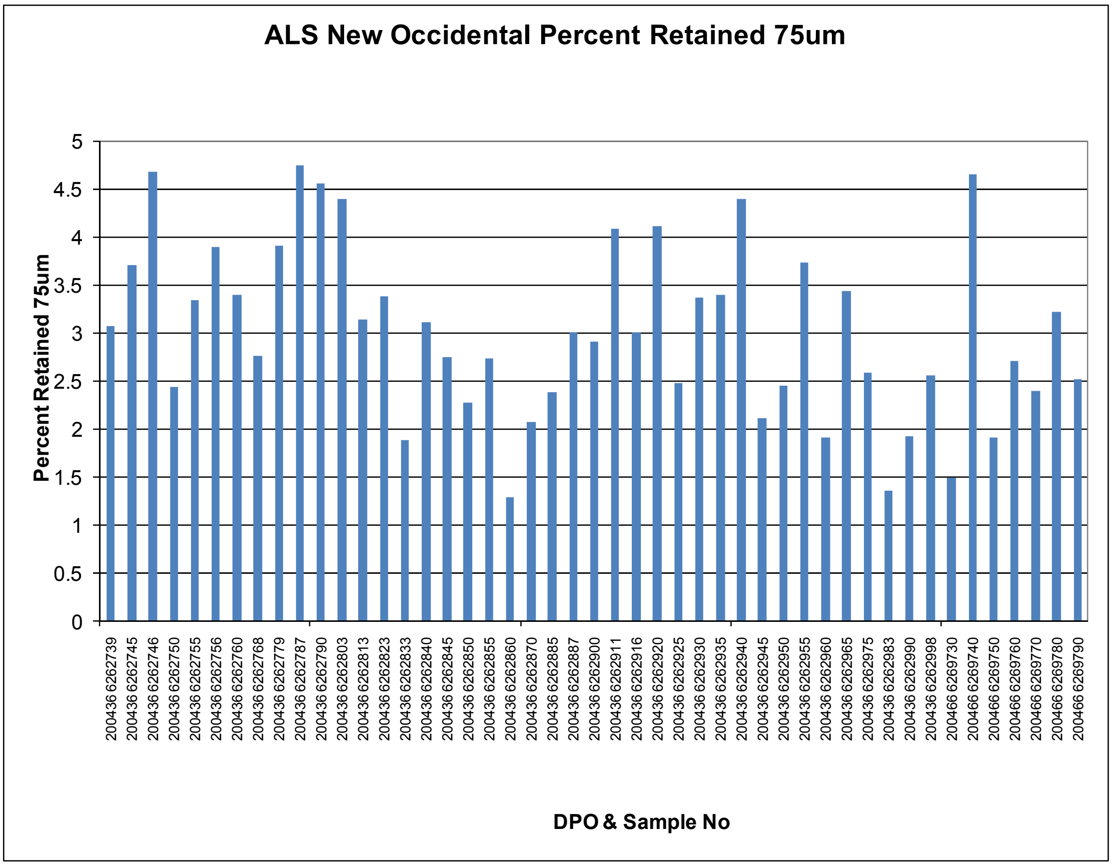
Appendix A - Standards



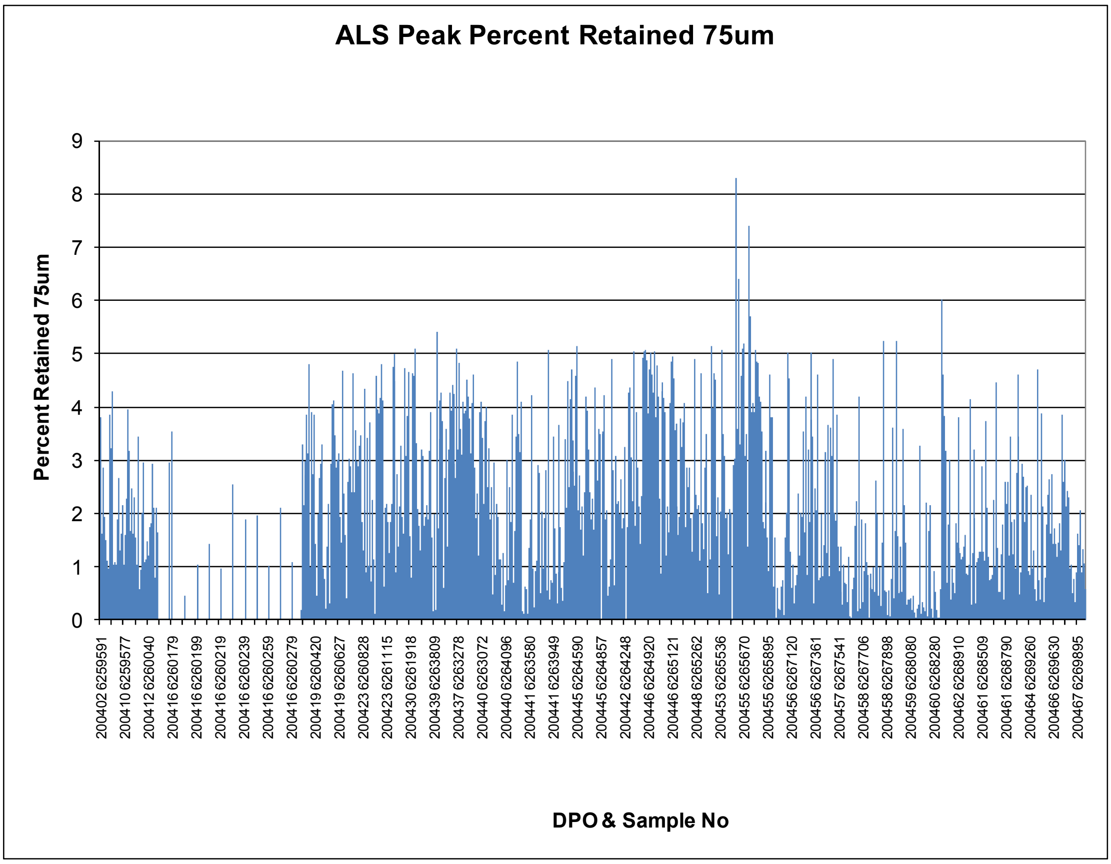


