Form 43-101F1

TECHNICAL REPORT

Mineral Resource and Reserve Estimate for the

Sukari Gold Project, Egypt

6 December 2010

Andrew Pardey
General Manager
Pharaoh Gold Mines NL
361 El-Horreya Road
Sedi Gaber
Alexandria
Egypt

Richard Osman
Assistant Mine Manager
Pharaoh Gold Mines NL
361 El-Horreya Road
Sedi Gaber
Alexandria
Egypt

Nic Johnson
Consulting Geologist
Hellman & Schofield Pty Ltd
102 Colin Street
West Perth
Western Australia 6005
Australia

Table of Contents

1 Summary 1
1.1 Introduction 1
1.2 Location 2
1.3 Ownership 2
1.4 Geology 3
1.5 Mineral Resources 4
1.6 Mineral Reserve 5
1.7 Mining 6
1.8 Metallurgy 8
1.9 Process Plant 9
1.10 Services and Infrastructure 10
1.11 Tailings Storage Facility 11
1.12 Environmental 11
1.13 Project Implementation 12
1.14 Operating Cost Estimate 12

2 Introduction and Terms of Reference 14
2.1 Scope of the Report 14
2.2 Principal Sources of Information 14
2.3 Qualifications and Experience 14
2.4 Independence 15
2.5 Abbreviations 16

3 Reliance on other Experts 17

4 Property Description and Location 18
4.1 Background Information on Egypt 18
4.1.1 Geography 18
4.1.2 Political System 19
4.1.3 Legal System 19
4.1.4 Economy 19
4.1.5 Transport and Communications 19
4.2 Project Location 20
4.3 The Project Tenement and Area 20
4.4 Royalties, Payments, Agreements and Encumbrances 22

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography 23
5.1 Access 23
5.2 Climate 23
5.3 Physiography 23
5.4 Local Resources and Infrastructure 23

6 History 25
6.1 Ownership History 25
6.2 Exploration History 25
6.3 Resource History 25
6.4 Production History 27

7 Geological Setting 31
7.1 Regional Geological Setting 31
7.2 Local Geology 32

8 Deposit Types 34

9 Mineralisation 35
9.1 Introduction 35
9.2 Mineralisation Geometry 35
9.3 Deformation Kinematics during Mineralisation 36
9.4 Geometry of Vein Arrays and Shears 37
9.5 Sulphide Development 38
9.6 Gold 39
9.7 Deposit-Scale Architecture 40
9.8 Post-Mineralisation Structures 43
9.9 Alteration 43
10 Exploration 45
10.1 Introduction 45
10.2 Exploration Methods Used 45
10.3 Use of Contractors 45
10.4 Interpretation of Exploration Information 46
10.4.1 Introduction 46
10.4.2 Geological Mapping 46
10.4.3 Rock Chip Sampling 46
10.4.4 Trenches 46
10.5 Data Reliability 46
11 Drilling 47
11.1 Introduction 47
11.2 Survey Controls 48
11.3 Topographic Survey Data 49
11.4 Down-hole Surveys 50
12 Sampling Methods 52
12.1 Surface Mapping and Sampling 52
12.2 Underground Mapping and Sampling 52
12.3 Logging and Sampling, RC Holes 52
12.4 Logging and Sampling, Core Holes 55
12.5 Sample Quality 57
13 Sample Preparation, Analysis and Security 59
13.1 Sample Preparation 59
13.1.1 Core Samples 59
13.1.2 RC Samples 59
13.2 Assaying 60
13.3 Rock Density Data 61
13.3.1 Site Measurements 61
13.3.2 Independent Checks 62
13.4 Quality Control 62
14 Data Verification 63
14.1 The Resource Dataset 63
14.1.1 Trench Samples 63
14.1.2 Surface Rock Chip Traverses 63
14.1.3 Drillhole Data 63
14.2 Sampling and Assaying pre August 2000 63
14.2.1 Routine Internal Laboratory Repeat Analyses 63
14.2.2 Internal Routine Fire Assay Checks 66
14.2.3 Reference Samples 67
14.2.4 Independent Checks 68
14.2.5 Further Independent Check on Assay Accuracy 72

14.3 Sampling and Assaying, between August 2000 and 2003 73
14.3.1 Internal Routine Laboratory Repeat Analyses 74
14.3.2 Internal Routine Fire Assay Checks 76
14.3.3 Certified Reference Samples 78
14.4 Sampling and Assaying, 2005 to 25th January 2009 81
14.4.1 Routine Laboratory Repeat Analyses 81
14.4.2 Routine Fire Assay Checks 83
14.4.3 Blanks 83
14.4.4 Certified Reference Samples 85
14.4.5 Field Duplicates, Diamond 95
14.4.6 Field Duplicates, RC 95
14.4.7 Re-assaying of Crushed Material 98
14.4.8 Duplicate Assaying of Pulverised Material 99
14.5 Sampling and Assaying, 25th January 2009 to 31st May 2010 100
14.5.1 Routine Laboratory Repeat Analyses 101
14.5.2 Routine Fire Assay Checks 103
14.5.3 Blanks 103
14.5.4 Certified Reference Samples 105
14.5.5 Field Duplicates, Diamond 115
14.5.6 Field Duplicates, RC 115
14.5.7 Re-assaying of Crushed Material 118
14.5.8 Duplicate Assaying of Pulverised Material 119
14.6 Conclusions 120
15 Adjacent Properties 121
16 Mineral Processing and Metallurgical Testing 122
16.1 Introduction 122
16.2 Metallurgical Sampling 122
16.3 Metallurgical Testwork 126
16.3.1 Process Water 126
16.3.2 Comminution 126
16.3.3 Gravity Separation 128
16.3.4 Whole of Ore Cyanidation 128
16.3.5 Flotation 129
16.3.6 Cyanidation - Flotation Tailing 130
16.3.7 Cyanidation - Flotation Concentrate 131
16.4 Testwork Summary 132
16.5 Process Plant 133
17. MINERAL RESOURCES AND MINERAL RESERVES 136
17.1. Mineral Resources 136
17.1.1. Introduction 136
17.1.2. Indicator Kriging for Recoverable Resource Estimation 136
17.1.3. The Resource Dataset 137
17.1.4. Geological Interpretation and Domaining of Data 137
17.1.5. Oxidation Interpretation 142
17.1.5.1. Oxidation 142
17.1.6. Exploratory Data Analysis 144
17.1.7. Spatial Continuity Analysis 150
17.1.7.1. Measures of Spatial Continuity 150
17.1.7.2. Sukari Indicator Variogram Models 150
17.1.8. Indicator Kriging Parameters 151
17.1.9. Block Support Adjustment (Variance Adjustment) 157
17.1.9.1. General 157
17.1.9.2. The Variance Adjustment 157

17.1.9.3. Shape of the Block grade Distribution 158
17.1.9.4. The Information Effect 158
17.1.9.5. Variance Adjustments Applied to the Sukari Model 158
17.1.10. Bulk Densities 159
17.1.11. Resource Classification 159
17.1.12. Resource Estimates 160
17.1.13. Plots of the Sukari Resource Model 161
17.2. Mineral Reserves 170
18.1 Mining 172
18.1.1 Mining Method 172
18.1.1.1 Load and Haul 172
18.1.1.2 Grade Control 173
18.1.1.3 Mining Dilution and Ore Loss 173
18.1.1.4 Drill and Blast 174
18.1.1.5 Other Mining Activities 174
18.1.2 Geotechnical Considerations 174
18.1.3 Hydrogeology and Hydrology Input 176
18.1.4 Pit Optimisation 177
18.1.4.1 Model Preparation 177
18.1.4.2 Mining Costs 177
18.1.4.3 Processing Costs 178
18.1.4.4 Administration Costs 178
18.1.4.5 Metallurgical Recoveries 179
18.1.4.6 Optimisation Results 179
18.1.5 Pit Designs 183
18.1.5.1 Design Basis 183
18.1.5.2 Haul Road Designs 183
18.1.5.3 Pit Slopes 183
18.1.6 Waste Dumps 185
18.1.7 10 Year Mining and Treatment Schedule 185
18.2 Proposed Processing Operations 190
18.2.1 Introduction 190
18.2.2 Primary Crushing and Stockpiling 190
18.2.3 Grinding and Classification 191
18.2.4 Leaching and Carbon in Leach (CIL) 192
18.2.4.1 Whole of Ore and Flotation Tails CIL 193
18.2.4.2 Flotation Concentrate CIL 194
18.2.5 Flotation 195
18.2.6 Concentrate Thickening and Regrind 195
18.2.7 Tailings Thickening and Disposal 195
18.2.8 Gold Recovery 196
18.2.9 Carbon Regeneration 196
18.2.10 Reagents 197
18.3 Tailings Storage Facility 197
18.3.1 Introduction 197
18.3.2 Geotechnical Investigation 198
18.3.3 Tailings Characteristics 198
18.3.4 Site Water Management 198
18.3.5 Tailings Storage Facility (TSF) 199
18.4 Infrastructure and Services 199
18.4.1 Plant Site Geotechnical Investigations 199
18.4.1.1 Introduction 199
18.4.1.2 Foundation Conditions 200
18.4.1.3 Geotechnical Issues 200
18.4.1.4 Safe Bearing Capacity 201
18.4.2 Buildings 201
18.4.3 Communications 201
18.4.4 Security 202
18.4.5 Power Supply 202

18.4.6 Water Supply 203
18.5 Personnel 203
18.6 Markets 204
18.7 Contracts 204
18.8 Environmental Considerations 204
18.8.1 Introduction 204
18.8.2 Legislative Background 205
18.8.3 Baseline Survey 205
18.9 Taxes, Duties and Royalties 205
18.10 Operating Costs 205
18.10.1 Mining Costs 206
18.10.2 Processing Costs 206
18.10.3 Labour 207
18.11 Economic Analysis 208
18.11.1 Project Economics 208
19 References 210
20 Date and Signatures 213

List of Tables

Table 1.1 – Estimated Mineral Resources 4
Table 1.2 – Break-even Cut-off Grade Calculation 5
Table 1.3 – Sukari Open Pit Mineral Reserve Estimate 6
Table 1.4 – Sukari Open Pit Mineral Reserve – By Design Stages 6
Table 1.5 – Recoveries by Ore Type and Processing Stage 8
Table 1.6 –Sulphide Ore at 5.0 Mt/y 14
Table 2.1 – List of Abbreviations 16
Table 4.1 – Sukari Project - List of Tenement Corners 21
Table 6.1 – Resource Estimate H&S November 2001 26
Table 6.2 – Resource Estimate H&S April 2003 26
Table 6.3 – Resource Estimate H&S November 2006 26
Table 6.4 – Resource Estimate H&S February 2007 27
Table 6.5 – Recent Production from SGM 30
Table 12.1 – Summary details of Sukari Drillhole Dataset 48
Table 16.1 – Metallurgical Testwork Samples 122
Table 16.2 – Mineralisation Composite Head Assays 125
Table 16.3 – Comminution Test Results 126
Table 16.4 – Comparison of CIL and Direct Cyanidation Test Results 128
Table 16.5 – Flotation Test Summary 129
Table 16.6 – Flotation Tails Cyanidation Results 130
Table 16.7 – Recoveries by Ore Type and Processing Stage 132
Table 16.8 – Summary of Process Plant Design Criteria 134
Table 16.9 – Major Equipment Items 134
Table 17.1.5.1_1 – Sukari Oxidation Logging Codes 143
Table 17.1.8_1 – Model Framework and Kriging Parameters 151
Table 17.1.8_2 – Indicator Class Means, Domain 1 154
Table 17.1.8_3 – Indicator Class Means, Domains 2 and 3 154
Table 17.1.8_4 – Indicator Class Means, Domains 4 and 5 155
Table 17.1.8_5 – Indicator Class Means, Domain 6 155
Table 17.1.8_6 – Indicator Class Means, Domain 7 156
Table 17.1.8_7 – Indicator Class Means Domain 8 156
Table 17.1.9.5_1 – Variance Adjustments 159
Table 17.1.10_1 – Bulk Densities 159
Table 17.1.12_1 – Estimated Mineral Resources 160
Table 17.2_1 – Sukari Open Pit Mineral Reserve Estimate – By Design Stages 170
Table 18.1.2_1 – Slope Design Specifications 176
Table 18.1.4.2_1 – Base Case Mining Unit Costs for Pit Optimisations 178
Table 18.1.4.5_1 – Pit Optimisation Metallurgical Recoveries and Processing Costs 179
Table 18.1.4.6_1 – Sukari Pit Optimisation Results 181
Table 18.1.5.2_1 – Final Pit In-Pit Ramp Design Specifications 183
Table 18.1.5.3_1 – Pit Slope Design Parameters 183
Table 18.1.7_1 – Mining and Treatment Schedule Annual Summary 187

Table 18.5_1 – Total Operations Manpower and Build-up 203
Table 18.10_1 – Currency Exchange Rates 205
Table 18.10.2_1 – Sulphide Ore at 5.0 Mt/y 207
Table 18.10.3_1 – Total Unit Labour Costs 208
Table 18.12.1_1 – 10 Year Schedule Cashflow Summary 209

List of Figures

Figure 1.1 – Current Sukari Exploitation Licence and Sukari Project Location 2
Figure 1.2 – Sukari Project Location 3
Figure 1.3 – Sukari Hill with designated geographical Zones, looking to the SE 4
Figure 1.4 – Site Layout Plan 7
Figure 1.5 – Process Block Flow Diagram – Stage 1 – Oxide Ore 9
Figure 1.6 – Process Block Flow Diagram – Stage 2 – Mixed Ore 9
Figure 1.7 – Process Block Flow Diagram – Stage 3 – Sulphide Ore 9
Figure 4.1 – Sukari Location and Road Access 20
Figure 4.2 – Concession Area 21
Figure 6.1 – Growth in Contained Ounces at each Resource Estimate Update 27
Figure 6.2 – View of Historic Mine Workings 28
Figure 6.3 – Underground Workings 29
Figure 7.1 – Sukari District Geology 31
Figure 7.2 – Sukari Hill with designated Geographical Zones 32
Figure 7.3 – Interpreted Geology, Sukari Porphyry and surrounding Country-rocks. 33
Figure 9.1 – Geometry of Main Zone and Hapi Zone within the Host Porphyry 36
Figure 9.2 – Sukari Main Reef – Sketch Diagram 37
Figure 9.3 – Oblique Schematic Section 10,300mN to 10,550mN 41
Figure 9.4 – Antithetic Reverse Shearing 42
Figure 9.5 – View of Sukari Ridge showing probable Antithetic Shears 43
Figure 9.6 – Progressive Alteration of Serpentinite 44
Figure 11.1 – Drillhole Locations 48
Figure 11.2 – Sukari Topography DTM 50
Figure 12.1 – Surface Rock Chip Sampling 53
Figure 12.2 – Histogram of RC Sample Recoveries 54
Figure 12.3 – RC Sample Recovery vs Gold Grade 55
Figure 12.4 – Histogram of Length-Weighted Core Recoveries 56
Figure 12.5 – Gold Grade vs Core Recovery 57
Figure 13.1 – Percentage of Particles passing 75µm 60
Figure 13.2 – Bulk Density Measurements Histogram 61
Figure 14.1 – Scatter plot original assays vs repeat assays, all data 64
Figure 14.2 – Scatter plot original assays vs repeat assays, subset 0.1 to 20.0g/t Au 64
Figure 14.3 – QQ Plot, original assay vs repeat assay subset 0.1 to 20.0g/t Au 65
Figure 14.4 – Precision plot original assay vs repeat assay, subset 0.1 to 20.0g/t Au 65
Figure 14.5 – Scatter Plot original vs FA repeats 66
Figure 14.6 – QQ Plot, original assays vs FA repeats 67
Figure 14.7 – Sukari Reference Samples (submitted with drillcore) 68

Figure 14.8 – Scatter Plot, Original Assays vs Aqua Regia Check Assays (MRL) 69
Figure 14.9 – QQ Plot, Original Assays vs Aqua Regia Check Assays (MRL) 70
Figure 14.10 – Scatter Plot, Original Assays vs FA (S & R) 70
Figure 14.11 – QQ Plot, Original Assays vs FA (S & R) 71
Figure 14.12 – Gannet Standards (submitted with check samples) 71
Figure 14.13 – Scatter Plot, Original Assays vs Pulp Re-assays (S & R) 72
Figure 14.14 – QQ Plot, Original Assays vs Pulp Re-assays (S & R) 73
Figure 14.15 – Gannet Standards (submitted with pulp reassays) 73
Figure 14.16 – Scatter Plot, Original Assays vs Repeat Assays, all Data 75
Figure 14.17 – QQ Plot, Original Assays vs Repeat Assays, all Data 75
Figure 14.18 – Precision Plot, Original Assays vs Repeat Assays, all Data 76
Figure 14.19 – Scatter Plot, Original vs FA Repeats 77
Figure 14.20 – QQ Plot, Original vs FA Repeats 77
Figure 14.21 – Gannet Standard Control Chart ST17 79
Figure 14.22 – Gannet Standard Control Chart ST73 79
Figure 14.23 – Gannet Standard Control Chart ST09 80
Figure 14.24 – Gannet Standard Control Chart ST274 80
Figure 14.25 – Scatter Plot, Original Assays vs Repeat Assays, all Data 82
Figure 14.26 – QQ Plot, Original Assays vs Repeat Assays, all Data 82
Figure 14.27 – Precision Plot, Original Assays vs Repeat Assays, all Data 83
Figure 14.28 – Scatter Plot, Original Assays vs FA Repeats, all Data 84
Figure 14.29 – QQ Plot, Original Assays vs FA Repeats, all Data 84
Figure 14.30 – Blanks assayed with Drill Samples 85
Figure 14.31 – Control Chart for ST353 87
Figure 14.32 – Control Chart for ST17 88
Figure 14.33 – Control Chart for ST01 89
Figure 14.34 – Control Chart for ST09_0285 90
Figure 14.35 – Control Chart for ST09_3320 91
Figure 14.36 – Control Chart for ST355 92
Figure 14.37 – Control Chart for ST274 93
Figure 14.38 – Control Chart for ST279 94
Figure 14.39 – Scatter Plot, Original vs Duplicate Samples Diamond Core 95
Figure 14.40 – QQ Plot, Original vs Duplicate Samples Diamond Core 96
Figure 14.41 – Precision Plot, Original vs Duplicate Samples Diamond Core 96
Figure 14.42 – Scatter Plot, Original vs Duplicate Samples RC 97
Figure 14.43 – QQ Plot, Original vs Duplicate Samples RC 97
Figure 14.44 – Precision Plot, Original vs Duplicate Samples RC 98
Figure 14.45 – Scatter Plot, Original vs Resplit crushed Material 98
Figure 14.46 – QQ Plot, Original vs Resplit crushed Material 99
Figure 14.47 – Scatter Plot, Original vs Resplit pulverised Material 99
Figure 14.48 – QQ Plot, Original vs Resplit pulverised Material 100
Figure 14.49 – Scatter Plot, Original Assays vs Repeat Assays 101
Figure 14.50 – QQ Plot, Original Assays vs Repeat Assays 102
Figure 14.51 – Precision Plot, Original Assays vs Repeat Assays 103
Figure 14.52 – Scatter Plot, Original Assays vs FA Repeats, all Data 104
Figure 14.53 – QQ Plot, Original Assays vs FA Repeats 104

Figure 14.54 – Blanks assayed with Drill Samples 105
Figure 14.55 – Control Chart for ST353 107
Figure 14.56 – Control Chart for ST17 108
Figure 14.57 – Control Chart for ST01 109
Figure 14.58 – Control Chart for ST09_0285 110
Figure 14.59 – Control Chart for ST09_6382 111
Figure 14.60 – Control Chart for ST355 112
Figure 14.61 – Control Chart for ST383 113
Figure 14.62 – Control Chart for ST279 114
Figure 14.63 – Scatter Plot, Original vs Duplicate Samples, Diamond Core 115
Figure 14.64 – QQ Plot, Original vs Duplicate Samples, Diamond Core 116
Figure 14.65 – Precision Plot, Original vs Duplicate Samples, Diamond Core 116
Figure 14.66 – Scatter Plot, Original vs Duplicate Samples, RC 117
Figure 14.67 – QQ Plot, Original vs Duplicate Samples, RC 117
Figure 14.68 – Precision Plot, Original vs Duplicate Samples, RC 118
Figure 14.69 – Scatter Plot, Original vs Resplit crushed Material 118
Figure 14.70 – QQ Plot, Original vs Resplit crushed Material 119
Figure 14.71 – Scatter Plot, Original vs Resplit pulverised Material 119
Figure 14.72 – QQ Plot, Original vs Resplit pulverised Material 120
Figure 16.1 – Long Section Displaying Metallurgical Sample Intervals 124
Figure 16.2 – Concentrate Cyanidation Optimisation 131
Figure 16.3 – Process Block Flow Diagram – Stage 1 – Oxide Ore 132
Figure 16.4 – Process Block Flow Diagram – Stage 2 – Mixed Ore 133
Figure 16.5 – Process Block Flow Diagram – Stage 3 – Sulphide Ore 133
Figure 17.1.4_1 – Domain 1, Footwall Waste 139
Figure 17.1.4_2 – Domain 2, Oxide Zone Sukari Porphyry South 139
Figure 17.1.4_3 – Domain 3, Sulphide Zone Sukari PorphyrySouth 140
Figure 17.1.4_4 – Domain 4, Oxide Zone Sukari Porphyry North 140
Figure 17.1.4_5 – Domain 5, Sulphide Zone Sukari Porphyry North 141
Figure 17.1.4_6 – Domain 6, Horus Zone 141
Figure 17.1.4_7 – Hangingwall Porphyry Mineralised Zones 142
Figure 17.1.6_1 – Histograms Au grade Domain 2 and 3 145
Figure 17.1.6_2 – Histograms Au grade Domain 4 and 5 145
Figure 17.1.6_3 – Histogram Au grade Domain 6 147
Figure 17.1.6_4 – Histograms Au grade Domain 7 148
Figure 17.1.6_5 – Histogram Au grade Domain 1 - Footwall 149
Figure 17.1.6_6 – Histogram Au grade Domain 8 - Hangingwall 149
Figure 17.1.8_1 – Variogram Models Domain 2 152
Figure 17.1.8_2 – Variogram Models Domain 3 152
Figure 17.1.8_3 – Variogram Models Domain 4 153
Figure 17.1.8_4 – Variogram Models Domain 5 153
Figure 17.1.13_1 – Section 10,325mN, Panel Estimated Proportion and Grades @ 0.7g/t Au Cutoff 162
Figure 17.1.13_2 – Section 11,125mN, Panel Estimated Proportion and Grades @ 0.7g/t Au Cutoff 163
Figure 17.1.13_3 – Section 10,325mN Panel Confidence Categories 164
Figure 17.1.13_4 – Section 11,125mN Panel Confidence Categories 165
Figure 17.1.13_5 – Panel Estimated Grades @ 0.7g/t Au cutoff 1103mRL 166

Figure 17.1.13_6 – Panel Estimated Grades @ 0.7g/t Au cutoff 1001mRL 167
Figure 17.1.13_7 – Panel Confidence Categories 1103mRL 168
Figure 17.1.13_8 – Panel Confidence Categories 1001mRL 169
Figure 18.1.2_1 – Pit Slope Regions (GOA) 175
Figure 18.1.4.2_2 – Haulage Cost Estimate with Depth 178
Figure 18.1.4.6_1 – Optimisation Results 180
Figure 18.1.4.6_2 – Optimisation Results 180

1 SUMMARY

1.1 Introduction

This report provides an updated estimate of Mineral Resources and Reserves for the Sukari Gold Project. The report is intended to comply with disclosure and reporting requirements set forth in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ('NI 43-101') of the Canadian Securities Administrators (including NI 43-101's Companion Policy and Form 43-101F1).

This report is the compilation of work performed by a number of engineering organisations for Centamin Egypt Limited ('Centamin') on its Sukari Gold Project as well as Centamin employees dating back to 2003 and draws heavily on a previous technical report dated 21st May 2009, which report has been filed on SEDAR at www.sedar.com. Technical information relating to project development, mining and processing has not changed significantly since reported in the technical report dated 21st May 2009 and as such is essentially repeated in this report with minor updates.

Pharaoh Gold Mines NL ('PGM') is a subsidiary of Centamin. Ausenco Limited performed a scoping study in 2003 and in May 2005 was engaged to undertake a feasibility study based on the Sukari resource delineated at that time. For various reasons, the Ausenco study was discontinued in July 2006 and their work was compiled into a technical report.

In July 2006, PGM became aware of a redundant high tonnage gold processing plant that was available for sale at the Kori Kollo mine in Bolivia and engaged Roche Process Engineering Pty Ltd ('RPE') to make an assessment of the plant condition and to produce a report and estimate of the cost of relocating the plant to Sukari. The report was favourable and in September 2006 PGM bought the plant.

At the same time PGM engaged RPE to undertake the engineering required to incorporate the Kori Kollo plant into the Sukari flow sheet and to produce a Definitive Feasibility Study ('DFS') incorporating the latest exploration drilling results and updating previous work and estimates carried out by Ausenco and others.

Commissioning of the Stage 1 (CIL) plant commenced in December 2009 along with initial gold production from this circuit. In April 2010 the Stage 2 (Flotation) Circuit commissioning also commenced.

Step-out and infill drilling carried out in 2009 and 2010 continued to identify significant additional mineralisation at Sukari in addition to the commencement of initial grade control (close spaced) drilling in the Sukari porphyry. As a result Hellman & Schofield Pty Ltd was commissioned by Centamin to update the estimate of Mineral Resources in June 2010. The Mineral Reserves were subsequently updated in September 2010.

1.2 Location

The Sukari gold deposit is located in the Eastern Desert of Egypt at 24º 56' 50"N 34º 42' 27"E, about 23 kilometres southwest of the Red Sea coastal town of Marsa Alam. The region has a very long history of mining and exploration carried out through all stages of history, from Pre-dynastic (Ca 3,200 BC), through Ptolemaic, Roman, Arab and British colonial to the present day.

1.3 Ownership

In 1994, PGM negotiated an exploration and mining agreement (Concession Agreement), with the Egyptian Geological Survey and Mining Authority ('EGSMA'; now the Egyptian Minerals and Resource Authority or 'EMRA') and the Egyptian Government, to explore for gold and associated minerals in the Eastern Desert of Egypt (Figures 1.1 and 1.2). The Concession Agreement ('CA') was declared into Law 222 of 1994 and came into effect on the 29th of January 1995 pursuant to which PGM had the sole right to explore and develop gold and associated metal deposits within the concession area.

Figure 1.1
Sukari Exploitation Licence and Sukari Project Location

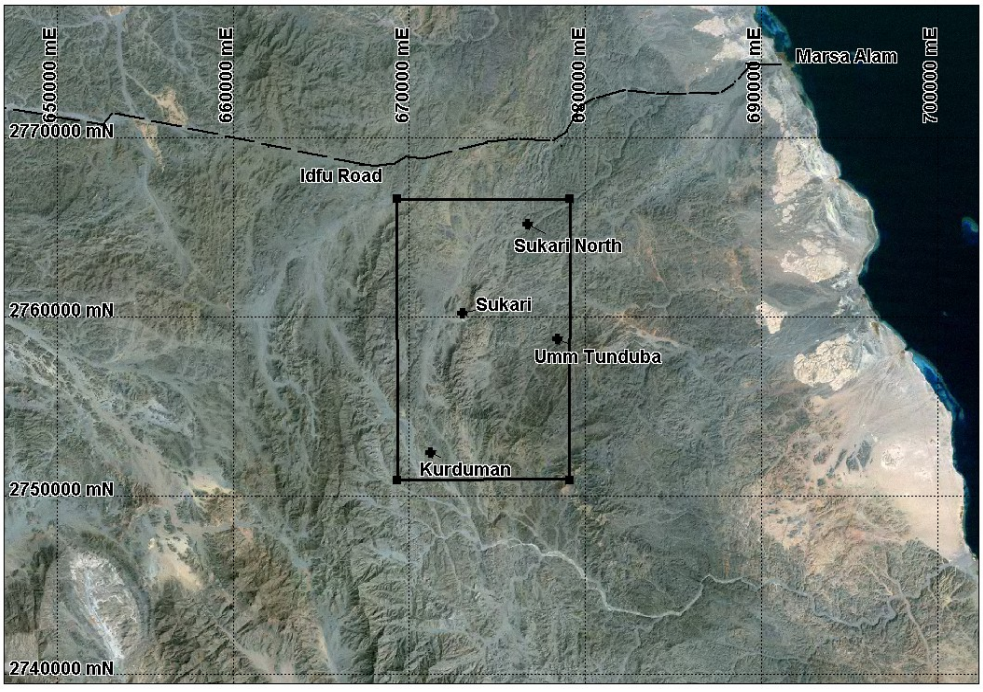


Figure 1.2
Sukari Project Location

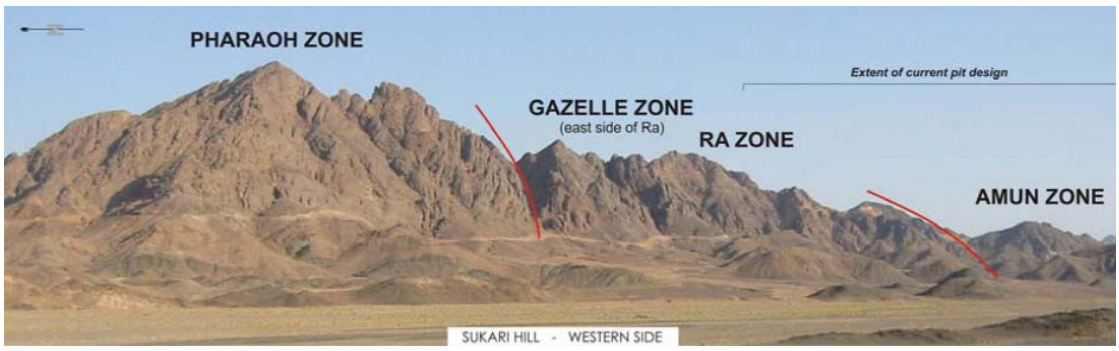
On the 4 November 2001 PGM was formally notified by EMRA that in accordance with the terms of the CA the feasibility study submitted by PGM and dated the 26th of October 2000 relating to the Sukari Gold Project, had been accepted by EMRA and had demonstrated the existence of a "Commercial Discovery" at the Sukari Project and that as such, in accordance with the CA, the Sukari Project had been converted to an Exploitation (Mining) Lease.

In April 2005, PGM recommenced operations on a new Exploitation Lease covering an area of 160km^2, containing the proposed Sukari mine site and surrounding prospects. Under the terms of the CA, the Company has title for a period of 30 years from 24 May 2005, being the date of approval by the Minister of the area covered by the Exploitation Lease, and is renewable at the Company's election for a further period of 30 years.

1.4 Geology

The Sukari felsic porphyry outcrop is located in an easterly dipping sequence of andesite flows, serpentinites and associated volcanoclastic sediments, mainly tuffs and epiclastics. It strikes for 2.3km and is 100 to 600m thick. It forms a jagged-toothed, strong topographic high up to 250m above wadi level (390m ASL) (Figure 1.3). Wadi drainage plains pass to the east and west of the outcrop, and the sharply incised green – brown Red Sea Hills surround that. The area is arid and almost bare of vegetation.

Figure 1.3
Sukari Hill with designated geographical Zones, looking to the SE



The host to gold mineralisation is the Sukari felsic porphyry unit that contains a variable series of sub-units ranging from minor acid and felsic rhyolite and dacite to coarser grained feldspar and quartz porphyries, quartz diorites and granodiorites, dipping from moderate east in the south, through sub-vertical to slightly overturned "hangingwall" contact in the north. The porphyry units are envisaged to have formed during a more acid event in an overall intermediate environment in which some rhyolite – dacite flows were later intruded by high level porphyry as a multiple event (Cavaney, 2005).

1.5 Mineral Resources

Hellman & Schofield Pty Ltd was commissioned by PGM to undertake an estimate of recoverable resources for the Sukari gold prospect. The work satisfies the reporting requirements of the JORC (2004) and CIM (2004) guidelines for reporting mineral resources.

Estimation of recoverable resources has been undertaken using Multiple Indicator Kriging with block support adjustment. The method estimates the histogram of drillhole sample grades in large panels (20m east x 25m north x 6m elevation), using variograms of gold grade indicators at 14 thresholds. The size of model panels was based on drillhole spacing. A block support adjustment was then applied to estimate the histogram of grades of selective mining units in each panel. For Sukari, a selective mining unit measuring 4m east x 8m north x 3m elevation has been assumed. The shape of the local block gold grade distribution has been assumed lognormal and an additional adjustment correction for Information Effect has also been applied, to make allowance for the fact that ore selection will be based on an estimate of the grade of a mining block generated from grade control sampling, not perfect knowledge of block grades.

The recoverable resources within each panel have been allocated a confidence category based on the spatial distribution of samples in the Kriging neighbourhood. This classification scheme takes into account the uncertainty in estimates resulting from the proximity and locations of the informing samples.

Recoverable gold resources at Sukari, estimated to approximately 1,350 metres depth below surface[1] are shown in the following Table 1.1. The resources shown have been adjusted to the end of May 2010 mining surface and for historical underground mining voids.

Table 1.1

Estimated Mineral Resources

Cutoff	Measured		Indicated		Measured plus Indicated		Inferred	
	Mt	g/t Au	Mt	g/t Au	Mt	g/t Au	Mt	g/t Au
0.5	84.01	1.42	151.72	1.47	235.73	1.45	68.9	1.6
0.6	71.29	1.57	130.31	1.62	201.60	1.60	58.4	1.8
0.7	61.23	1.72	112.85	1.77	174.08	1.75	50.1	2.0
0.8	53.05	1.87	98.40	1.92	151.45	1.90	43.7	2.1
0.9	46.27	2.02	86.25	2.07	132.52	2.05	38.3	2.3
1.0	40.54	2.17	75.95	2.22	116.49	2.20	33.8	2.5
1.2	31.44	2.48	59.73	2.53	91.18	2.51	26.7	2.9

1.6 Mineral Reserve

The Mineral Reserves for the Sukari Gold Project were estimated by PGM mining and geological staff. This Mineral Reserve estimate was based upon the Mineral Resource estimate prepared by H&S in August 2010.

The resource model contains an allowance for dilution within the limits of the chosen SMU. To date, only a small amount of ore has been mined and as such the reconciliation results from this ore to be representative of the resource estimate. It is assumed that the resource estimate contains the appropriate dilution allowance and that ore recovery is 100% of the estimate.

The Sukari pit will produce ores of varying stages of weathering. The processing costs are expected to vary with the extent of the weathering and hence the cut-off grade applied also varies with the weathering state. The estimated cut-off grades are summarised in Table 1.2.

Table 1.2

Break-even Cut-off Grade Calculation

	Oxide	Transitional	Sulphide
Revenue (US$900/oz) ($/g Au)	28.94	28.94	28.94
Processing Cost ($/t)	1.36	9.03	9.03
Overhead Cost ($/t)	1.90	1.90	1.90
Processing Recovery (%)	50	93.4	93.4
Incremental Cut-off Grade (g/t Au)	**0.23**	**0.41**	**0.41**

[1] Surface is defined as the initial topography which includes mountain and base ground below mountain.

The Sukari pit will be developed in a number of Stages and the Mineral Reserve is tabulated by mining stage in Table 1.3.

Table 1.3[(2) (3)]

Sukari Open Pit Mineral Reserve Estimate (as at 01 August 2010)

Stage	Proven		Probable		Mineral Reserve[(4)]			Inferred Resource		Total Rock	Strip Ratio
	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Cont Au (Moz)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	(t/t)
Total[(1)]	**102.4**	**1.09**	**142.9**	**1.19**	**245.4**	**1.15**	**9.1**	**36**	**1.2**	**1,130.1**	**5.6**

Notes to Table: (1) Includes surface stockpiles totalling 3.1Mt @ 0.58 g/t
(2) Based on as mined surfaced as at 01 August 2010
(3) Ultimate Pit design has a waste to ore ratio of 5.6:1
(4) Includes 42.3Mt @ 0.37g/t of low grade oxide ore to be stockpiled for end of mine life feed.

The total Mineral Reserves at Sukari were estimated at 245.4Mt of ore at an average grade of 1.15g/t Au for 9.1M contained ounces of gold.

The Mineral Reserves that have been declared are based on a gold price of US$900/oz and a gold cut-off grade of 0.17g/t Au for oxide and 0.36g/t for sulphide transitional material.

The work satisfies the reporting requirements of the JORC (2004) and CIM (2004) guidelines for reporting mineral resources.

Table 1.4

Sukari Open Pit Mineral Reserve [(2)] – By Design Stages

(reported at a cut-off grade of 0.4g/t Au for oxide, transitional and sulphide material)

Stage	Oxide Ore		Sulphide Ore		Transitional Ore		Total Ore		Waste	Strip Ratio
	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	(t/t)
1	0.03	0.67	1.93	2.65	1.23	1.75	4.43	1.65	3.33	0.75
2	2.20	0.85	11.47	1.42	3.26	1.17	16.94	1.30	34.88	2.06
3	2.19	0.76	15.34	1.59	0.94	0.79	18.47	1.45	66.18	3.58
Rem	12.96	0.82	139.28	1.37	11.10	0.75	163.34	1.28	1,025.70	6.28
Total	**17.38**	**0.82**	**168.02**	**1.41**	**16.53**	**0.91**	**201.93[(1)]**	**1.31**	**1,130.09**	**5.60**

Notes to Table: (1) Ore within design pit shells at 0.4g/t cut-off for all types
(2) Based on as mined surfaced as at 01 August 2010

1.7 Mining

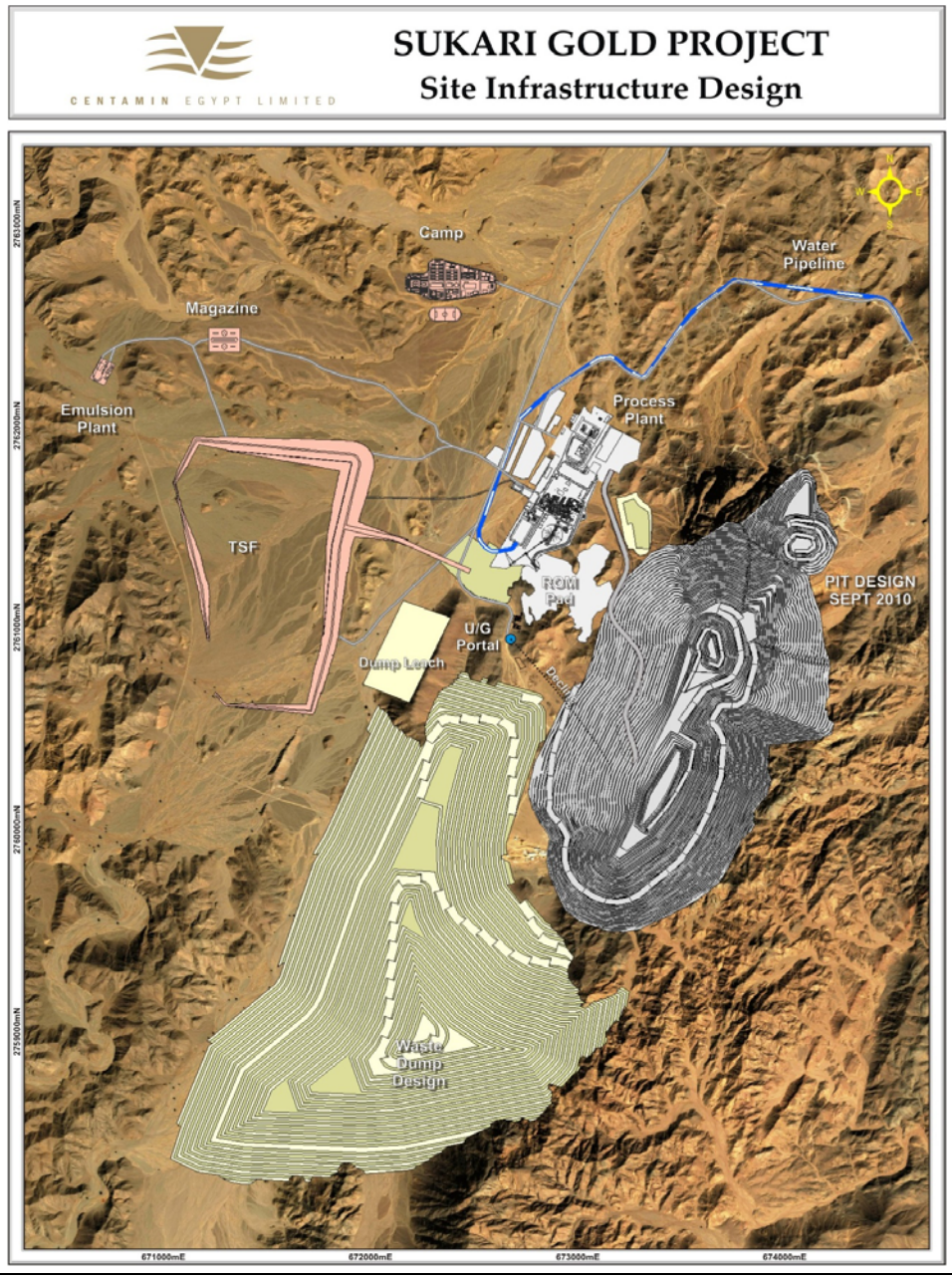
Ore and waste will be mined using conventional open pit mining methods. The operation is planned to utilise selective mining techniques to separate ore and waste. Provision has been made for drilling and blasting all primary and oxide materials.

Ore will be hauled to the Run of Mine ('ROM') pad next to the processing plant and either direct tipped to the crusher or stockpiled for future reclaim at the 5Mtpa process plant throughput rate.

Waste from the mine is being used to construct the tailings storage facility (TSF) and haul roads and will later form a single waste dump along the western and southern sides of the pit.

Compared to the previous design prepared for the 2007 DFS and subsequent March 2007 technical report, it is significantly deeper and extends further to the north and to the east. Figure 1.4 shows the site layout plan including the final pit design and waste dumps.

Figure 1.4
Site Layout Plan



1.8 Metallurgy

The metallurgical behaviour of Sukari ores has been established through various test work programmes dating back to 2000 performed variously by Lakefield Oretest Pty Ltd, Independent Metallurgical Testing Laboratories and AMMTEC. Comminution, flotation and cyanidation of flotation concentrate and tailings were the dominant processes tested.

The ore is relatively hard and competent being hosted in porphyry and is suitable for SAG milling. The gold is fine and associated with pyrite which is readily floated and ultra fine grinding renders the gold amenable to cyanidation.

Near surface ore has undergone varying degrees of oxidation and core logging has classified the ore into five stages of oxidation ranging from M1 (un-oxidized sulphide ore) through M5 (highly weathered oxide ore). Definitive test work by AMMTEC on the five ore classifications provided a basis for prediction of gold recovery based on flotation response and resulted in selection of three processing routes:-

- Stage 1 - Whole ore direct cyanidation of “oxide ore” (M5).
- Stage 2 – Flotation of “mixed ore” with separate cyanidation of reground concentrate and flotation tail (M2, M3 and M4).
- Stage 3 – Flotation of “sulphide ore” with cyanidation of reground concentrate and discarding flotation tail (M1).

Gold recoveries for the three process Stages were derived from the metallurgical test work and are shown in Table 1.5.

Table 1.5
Recoveries by Ore Type and Processing Stage

Ore Type	**Stage 3** Float + Conc CIL	**Stage 2** Float + Conc CIL + Float Tails CIL	**Stage 1** Direct CIL
M1 - Sulphide	89.7%	92.2%	70%
M2, M3, M4 - Mixed	69.4%	87.4%	81.6%
M5 - Oxide	62.8%	88.8%	90.8%

Block flow diagrams depicting the process routes for each Stage are shown in Figures 1.5, 1.6 and 1.7.

Figure 1.5
Process Block Flow Diagram – Stage 1 – Oxide Ore

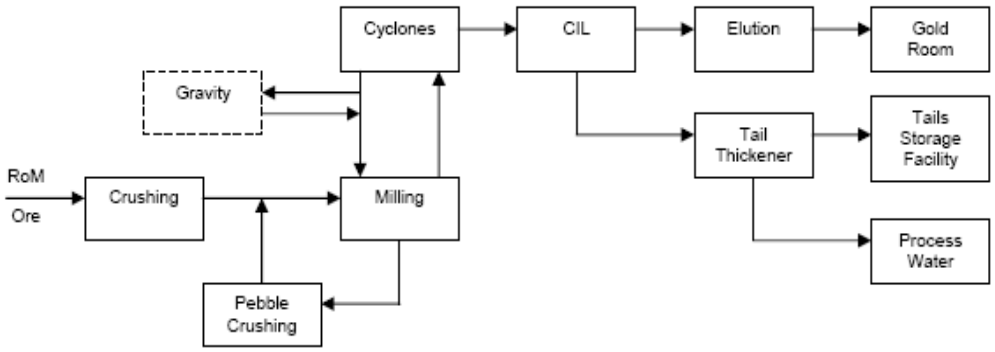


Figure 1.6
Process Block Flow Diagram – Stage 2 – Mixed Ore

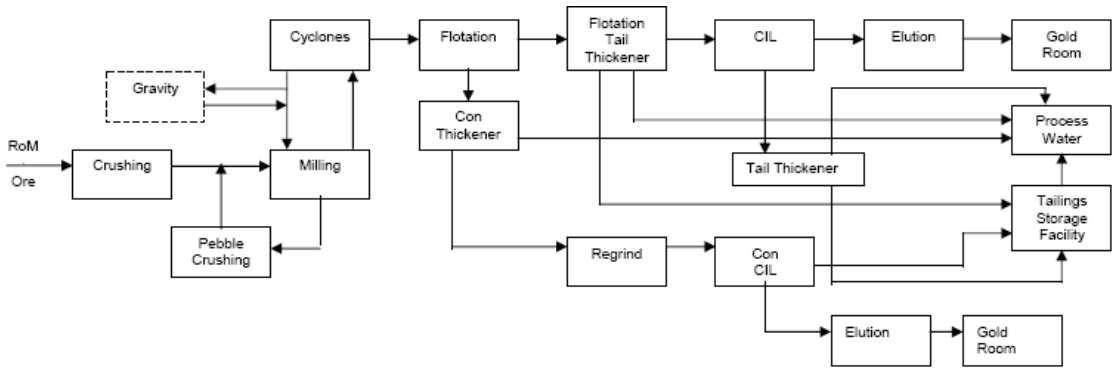
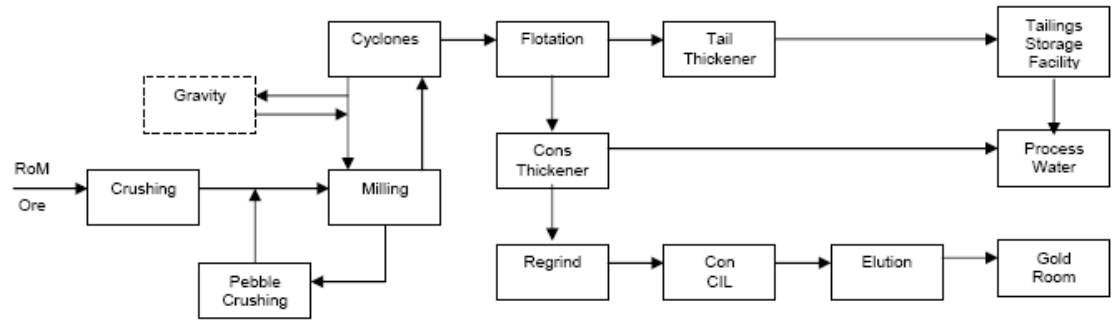


Figure 1.7
Process Block Flow Diagram – Stage 3 – Sulphide Ore



1.9 Process Plant

The process plant design comprises coarse ore stockpile, milling and CIL circuits relocated from Kori Kollo combined with a new primary crusher, a new flotation, regrind and concentrate CIL circuit and new elution/gold room facilities for each of the CIL circuits.

The crushing and milling circuits are common to all three process Stages. The Kori Kollo CIL circuit will be used to treat Stage 1 oxide ore and the Stage 2 flotation tail while the new flotation, regrind and concentrate CIL circuit will treat

Stage 2 mixed ore concentrate and Stage 3 sulphide ore. The plant circuits will be arranged and interconnected such that processing can be easily switched from one Stage route to another without the need for lengthy shutdown.

There will be two separate gold room facilities associated with each Zadra elution circuit each comprising electrowinning cells and smelting furnace. The reasons for separating these facilities are the significant differential in required carbon handling capacity through each circuit and the security risk in transporting loaded carbon or cathodes.

Because the plant throughput is limited by SAG milling capacity, provision has been made in the design for possible future inclusion of a second SAG mill. Allowance has also been made for future retro fit of a gravity circuit.

Due to the lack of a local source of fresh water in the Sukari area, seawater will be used for process water and will be pumped a distance of 26km from the Red Sea. Storage capacity at the plant is provided in a lined pond of 30,500m^3 capacity equivalent to two days at maximum demand. A Reverse Osmosis plant is provided to produce fresh water for the elution process and for domestic uses on site. The reject stream from the RO plant reports back to the seawater storage pond.

Power will be distributed to area sub-stations at 11kV where it is transformed to 4160v for the mill drives and 400v for the balance of drives. Electrical power systems are standard at 400v 50 Hz which coincides with the electrical system standards of the Kori Kollo plant.

1.10 Services and Infrastructure

On site infrastructure comprises the following facilities:-

- A 35MW power station located at the plant site including 7 x 4MW and 1 x 7MW engine/alternator sets with associated air coolers, fuel cleaning equipment, switchgear etc.
- A fuel storage facility comprising 2 x 2,500m^3 and 1 x 500m^3 diesel tanks complete with off-loading facilities and distribution pumps.
- On site buildings for administration offices, first-aid clinic and security.
- On site buildings for the process plant including sub-stations, control rooms, plant offices, plant maintenance workshop, main warehouse, laboratory and reagent storage buildings.
- Mine facilities including mine offices, mine maintenance workshop and change room.
- Construction camp, capacity 500 man, to accommodate the construction labour force complete with kitchen/mess, ablution blocks, laundry and recreation area.
- Fire protection systems including fire pump with reserve storage and hydrants.

- A low security area fence is provided around the plant site offices/buildings, fuel depot, power station and water ponds. A high security area double fence is provided around the process plant and gold room area.
- A communications network with satellite and terrestrial connections to existing Egyptian communication networks.
- An accommodation village to house up to 500 permanent operations personnel.

Off site infrastructure comprises the following facilities:-

- The main access road, 26km in length, running through the Umm Tunduba wadi from Marsa Alam to the project site.
- Seawater supply system and pipeline. The pipeline is twin 310mm diameter, HDPE pipes running parallel from the Red Sea coast to Sukari. Six pumping stations are located at intermediate locations along the pipeline route. The pipelines have been buried to a depth of 1m.

1.11 Tailings Storage Facility

Knight Piésold was engaged by PGM to design the tailings storage facility for the Sukari mine site.

Site surveys and geotechnical investigations were carried out at alternate locations and a site was chosen immediately adjacent to the proposed plant site.

To avoid tailings and strongly saline water penetrating the wadi groundwater, the TSF basin will be lined with 1.5mm thick HDPE material.

Tailings are deposited by sub-aerial techniques and decant water will be pumped from a floating barge back to the plant process water tank.

The embankment is designed to be raised in annual increments by upstream construction methods. Although the design carried out was for a total 48Mt tailings capacity it is predicted, subject to satisfactory engineering evaluation at the time, that the ultimate capacity of 78Mt will be achieved by continuing with staged upstream construction.

1.12 Environmental

PGM commissioned Environics, Giza, Egypt to complete an Environmental and Social Impact Study ('ESIA') for the Sukari Project in accordance with the Equator Principles of the International Finance Corporation ('IFC').

The ESIA was completed in February 2007 and submitted to the Egyptian Environmental Affairs Agency ('EEAA') through the Egyptian Mineral Resources Agency ('EMRA') in March 2007. The ESIA was approved by the EEAA in April 2007.

Public Consultation meetings were held in Marsa Alam during the course of preparation of the ESIA. All concerns raised were satisfactorily answered or mitigated and no significant objections are outstanding.

1.13 Project Implementation

Stage 1 and 2

PGM completed the Stage 1 and 2 projects for a capital cost of US$267.5M. Production ramp up continued throughout 2010.

Stage 3 Expansion

Construction has commenced for the Stage 3 expansion of the process plant which will entail the inclusion of two secondary crushers and associated infrastructure to increase plant throughput to approximately 5.0Mtpa.

Construction contracts will be a combination of vertical and horizontal packages tendered and awarded in Egypt.

Total capital cost for the Stage 3 expansion is expected to be $15M with commissioning expected mid 2011.

1.14 Operating Cost Estimate

Operating Costs were derived from actual production and cost performance data and projected costs and input into the optimisations. Costs detailed in completed and active contracts, updated labour and consumable costs (mainly diesel fuel) and recent quotations for reagent supply and maintenance consumables.

The mine activity costs include all operating costs for excavating, loading and dumping of ore and waste as well as drill and blast costs, grade control costs and low grade rehandling costs.

The average unit mining cost over the first 10 years of the mine life is US$1.25/t of material mined based on 1100mRL and increasing in depth and height.

Summaries of the total annual processing costs for the three ore types are as follows:-

Table 1.6 Sulphide Ore at 5.0 Mt/y		
Cost Centre	**Total Plant Cost (Gold Price US$900/oz) @5mtpa Mill throughput**	
	US$/Year	**US$/t**
Mining Cost		
At 1100mRL	45,000,000	1.25
Each 6m Above 1100mRL		+0.006
Each 6m Below 1100mRL		+0.016
Process Cost		
Sulphide/Transitional	42,500,000	8.50
Oxide	1,100,000	1.36
TCRC	450,000	1.50/oz
Administration cost	6,000,000	1.20
Total	**95,050,000**	

The Operating Cost estimate excludes contingencies, escalation, accuracy provisions, and corporate overhead charges, financing costs, royalties, expenditures classified as capital, gold shipping and insurance. Sustaining capital, rehabilitation and closure costs, and other Owner's Costs are included in the Financial Analysis.

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 Scope of the Report

This report provides an updated estimate of Mineral Resources and Reserves for the Sukari Gold Project in the country of Egypt, Africa.

This report is intended to comply with disclosure and reporting requirements set forth in NI 43-101 of the Canadian Securities Administrators (including NI 43-101's Companion Policy and Form 43-101F1).

The report complies with NI 43-101 and the resource and reserve classifications adopted by CIM Council in November 2004. The report is also consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of December 2004 (the Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia (JORC).

Furthermore, this report has been prepared in accordance with the 'Code for the Technical Assessment and Valuation of Mineral and Petroleum Assets and Securities for Independent Expert Reports of 2005 (the Valmin Code) as adopted by the Australasian Institute of Mining and Metallurgy (AusIMM). The satisfaction of requirements under both the JORC and Valmin Codes is binding on the authors as Members of the Australasian Institute of Mining and Metallurgy.

All monetary amounts expressed in this report are in United States of America dollars (US$) unless otherwise stated.

2.2 Principal Sources of Information

This report is the compilation of work performed by a number of engineering organisations for Centamin on its Sukari Gold Project dating back to 2003 and draws heavily on a previous technical report dated 23 March 2007. Information relating to project development, mining and processing has not changed significantly since reported in the technical report dated 23 March 2007 and as such is essentially repeated in this report with minor updates as well as the provision of operating data during 2010.

Reference is made to Section 22 of this report which details all documents used in the preparation of this report.

2.3 Qualifications and Experience

The authors of this report are:-

- Nic Johnson of Hellman & Schofield Pty Ltd ('H&S')
- Richard Osman a full-time employee of PGM
- Andrew Pardey a full-time employee of PGM

H&S provides a wide range of geological services for all phases of mineral projects from advanced exploration through evaluation and bankable feasibility to mine production. H&S was established in 1998 by combining the resources and experience of FSS International and PL Hellman & Associates. H&S has offices in Sydney, Brisbane and Perth in Australia, and through FSSI has affiliated offices in Canada (Toronto) and Switzerland (Geneva).

Mr Nic Johnson is Consulting Geologist of H&S. Mr Johnson is a professional geologist with 21 years experience in mining geology and the geostatistical estimation of mineral resources. The resource estimates referred to in this report were prepared by Mr Johnson. He is a Member of the Australian Institute of Geoscientists ('AIG') and has the appropriate relevant qualifications, experience and independence to be considered a Competent Person as defined in the Australasian JORC Code and a Qualified Person as defined in Canadian NI 43-101. Mr Johnson last visited the Sukari Gold Project from March 30 to April 7, 2009.

Richard Osman is a professional geologist with 11 years experience in the exploration and evaluation of mineral properties internationally. Mr Osman is Assistant Mine Manager of Sukari Gold Mine and has been involved with the Sukari Gold Project since 2002. The project background and geology sections (Sections 4 to 14) of this report were completed by Mr Osman. Mr Osman is a Member of the AusIMM and has the appropriate relevant qualifications and experience to be considered a Competent Person as defined in the Australasian JORC Code and a Qualified Person as defined in Canadian NI 43-101. Mr Osman is based at the Sukari Gold Project.

Andrew Pardey is a professional geologist with 23 years experience in the exploration and evaluation of mineral properties internationally. Mr Pardey is General Manager of Sukari Gold Mine and has been involved with the Sukari Gold Project since September 2008. The mineral reserve and mining sections (Sections 17.2 to 18) of this report were completed by Mr Pardey. Mr Pardey is a Member of the AusIMM and has the appropriate relevant qualifications and experience to be considered a Competent Person as defined in the Australasian JORC Code and a Qualified Person as defined in Canadian NI 43-101. Mr Pardey is based at the Sukari Gold Project.

2.4 Independence

Nic Johnson is independent of PGM and Centamin as defined by NI 43-101.

Neither H&S nor Nic Johnson, have any material interest in Centamin or related entities or interests. Their relationship with Centamin is solely one of professional association between client and independent consultant. Their input into this report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

Richard Osman and Andrew Pardey are not independent of PGM and Centamin as defined by NI 43-101; they are both full-time employees of PGM, which is a subsidiary of Centamin.

2.5 Abbreviations

A full listing of abbreviations used in this report is provided in Table 2.1 below.

Table 2.1
List of Abbreviations

	Description			Description
"	inch		kWhr/t	kilowatt hours per tonne
$	United States of America dollars		l/hr/m^2	litres per hour per square metre
µ	microns		LM2	Labtechnics 2kg (nominal) pulverising mill
3D	three dimensional		M	million
AAS	atomic absorption spectrometry		m	metres
AR	aqua regia		Ma	million years
ASL	above sea level		MIK	Multiple Indicator Kriging
Au	gold		ml	Millilitre
bcm	bank cubic metres		mm	millimetres
CC	correlation coefficient		MMI	mobile metal ion
cfm	cubic feet per minute		Moz	million ounces
CIC	carbon in column		Mtpa	million tonnes per annum
CIL	carbon-in-leach		N (Y)	northing
cm	centimetre		NaCN	sodium cyanide
CN	cyanide		NATA	National Association of Testing Authorities
cusum	cumulative sum of the deviations		NPV	net present value
CV	coefficient of variation		NQ	size of diamond drill rod/bit/core
DTM	digital terrain model		ºC	degrees centigrade
E (X)	easting		OK	Ordinary Kriging
EDM	electronic distance measuring		oz	troy ounce
EMRA	Egyptian Minerals and Resource Authority		P90	90% passing
EV	expected value		PGM	Pharaoh Gold Mines
FA	fire assay		ppb	parts per billion
g	gram		ppm	parts per million
g/kWh	Gallons per kilowatt hour		psi	pounds per square inch
g/m^3	grams per cubic metre		PVC	poly vinyl chloride
g/t	grams per tonne of gold		QA	quality assurance
GOA	George, Orr and Associates		QC	quality control
HDPE	high density poly ethylene		QQ	quantile-quantile
HQ	size of diamond drill rod/bit/core		RAB	rotary air blast
hr	hours		Rb-Sr	Rubidium-Strontium
H&S	Hellman and Schofield		RC	reverse circulation
ICP-MS	inductivity coupled plasma mass spectroscopy		RL	reduced level
ID	Inverse Distance weighting		ROM	run of mine
ID^2	Inverse Distance Squared		RQD	rock quality designation
IPS	integrated pressure stripping		SD	standard deviation
IRR	internal rate of return		SG	Specific gravity
ISO	International Standards Organisation		Sm-Nd	Samarium-Neodymium
ITS	Inchcape Testing Services		SMU	selective mining unit
kg	kilogram		t	tonnes
kg/t	kilogram per tonne		t/m^3	tonnes per cubic metre
km	kilometres		tpa	tonnes per annum
km^2	square kilometres		Var	variance
KP	Knight Piésold Pty Ltd		w:o	waste to ore ratio
kW	kilowatts			

3 RELIANCE ON OTHER EXPERTS

The authors of this report are not qualified to provide extensive comment on legal issues associated with the Sukari Gold Project and included in Sections 1 and 4 of this report. The assessment of data pertaining to these sections relies on information provided by PGM, which has not been independently verified by the authors.

Similarly, the authors of this report are not qualified to provide extensive comment on environmental issues associated with the Sukari Gold Project and included in Section 18 of this report. The assessment of data pertaining to these disciplines relies on information provided by Environics, an environmental management consulting firm based in Giza, Egypt, which has not been independently verified by the authors.

This report is based on information provided by PGM and various consultants which reflect various technical and economic conditions prevailing at the time of compilation of the report. These conditions can change significantly over relatively short periods of time and as such the information and opinions contained in this report may be subject to change.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Background Information on Egypt

4.1.1 Geography

The Arab Republic of Egypt is a Northern African country of approximately 1,000,000km^2. It borders the Mediterranean Sea (to the north), Libya (to the west) and the Gaza Strip and Israel (to the east), and the Red Sea north of Sudan, and includes the Asian Sinai Peninsula. It lies approximately at geographic coordinates 27ºN and 30ºE and operates on time zone GMT+2h00. Egypt has a population of approximately 79 million people generally concentrated on the fertile banks of the Nile River. Around half of Egypt's residents live in urban areas, with the majority spread across the densely populated centres of greater Cairo (which is the largest city in Africa and the Middle East), Alexandria and other major towns in the Nile Delta.