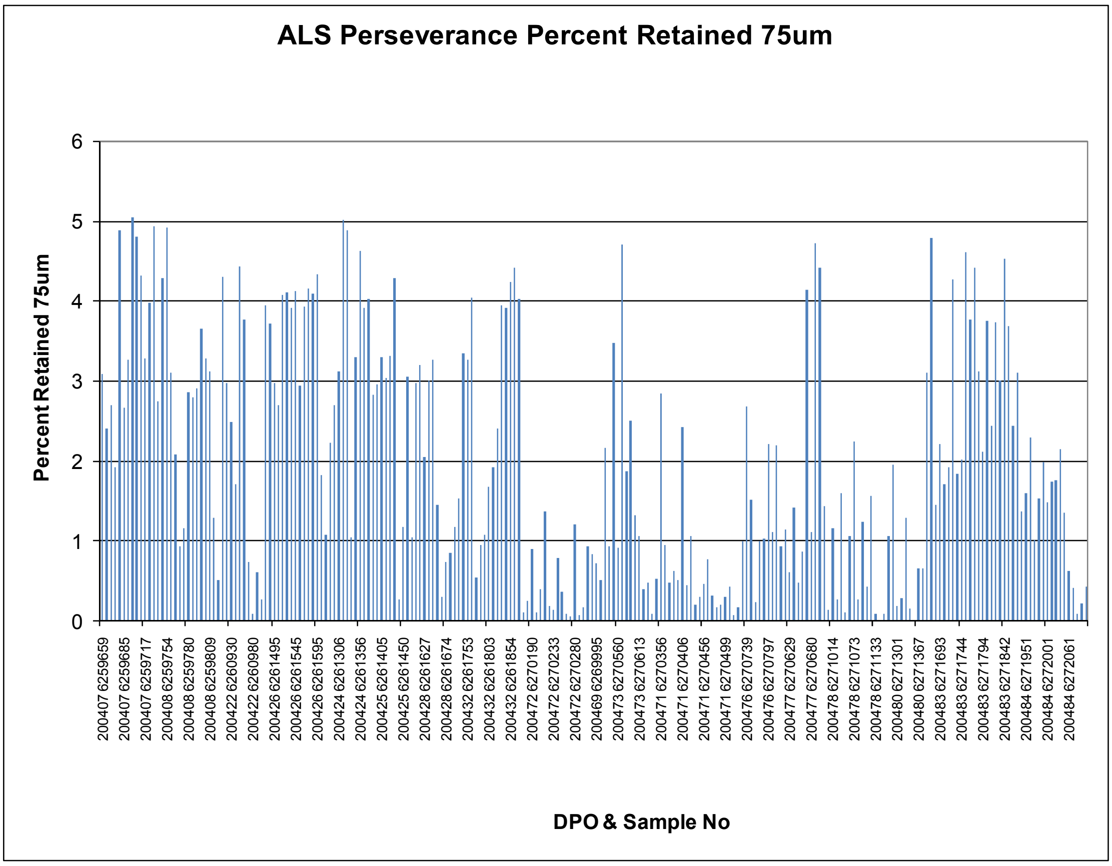
Appendix A - Standards