Arabic is the official language with English and French widely understood by educated classes. The main ethnic group is Egyptian 98%, followed by Berber, Nubian, Bedouin, and Beja 1%, Greek, Armenian and other European (primarily Italian and French) 1%. The rapidly growing population is young, with approximately one-third of the total under 15 years of age. Despite improvements in health care, infant mortality is high and about half of all deaths occur among children less than five years of age. Life expectancy, however, increased from only about 33 years in 1927 to almost 72 years by 2008. Almost half of the population lives in urban areas.

Egypt is predominantly Muslim, at approximately 90% of the population, with the majority being adherents of the Sunni branch of Islam. A significant number of Muslim Egyptians also follow native Sufi orders, and a minority of Shi'a. Christians represent about 10% of the population.

Egypt is bisected by the highly fertile Nile valley where most economic activity takes place. The Nile divides the desert plateau through which it flows into two unequal sections, the Western Desert between the river and the Libyan frontier; and the Eastern Desert extending to the Suez Canal, the Gulf of Suez, and the Red Sea. The Western Desert is arid and without wadis (dry beds of seasonal rivers), while the Eastern Desert is extensively dissected by wadis and fringed by rugged mountains in the east. The desert of central Sinai is open country, broken by isolated hills and scored by wadis. The coastal regions of Egypt, with the exception of the Delta, are everywhere hemmed in either by desert or by mountain; they are arid or of very limited fertility. The coastal plain, in both the north and east, tends to be narrow; it seldom exceeds a width of 50km. With the exception of the cities of Alexandria, Port Said, and Suez and a few small ports and resorts, the coastal regions are sparsely populated and underdeveloped.

4.1.2 Political System

Although Egypt is a democracy, government is highly centralised with the president having extensive executive powers. The last presidential election took place in 2005. President Mubarak has been in power since 1981 and is widely considered to have kept the country's internal situation quite stable. His greatest opposition comes from Islamic extremists. The government has gone to great lengths to keep groups such as these from employing terrorist tactics.

4.1.3 Legal System

The Legal System is based on English common law, Islamic law, and Napoleonic codes. Judicial review is by Supreme Court and Council of State (oversees validity of administrative decisions).

4.1.4 Economy

Egypt has one of Africa's most prosperous economies despite the fact that it is extremely vulnerable to external factors such as the continuing conflict in the surrounding region and the subsequent drop in tourism, which is one of the country's main economic sectors in addition to the oil and gas sectors.

The country is classified as a middle-income and continues to record economic growth although there is increased pressure by international donors to implement reforms. These include the lifting of price controls, the reduction of subsidies as well as the relaxation of restrictions on trade and investment. The government has instituted a privatisation program and has committed itself to the implementation of reforms requested by donors.

Egypt was rated 116th out of 179 countries on the Human Development Index in 2006 and faces challenges of water scarcity and high instances of unemployment and poverty. The country's GDP per capita was US$4,953 (PPP) in 2006.

4.1.5 Transport and Communications

Egypt is served by a range of international airlines, which fly to both Cairo and Alexandria, and internal air services service other points of the country. The nearest airport to the Sukari Project is located at Marsa Alam, which is 30km from the site. There are bus and rail services along the Nile River valley. The road system in the country is good with only small amounts remaining unpaved.

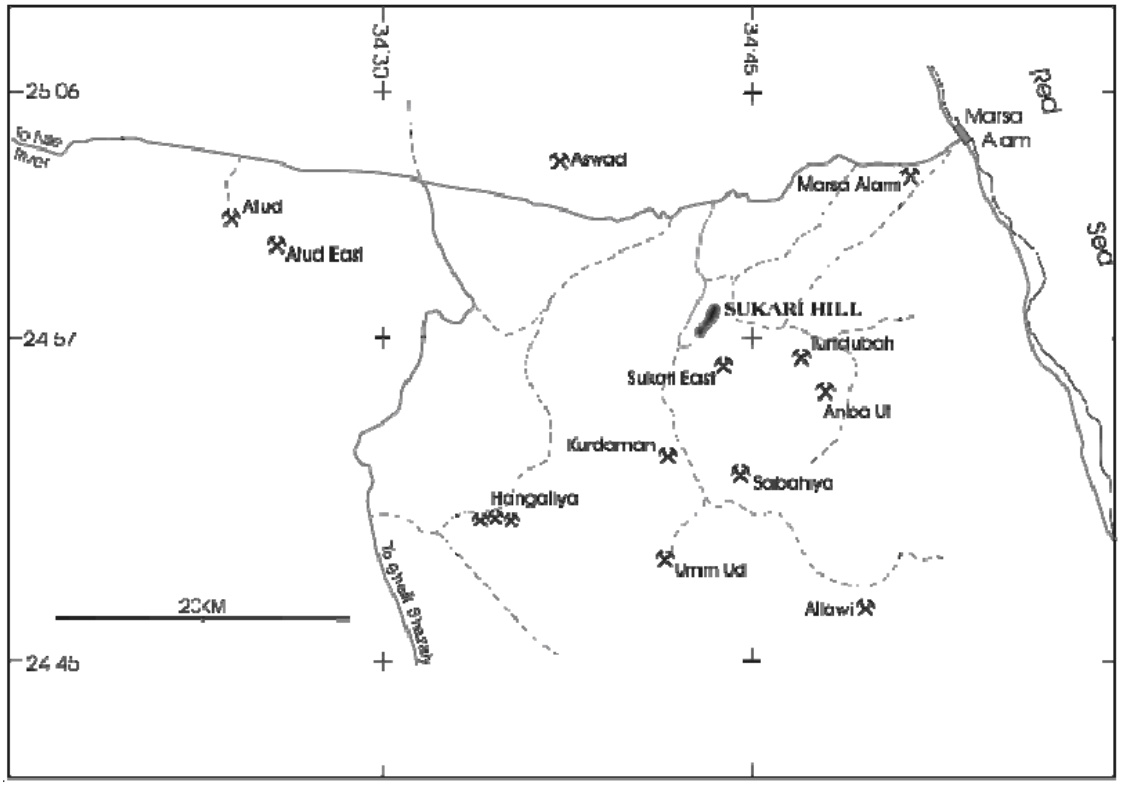
In spite of its long coastline, Egypt has only three ports of any significance – Alexandria, Port Said, and Suez. Alexandria handles most of the country's imports and exports, as well as the bulk of passenger traffic. Port Said, at the northern entrance to the Suez Canal, lacks the berthing and loading facilities of Alexandria. Suez's main function is that of an entry port for petroleum and minerals from the Egyptian Red Sea coast and for goods from the Far East.

There is a large, well-established telephone network covering the whole country, which underwent extensive upgrading during 1990s. Cellular phone services are available in the major centres. Internet access is available in most centres.

4.2 Project Location

The Sukari Gold Project is located in bare hills in the arid Eastern Desert of Egypt some 30km westerly from the growing township of Marsa Alam on the Red Sea (Figure 4.1).

Figure 4.1
Sukari Location and Road Access



4.3 The Project Tenement and Area

According to information supplied by Centamin, PGM obtained a concession agreement in 1994 with the Egyptian Geological Survey and Mining Authority ('EGSMA', now known as the Egyptian Minerals and Resource Authority or 'EMRA') and the Egyptian Government for gold and associated minerals in three areas in the Eastern Desert. The agreement is in Law 222 of 1994 which came into effect on 29 January 1995. With a total expenditure commitment of US$ 6,500,000, tenure was for one year with three extensions of two years with areal relinquishment conditions.

From the date of a commercial discovery, a period of 30 years is permitted for exploitation, with one 30 year renewal period available at PGM's option. A commercial discovery is defined in the concession agreement as meaning "a discovery of Gold and Associated Minerals worthy of being developed and exploited commercially in accordance with the principles of good mining practices.

A concession agreement with the Egyptian Government was negotiated in April 2005 for a 30 year lease over 160km^2 in the Eastern desert region to accommodate the company's exploration and mining activities.

The Sukari concession is shown in Figure 4.2. The concession covers an area of 160km^2, containing the proposed Sukari mine site and surrounding prospects. The points defining its boundary are listed in a clockwise fashion from the northwest corner in Table 4.2.

Figure 4.2
Concession Area

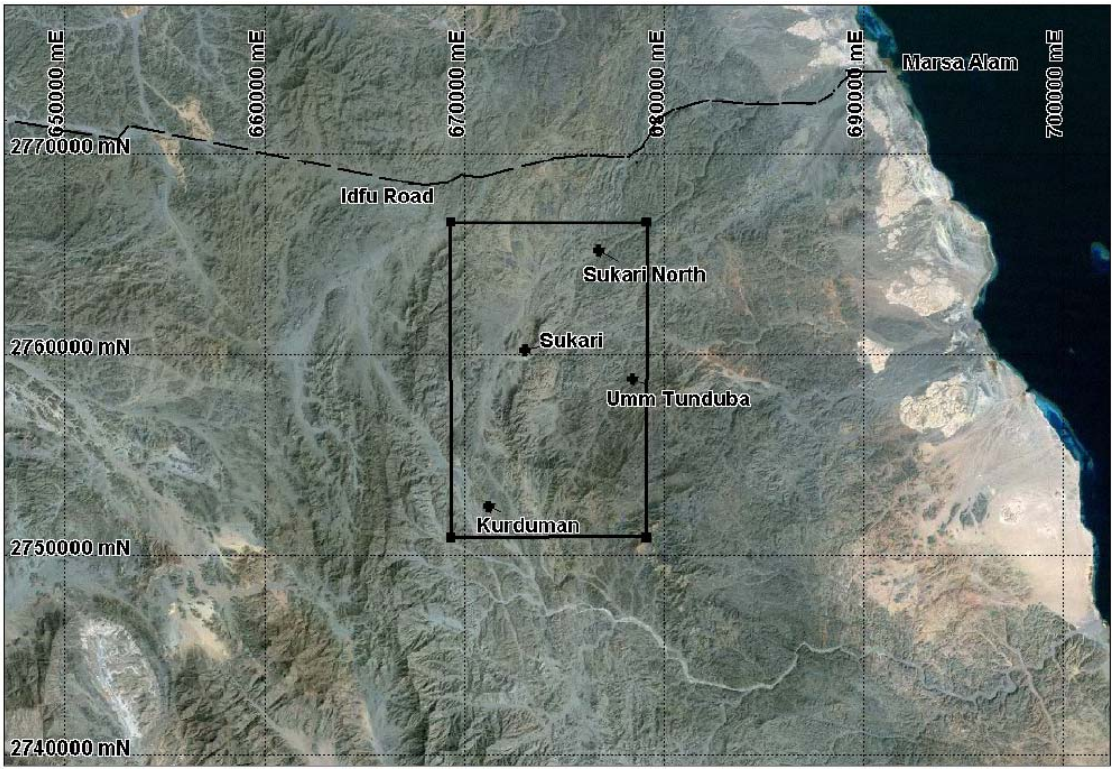


Table 4.1
Sukari Project
List of Tenement Corners

Pillar	Longitude (east)	Latitude (north)
A	34° 40' 40"	25° 00' 20"
B	34° 46' 30"	25° 00' 20"
C	34° 46' 30"	24° 51' 45"
D	34° 40' 40"	24° 51' 45"

Information pertaining to the project tenement and area has been provided by Centamin and has not been independently verified by the authors.

4.4 Royalties, Payments, Agreements and Encumbrances

According to information supplied by Centamin, PGM and EMRA have established an operating company, Sukari Gold Mines, 50% owned by each party. The operating company will have no funding separate from the obligations of PGM under the concession agreement. The articles of association of this company have not been translated from Arabic and have therefore not been independently verified by the authors.

The Government of Egypt receives a royalty of 3% of net sale revenue, payable in cash each calendar half year.

PGM shall be entitled to recover the following costs and expenses payable from sales revenue (excluding the government royalty):-

- All current operating expenses incurred and paid after the initial commercial production;
- Exploration costs, including those accumulated prior to the commencement of commercial production at the rate of 33.3% per annum;
- Exploitation capital costs, including those accumulated prior to the commencement of commercial production at the rate of 33.3% per annum.

If costs recoverable by PGM exceed the sales revenue (excluding the royalty payable to the government) in any financial year, the excess shall be carried forward for recovery in the next financial year or years until fully recovered but in no case after the termination of the agreement.

After deduction of royalty payments and recoverable expenses, the remainder of the sale revenue will be shared equally between PGM and EMRA. Except that for the first and second years in which there are net proceeds for the entire year, an additional 10% of those proceeds shall be paid to PGM as incentive (PGM 60%, EMRA 40%). For each of the next two years in which there are net proceeds for the entire year, 5% of the net proceeds shall be paid to PGM (PGM 55%, EMRA 45%), thereafter the net proceeds shall be shared equally (PGM 50%, EMRA 50%).

Starting from the date of commercial production PGM will enjoy a 15 year exemption from taxes imposed by the Egyptian government. PGM and EMRA also agree that the operating company will in due course file an application to extend the tax free period for a further 15 years.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

A coastal highway runs along the west coast of the Red Sea from the border with Sudan in the south to Suez in the north, passing through Marsa Alam, Qseir, Safaga, Hurghada, Ras Gharib and Ein Sokna. Another highway connects Cairo directly to Ein Sokna. From Cairo to Marsa Alam by highway is about 750km, about ten hours by supply truck. There is also a bitumen highway from Idfu on the Nile to Marsa Alam. There is a new international airport north of Marsa Alam.

From Marsa Alam the Idfu bitumen highway first runs west up gravel outwash plains to the mouth of Wadi Khariga, then it follows the winding narrow wadi to Bir Umm Khariga, some 20km from Marsa Alam. From Umm Khariga Well a corrugated gravel road runs southerly for 10km where it crosses a low divide and goes down into Wadi Sukari, and then on to the old Sukari gold mine. The Sukari road then turns east from the Wadi Sukari road and continues on for about 1.5km north-west to the Sukari camp.

5.2 Climate

Egypt has a dry climate. It is hot in the summer, with temperatures averaging between 80 and 90°F (27-32°C). Winters are warm, with temperatures averaging between 55 and 70°F (13-21°C). A steady wind from the northwest helps to lower the temperature near the coast. The Khamaseen is a wind that blows from the south in Egypt, usually in spring or summer, bringing sand and dust, and sometimes raises the temperature in the desert to more than 100°F (38°C). Rain seldom falls in Egypt. Along the Mediterranean Coast, the average yearly rainfall is 8 inches (20cm). Farther south, only about one inch of rain falls every year. During winter snow falls on Sinai's mountains and some coastal cities, such as Baltim, Damiatta, Sidi Barrany and Alexandria.

5.3 Physiography

The old Sukari gold mine is in the Eastern Desert of Egypt and it is estimated that nearly all of the gold mineralisation occurs within a porphyry outcrop which is expressed as a 2,500m long jagged-toothed, strong topographic high rising to 350m above the local wadi (intermittent water course) level. Wadi drainage plains pass to the east and west of the outcrop and the sharply incised green – brown Red Sea Hills surround that. The area is arid and almost bare of vegetation.

5.4 Local Resources and Infrastructure

The Sukari site is located in stark desert with little or no vegetation. There is no permanent population in the immediate area and it has been visited only by people tending nomadic livestock herds in recent times. No existing services and infrastructure exist suitable to support a mining operation of the proposed magnitude. Access roads, water supply, power supply and distribution will all

have to be built from the ground up. There is suggestion that grid power maybe available in 3 to 5 years, however, there are no firm or fixed plans, hence a local power station will be required.

6 HISTORY

6.1 Ownership History

Gold was mined at Sukari in Pharaonic and Roman times, with perhaps 1,000kg of gold won. Small-scale mining was re-established in 1912 to 1914. More substantial operations were undertaken in the period 1937 to 1951, with recorded production of 4,768kg of gold from underground workings.

PGM negotiated a Concession Agreement (CA) in 1994 to explore for gold and associated minerals. On October 2000 PGM submitted a feasibility study, relating to the Sukari Gold Project, in accordance with the terms of the CA. This was accepted and on 24 May 2005 an Exploitation Lease covering an area of 160km^2, containing the proposed Sukari mine site and surrounding prospects, was officially granted.

6.2 Exploration History

The first systematic modern exploration in the Sukari area was carried out in the 1970's by the Egyptian Government with assistance from the USSR. Work completed consisted of geological mapping, trenching, geochemical sampling and finally, five diamond drillholes were completed at Sukari during the period 1975-1977. Assaying of the drill core confirmed the presence of gold mineralisation at depth.

Work by PGM commenced in 1995 with the establishment of a camp and work completed consisted of a detailed literature search prior to gridding, traversing, mapping, geochemical sampling, trenching, channel sampling, heavy mineral sampling, augering and surveying.

Drilling by PGM at Sukari commenced in April 1997 using two Atlas Copco Craelius Diamec 252 underground diamond drill rigs selected for their portability. In August 2000 an Atlas Copco 262 wireline rig mounted on tracks was brought into the operation to drill deeper holes. These rigs are owned by PGM and remain on site. A drilling contractor was mobilized to site in April 2005, initially with four drilling rigs. The drilling contractor currently has four diamond drilling rigs and one multipurpose rig on site.

6.3 Resource History

H&S's first involvement was in early 2000, when a preliminary resource model was reported. Subsequently, in July 2000, an H&S geologist travelled to site to verify logging, survey and sampling procedures employed by Sukari geologists at that time. A program of check sampling and assaying was also undertaken under chain of custody conditions supervised by H&S.

Late in 2001 a complete review of the resources and all geological data available at that time was undertaken by H&S. The mineral resources estimated in the November 2001 study are shown in the table below (Table 6.1) and were based on the sampling and assaying from some 32,627 metres of drilling (all diamond core) and 1,112 assays from surface rock chip traverses.

Table 6.1

Resource Estimate, H&S November 2001

Cutoff g/t Au	Measured			Indicated			Inferred		
	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au
0.5	8.86	1.40	397.4	12.66	1.37	556.7	15.48	1.53	760.4
0.7	6.74	1.65	356.9	9.45	1.63	495.3	11.68	1.83	687.8
1.0	4.60	2.02	299.3	6.29	2.03	410.3	7.92	2.30	586.0
1.2	3.64	2.27	265.5	4.87	2.30	360.2	6.19	2.64	525.2

The resource model was subsequently updated several times, utilising additional drilling information as it became available, up to April 2003 at which time a hiatus in drilling activity ensued due to withdrawal of work permits by the Egyptian Government. Table 6.2 shows the resource estimates reported in April 2003.

Table 6.2

Resource Estimate, H&S April 2003

Cutoff g/t Au	Measured			Indicated			Inferred		
	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au
0.5	14.79	1.29	613.4	25.68	1.35	1114.6	24.06	1.72	1330.5
0.7	10.89	1.54	539.2	19.75	1.58	1003.3	18.70	2.04	1226.5
1.0	7.17	1.91	440.3	13.23	1.94	825.2	13.03	2.57	1076.6
1.2	5.52	2.15	381.6	10.13	2.20	716.5	10.23	2.97	976.8

Access to the Sukari Gold Project was re-instated by the Egyptian Government in early 2005 and PGM had recommenced drilling by mid-May utilising a fleet of seven drill rigs acquiring a significant amount of additional sampling information, both infilling the present drill spacing and drilling to expand the down dip and along strike continuations to the Sukari mineralisation. During November 2006 all available resource data was incorporated into a new resource estimate. Table 6.3 shows the resource estimates reported in November 2006.

Table 6.3

Resource Estimate, H&S November 2006

Cutoff g/t Au	Measured			Indicated			Inferred		
	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au
0.5	39.212	1.43	1802.8	59.147	1.47	2795.4	68.8	1.4	3096.8
0.7	29.506	1.70	1612.7	45.263	1.74	2532.1	47.9	1.8	2772.0
1.0	19.915	2.12	1357.4	30.916	2.16	2147.0	30.8	2.3	2277.6
1.2	15.524	2.41	1202.8	24.181	2.46	1912.5	24.0	2.6	2006.2

A resource update was completed in February 2007 and formed the basis for the previous Technical Report dated 23 March 2007. The February 2007 resource update was based on data coming from 817 diamond and RC drill holes combining to give approximately 168,000 meters of drilling. Table 6.4 shows the resource estimates reported in February 2007.

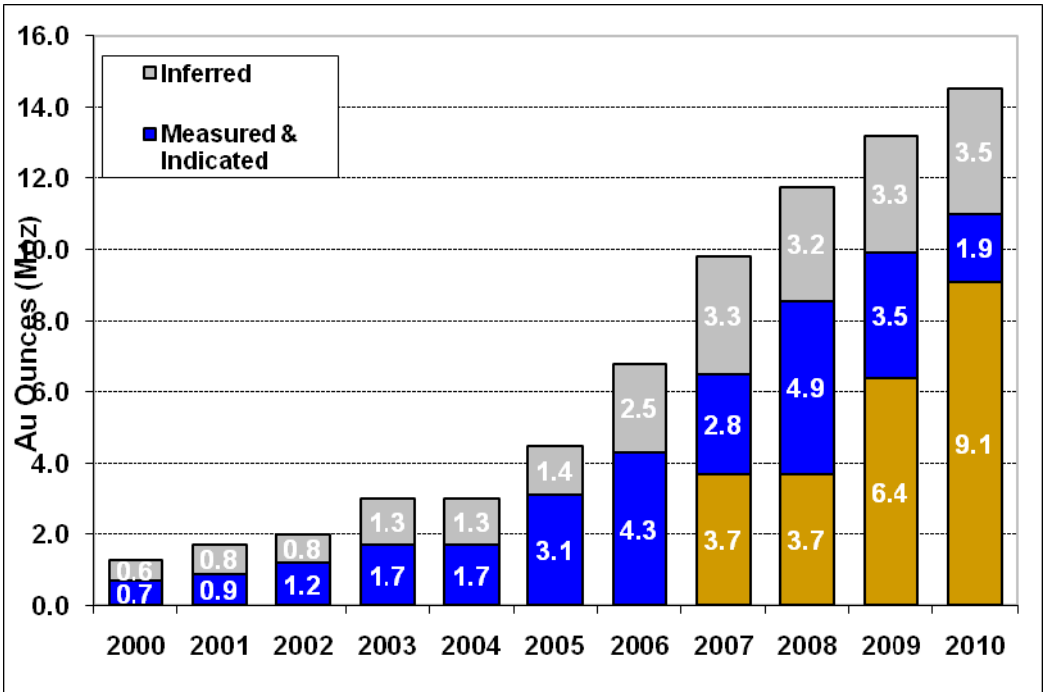
Table 6.4

Resource Estimate, H&S February 2007

Cutoff g/t Au	Measured			Indicated			Inferred		
	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au	Mt	g/t Au	k oz Au
0.5	47.389	1.40	2133.1	73.984	1.39	3306.4	52.82	1.7	2887.0
0.7	34.440	1.70	1882.4	53.445	1.69	2904.0	38.64	2.1	2608.9
1.0	22.399	2.16	1555.5	34.364	2.16	2386.5	25.88	2.7	2246.6
1.2	17.194	2.48	1371.0	26.170	2.50	2103.5	20.38	3.1	2031.3

The resource estimate has effectively been updated on a quarterly basis since February 2007. The consequent growth in contained ounces is described in Figure 6.1.

Figure 6.1
Growth in Contained Ounce s at each Resource Estimate Update



The resource estimates have consistently complied with the requirements of both NI 43-101 and the Australasian JORC Code. Details of the current resource are provided in Section 17 of this report. The resource estimate reported herein uses all drilling and assaying data available as at 31 May 2010.

6.4 Production History

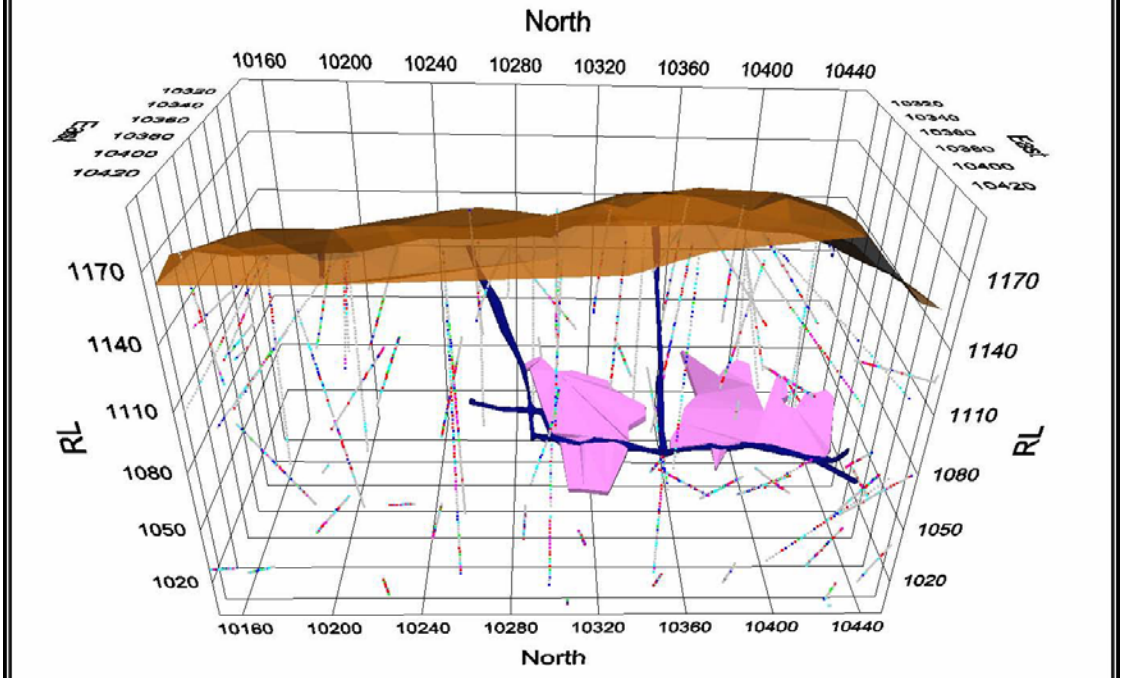
Gold was mined at Sukari in Pharaonic and Roman times. Numerous small pits are located over about two kilometres strike on Sukari Ridge. There are also small pits in wadi colluvium along the flanks of the ridge, most notably in Wadi Pharaoh to the east of the northern part of the ridge. It is believed that about 1,000kg of gold may have been won.

The old Sukari Mine was established on an outcropping quartz vein (the "Sukari Main Lode") on the south-western flank of Sukari Ridge (Figure 6.2). In Pharaonic times, mining of this vein extended to about 50m from surface, intermittently, along about 200m strike, with stopes about one metre wide. Small-scale mining was re-established in 1912 by British concerns but appears to have ceased at the outbreak of World War 1.

Figure 6.2
View of Historic Mine Workings

In 1936, a renewed effort by government authorities to re-establish Egypt's gold mining industry saw Sukari selected as the first mine to be brought back into production. In 1934, "developed ore reserves" were estimated to be about 12,000t @ 20g/t Au (Hume, 1937), with a further 65,000t estimated "ore in sight". After preparatory work, production commenced in August 1937 and continued intermittently until February 1951. Recorded gold production for this period is 4,768kg (Azzaz, et.al., 1978). Ore was sourced from the Main Lode, with the ancient underlay shaft being refurbished and extended to about 185m depth (on the underlay). An extraction level was established at 110m depth and stoping above this level extended over about 100m strike length. Several subsidiary adits and underlay shafts access stopes along the length of the mined strike. Ore below the 110m level has also been stoped over about 50m strike length. Stopes are generally 2m to 3m wide. Figure 6.3 shows a 3D view, looking west, of the surveyed underground workings. The total tonnage reported within the surveyed volumes is approximately 26,000 tonnes.

Figure 6.3
Underground Workings



A battery for ore treatment was located at Sukari from 1936 to 1944, at which time it was moved to Marsa Alam and used to treat ore from other prospects in the district, in addition to Sukari ore production Year to date for 2010.

Table 6.5 Recent Production from SGM					
		September 2010 Qtr	June 2010 Qtr	March 2010 Qtr	December 2009 Qtr
Ore Mined	('000t)	1,682	1,400	984	894
Total Mined	('000t)	4,916	5,305	4,326	3,914
Strip Ratio	waste/ore	1.9	2.8	3.4	3.4
Ore Processed	('000t)	605	847	919	140
Head Grade	(g/t)	1.75	1.37	1.41	1.11
Gold Recovery	(%)	82.6	86.5	90.0	70.6
Gold Produced - Total [(1)]	(oz)	30,243	30,236	36,621	244
Gold Produced - Dump Leach	(oz)	3,049	2,791	-	-
Cash Operating Cost of Production [(2)]	US$/oz	638	569	403	n/a
Gold Sold	(oz)	31,228	30,759	32,994	n/a
Average Sales Price	US$/oz	1,239	1,203	1,105	n/a

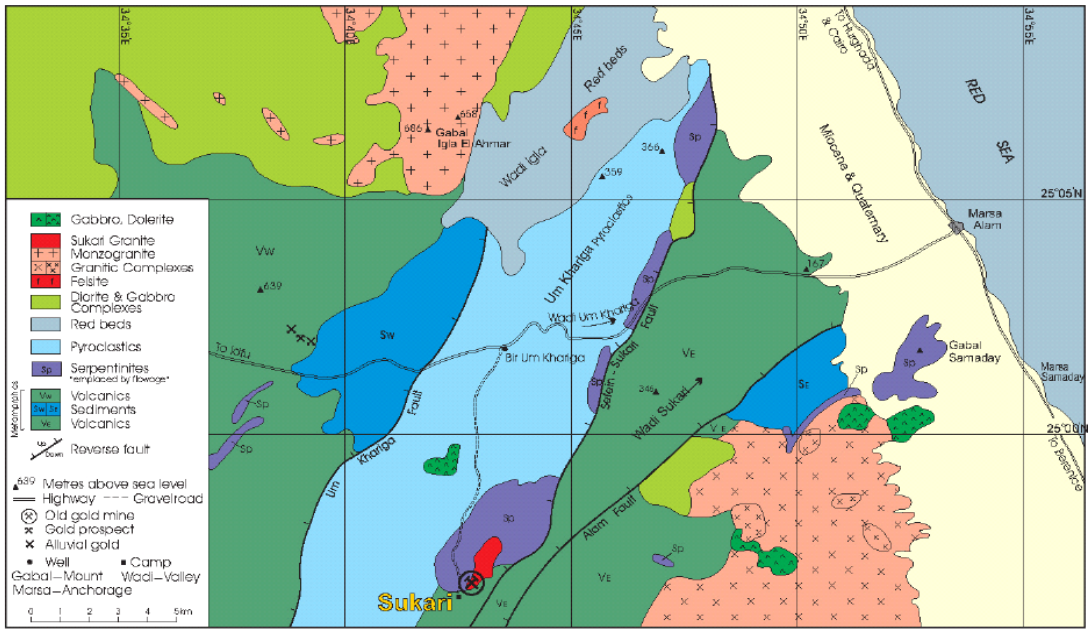
7 GEOLOGICAL SETTING

7.1 Regional Geological Setting

The Sukari gold deposit is located in the Eastern Desert of Egypt at 24º 56' 50"N 34º 42' 27"E, about 23km due west of the Red Sea coastal town of Marsa Alam. The regional geological setting is shown in Figure 7.1.

The rock sequence at Sukari comprises part of the Neoproterozoic (900-650Ma) Arabian-Nubian Shield, one of a number of areas of African continental crust that accreted and stabilised during the Pan-African Orogeny. At a district scale, the host sequence at Sukari comprises a NNE striking mélange of predominantly calc-alkaline igneous rocks and metasediments representing an accreted island arc or arcs. Several bodies of serpentinite, representing accreted slivers of highly deformed oceanic crustal rocks, occur in the hangingwall of the NNE striking, ESE verging, Sefein-Sukari thrust (Akaad, et al, 1993). This district-scale (~25km) structure is mapped as passing immediately to the east of Sukari, where it separates rocks of the Um Khariga Metapyroclastics (west of Sukari granitoid and enveloping serpentinite) from the Sukari Metavolcanics (east of Sukari). Vail (1983) assigns an age of 770-660Ma to rocks of the region. The entire sequence has undergone regional metamorphism to mid-upper greenschist facies.

Figure 7.1
Sukari District Geology



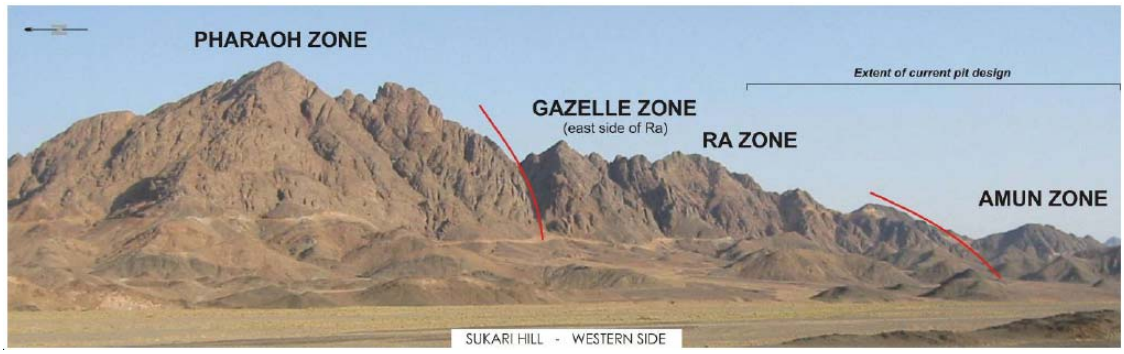
Note geology map from Cavaney 2005

7.2 Local Geology

The Sukari felsic porphyry outcrop is located in an easterly dipping sequence of andesite flows, serpentinites and associated volcanoclastic sediments, mainly tuffs and epiclastics. It strikes for 2.3km and is 100m to 600m thick. Drilling to date indicates that the Sukari Porphyry dips toward the east at between 50º and 75º. The western and eastern contacts of the porphyry are thus regarded as footwall and hangingwall contacts respectively. Porphyry/wall rock contacts are, in places, vertical or overturned.

The Sukari area has been designated four geographical zones namely Amun, Ra, Gazelle and Pharaoh Zones from south to north respectively (Figures 7.2 and 9.1).

Figure 7.2
Sukari Hill with designated Geographical Zones (looking to the SE)



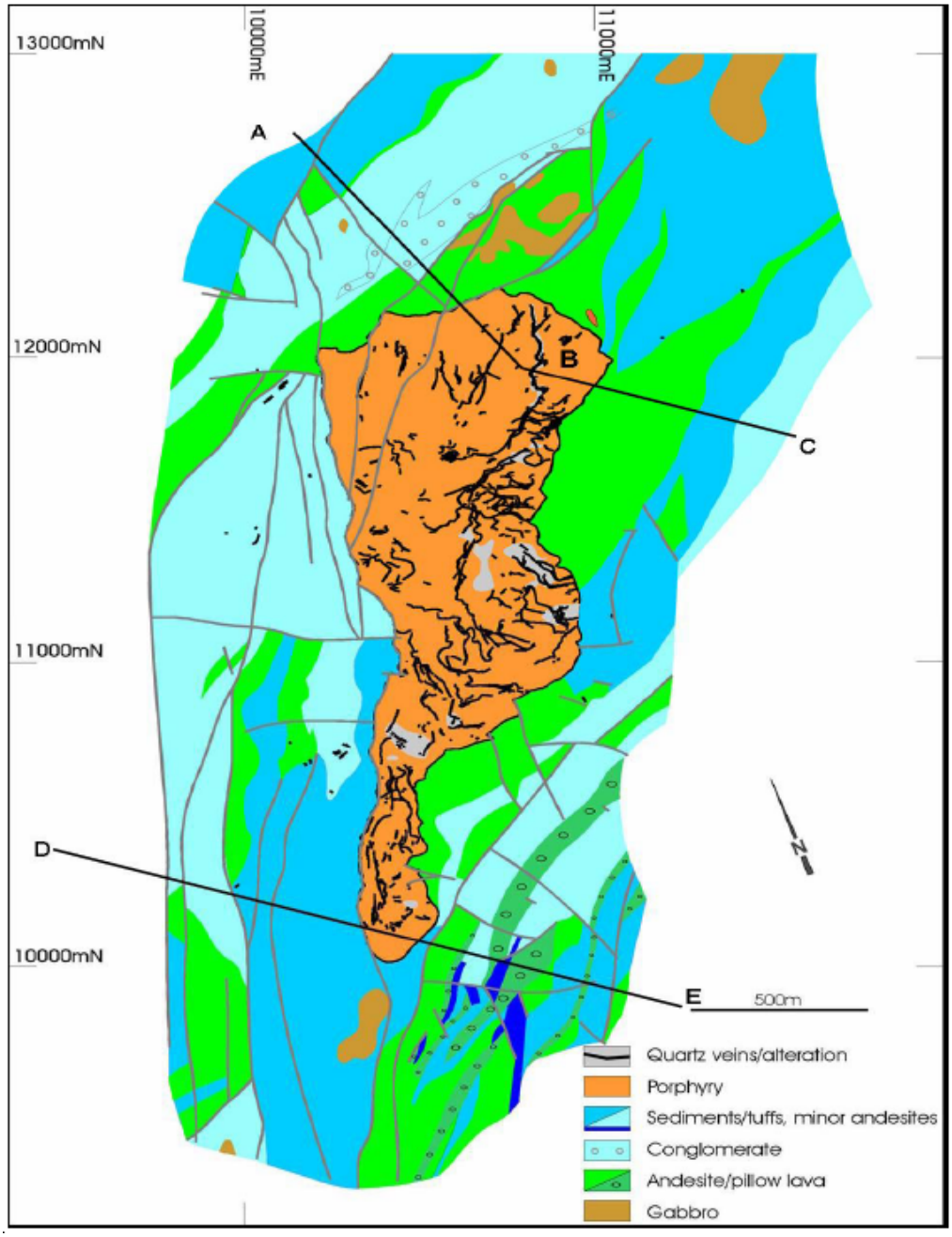
The hangingwall sequence comprises a mixture of serpentinite, meta-conglomerate, lesser fine-grained metasediments, minor basalt and porphyry dykes or sills (Figure 7.3). Drillhole logging has failed to define any consistent sequence in these rocks. Although this may be due in part to inconsistencies in the logging, surface exposures indicate that these rocks are complexly deformed. It is reasonable to assume that the porphyry dykes in the hangingwall sequence are genetically and temporally related to the main Sukari Porphyry. The footwall sequence is devoid of porphyry dykes. This may indicate that the entire sequence is overturned as it would be reasonable to expect subsidiary or feeder dykes in the footwall of the main intrusion rather than the hangingwall.

On the eastern slope of Sukari Ridge individual stratigraphic units, normally comprising more competent units in serpentinite, can be traced over tens of metres. Surface mapping is, however, complicated by a lack of persistent stratigraphic markers and the fact that, in many places, schistosity is at a high angle to bedding.

Few drillholes penetrate more than about 50m into the footwall sequence, where they have intersected fine-grained pelitic metasediments with lesser basalts. Outcrop is poor at the foot of the western slope of Sukari Ridge. Low ridges and mounds in the wadi to the west of Amun Zone are formed by outcropping serpentinite and it is likely that this lithology dominates the footwall sequence.

Similar to hangingwall rocks, footwall metasediments and serpentinites show development of very strong schistosity and a lack of stratigraphic markers, combined with poor outcrop, prevents mapping of a stratigraphic sequence.

Figure 7.3
Interpreted Geology, Sukari Porphyry and surrounding Country-rocks



8 DEPOSIT TYPES

Gold mineralisation is hosted by a granitoid body of approximately granodiorite-tonalite composition referred to as the Sukari Porphyry. Granitoids of the Egyptian portion of the Arabian-Nubian Shield are described as “older granites”, these being “syntectonic” intrusions related to arc magmatism, and “younger granites” that clearly post-date arc accretion and have monzonitic, syenitic or potassic compositions (Moghazi et al, 1999). Sharara (1999) ascribes the Sukari Porphyry to the latter, based on an age of 615-570Ma by Stern and Hedge (1985). Ghoneim et al. ascribe an age of 559Ma ± 6Ma to the Sukari Porphyry based on Rb-Sr and Sm-Nd isotopic dating of “fresh granite” and mineralisation age of 522 ± 11Ma based on the same methods applied to albitised granite and separated albite. In contrast, Sharara and Vennemann (1999) state that gold mineralisation at several localities in the Eastern Desert region, including Sukari, pre-dates intrusion of “younger granites”.

Considering the composition of Sukari Porphyry, its deformed nature and its contact relationships with surrounding rocks, it is considered to be a sub-volcanic, “syn-tectonic” granitoid, representing the roots of a rhyolitic arc volcano.

Based on fluid inclusions and stable isotope analyses, Sharara (1999) and Sharara and Venneman (1999) estimate a mineralisation temperature of 270ºC to 370ºC for Sukari and other gold deposits in the district, with gold deposited from a low-salinity carbonic fluid similar to that typically invoked for many Archaean gold deposits.

9 MINERALISATION

9.1 Introduction

Gold mineralisation at Sukari is hosted exclusively by porphyry. The lack of significant gold grades in chemically reactive serpentinitic wall rocks can be explained in one of two ways:-

- The porphyry represented a favourable host either because of its composition, relative to mineralising fluids, or its mechanical properties or both.
- The Sukari Porphyry was relocated, relative to wall rock sequences, by faulting after gold mineralisation.

While significant post-mineralisation faulting has possibly occurred, the second scenario is unlikely. Porphyry dykes in the hangingwall of the main porphyry body show gold mineralisation of essentially the same character as that in the main porphyry and wall rocks immediately adjacent to those dykes are also barren. Those dykes range in thickness from a few centimetres to several metres. It is not reasonable to postulate that they, along with the main porphyry body, were all relocated by faulting after mineralisation.

Certainly it is evident that the Sukari Porphyry has acted as a rigid body surrounded by weaker rocks. Footwall and hangingwall rocks have taken up strain by development of strong schistosity, almost certainly accompanied by large decreases in volume. The porphyry has taken up strain by development of predominantly brittle fault structures.

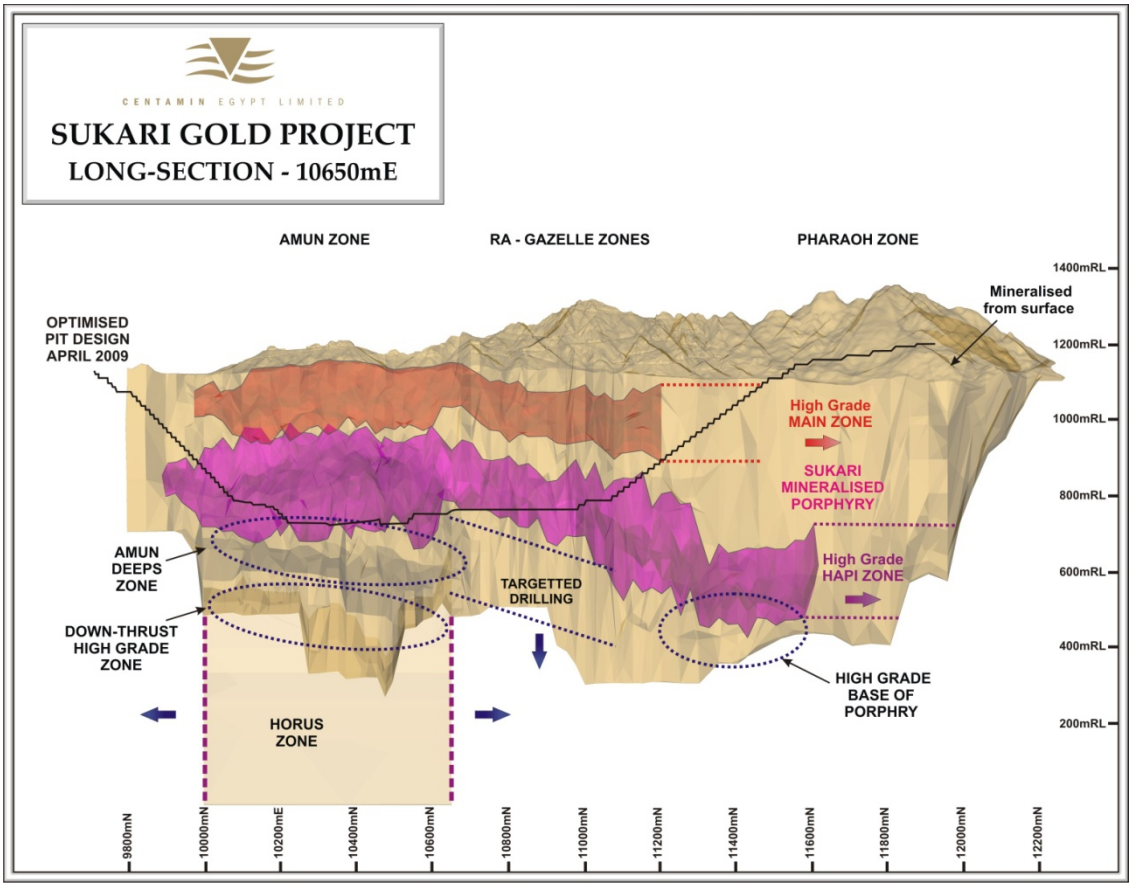
9.2 Mineralisation Geometry

The porphyry host for the mineralisation has a strike length of approximately 2,300m, and ranges in thickness from 100m to approximately 600m (Cavaney 2005). Gold mineralisation within this is not continuous and its deposition has been influenced by major long lived structures, the most important of which are tabular sheets of crackle breccia. Figure 9.1 illustrates the overall shape and size of the porphyry host and the geometry of the high grade Main Reef and Hapi Reef.

Drilling to date indicates that the Sukari Porphyry dips toward the east at between 50º and 75º. The western and eastern contacts of the porphyry are thus regarded as footwall and hangingwall contacts, respectively. Porphyry/wall rock contacts are, in places, vertical or overturned.

The maximum depth of drilling at Sukari is 1176m below surface, the median depth is approximately 240m and the majority (97%) of all drilling is less than 800m deep.

Figure 9.1
Geometry of Main Zone and Hapi Zone within the Host Porphyry

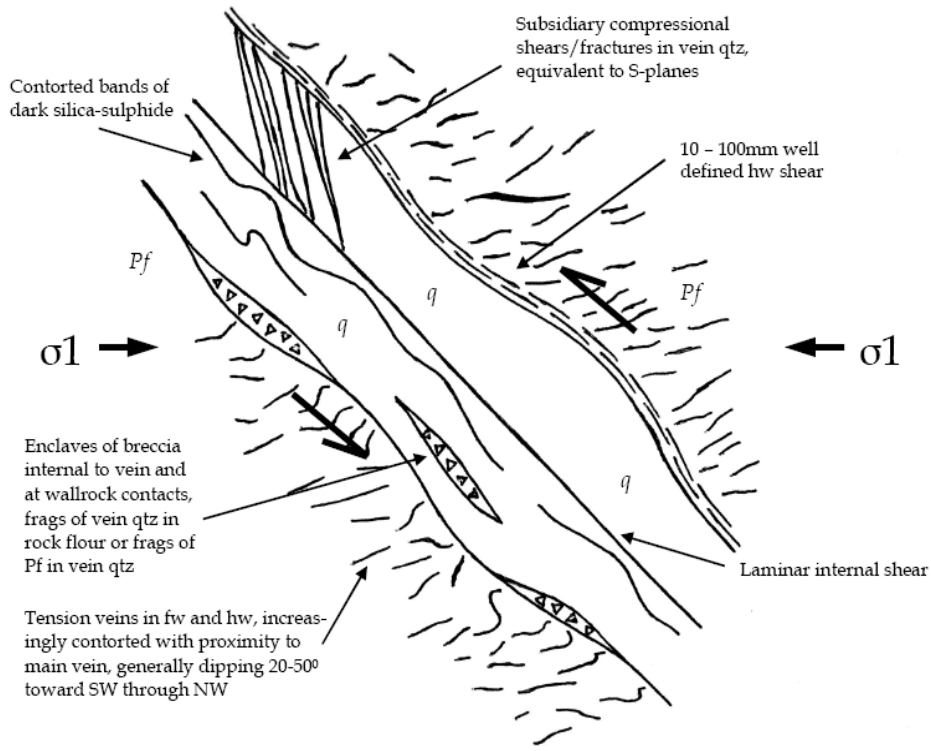


9.3 Deformation Kinematics during Mineralisation

There are several features that indicate that mineralisation at Sukari took place under a regime of near-horizontal, east-west directed stress.

Figure 9.2 shows a sketch diagram of a portion of the Sukari Main Reef. The reef is a complex, multiphase, antitaxial vein. Early vein quartz bands have, in places, been contorted into recumbent folds indicating reverse movement. The vein, and in places the wall rock immediately adjacent to it, shows development of sub-vertical compressional S foliation planes. The orientation of those planes is also consistent with reverse movement.

Figure 9.2
Sukari Main Reef - Sketch Diagram



Large-scale slickensides are present in sheared porphyry immediately adjacent to the hangingwall of the Main Reef. These slickensides, and associated pull-apart terminations, are common in underground exposures and their orientation is consistently steep, indicating predominantly reverse movement. A variety of plunges were observed, some indicating a component of dextral shear and some a component of sinistral shear. All, however, indicate a ratio of dip-slip to strike-slip components of 8:1 or greater.

The majority of small-scale extensional veins are sub-horizontal, generally dipping at shallow angles to the SW or NW. Where contorted, such as in the vicinity of the Main Reef, their tips are often sub-horizontal. Being purely extensional veins, with essentially no displacement across them, they must have formed parallel to the direction of maximum principle stress (σ1) and perpendicular to the direction of minimum principle stress (σ3). The variation in dips, from SW to NW, is consistent with the horizontal component of shear displacement varying between dextral and sinistral at various times during mineralisation.

9.4 Geometry of Vein Arrays and Shears

A key to understanding brittle vein arrays, and consequently being able to predict mineralisation geometry, is to differentiate between through-going structures and discontinuous, extensional veins.

Four types of mineralised quartz veins were observed at Sukari:

- Contorted and banded veins.
- Brecciated veins.
- Shear veins.
- Spaced extensional veins.

The contorted veins are the result of multiple deformation and veining episodes and are similar in character to veins found in low-sulphidation epithermal Au-Ag deposits. Sukari Main Reef represents this type of vein. They represent through-going, long-lived structures on which significant displacement (metres to tens of metres) has occurred. Arsenopyrite mineralisation tends to be concentrated in these veins and they represent the highest-grade mineralisation at Sukari.

Brecciated veins consist of brecciated vein quartz and porphyry rock fragments or porphyry fragments in a matrix of vein quartz +/- sulphides +/- haematite. They are a variant of the contorted veins and, in some exposures, breccia zones can be seen developed at the contacts of contorted veins and host porphyry. The breccia veins also represent through-going structures.

Shear veins appear to be rare. In the few observed occurrences they are centimetre-scale laminar veins with vague contact-parallel layering possibly formed by aligned inclusion trails. They some-times have bands of sulphides and or haematite developed at vein-wallrock contacts.

Extensional veins are distinguished by their short strike lengths and, in places, by internal fabrics that indicate purely extensional opening with no displacement across the vein. Extensional veins normally form stacked arrays between thin linking shears. The linking shears are commonly sericite altered and may contain fine sulphides +/- haematite. They have acted as fluid conduits but their orientation relative to the applied stress field has prevented any infill by vein quartz – they were under compression during the deformation that accompanied mineralisation. The orientation of the shears, not the extensional veins, indicates the large-scale direction of continuity of a stacked vein array.

9.5 Sulphide Development

Gold mineralisation at Sukari is intimately related dominantly to sulphides; pyrite is the most abundant sulphide, followed by arsenopyrite. High gold grades are associated with increased arsenopyrite concentration. The sulphides occur as fine grained, subhedral disseminations in altered porphyry and as blebby sub- to euhedral crystals and finer disseminations in quartz veins, fractures and breccias. Pyrite is found in all the mineralised zones. Deposition of pyrite was continuous throughout the various mineralising stages. Rust coloured goethite is commonly seen to rim pyrite crystals in a semi-weathered environment or fracture surfaces that have been exposed to surficial weathering conditions.

Arsenopyrite is most common in the zones of higher-grade gold mineralisation, so it is common in the Main Reef and in the Hapi Zone, and breccias.

Arsenopyrite is not abundant in the stacked extensional zones and minor quartz veins. It occurs primarily as fine, sub- to euhedral crystals (sometimes characteristically as diamond shaped needles in cross section) in fine, anastomosing stylolitic F veins, and rimming breccia clasts.

Petrographic study has shown that inclusions of pyrite in arsenopyrite and vise versa are common indicating that they belong to the same paragenesis (Khalil, 2006). Both minerals contain numerous inclusions of rutile and wall rock materials suggesting their formation as a result of sulphidisation of pre-existing rocks during the hydrothermal stage. Pyrite and arsenopyrite exhibit deformation and even brecciation textures. Abundant younger native gold is filling stringers and tiny holes in these deformed pyrite and arsenopyrite.

Other sulphides such as galena, chalcopyrite, sphalerite, pyrrhotite have been noted. Some pyrite crystals contain relicts of pyrrhotite and chalcopyrite suggesting the formation of pyrite by pyritization of pre-existing pyrrhotite and probably chalcopyrite. Galena occurs as coarse subhedral crystals in milky extensional quartz veins, generally in areas of strong gold mineralisation, but is rare. Sphalerite is sometimes a significant sulphide mineral. Abundant exsolved chalcopyrite bodies are randomly distributed in the sphalerite host. The sphalerite-chalcopyrite association seems to be filling and replacing the older pre-existing pyrite. The observed galena in pyrite seems to be belonging to the same age of the sphalerite-chalcopyrite association

9.6 Gold

Visible gold in core occurs as anhedral grains in milky white extensional and breccia quartz veins and as intergrowths with pyrite and arsenopyrite, commonly in narrow shear veins at quartz vein margins and margins to clasts in hydraulic quartz vein breccias

Silver is routinely assayed in drill samples; there is no correlation between it and gold and only very low levels returned (generally less than 1ppm Ag). Out of 70,500 Ag assays in the database, 32 are >10ppm, one with a peak value of 165ppm.

SEM and mineragraphic work (Mintek, 2000), determined that high-purity gold occurs free in quartz, on the margins of pyrite and arsenopyrite crystals, and as microfracture fillings. Gold as electrum is paragenetically first as it is often occluded in pyrite and followed secondly by high purity gold (>900 fine) depleted in silver. Gold is fine grained and ranges from 1µm to 40µm.

Based on the microscopic investigation, the following paragenesis is postulated (Khalil, 2006):-

1- Magnetite and Ti-bearing mineral (probably titanomagnetite or ilmenite) were formed during the magmatic stage. Pyrrhotite and chalcopyrite relicts seem to belong to this magmatic stage.

2- Under contact metamorphism and high strain conditions, probably at the site of shear zones, platy hematite and rutile were formed.

3- Pyrite and arsenopyrite are of hydrothermal origin and seem to be formed as a result of sulphidisation of pre-existing rocks.

4- Sphalerite, chalcopyrite, galena and native gold are fracture filling minerals and are syn – to post the host deformed pyrite and arsenopyrite assemblage.

5- Goethite and anatase were formed under weathering conditions.

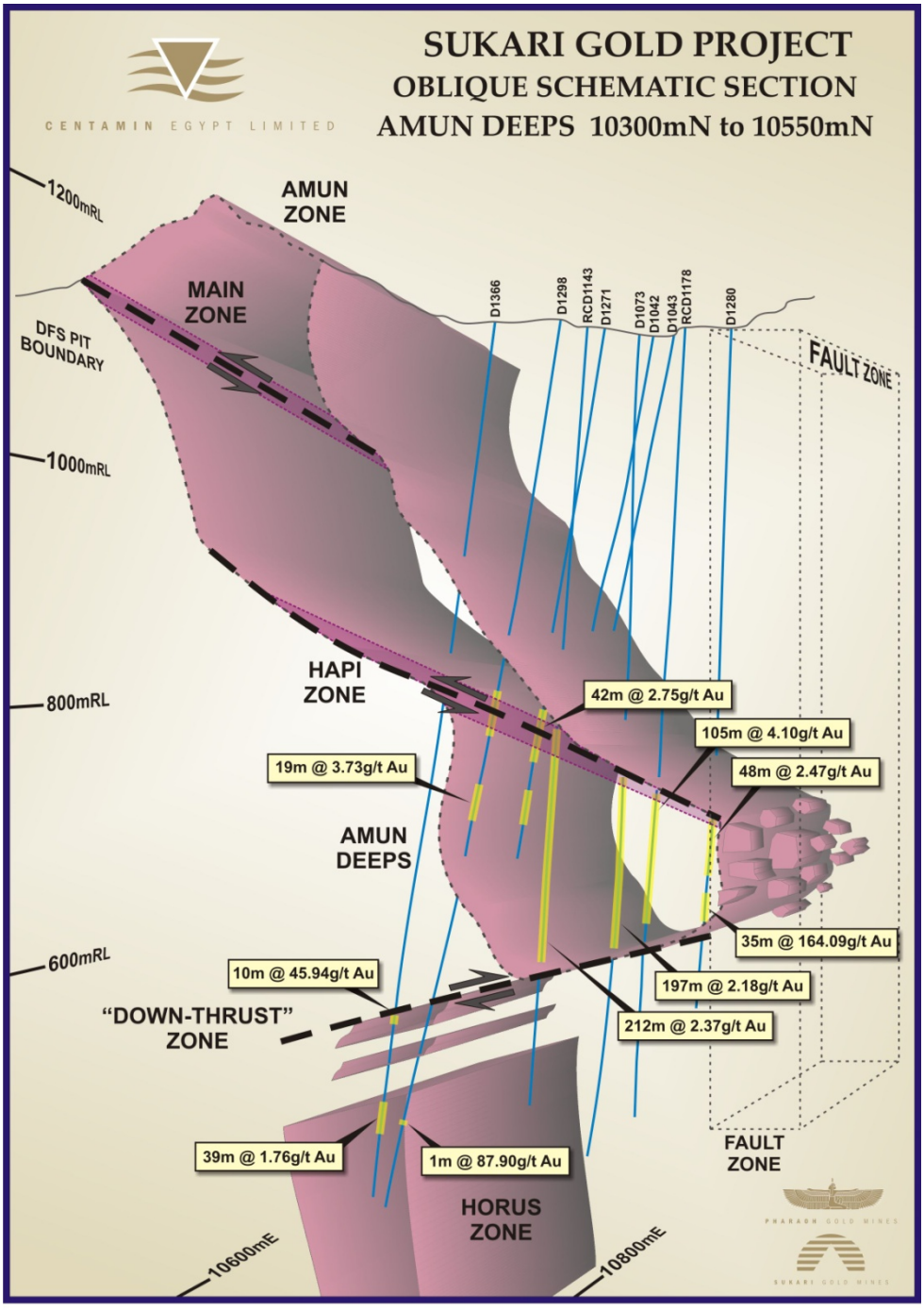
9.7 Deposit-Scale Architecture

Drillhole logging completed by PGM geologists has provided the geological interpretation illustrated in Figure 9.3. Veins and shears that comprise through-going structures have been differentiated from zones of stacked extensional veins. Shear veins can generally be correlated between drillholes although it must be recognised that their character may change over tens of metres, e.g. a structure filled with brecciated porphyry cemented by vein quartz in one hole may appear as a contorted vein in an adjacent hole. Lithologic logging was simplified by recognition of the variable carbonate-silica alteration in serpentinites.

Although several published papers describe the Sukari Main Reef as occurring at the footwall contact of the Sukari Porphyry, that interpretation has been disproved by drilling. The reef and other similar through-going shears (e.g. Hapi Reef) dip at 35º to 50º toward grid east whereas the overall dip of the porphyry body is about 65º. The nature of the mineralised shears almost certainly changes when they pass from the porphyry into surrounding metasediments and serpentinite – their extensions cannot be traced into footwall and hangingwall rocks. It appears likely that they become re-oriented to steeper dipping, compressional structures. Where they cut the porphyry they assume a flatter orientation, possibly comprising Reidel shears in the deposit-scale deformation scheme. It is likely that they cause relatively small (metres to tens of metres) offsets in both footwall and hangingwall porphyry contacts.

A large volume of mineralisation is associated with stacked brittle veins. These clearly occur in zones proximal to the through-going mineralised shears but they also occur remote from them and in such areas the overall orientation of vein packages is open to conjecture. Under the deformation regime, extensional veins should occur in stacks dipping at between 35º and 60º to both the east and west. The west-dipping stacks form in reverse faults that are antithetic to the through-going mineralised shears such as the Main Reef and Hapi Reef. Strike directions of both east- and west-dipping stacks are likely to be diverse because of local strain partitioning and their dip and strike extents are also likely to vary from a few metres to tens of metres.