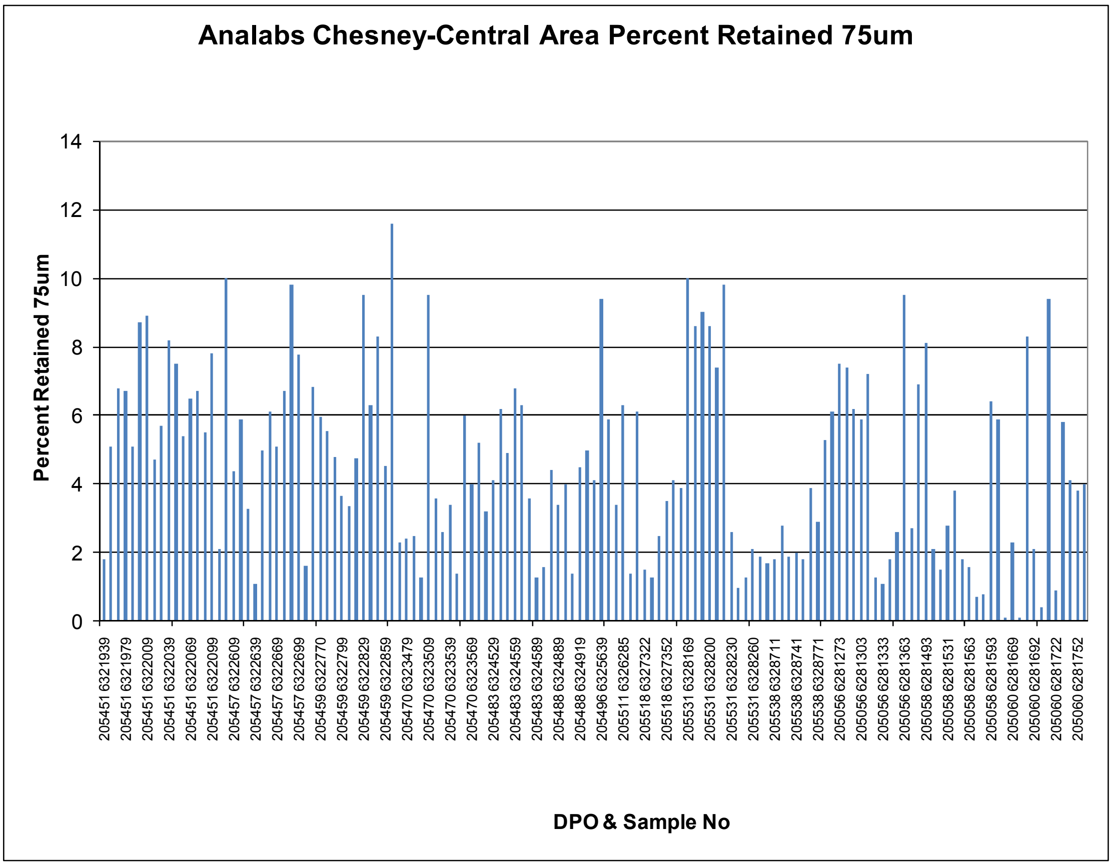


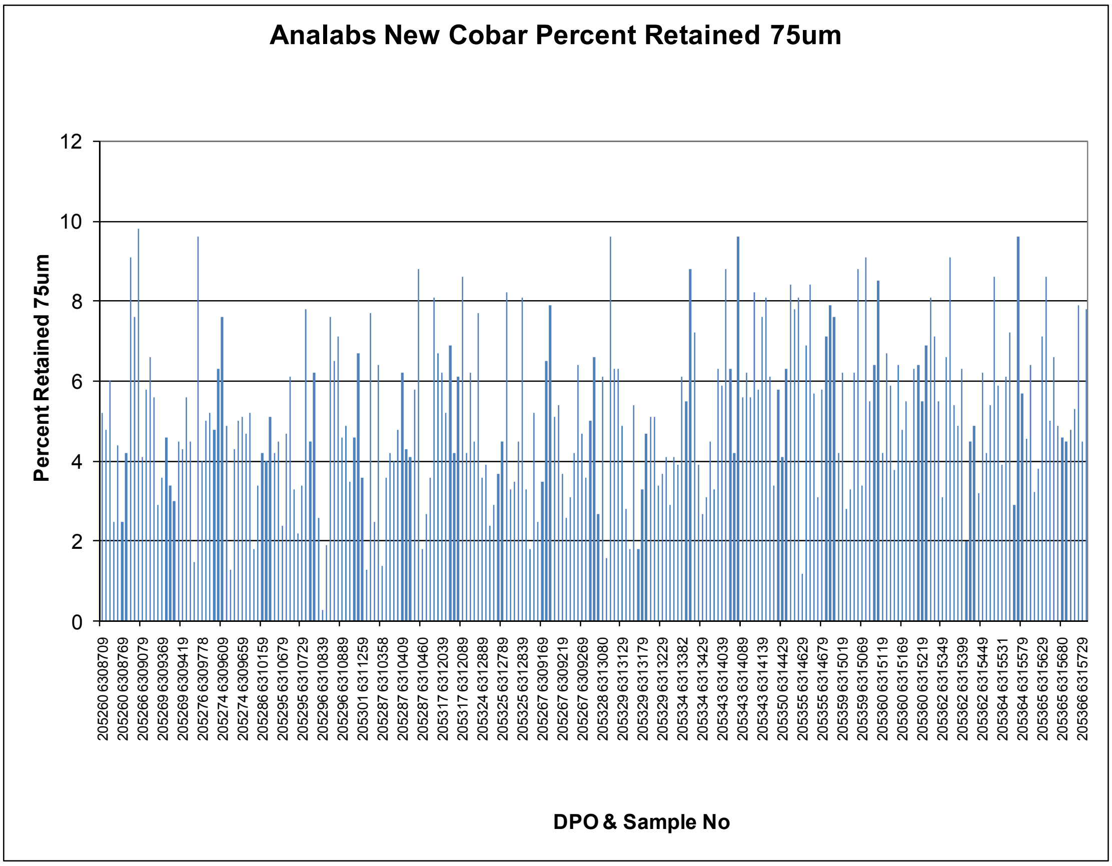