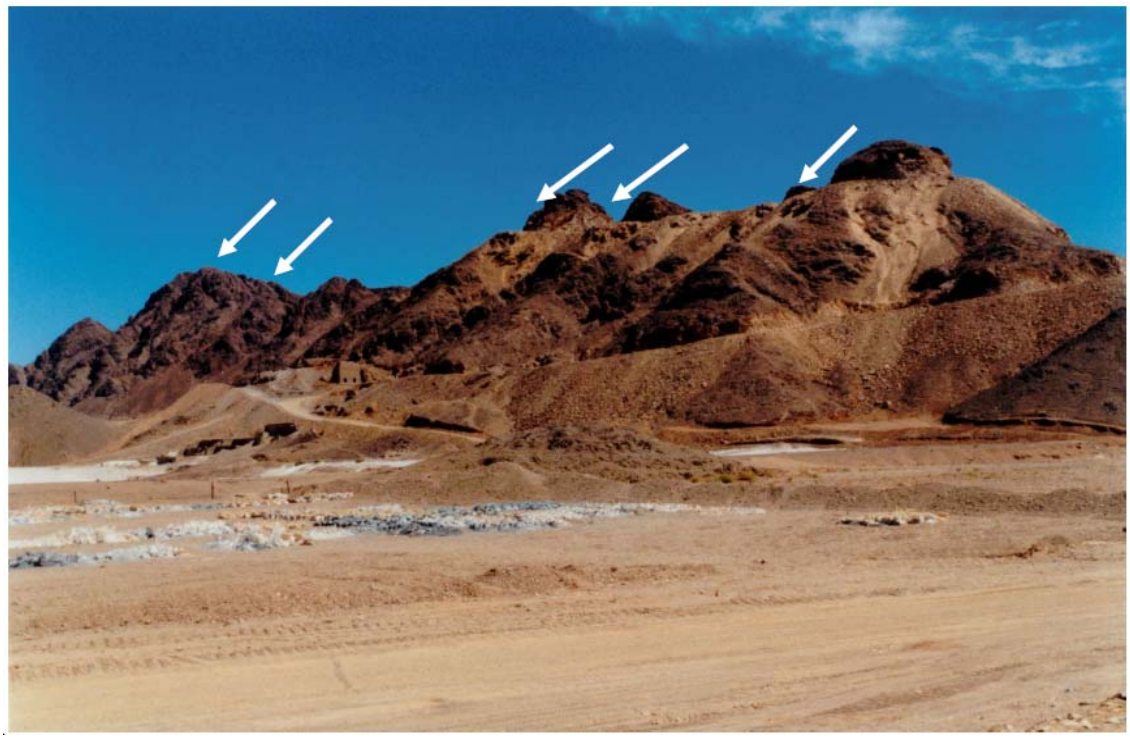
Figure 9.3
Oblique Schematic Section 10,300mN to 10,550mN



There is good evidence for antithetic shears, at least some of which host mineralised vein arrays. Figure 9.4 shows a fault plane dipping at 35º/260º, part of a foliated shear zone that is about 12m wide. The shear zone is replete with millimetre to centimetre thick contorted, extensional quartz veins dipping at shallow angles to the west and these veins appear to be coeval with reverse movement on the shear. Topographic relief on the western side of Sukari Ridge (Figure 9.5) indicates that a number of large-scale faults occur in this orientation but whether all are coeval with mineralisation is not known.

Figure 9.4
Antithetic Reverse Shearing

Figure 9.5
View of Sukari Ridge showing probable Antithetic Shears



9.8 Post-Mineralisation Structures

The only significant post-mineralisation structures observed at Sukari to date belong to a set of ENE-WSW trending sub-vertical fractures associated with andesite dykes. The thickness of these dykes ranges from centimetres to one or two metres and the fractures they fill appear to have negligible offset across them. The dykes are not always planar and, being sub-parallel to drill sections, confident interpretation of them between drillholes is not always possible.

The dykes and associated fracture set are clearly visible on Landsat images, forming an ENE trending swarm over about 10km width, extending to the Red Sea coast. They are possibly related to an incipient transform structure associated with the opening of the Red Sea rift.

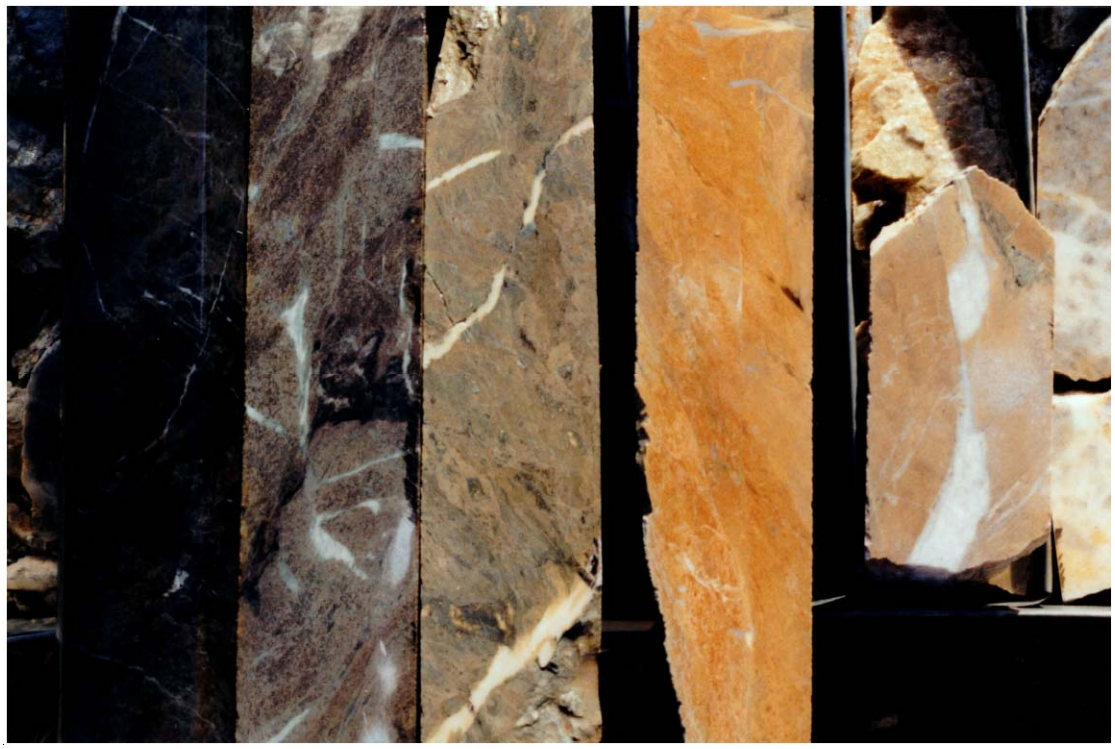
9.9 Alteration

Serpentinites in the footwall and hangingwall sequences have been variably carbonatised and silicified to listwaenite, typically a mixture of ferroan or magnesian carbonate and cryptocrystalline silica. Buisson & LeBlanc (1985) describe this type of alteration, sometimes associated with gold mineralisation, as being commonplace in serpentinitic rocks of the Arabian-Nubian Shield in Egypt, the Sinai Peninsular and Arabian Peninsular.

Figure 9.6 illustrates the progression of this alteration. The barely visible black-green core on the left of the photograph is least altered serpentinite. This rock often shows fine blood-red haematite veining. From left to right the cores show increasing carbonate and silica replacement until the rock is altered entirely to

listwaenite. At its most extreme, and if magnesian carbonate dominates, the altered rock may not be reactive to cold acid. It appears that most, if not all, of the rocks logged in Sukari drill core as rhyolite are in fact altered serpentinite. This misidentification has introduced unwarranted complications in the geology of the wallrock sequences.

Figure 9.6
Progressive Alteration of Serpentinite



The timing of carbonate alteration relative to gold mineralisation is unknown. Carbonatised serpentinite does, in places, contain fine-grained pyrite and quartz veins but no detectable gold. Although it would make sense to postulate that carbonate alteration of serpentinite wallrocks accompanied the gold mineralisation event it is difficult to conceive of a process that would so efficiently partition gold deposition into only the porphyry host.

10 EXPLORATION

10.1 Introduction

Exploration at Sukari is at a very advanced stage. It has culminated in the discovery of a gold deposit, and the current emphasis is on the development and construction of a gold mining and centralised processing operation. However, significant potential remains to increase the known Mineral Resources and Mineral Reserves, and exploration will continue during the life of the mining operation.

10.2 Exploration Methods Used

Gridding and traversing was carried out at 1:10,000 scale.

Mapping was carried out on a 1:500 scale (Amun Zone) and 1:1,000 scale.

Trenching and channel sampling within the cut trenches was undertaken, mainly within zones of intense silicification and sulphidation. Total length of trenching was 1,143m.

Channel sampling from historical underground workings was completed. A total of 982 samples were submitted for analysis.

Augering was undertaken on two heaps of tailings on a 10m by 10m grid to a maximum depth of 1m. A total of 327 samples were taken for gold analysis.

Rock chips were taken initially on 160m spaced lines with some supplementary infill lines. In addition, dykes, quartz veins and zones of hydrothermal alteration were grab-sampled. Later rock chip sampling was undertaken on 100m spaced lines and samples were approximately 1m to 2m in length.

Regional sampling and prospecting consisted of rock chip and channel sampling at various small mines in the vicinity.

Heavy mineral sampling was undertaken at various suitable sites in wadis.

Both reverse circulation ('RC') and diamond drilling have been employed for exploration and resource definition purposes.

10.3 Use of Contractors

All exploration work has been carried out by, or supervised by PGM geological personnel. The only significant use of contractors has been for drilling and assaying.

The contract drilling is currently being undertaken by Capital Drilling (Bermuda) Ltd.

Preparation and assaying of samples produced during the various exploration programs has been undertaken at either Minesite Reference Laboratories (earlier phases of drilling prior to 2005) or Ultratrace Analytical Laboratories

(used during the 2005 and subsequent drilling), both situated in Perth, Western Australia.

10.4 Interpretation of Exploration Information

10.4.1 Introduction

At this stage of the project, interpretation of the exploration information has been largely superseded by the estimation of resources and reserves, however some general comments may be made on interpretation of the exploration results.

10.4.2 Geological Mapping

Geological mapping in the Sukari area has been successful, despite difficult topography (steep cliffs). The mineralisation and alteration can usually be seen at the surface and mapping has been useful in defining the extent and morphology of the porphyry that hosts the mineralisation

10.4.3 Rock Chip Sampling

Rock chip samples have been useful in confirming the position of the lode horizon. Rock chip sampling conducted by PGM has been surveyed and has been included in the resource estimation sampling database.

10.4.4 Trenches

Excavation of trenches as an exploration technique has been very successful. Significant gold intersections in trenches typically overlie sub-surface zones of similar grade and width, as defined by subsequent drilling. Trench sampling positions have not been surveyed and have not been included in the resource estimation sampling database.

10.5 Data Reliability

Data acquired during exploration at Sukari is considered to be very reliable. All work has been carried out by technically qualified personnel, and has been planned and supervised by highly trained and experienced geoscientists. The location of all exploration data is known with adequate accuracy. The quality of geochemical analyses has been monitored by the use of blanks, standards, field duplicates, and check analyses via primary and umpire laboratories.

11 DRILLING

11.1 Introduction

The first known drillholes at Sukari are five holes drilled by the EGSMA-Soviet research team in 1976-78 (Azzaz, et.al., 1978). The locations of these holes are known and logging and assay records are available, however, a lack of confidence in the quality of the sampling and assaying from these holes preclude their use in the resource dataset. The geological logs are included in the drilling database and have been used to aid in the interpretations of the Sukari geology.

Drilling by PGM commenced in April 1997 and is ongoing at the time of this report. PGM's drilling at Sukari has been by diamond core drilling, using two Atlas Copco Craelius 252 rigs drilling with standard (i.e. not wire-line) TT46 gear to produce 35.3mm diameter core. These rigs are skid-mounted electric-hydraulic drills normally used for drilling in underground mines. Several holes were drilled from the No. 1 (110m) level of the old Sukari mine workings. In August 2000 the drilling effort was augmented by a track-mounted Atlas Copco 262 diesel-hydraulic rig. This rig uses standard (i.e. not wire-line) TT46 gear to produce 35.3mm diameter core.

During the first half of 2002 the drilling capacity at Sukari was significantly enhanced with the introduction of a drilling contractor util ising larger, more flexible, drilling rigs; two multi-purpose diamond and reverse circulation (RC) capable drilling rigs (one KL600 and one LMP850). In the last quarter of 2002 two CS1000 rigs, in second quarter of 2007 two CS14 (wire-line) configured for HQ and NQ core drilling only were added to the drilling fleet at Sukari, and in 2009 and additional HQ and NQ Atlas Copco CS3001 core drilling rig was added to the fleet.

South of approximately 11,300mN drilling has been conducted on 25m spaced sections oriented grid east – west. North of 11,300mN and South of 12,200mN drilling is primarily spaced at 50m sections. Locations of drillholes used for resource estimation are shown in Figure 11.1.

At the time of writing the drillhole database comprises approximately 488,795m of drilling in 1700 holes, including some holes abandoned because of drilling difficulties and some intervals that are awaiting assays. Five historical diamond holes (D001 to D005) are included in the drill database statistics, however, neither their geological nor sampling information has been used as the recorded information for these holes can not be verified. Some summary details of the available drillhole dataset are provided in Table 11.1.

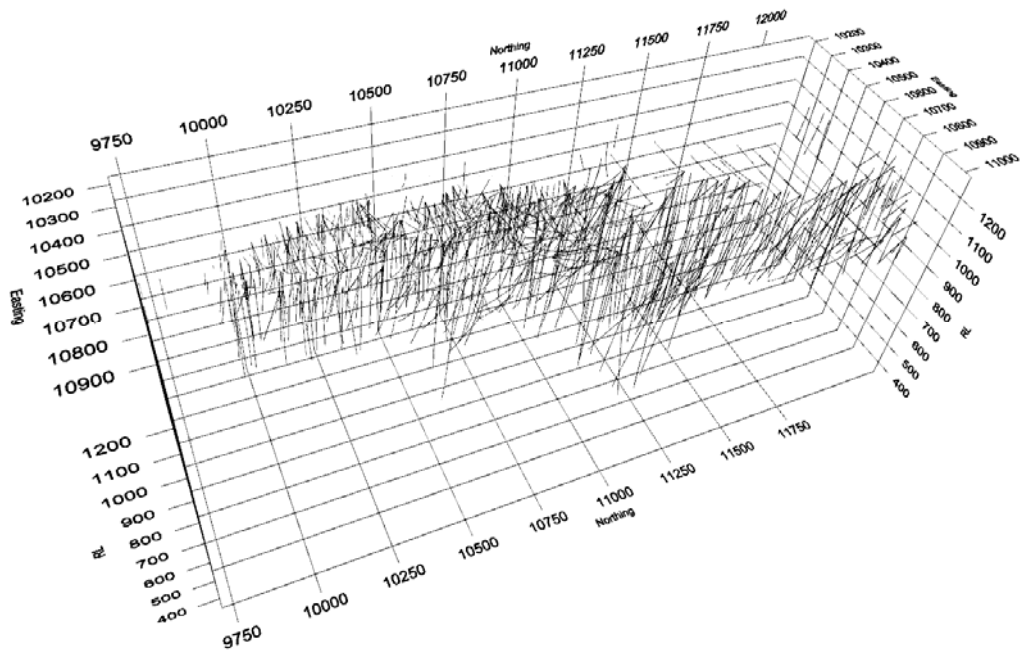
Table 11.1

Summary details of Sukari Drillhole Dataset

Total Number of Drillholes	1700
Total metres (all holes)	488,795
RC total metres	126,073
DD total metres	362,722

Commencing October 2005 the drill related data has been stored in Century Systems geological database, the complete assay database was imported during August 2006. Century Systems is used to collect geological data in the field, validating data on entry, store the data in a secure open database, and validate quality control from the lab. Data is migrated to mine software by ODBC link and CSV exports.

Figure 11.1
Drillhole Locations



Note: View looking down towards grid west

11.2 Survey Controls

A 50mN by 20mE control grid was established by a contract surveyor using a point located about 200m south-west of the EGSMA garages as grid origin (10,000mN, 10,000mE, 1100mRL). A base-line striking 020º magnetic and cross-lines were established using a theodolite. Stadia measurements were used for distances, with slope corrections as required.

The highest point of Sukari Ridge is marked by a spot height on the Jabal Nuqrus 1:50,000 scale topographic map sheet, with elevation 630m. The map sheet also shows a survey station at about the middle of Sukari Ridge with

elevation 591m. This point is ground marked by a cairn of stones. Using these two points as references, the elevation of the grid origin is approximately 380m above sea level.

GPS observations place the grid origin at 24º 56' 48"N 34º 42' 36"E. This location reconciles well with the location of a survey control point shown on the 1972 1:1000 scale geological map of Sukari mine area compiled by EGSMA. The location of that point, marked by a small cairn of stones near the southern end of Sukari Ridge, above the old mine workings, is listed as being 24º 56' 48.78"N 34º 42' 48.71"E. Location of that point was presumably by sun observations or by carrying survey from distant controls.

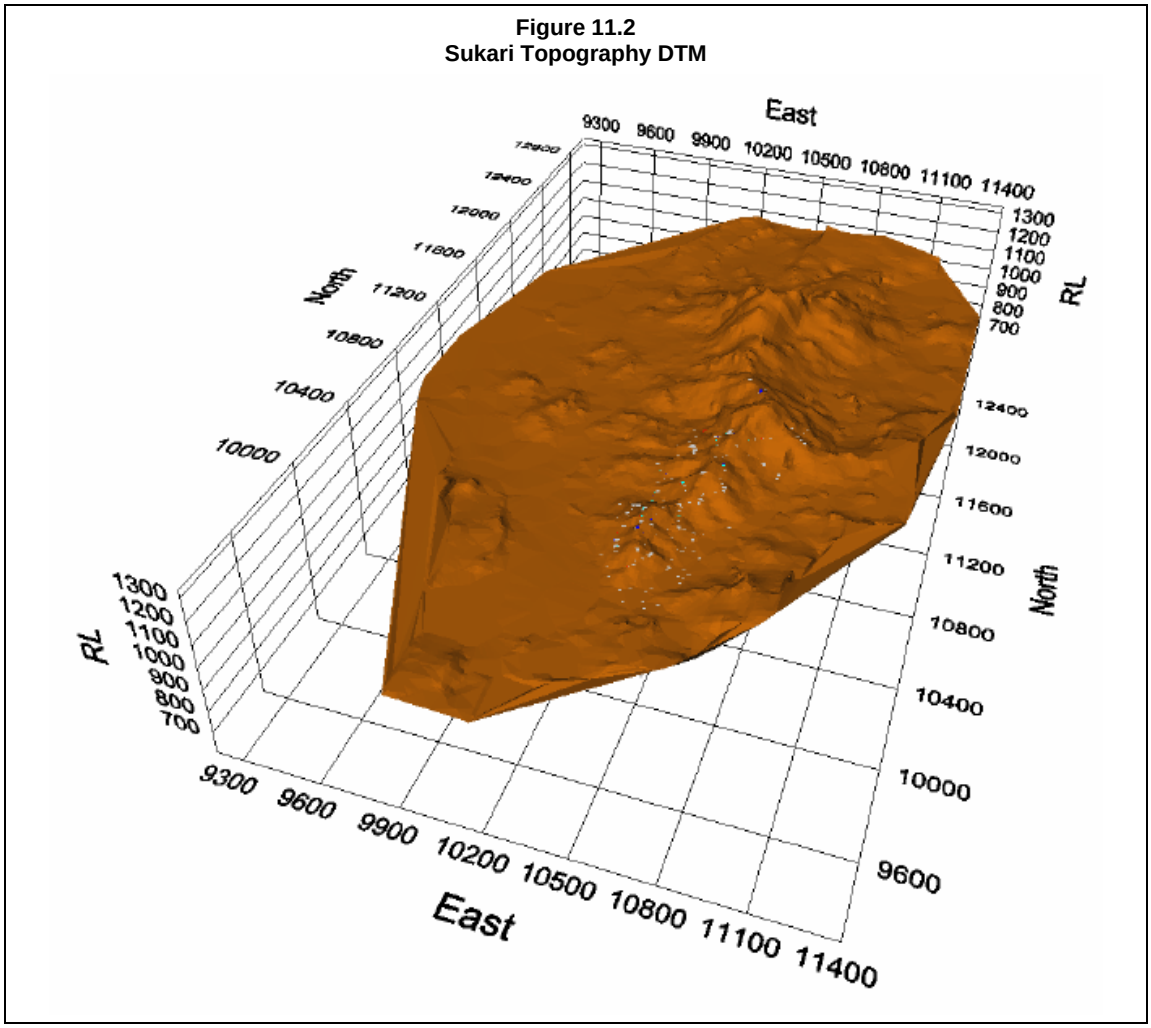
In February 1998, licensed surveyor Mr G. Robinson placed and surveyed control stations around Sukari Ridge and determined the bearing of the Sukari exploration grid base-line. The true bearing of the grid base-line determined from sun observations at the grid origin was calculated as 22º 29' ± 10''. This is very close to 020º magnetic as at year 2000.

In 1999, qualified surveyor Mr F. Howard established several additional local control points, using Robinson's controls.

11.3 Topographic Survey Data

Spot heights and features have been surveyed by PGM surveyors using Sokkia total station equipment, controlled by reference to pillars established by Robinson and Howard. Additional local controls have been established by closing to earlier control points. Surveyed spot heights form the basis for the digital terrain model used in resource estimation and mine design (Figure 11.2).

Figure 11.2
Sukari Topography DTM



11.4 Down-hole Surveys

Most PGM drillholes drilled prior to 2003 have been down-hole surveyed at depth intervals of 50m to 100m using a single shot camera device. The first survey is normally at about 4m depth and the dip and azimuth at that point is allocated to the collar point (i.e. zero depth). Magnetic azimuths recorded by the survey instrument are reduced to grid bearing by subtraction of 22º, this correction being derived by taking shots on surface along the grid baseline.

Holes less than 50m long are generally not surveyed. In addition, several holes have not been surveyed due to unstable ground conditions and twelve holes (holes 92 to 103) were not surveyed because of survey equipment breakdown. For unsurveyed holes, dip and azimuth of the rig frame is measured on surface and those readings allotted to the top and bottom hole depths in the drillhole database. There has been no attempt to estimate deviations in unsurveyed holes.

During a site visit in 2000, an H&S representative cross-checked 50 survey discs with records in the computer database. Two azimuth readings in the field

book and computer database were five degrees different to disc readings. All other readings were in agreement.

Even given the small drillhole diameter, drill path deviations are small. Average change in dip is about 1º per 50m; average change in azimuth is about 1.5º per 50m. This is thought to be due to the requirement to drill with low pull-down force because of ground conditions and the use of lightweight rigs. In H&S's opinion, the lack of down-hole surveys for a number of holes drill earlier period of PGM's exploration does not introduce any significant error into the resource estimate.

For the recent drilling downhole surveying is conducted on a more rigorous basis. For all PGM and contractor diamond drillholes the first survey is taken at 10m and then at 50m intervals thereafter, where ground conditions allow. Dip only surveys are taken for RC holes. If the RC hole is used as a pre-collar for diamond the RC component is re-surveyed by the diamond rig crew to obtain surveyed azimuths for this proportion of the hole.

12 SAMPLING METHODS

12.1 Surface Mapping and Sampling

Two EGSMA geological maps are available for Sukari. A 1:1,000 scale map covers the Amun Zone, in the vicinity of the old Sukari Mine. A 1:2,000 scale map covers the entire 2.5km strike length of the Sukari Ridge. These maps were used as bases for early PGM evaluation work.

In 1998 surface geology of the Amun Zone was grid mapped at 1:500 scale by consulting geologist Mr. R.J. Cavaney. PGM geologists have also mapped several surface sample traverses in detail.

Sample trenches excavated by the EGSMA-Soviet study team in the 1970s have been re-sampled by PGM and additional lines of surface rock chip samples have been taken also. Surface rock chip sample traverses are shown in Figure 12.1 and have been included into the resource dataset.

Trench sample locations have not been surveyed and these data have been excluded from the sample database used for resource estimation. Surveyed locations are available for surface rock chip samples and they have been used in resource estimation. The surface rock chip samples sum to 4,908 (10,540m of two metre samples) originating from 180 sample lines.

12.2 Underground Mapping and Sampling

Rock chip and channel samples have been taken from adits, underlay shafts, drives and stope pillars in the old Sukari gold mine and other workings on Sukari Ridge. Sample locations were surveyed by tape and compass. Assays of underground sampling have not been used for resource estimation.

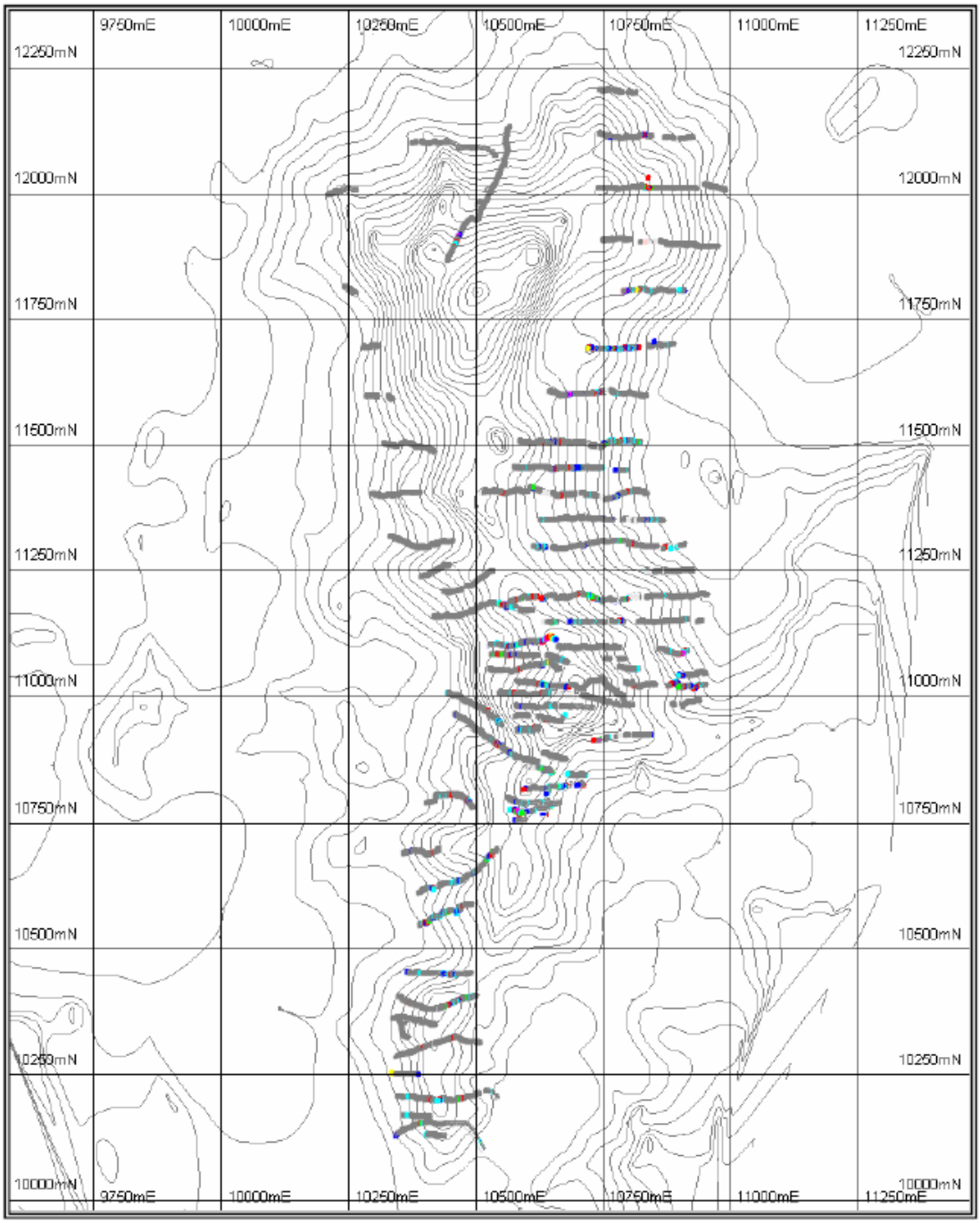
12.3 Logging and Sampling, RC Holes

Sukari geological and technical staff are present during the drilling of all RC holes and are responsible for the taking of the samples for assay. In addition to sample collection, the drill rig geologist is also responsible for recording:-

1. Geology – logged at the drill rig onto RC logging sheet.
2. Sample Recovery – recorded onto the RC logging sheet as a mass (kg) and as a percentage of theoretical sample return.

Samples are routinely taken at one metre intervals. Prior to drilling and sampling the next drilling metre the driller is instructed to clean the sample system on the rig. This is undertaken by lifting the face off the hammer off the bottom of the hole and blowing back through the rods and out through the cyclone to clear all sample from the previous meter. This is undertaken to minimise any potential down-hole contamination.

Figure 12.1
Surface Rock Chip Sampling



The drilled material for a single one metre sampling interval is collected into a large green plastic bag taken from the bottom of the drill rig cyclone. The total recovered sample is weighed prior to splitting. The sample weight is recorded in kilograms (kg) and also as a percentage recovered on the geological log sheet.

Table 12.1 below shows the expected recovery with different RC hammer diameter using a BD of 2.68t/m^3.

Table 12.1
Expected Recovery from 1m RC Sample by Weight

Inch	6	5.38	5.06	5.00	4
mm	**152.4**	**136.7**	**128.6**	**127.00**	**100.16**
130%	63.6	51.1	45.2	44.1	28.2
120%	58.7	47.2	41.8	40.7	26.00
110%	53.8	42.2	38.3	37.3	23.9
100%	***48.9***	***39.3***	***34.8***	***33.9***	***21.7***
90%	44.00	35.4	31.3	30.5	19.5
80%	39.10	31.4	27.8	27.10	17.40
70%	34.2	27.5	24.4	23.7	15.20
60%	29.3	23.6	20.9	20.3	13.00
50%	24.5	19.7	17.4	17.00	10.9
40%	19.6	15.7	13.9	13.6	8.70
30%	14.7	11.8	10.4	10.20	6.50

At the time of writing, RC sample recoveries were available for 56,986 RC sample intervals. Statistics of RC recoveries are shown in Figure 12.2; average RC sample recovery is about 86 percent and is considered good. Gold assays and RC recoveries are plotted as a scatter in Figure 12.3 and shows that there is no significant correlation between grade and RC sample recovery.

Figure 12.2
Histogram of RC Sample Recoveries

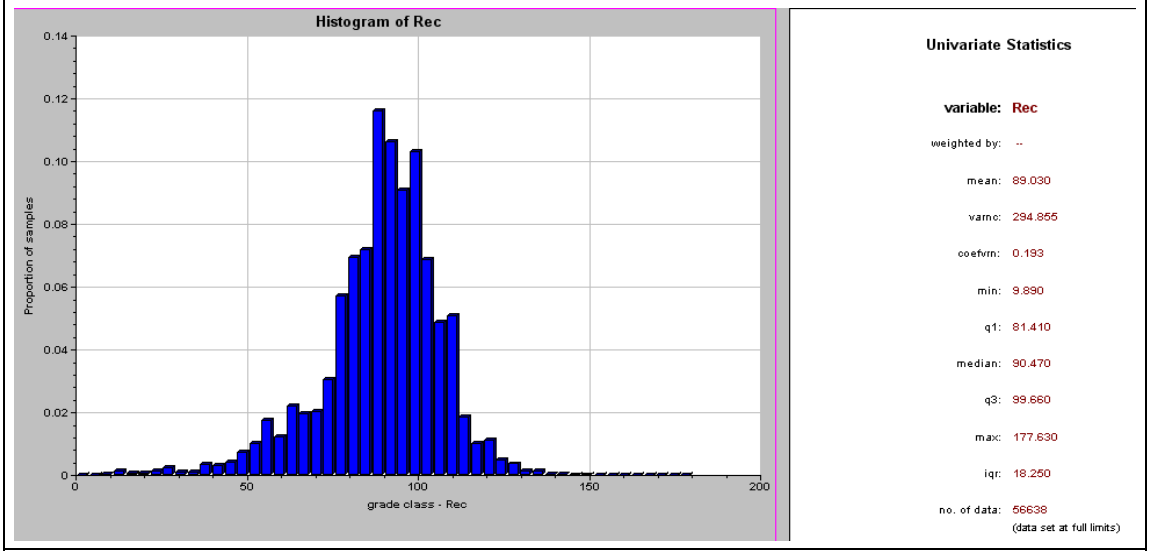
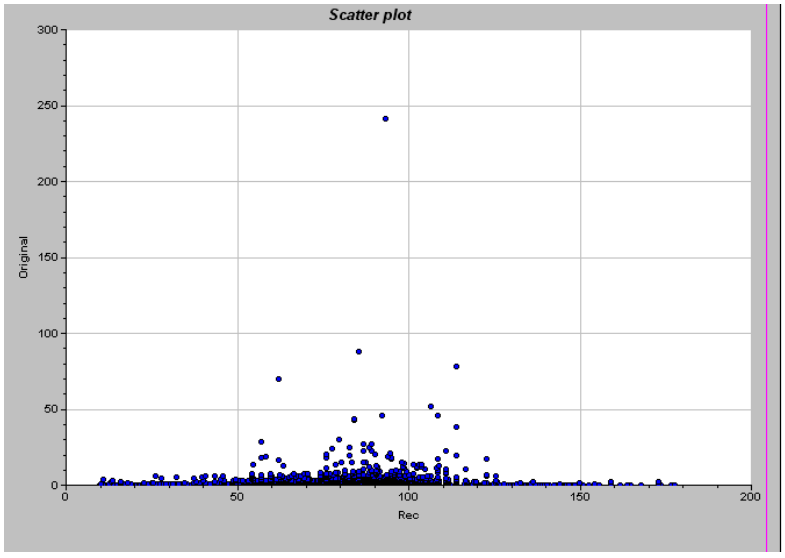


Figure 12.3
RC Sample Recovery vs Gold Grade



The bulk RC sample (collected in large green plastic bag) is split at the drill rig site to obtain the sample for assay. The bulk sample is reduced in mass by passing the sample through a 50-50 splitter (with 25mm spaced riffles), until a 4kg to 5kg sample is collected. The final split sample is placed in a numbered calico bag for dispatch to the onsite sample preparation facility. The method for splitting is described as;

"*prior to passing the sample through the splitter place a sheet of plastic over the top of the splitter and pour the sample on to this plastic sheet. Then quickly pull away the plastic sheet to allow the sample to drop through all together. This is important as the sample will be layered in the green plastic and ensures a good mix of the sample, before decreasing the sample mass*."

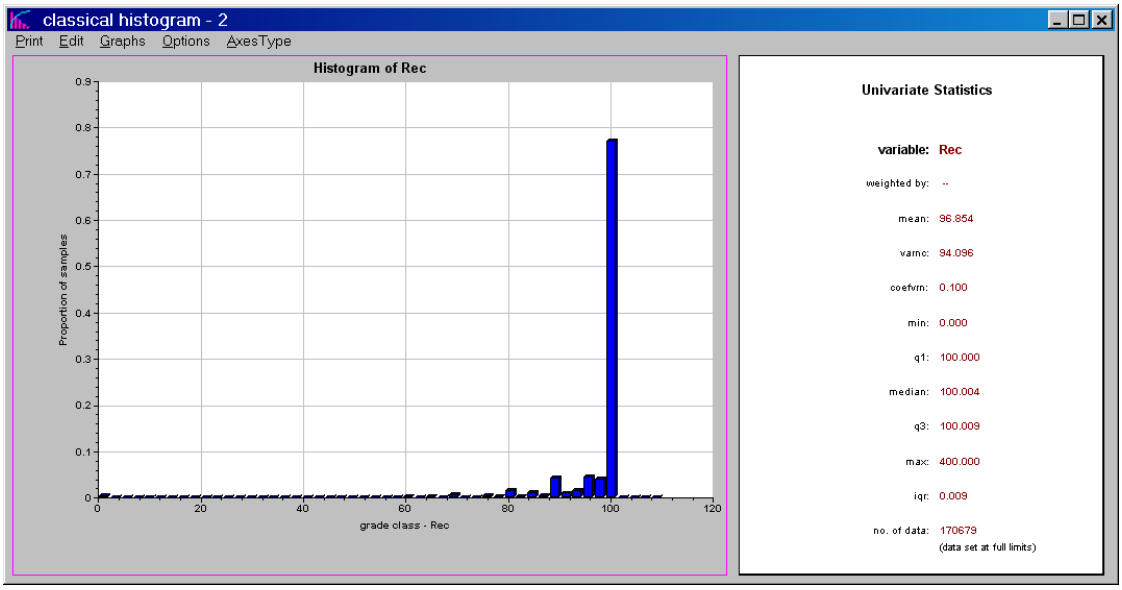
In the case where a wet sample is encountered the sample is collected in a poly weave bag identified by the full BHID (Bore Hole Identification) and meter interval marked in aluminium tag. The sample is allowed to dry for 2 days and then split in the same way as the dry sample described above. It is understood by H&S that wet drilling conditions are rarely encountered in RC holes at Sukari.

12.4 Logging and Sampling, Core Holes

Core is laid out in trays, each of which carries 6m to 7m of core. Drillers mark each core tray with drillhole number and tray number, and a wooden core block showing down-hole depth is placed at the end of each core run. Sampling is normally on a 1m basis.

The geologist checks down-hole depths marked on core blocks, then measures the length of core recovered per drill run (i.e. between consecutive core blocks). At the time of writing, measured core recoveries were available for a significant proportion of the total diamond drillholes. Approximately 306,364m of diamond core metres have been assessed for recovery. Statistics of length-weighted core recoveries are shown in Figure 12.4; average core recovery is about 96%. Generally the core recoveries are acceptable.

Figure 12.4
Histogram of Length-Weighted Core Recoveries



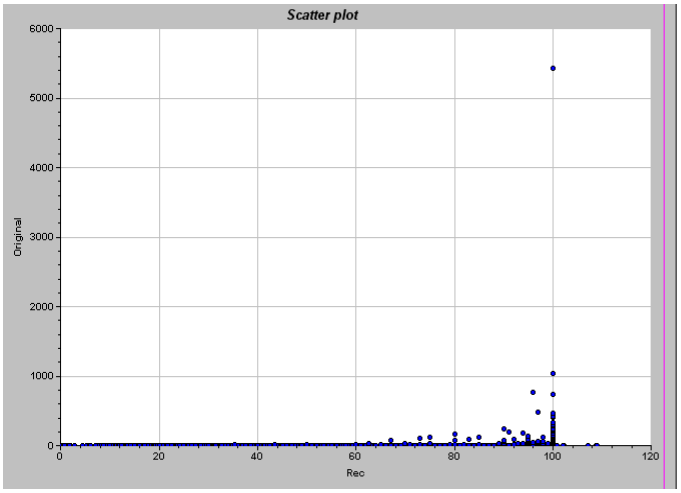
Gold assays and core recoveries were composited into 1m intervals to allow direct comparison. Figure 12.5 shows a scatter plot of gold grades against core recoveries. There is no significant correlation between grade and core recovery.

Core is marked in (normally) one metre intervals and logged for rock mechanical properties, e.g. number of fractures, number of pieces greater than 10cm length. The core is then photographed (both wet and dry).

At approximately 50m intervals, a 10cm to15cm length of core is selected for bulk density determination. The analysis of this bulk density information is provided in Section 13.3.

The core is then logged for lithology, alteration and mineralisation features, with descriptors recorded for each sample interval. A sample number is then assigned to each interval that is to be assayed. Sample “from-to” intervals are recorded in a sample ticket book and two numbered tickets are inserted in a calico sample bag. The hole number, interval and sample number are also written on the calico bag by the geologist.

Figure 12.5
Gold Grade vs Core Recovery



Data Statistics

variable: Rec		Original
weight by:	--	--
mean:	96.849	0.405
varnc:	93.216	201.065
coefvrn:	0.100	35.038
min:	0.000	0.001
q1:	100.000	0.001
median:	100.004	0.009
q3:	100.009	0.129
max:	109.000	5420.000
iqr:	0.009	0.128
covarnc:	0.191	
Pearson:	0.001	
Spearman:	-0.029	
no. data:	170672 / 170679	

Core is sawn in half with a diamond saw under supervision of a geologist. Half core is bagged in numbered calico bags. Sample intervals are regularly checked against the intervals written on both the calico bag and on sample tickets contained in the bags. Intervals of core too broken to be sawn are sampled by taking every second piece of core in the sample interval.

12.5 Sample Quality

The production of high quality samples has been a key aspect of the entire exploration programme and this philosophy has been applied to all aspects of sampling.

During rock chip sampling great care was taken to ensure that a sampling channel of consistent width and depth was collected.

Diamond core recoveries are routinely recorded as part of the standard geological logging practice. The vast majority of diamond core is highly competent and an average recovery of 96% has been achieved. Due to the consistently high core recoveries, the opportunity for over or under-reporting grades from diamond drill samples is considered minimal.

RC sampling weights have been recorded routinely allowing for calculation of a theoretical recovery. An average recovery of around 89% has been achieved and this is considered excellent. All RC drilling was carried out using face sampling hammers, employing rigs equipped with a large air capacity and pressure to effectively flush samples from the hole face through the rod line and hoses. The RC string and hole were cleaned after each drill rod was attached and prior to continuing drilling.

RC and diamond holes have been routinely sampled at one metre intervals. This sampling interval is considered to be small compared to the typical width of the deposit and the spatial variation in gold grade within them. In addition, higher grade zones are common within the deposits, and are generally thick enough to be defined by a series of adjacent one metre samples.

13 SAMPLE PREPARATION, ANALYSIS AND SECURITY

13.1 Sample Preparation

All diamond core and RC samples are prepared at the Sukari sample preparation facility.

13.1.1 Core Samples

The half core sample is put in an oven for two hours (110ºC). The dried material is initially passed through a jaw crusher to produce fragments less than 10mm. The crushed material is then passed through a secondary crusher, to reduce the sample further (down to 4mm) prior to pulverisation. The sample is split into two equal halves, one half returned to the sample calico bag, using a riffle splitter. One of the equal sized splits is used for pulverizing and the other added to the calico bag for storage. The current procedure for cleaning the sample preparation equipment between samples is by high pressure air guns with dust extraction fans. Previous to this procedure (circa 2003) barren quartz wash was used.

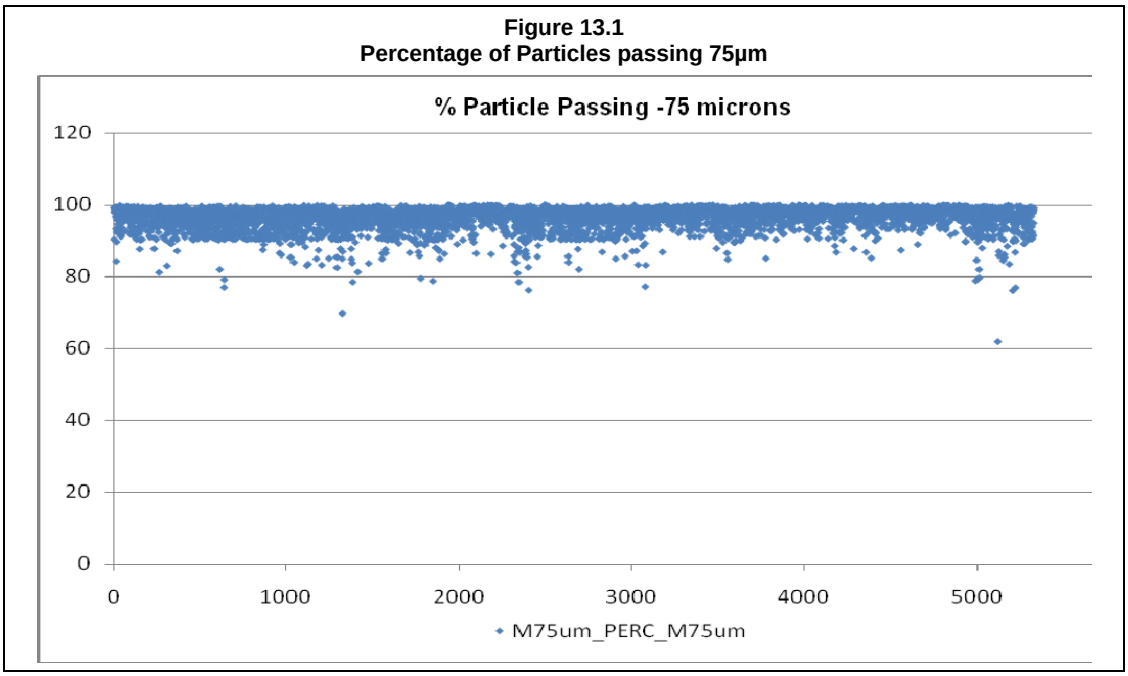
13.1.2 RC Samples

The RC samples are split upon receipt at the sample preparation facility to give an adequate size sample for the pulveriser.

All crushed material is pulverised as a single pass in a ring mill. The Sukari sample preparation facility has four Labtech Essa pulverisers (LM2) available. The pulverised sample is finger tested for powder consistency. If there is any roughness or coarse material, the sample is re-pulverised. Two approximately 250g craft packets of sample are taken by dipping a small plastic scoop from several places into the pulverised material within the bowl once the puck has been removed. One sample is submitted to the contract assay laboratory and the other is submitted to the Egyptian Geological Mining and Survey Authority. Standards and blanks are submitted with the samples for assaying (Section 14).

Figure 13.1 shows the results of 5,327 sieve analyses by Ultratrace Analytical Laboratories of Perth, Western Australia, conducted during the 2005 and 2006 drilling campaign. The plot shows that the Sukari sample preparation facility is performing well in terms of ensuring that pulverisation of the sample is being undertaken according to design criteria, i.e. better than 90% samples passing 75µm. The Sukari sample preparation facility is achieving approximately 96.23% of the samples passing 75µm.

Figure 13.1
Percentage of Particles passing 75µm



13.2 Assaying

All gold assays on samples collected by PGM have been conducted by either Minesite Reference Laboratories (earlier phases of drilling prior to 2005) or Ultratrace Analytical Laboratories (used during the 2005 drilling), both situated in Perth, Western Australia.

Upon receipt of samples, pulps are finger tested for fineness and if grinding is considered insufficient, individual sample pulps are selected for re-pulverising.

Earlier assaying by Minesite Reference Laboratories (before recommencement of exploration activities in 2005) was conducted by Aqua-Regia digest ('AR') of a 25g portion selected by taking multiple dips from the sample pulp packet. Gold is extracted from the aqua-regia liquor into DIBK solvent and gold content determined by AAS. At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system ('LIMS').

Ultratrace Analytical Laboratories have conducted all assaying of samples processed since the recommencement of exploration in 2005. Assaying by this laboratory has continued to be by AR, however, using a 40g charge where the digest is diluted, mixed and an aliquot of the acid solution is taken and analysed directly by ICP-MS for gold and other elements.

Of samples that return assays more than 1g/t Au, a selection are subsequently analysed by 30g (Minesite Reference Laboratories) or 40g (Ultratrace Analytical Laboratories) fire assay with AAS finish.

13.3 Rock Density Data

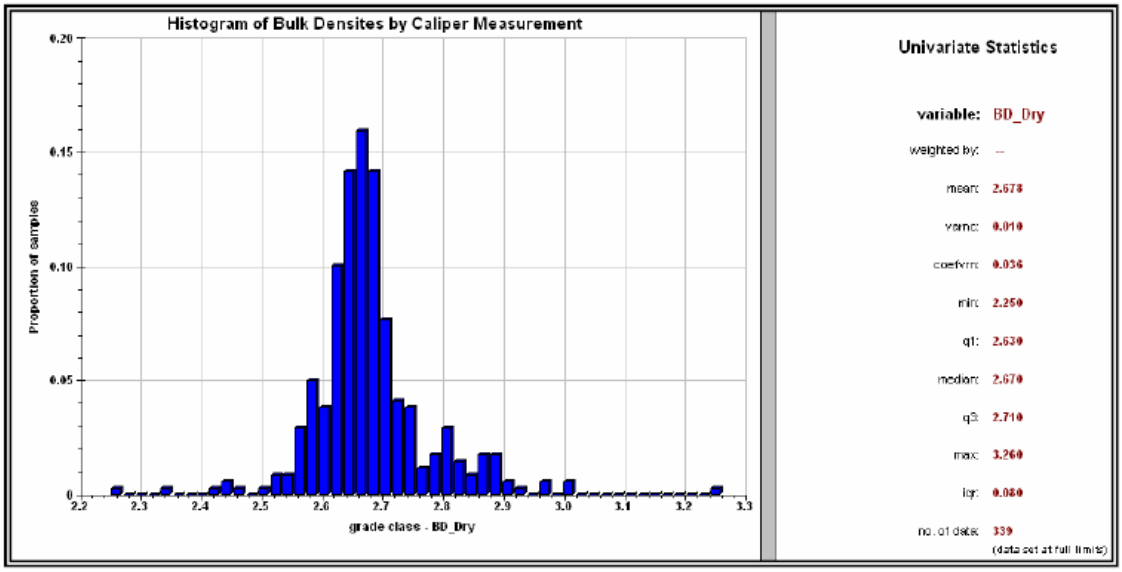
13.3.1 Site Measurements

Samples comprising intact pieces of core 10cm to 15cm length are taken at approximately 50m down-hole intervals. Bulk density ('BD') is measured on site by two techniques:-

- Each sample is weighed twice on a laboratory electronic balance. Sample volume is estimated by measuring sample length (two measurements, rotating core each time) and core radius (six measurements, two at each end and two in the middle of the sample). Radii are measured using a vernier caliper. An effort is made to cut the ends of core samples flat in order to make length measurements more accurate.
- Weigh the sample as for (1), then immerse the sample in a measured volume of water in a graduated cylinder in order to measure sample volume by displacement. The graduated cylinder allows reading of volumes to about ± 0.2cc.

Prior to 2005 a database of 339 BD measurements had been generated. Bulk densities calculated by the two methods rarely disagree by more than 5% and normally agree to within 2%. The mean of BD calculated from calliper measurements is 2.68g/cc, compared to 2.69g/cc for BD's using displacement volumes. Because samples have not been waxed prior to immersion for measurement of volume by displacement, H&S regard the volumes calculated from calliper measurements likely to be more accurate. A histogram of BD's calculated from calliper measurements are shown in Figure 13.2.

Figure 13.2
Bulk Density Measurements Histogram



A bias has possibly been introduced into BD measurements by the process of sample selection. The methods used for bulk density measurements at site

require solid lengths of drill core. In places, drilling has encountered broken and sheared ground. These zones almost certainly contain rock of lower bulk density but the volume of gold resources contained in such zones is thought to be small.

Bulk densities used in the current resource model are detailed Section 17.1.10. For the latest resource model BD's for the various modelling domains have been calculated. The BD data available at the time of writing has been augmented with 1779 additional measurements (2118 in total). Review of the expanded BD dataset show no material change in the average BD described above, however, the latest study has separated and calculated average BD's for use in the various modelling domains.

13.3.2 Independent Checks

Two composite metallurgical samples were subjected to bulk density determinations (Lakefield Oretest, 2000a), yielding 2.696g/cc and 2.69g/cc. In addition, bulk density measurements of 2.644g/cc, 2.720g/cc and 2.571g/cc on lump rock samples used for comminution test work (Lakefield Orestest, 2000b) support the drill core bulk density measurements. Samples for comminution test work were taken from the old Sukari mine workings.

In August 2000, pulp material remaining from samples collected from site by H&S was submitted to Minesite Reference Laboratories for pulp density determinations. A Beck-man Model 930 vacuum pycnometer was used for measurements on 50g pulps. Claimed accuracy for the instrument and method is ± 10 per cent. Fifteen samples of quartz wash material reported bulk densities ranging from 2.59g/cc to 2.74g/cc, with a mean of 2.68g/cc being very close to the expected bulk density of clean quartz (2.65 g/cc).

Pulp bulk densities measured by pycnometer average 2.74g/cc. Bulk density determinations by pycnometer are, on average, two per cent higher than bulk densities measured on site. Measurements by pycnometer fail to account for in-situ porosity and voids in sample material so the results are as expected. The bulk densities measured at site are considered reliable and adequate for the purpose of resource estimation.

13.4 Quality Control

Quality control issues are discussed in Section 14.

14 DATA VERIFICATION

14.1 The Resource Dataset

14.1.1 Trench Samples

Trench sample assays are available as point data located by nominal northing's, and with easting's and elevations estimated by interpolation between trench end locations measured by tape and compass. There is some doubt about the true locations of trench samples and they have been excluded from the sample database used for resource estimation.

14.1.2 Surface Rock Chip Traverses

Surface rock chip lines are available as point data located by survey. All available rock chip samples have been included in the sample database used for resource estimation. Most surface rock chip samples were collected over 2m intervals and are deemed compatible with the 2m drillhole composites without any further manipulation.

14.1.3 Drillhole Data

The resource estimate uses assay data for drillholes up to 31st May 2010, where available.

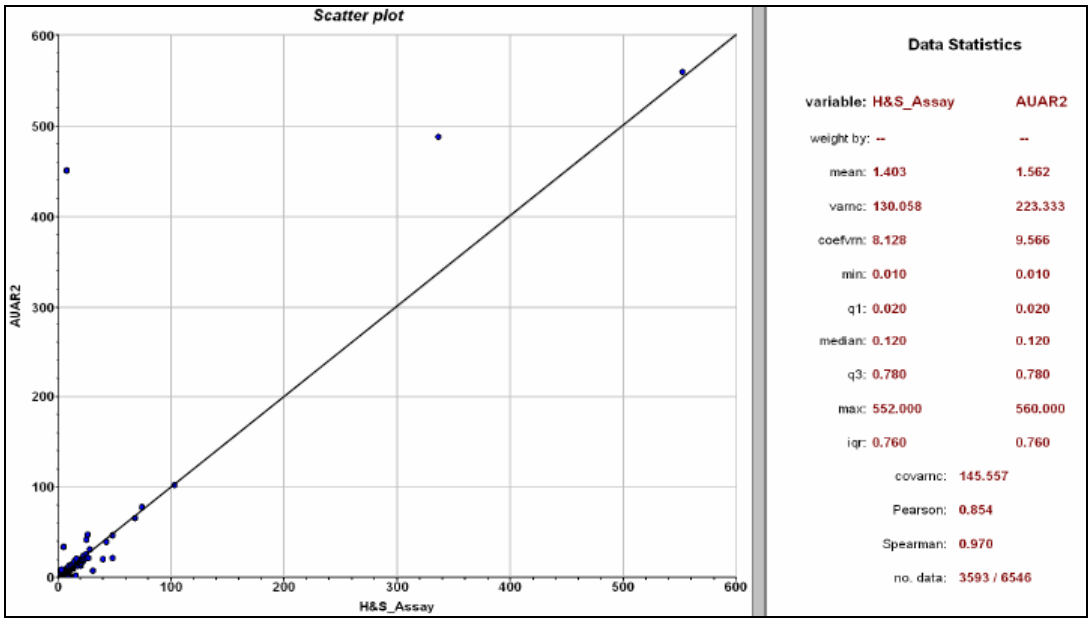
Assays were composited to 2m down-hole lengths with residual intervals less than 2m length being discarded. The resulting file contained 189,983 sample composites (including rock chip samples) with gold grades after trimming to the resource study area. Samples assaying less than detection limit were allocated 50% of the gold grade.

14.2 Sampling and Assaying pre August 2000

14.2.1 Routine Internal Laboratory Repeat Analyses

At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system ('LIMS'). Samples returning high assays are also selected for subsequent repeat analyses. The drillhole assay database contains duplicate analyses for 3,593 samples, approximately 1 in 7 samples. Figure 14.1 shows a scatter plot comparing original gold assays with repeat assays for the complete repeat assaying dataset. There is apparent bias towards the repeat assay being higher (by about 11%) than the original assay. This result is being produced by a few poorly repeating pairs.

Figure 14.1
Scatter plot original assays vs repeat assays, all data



Figures 14.2 and 14.3 show scatter and quantile-quantile ('QQ') plots for the repeat assay only for pairs with gold grades within the range of 0.1g/t Au to 20.0g/t Au. 1839 pairs remain after applying the grade range with the vast majority of pairs excluded falling below the lower threshold. Mean grades of the sub-setted pairs are very close; 1.778g/t Au versus 1.775g/t Au, original and repeat, respectively. Repeat assays correlate well with original assays and there is no clear bias to higher grades in either sample set, indicating that sample extraction and analytical errors are within acceptable limits.

Figure 14.2
Scatter plot original assays vs repeat assays, subset 0.1 to 20.0g/t Au

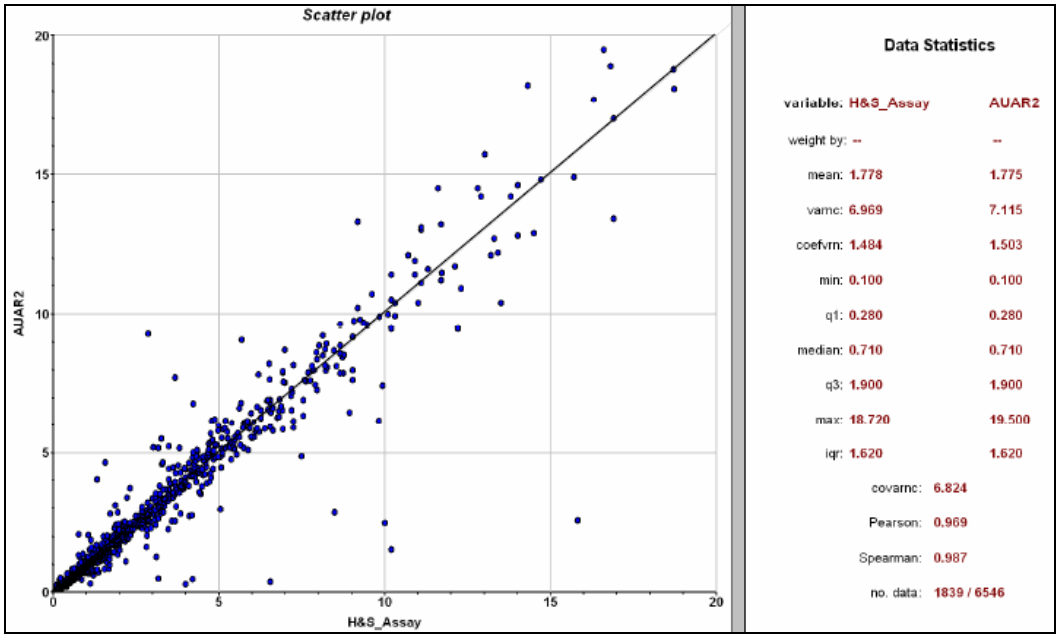
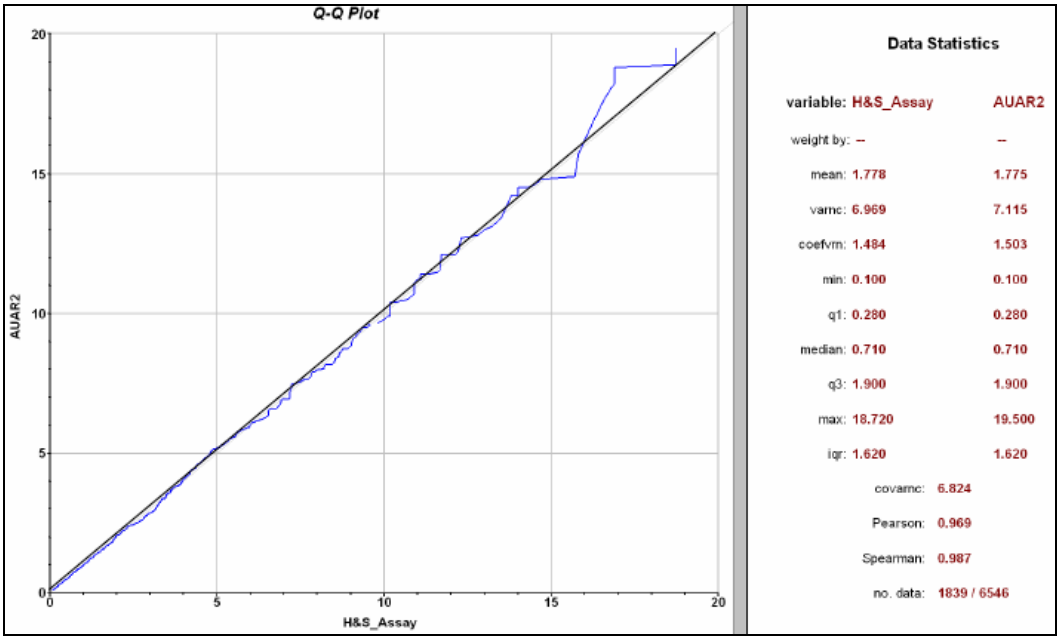
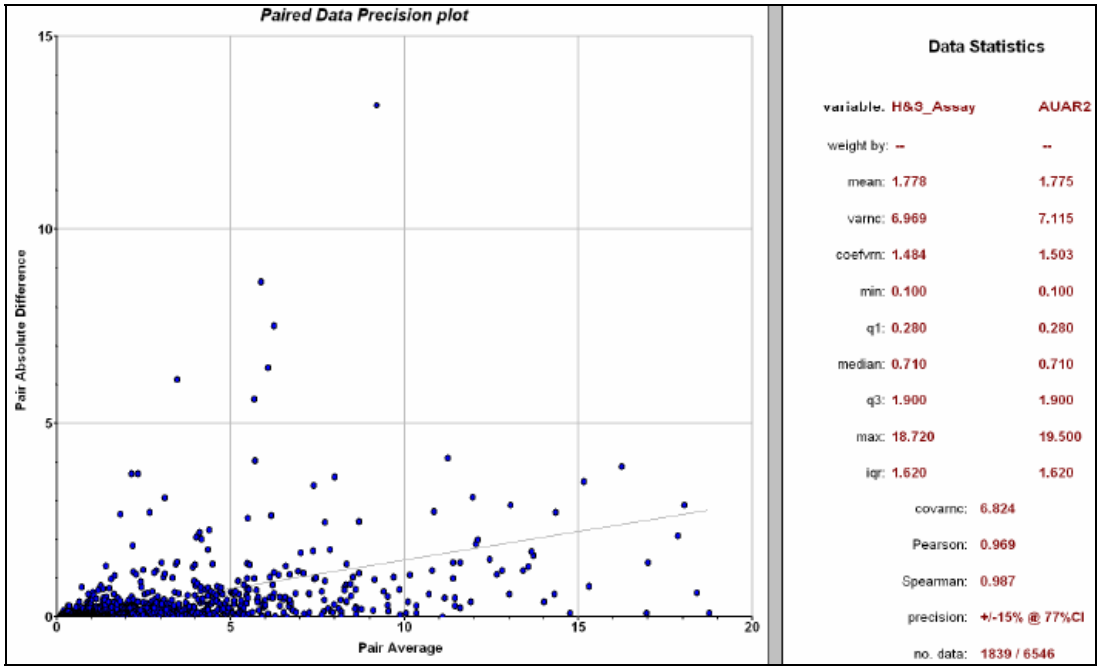


Figure 14.3
QQ Plot, original assay vs repeat assay subset 0.1 to 20.0g/t Au



The relative precision of pairs of repeat analyses describes the sum of errors arising in the sub-sampling (extraction) and analytical processes at the assay laboratory. Figure 14.4 present a precision plot and at ±15%, the precision of the sub-setted assay pairs is relatively high for this type of comparison but is acceptable considering the assay method being used.