Appendix A - Standards



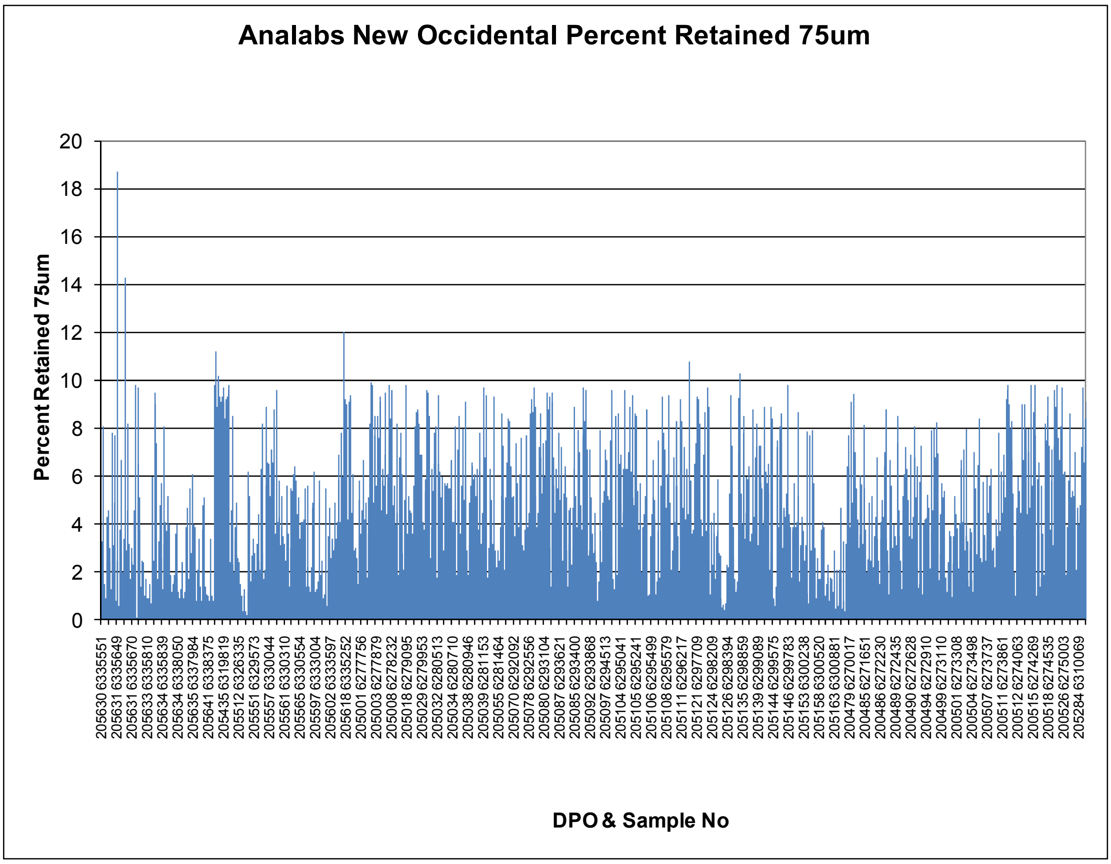


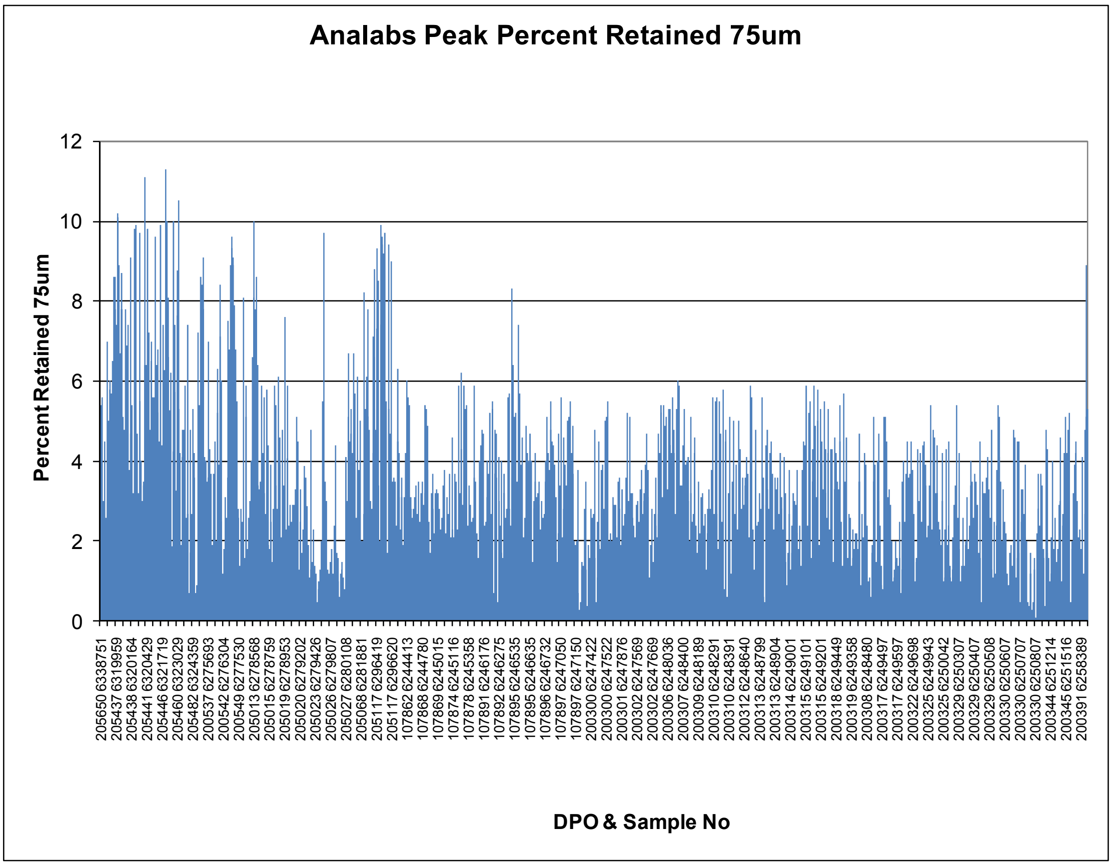
Appendix A - Standards





Appendix A - Standards





Appendix A - Standards