Figure 14.4
Precision Plot original assay vs repeat assay, subset 0.1 to 20.0 g/t Au



14.2.2 Internal Routine Fire Assay Checks

Of samples that return assays more than 1g/t Au, a selection are subsequently analysed by 30g fire assay with AAS finish. The drillhole assay database contains 2,718 check fire assays, amounting to 2 in every 3 samples assaying over 1g/t Au.

Figures 14.5 and 14.6 show scatter and quantile-quantile plots comparing original gold assays with fire assay repeats. For the present analysis the two highest grade sample pairs have been excluded. These pairs have grades in excess 2,000g/t Au and removed as they have a large influence on the means calculated for the two grade distribution. Fire assays correlate well with original assays and there is no clear bias to higher grades in either sample set, indicating that acid digestion of samples is liberating gold into solution. Note that for the quantile-quantile plot the axis to the graph have been reset to be a maximum of 100g/t Au. This is purely for presentation purposes and no sub-setting of the data has occurred.

Figure 14.5
Scatter Plot original vs FA repeats

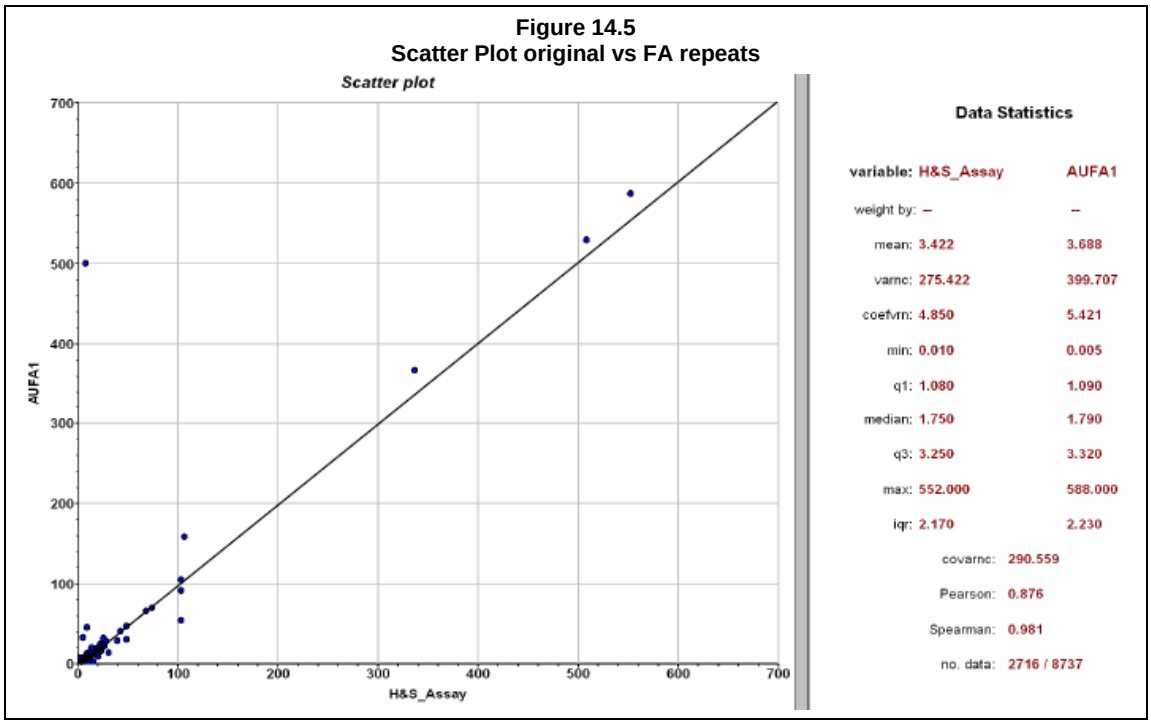
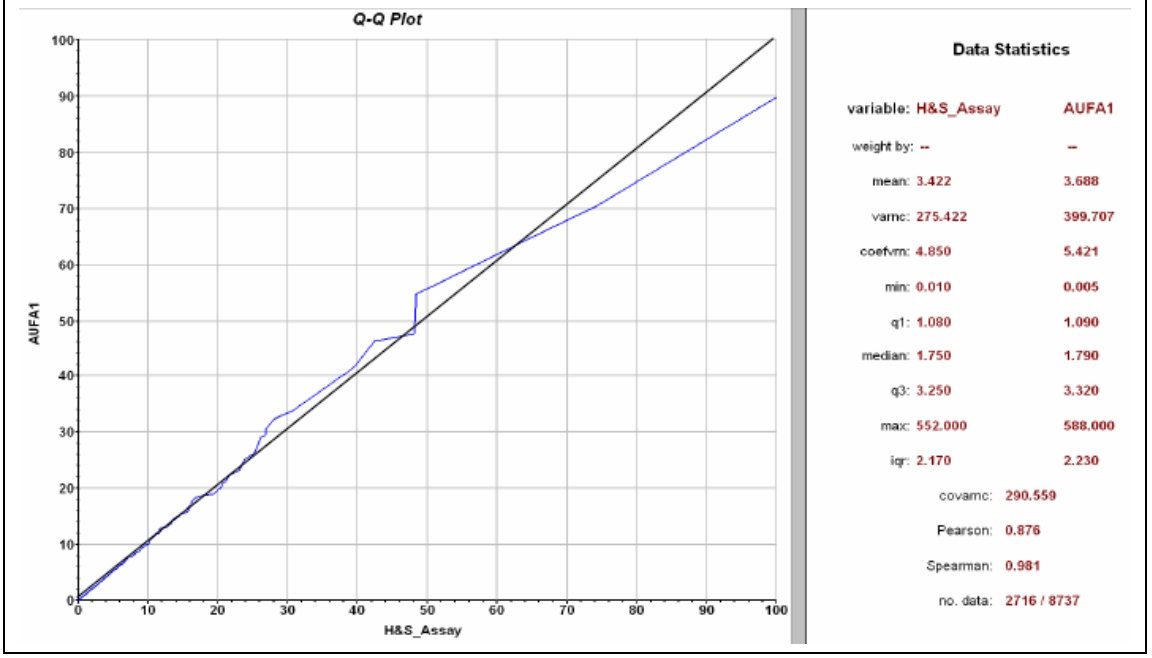


Figure 14.6
QQ Plot, original assays vs FA repeats



14.2.3 Reference Samples

Certified reference standard materials have not been used to monitor assay accuracy. Two materials have been used as reference samples:

1. Tailings slimes from the old Sukari mine dumps. Samples of this material were submitted intermittently with batches of drill core samples despatched between December 1997 and November 1999. Samples have assayed between 2.68g/t Au and 3.82g/t Au, with a mean assay of 3.30g/t Au.

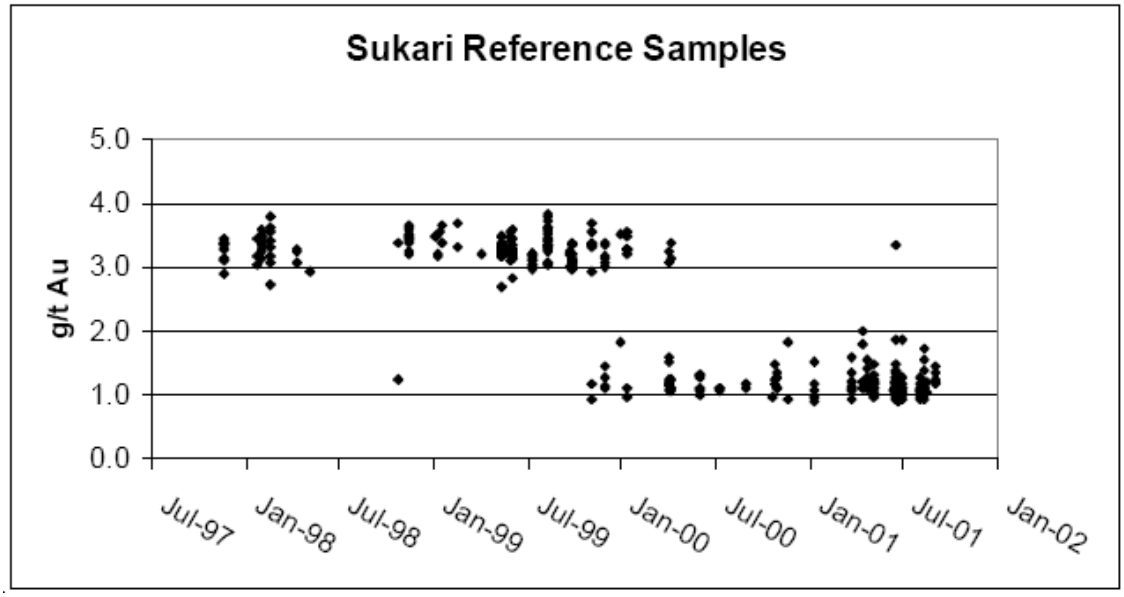
2. Tailings sands from the old Sukari mine dumps. Samples of this material were used as reference standards after April 1999. Samples have assayed between 0.88g/t Au and 2.01g/t Au, with a mean assay of 1.19g/t Au.

Both materials were used in batches of samples despatched between April 1999 and November 1999.

Drill core samples are normally despatched to the assay laboratory in batches of 200-250 samples. Reference samples were submitted with only about half the batches assayed prior to mid-1999, and those batches containing reference samples normally contained only one or two. After mid-1999, reference samples have been included in all sample batches at a ratio of between 1 in 40 and 1 in 100 samples, resulting in most sample batches containing at least four reference samples.

Assays of reference samples through time are shown in Figure 14.7. Because the reference materials are not homogenised, the assays show considerable variations in gold grade. There is, however, no trend to consistently higher or lower grades throughout time and there are no sample batches that have suffered gross errors (e.g. order of magnitude errors) in the assaying process.

Figure 14.7
Sukari Reference Samples (submitted with drillcore)



14.2.4 Independent Checks

In June 2000, an H&S consulting geologist visited Sukari site and collected 100 samples for independent verification of gold grades. Samples of quarter core were selected from eleven drillholes, the locations of which spanned the extent of drill coverage at that time. Core was selected, sawn and bagged in tamper-evident bags under constant supervision by the H&S representative. Each sample weighed about 0.5kg. Approximately 8kg of quartz used for wash material at the Sukari sample preparation facility was shipped with the check samples.

The quarter core samples were sent to Minesite Reference Laboratories in Perth, Western Australia. Tamper-evident bags were opened in the presence of the H&S geologist and the samples were crushed and pulverised. The quartz wash material from site was also crushed and pulverised. After pulverising, sample pulps were shuffled and re-numbered, and standards and blank samples were inserted in the sample sequence at a ratio of 1:10. Standards were sourced from Gannet Holdings, Perth, with two materials being used: ST06/8222 with a reference grade of 1.06g/t Au, and ST37/8239 with a reference grade of 1.69g/t Au.

The re-numbered sample pulps were then each split into three 150g portions using a rotary divider. One pulp portion was submitted to Minesite Reference Laboratories for gold assays by the normal 25g aqua regia digest ('AR') method. The second pulp portion was submitted to Standard & Reference Laboratories, Perth, for fire assay using a gravimetric finish. Standard & Reference Laboratories specialise in high precision, high accuracy assaying and are associated with Alex Stewart Assayers of the U.K.

Figures 14.8 and 14.9 show scatter and quantile-quantile plots comparing original AR gold assays by Minesite Reference Laboratories with check assays

also conducted by Minesite Reference Laboratories using AR. Plots and summary statistics have been limited to pairs with both assays reporting less than 20g/t Au. The width of the scatter in Figure 14.8 reflects variations between gold contents of the pairs of core portions, expected in a gold deposit. Figure 14.10 indicates that the check AR assays report 6% to 8% lower grades than the original assays, particularly for samples assaying more than about 1.2g/t Au.

Figure 14.8
Scatter Plot, Original Assays vs Aqua Regia Check assays (MRL)

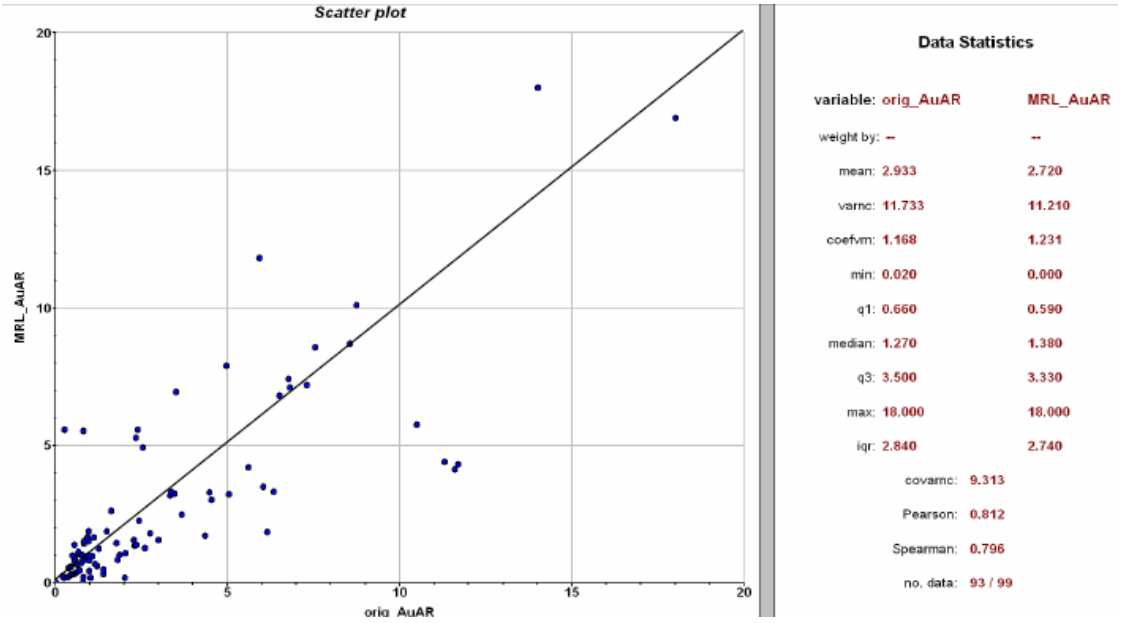


Figure 14.9
QQ Plot Original Assays vs Aqua Regia Check Assays (MRL)

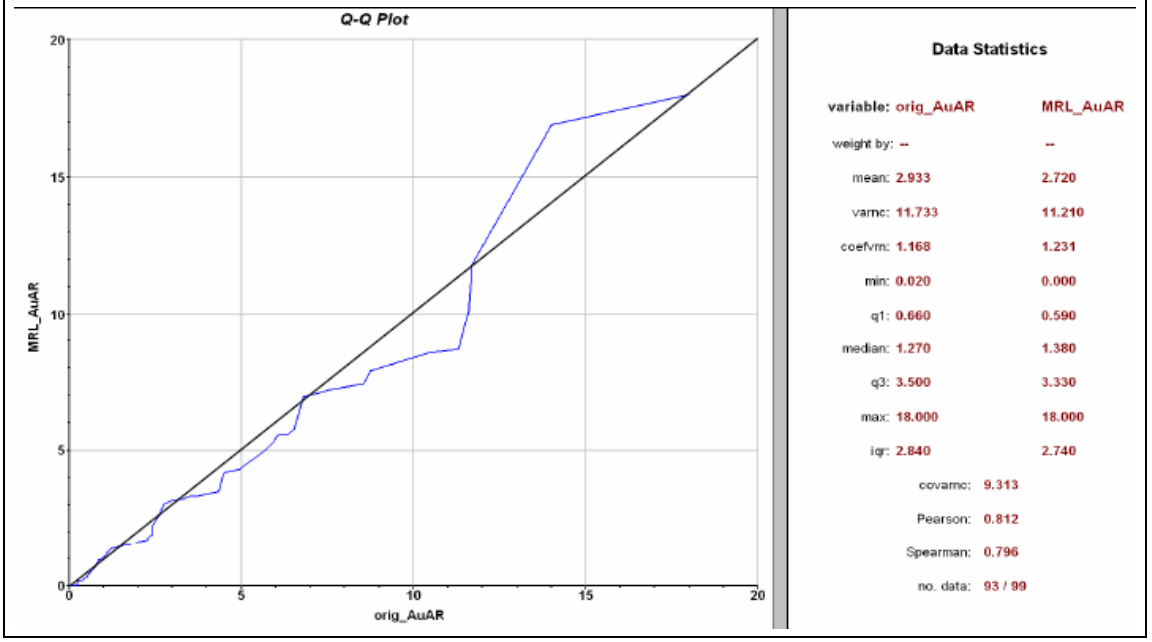
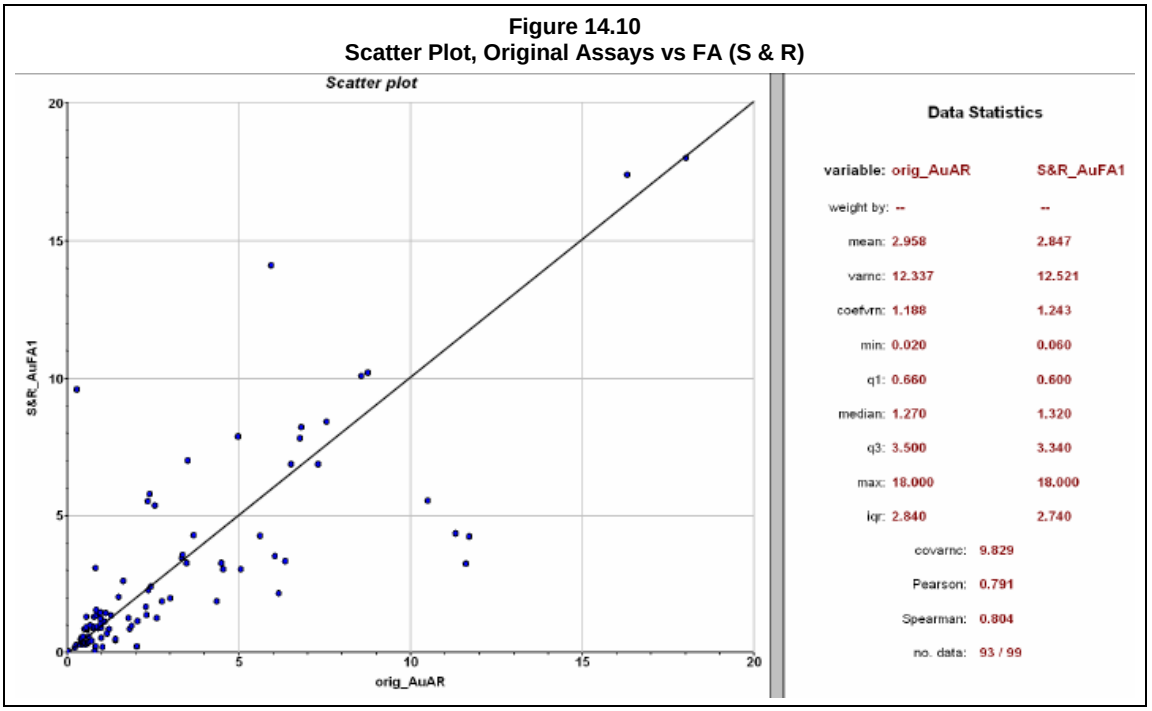
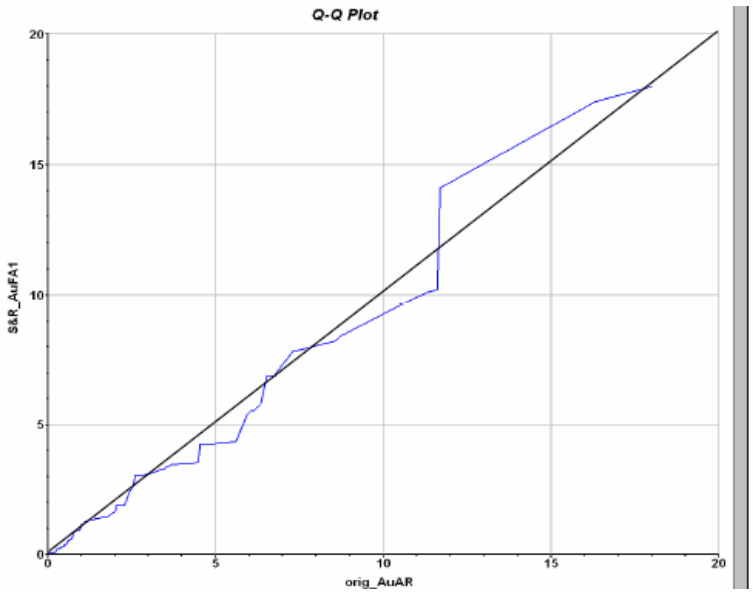


Figure 14.10
Scatter Plot, Original Assays vs FA (S & R)



Figures 14.10 and 14.11 show scatter and quantile-quantile plots comparing original AR gold assays by Minesite Reference Laboratories with check assays conducted by Standard & Reference Laboratories using 50g fire assay with gravimetric finish. The slight bias to higher grades in the original assays is almost identical to that seen in the comparison of aqua regia digest check assays.

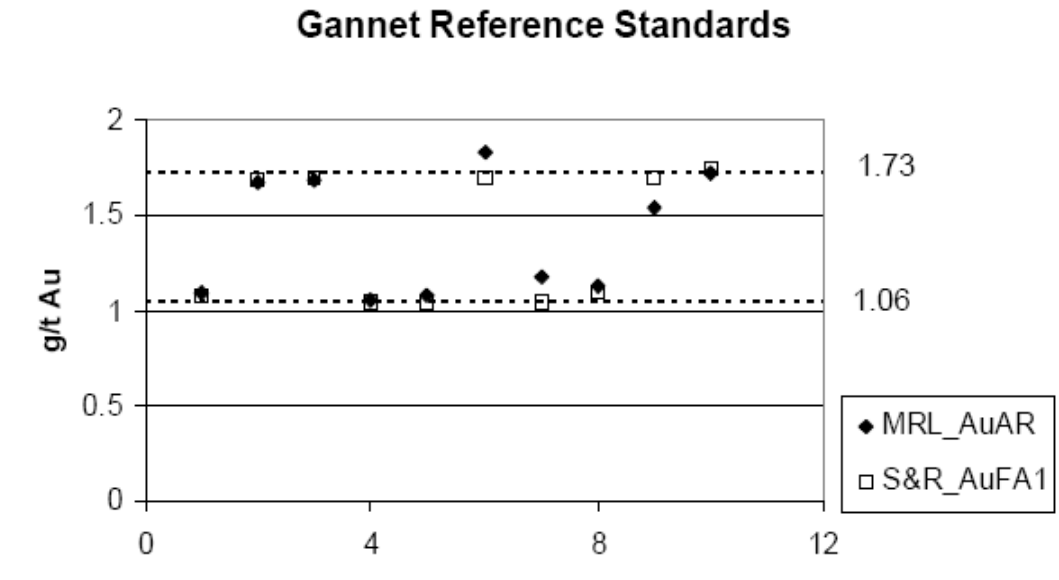
Figure 14.11
QQ Plot, Original Assays vs FA (S & R)



That original assays report slightly higher gold grades in both the above comparisons indicates that a bias is possibly arising in on-site sample selection or preparation procedures. Without additional check work the source of bias cannot be pinpointed.

Figure 14.12 shows assays for reference standards that were interleaved with re-numbered check samples. The accuracy of assays from both laboratories is within acceptable limits. The enhanced accuracy and precision of the fire assay method is evident.

Figure 14.12
Gannet Standards (submitted with check samples)



14.2.5 Further Independent Check on Assay Accuracy

Because no certified reference materials have been used to monitor the accuracy of Sukari assays, it was thought prudent to conduct an independent check of assay accuracy. In October 2001, H&S selected 223 remnant sample pulps and submitted them for gravimetric fire assay by Standard & Reference Laboratories. Five sample pulps were selected from every second batch of samples submitted by PGM, dating back to October 1997. Samples reporting significant gold grades (generally >0.3g/t Au) were selected. Reference standards were inserted in the sample sequence at a ratio of 1:10. Standards were sourced from Gannet Holdings, Perth, with two materials being used: ST73/7192 with a reference grade of 1.51g/t Au, and ST43/7194 with a reference grade of 3.64g/t Au.

Figures 14.13 and 14.14 show scatter and quantile-quantile plots comparing original aqua regia digest gold assays by Minesite Reference Laboratories with check assays conducted by Standard & Reference Laboratories. Although a few individual pairs show poor agreement, the check assays correlate very well with original assays by Minesite Reference Laboratories.

Figure 14.15 shows assays for reference standards that were interleaved with sample pulps. Assays of reference standards confirm the accuracy of S&R gravimetric fire assays.

Figure 14.13
Scatter Plot, Original Assays vs Pulp Re-assays (S & R)

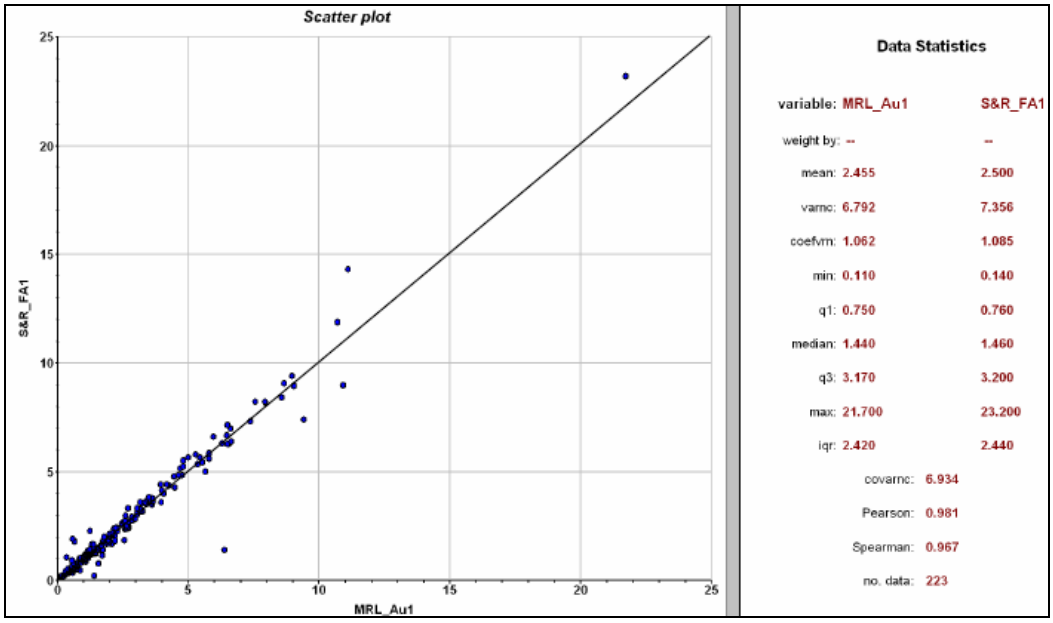


Figure 14.14
QQ Plot, Original Assays vs Pulp Re-assays (S & R)

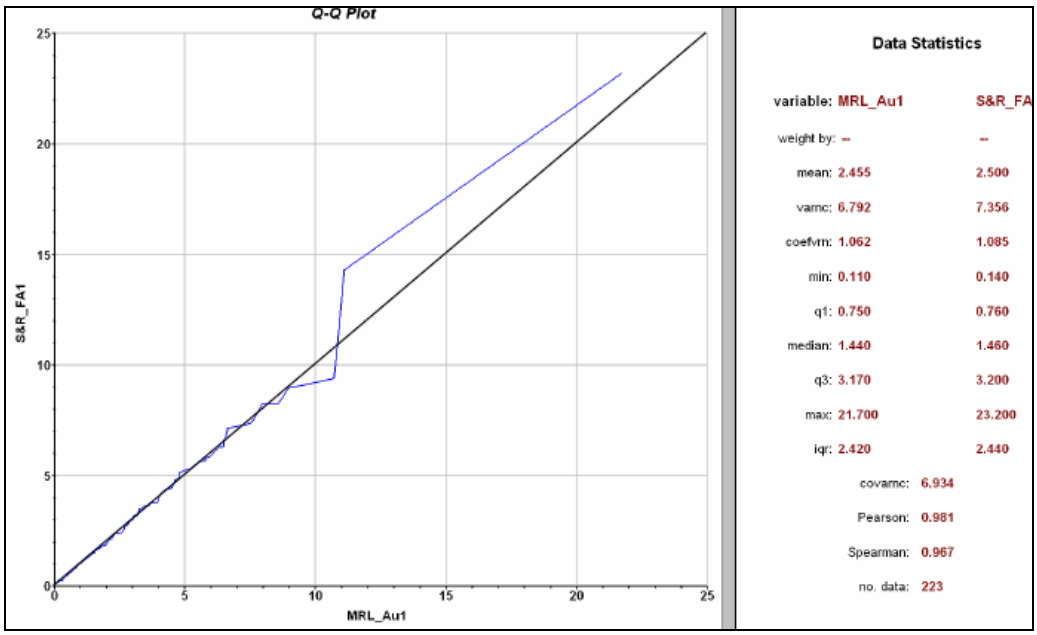
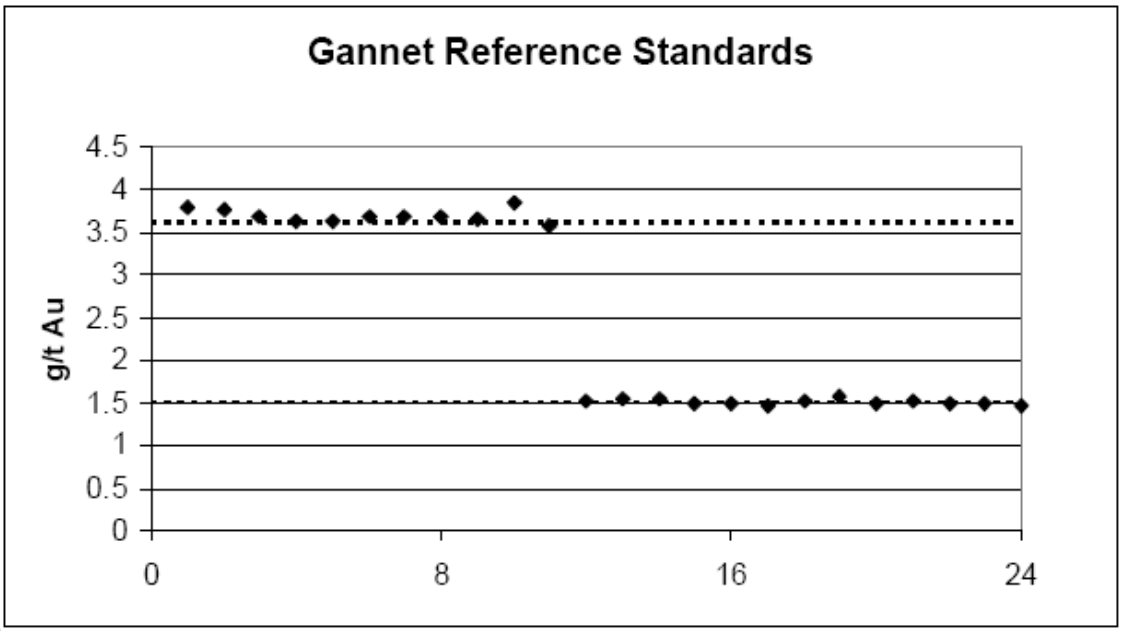


Figure 14.15
Gannet Standards (submitted with pulp reassays)



14.3 Sampling and Assaying, between August 2000 and 2003

Recommendations on QAQC procedures were made by H&S in 2000 report and reviewed during a site visit conducted by H&S in February 2003. The laboratory contracted to undertake all assay during this period was Minesite Reference Laboratories, Perth. The QAQC procedures introduced after 2000, and continued to the present, are summarised below:-

- Insertion of blank samples is used to check for cross sample contamination at the sample preparation stages.
- Standards are employed to check precision and accuracy of the assaying.
- Routine RC sample duplicates provide a measure of precision of all stages from the collection of the sample in the field to the final assay.
- Repeat analysis by the same assay method of the original ('AR') and are checks on errors that arise from sub-sampling and processes thereafter and can provide useful checks on the assaying precision. Samples for repeat analyses are selected at random by the laboratory information management system ('LIMS').
- Fire assay checks are a monitor that acid digest (AR technique) is effectively liberating gold into solution. Fire Assay checks of samples are completed when they return more than more 1g/t.

Blank, standards and RC/DD duplicates are inserted into the sample sequence at Sukari with the duplicate and fire assay checks conducted by the assay laboratory. Check samples are inserted in the sample runs at a frequency of approximately 1:50. The intention is to place the samples into logged zones of mineralization to maximize the effectiveness of the checks samples. They are sequenced in the following pattern, for:-

- Diamond Drilling; SAMPLE – BLANK – STANDARD – SAMPLE
- RC drilling; Field duplicates are inserted at a ratio of 1:25 samples with blanks and standards inserted at a ratio of 1:50.

14.3.1 Internal Routine Laboratory Repeat Analyses

At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system ('LIMS'). Samples returning high assays are also selected for subsequent repeat analyses. The drillhole assay database contains duplicate analyses for 2,324 samples. Figures 14.16 and 14.17 show scatter and quantile-quantile plots for the repeat assay pairs for all available data. Mean grades of the pairs are very close; 0.985g/t Au versus 0.953g/t Au, original and repeat, respectively. Repeat assays correlate well with original assays and there is no clear bias to higher grades in either sample set, indicating that sample extraction and analytical errors are within acceptable limits.

The relative precision of pairs of repeat analyses describes the sum of errors arising in the sub-sampling (extraction) and analytical processes at the assay laboratory. Figure 14.18 presents a precision plot and at ±13%, the precision of the sub-set of assay pairs is good for this type of comparison.

Figure 14.16
Scatter Plot, Original Assays vs Repeat Assays, all Data

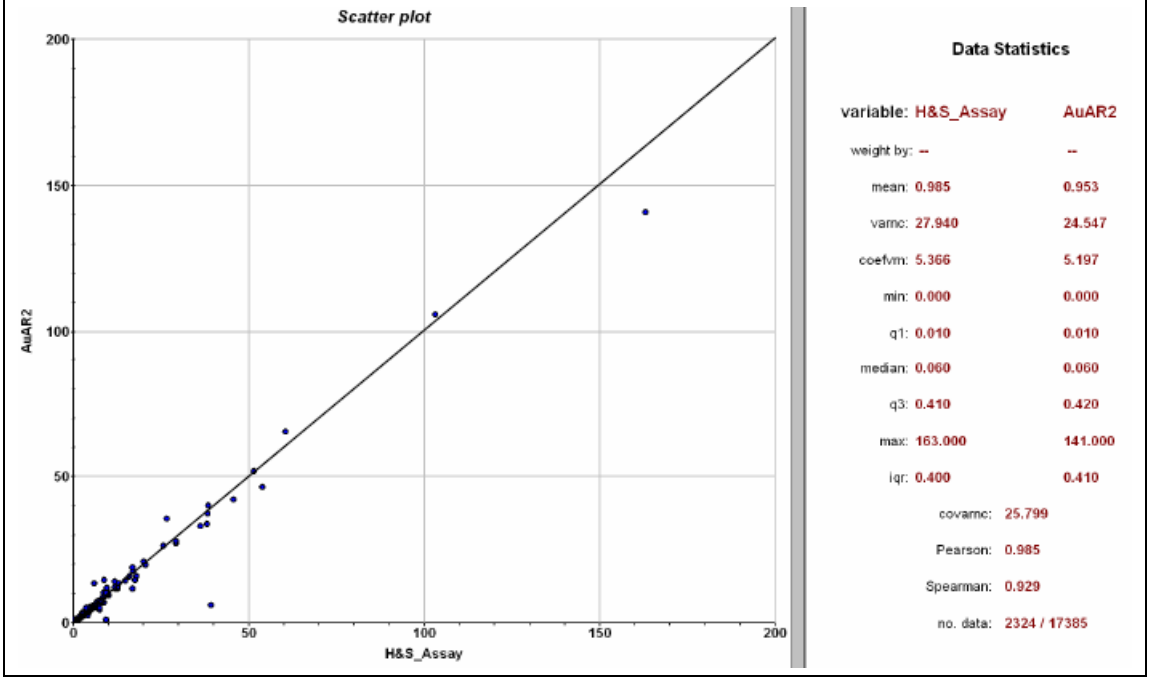


Figure 14.17
QQ Plot, Original Assays vs Repeat Assays, all Data

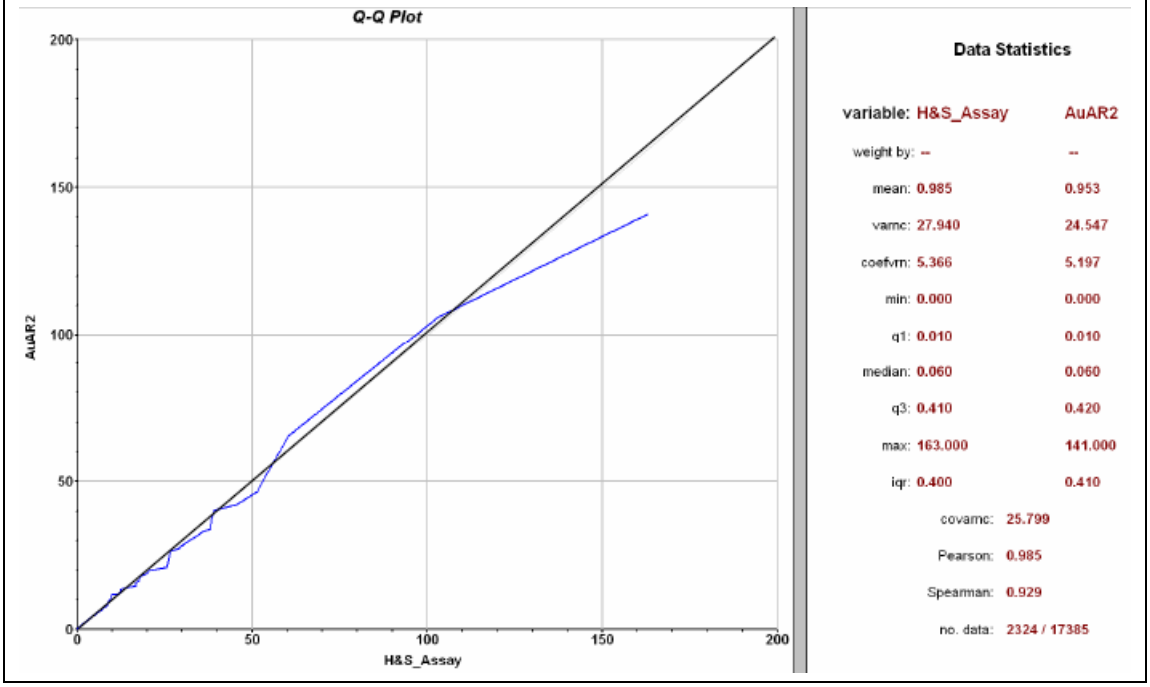
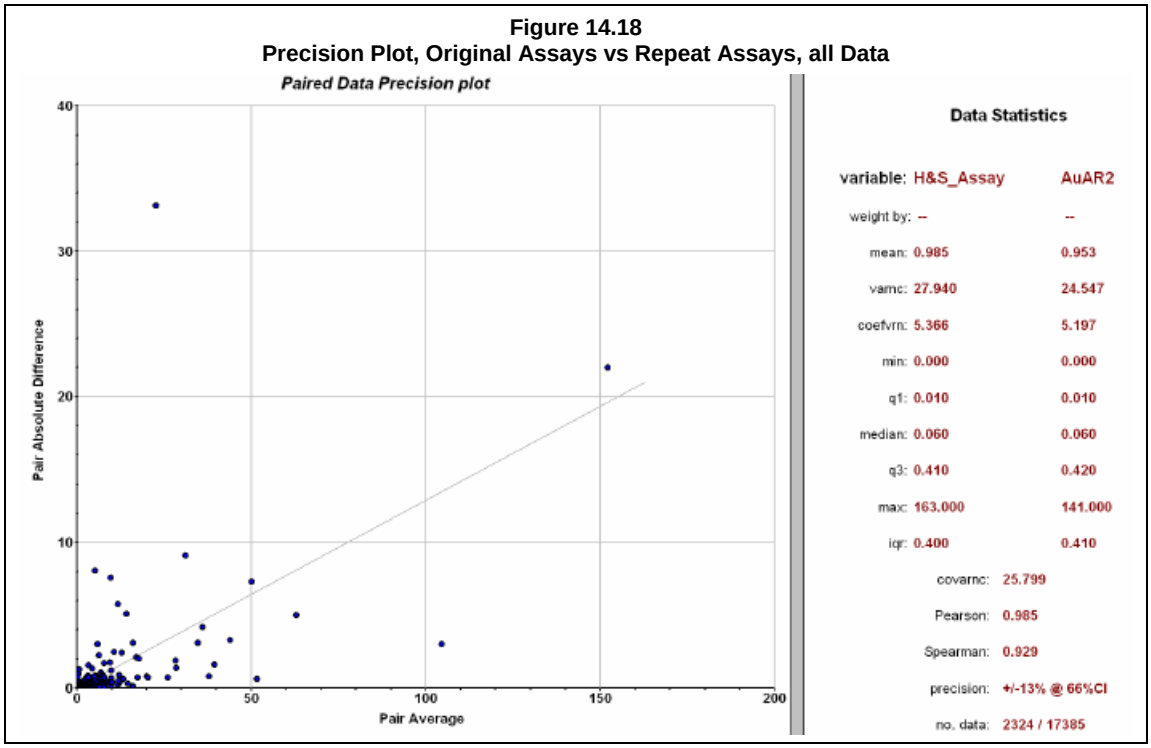


Figure 14.18
Precision Plot, Original Assays vs Repeat Assays, all Data



14.3.2 Internal Routine Fire Assay Checks

Of samples that return assays more than 1g/t Au, a selection are subsequently analysed by 30g fire assay with AAS finish. The drillhole assay database contains 2,986 check fire assays, amounting to approximately 2 in every 3 samples assaying over 1g/t Au.

Figures 14.19 and 14.20 show scatter and quantile-quantile plots comparing original gold assays with fire assay repeats. Fire assays correlate well with original assays and there is no clear bias to higher grades in either sample set, indicating that acid digestion of samples is liberating gold into solution. Although the mean grade of the FA repeats are slightly higher than the original AR assays, 3.391g/t Au versus 3.396g/t Au, respectively, the quantile-quantile plots shows that this difference is only due to a few poor repeating pairs.

Figure 14.19
Scatter Plot, Original vs FA Repeats

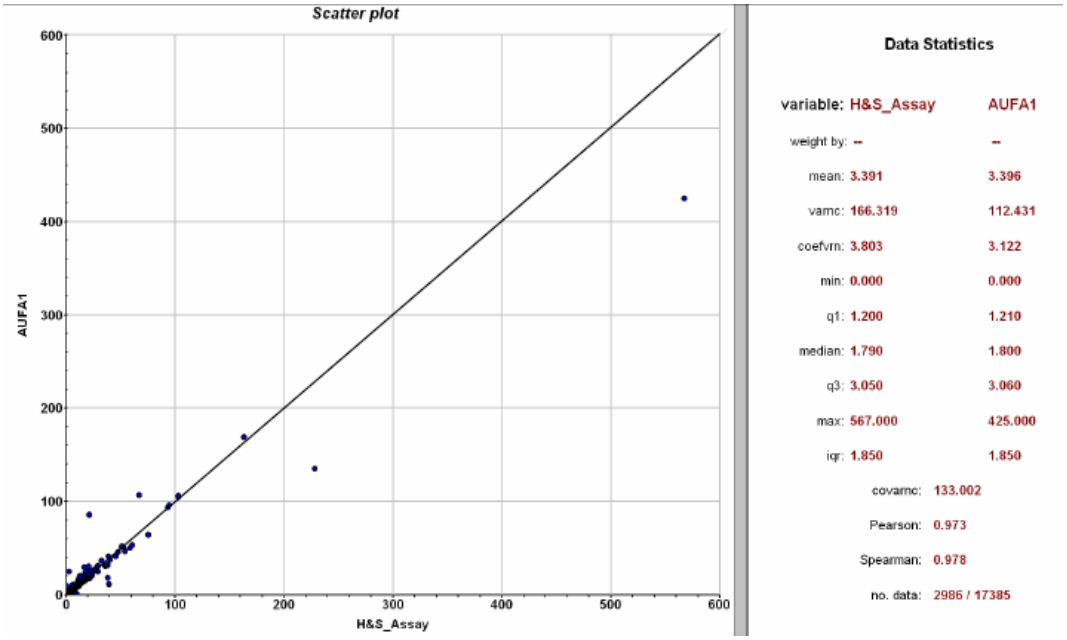
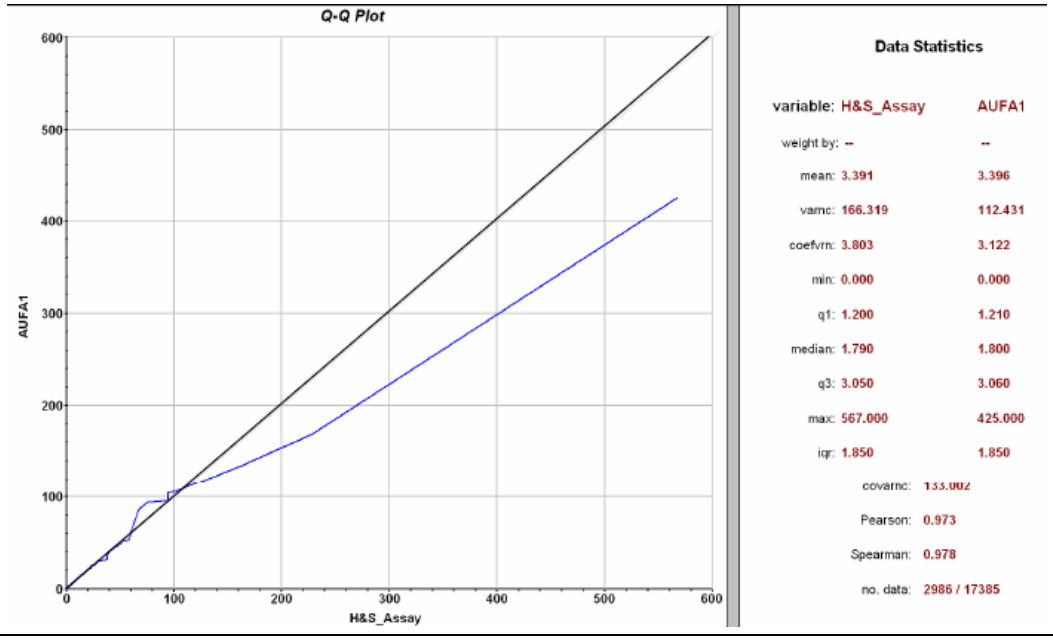


Figure 14.20
QQ Plot Original vs FA Repeats



14.3.3 Certified Reference Samples

A series of certified reference samples were routinely inserted within the assaying batches at Sukari during the pre 2003 drilling and have been sourced from Gannet Laboratories, Perth. The standards used are listed below together with there recommended value (by AR assaying);

- ST17, with a recommended gold assay of 0.78g/t Au
- ST73, with a recommended gold assay of 1.44g/t Au
- ST09, with a recommended gold assay 1.98g/t Au
- ST274, with a recommended gold assay of 6.08g/t Au

Figure 14.21 through 14.24 shows the control charts (with ±10% control lines plotted either side of the recommended grade) for each of the standards used pre 2003. General observations that can be made from viewing these control charts are;

1. For ST17 the average grade returned for the 42 assays available is 0.78g/t Au and is a very good result.
2. For ST73 the average grade returned for the 243 assays available is 1.40g/t Au, this being slightly lower than the expected value but well within acceptable limits. The notable observation for the assaying of this standard is during the middle period (between sequence numbers 100 to 170 or approximately March 2002) where a high number of assays fall outside the ±10 control lines. There is an equal number above the +10% line as there is below the -10% line so not indicating a bias, but this period was certainly a time of poor precision.
3. For ST09 the average grade returned for the 50 assays available is 1.98g/t Au and is a good result.
4. For ST274 the average grade returned for the 278 assays available is 6.10g/t Au, overall showing a good correlation with the expected value. There is an indication from the control chart that the grades returned for this standard have been steadily increasing over time with a notable step up in grade at about sequence number 120 (approximately December 2002). Calculating the average grade before and after this point shows yields average grades of 6.02g/t Au and 6.17g/t Au, respectively. Both results are still acceptable, however, the reason for the change should be investigated, if presently unknown (see discussion in next section).

Figure 14.21
Gannet Standard Control Chart ST17

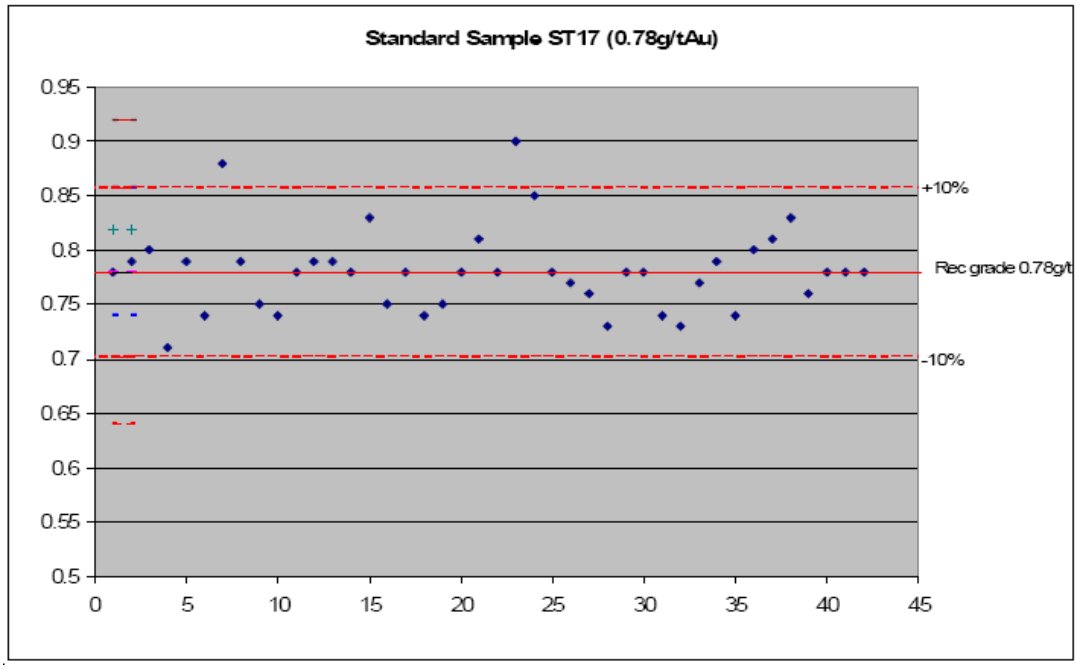


Figure 14.22
Gannet Standard Control Chart ST73

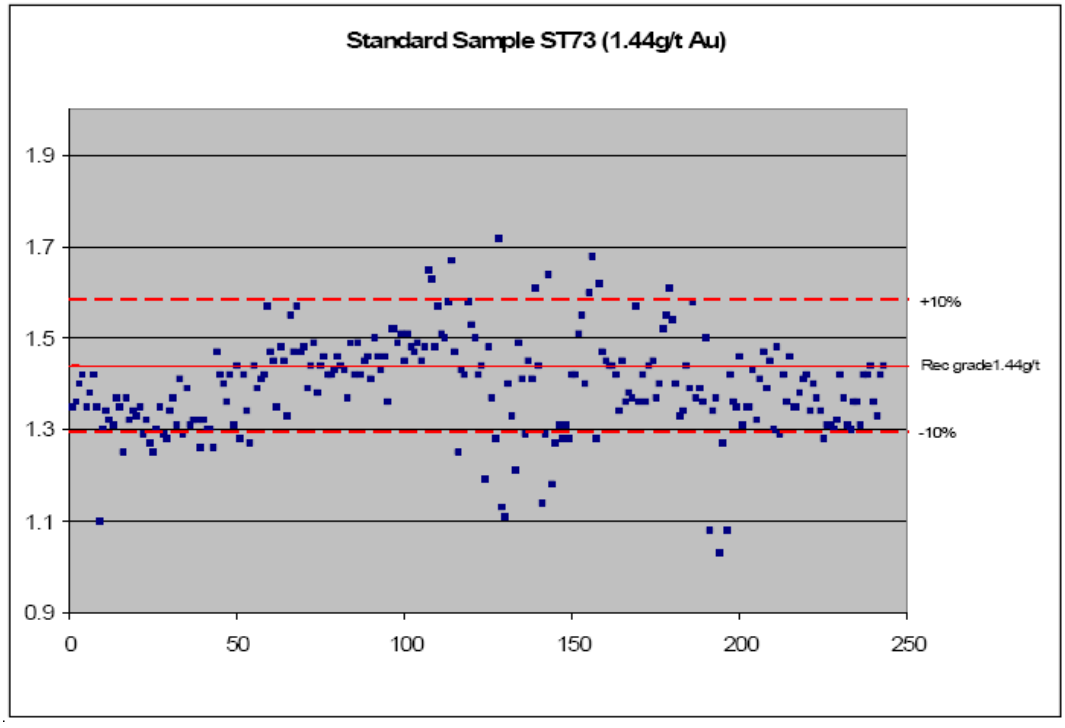


Figure 14.23
Gannet Standard Control Chart ST09

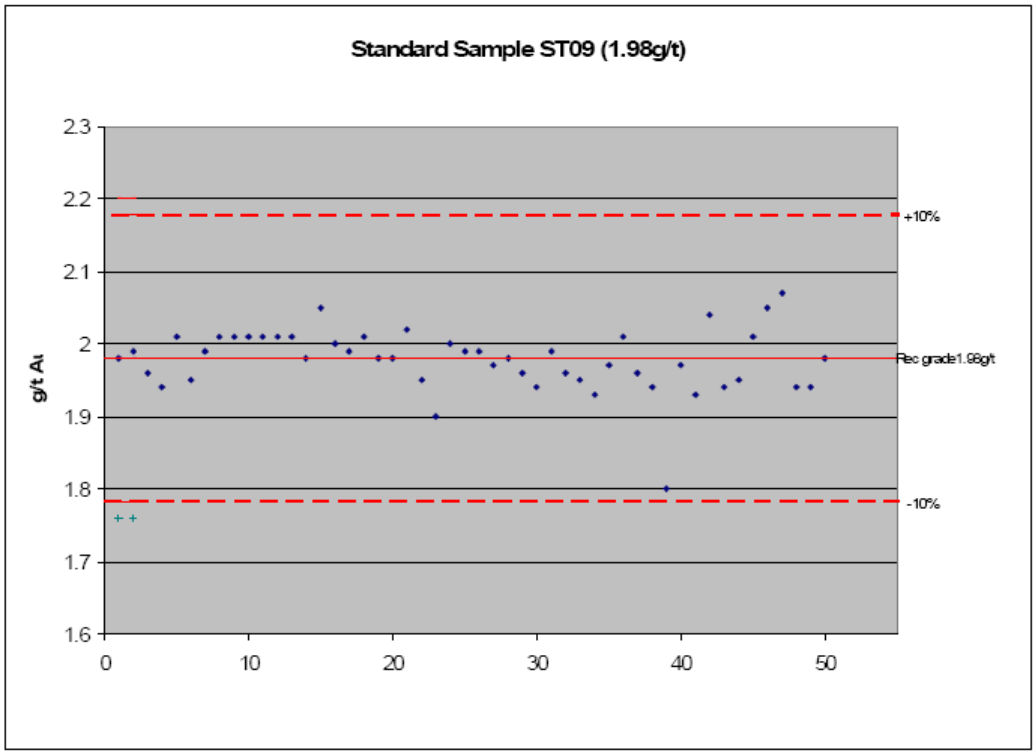
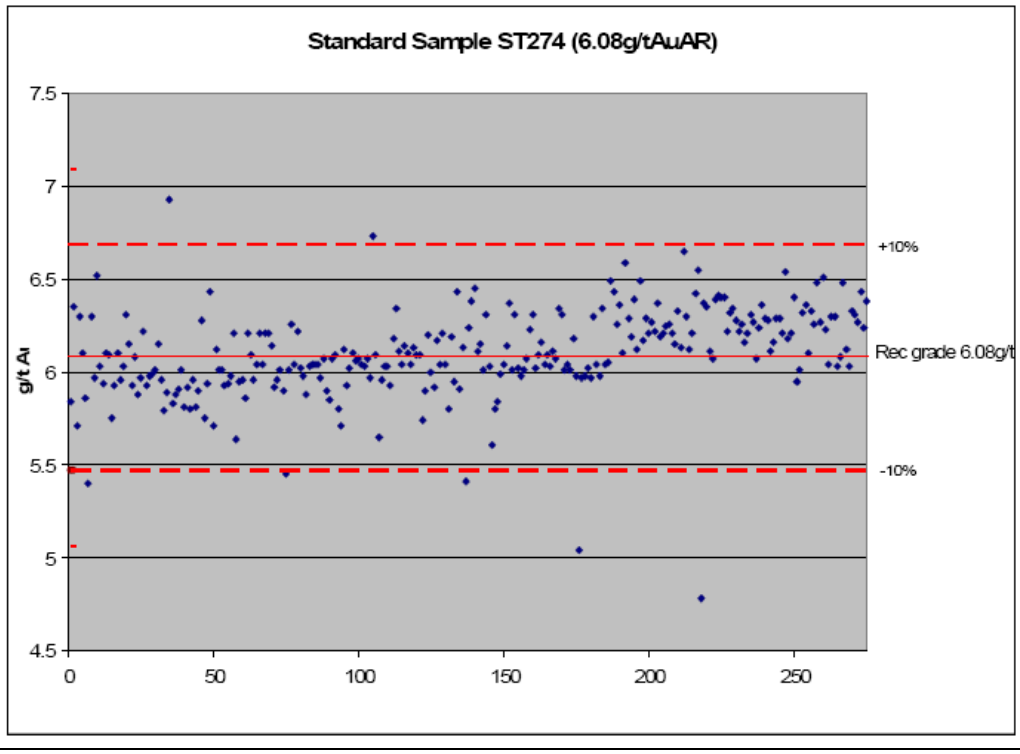


Figure 14.24
Gannet Standard Control Chart ST274



14.4 Sampling and Assaying, 2005 to 25th January 2009

The QAQC programmes to verify the sampling and assaying conducted during 2005 to 25th January 2009 are exhaustive and include:-

- Routine laboratory repeat assaying, routine Fire Assay ('FA') checks of Aqua Regia ('AR') original assaying, consistent with the previous drilling periods except that the assay laboratory had been changed to Ultratrace Analytical Laboratories, Perth, with some changes to the AR procedure and charge sizes (discussed earlier in Section 13.2).
- Insertion of blanks into the sample preparation and assaying batches.
- Insertion of certified standards into the assaying batches.
- Particle size analysis of the pulverised sample (presented in Section 13.1.2).
- Duplicate sampling of both diamond core and RC samples (field duplicates).
- Sample preparation duplicates i.e. a number (about 2099 samples) of second split obtained at the sample preparation facility after initial crushing but before pulverisation. These samples are dispatched to Ultratrace for pulverisation and assaying.
- Sample preparation pulp duplicates, commenced in August 2006. These samples are a second split from the pulverised material obtained at the sample preparation facility. These samples are dispatched to Ultratrace for assaying.

The insertion of QAQC samples into the assaying batches was augmented in August 2006 with the inclusion of duplicate pulp assaying. The current sequencing of the QAQC samples proceeds at frequency of 1:25 for RC samples and for the diamond samples the QAQC samples are inserted at a ratio of at least 1:50 however, this is increased in the mineralised zones. The sequencing of the QAQC samples is shown below for both the diamond and RC sampling;

- Diamond Drilling; SAMPLE – BLANK – STANDARD – PULP SAMPLE – DUPLICATE (1/4 Core) - SAMPLE
- RC drilling; SAMPLE – BLANK – STANDARD – PULP SAMPLE – DUPLICATE (Field re-split) - SAMPLE.

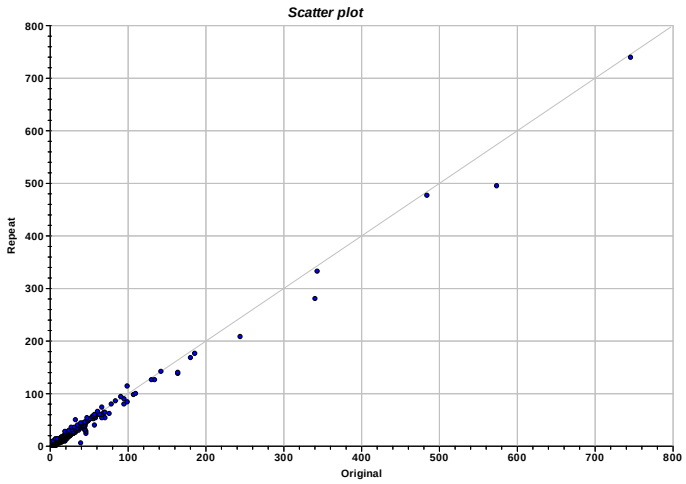
The following section summaries the results of the above QAQC programmes.

14.4.1 Routine Laboratory Repeat Analyses

At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system (LIMS). Samples returning high assays are also selected for subsequent repeat analyses. The drillhole assay database contains duplicate analyses for 36,442 samples. Figures 14.25 and 14.26 show

scatter and quantile-quantile plots comparing original gold assays with assay repeats. The means of the two assay datasets are equivalent, suggesting that sample extraction and analytical errors are very low. This is supported by the precision plot (Figure 14.27) that shows the assaying precision to be ±7%.

Figure 14.25
Scatter Plot, Original Assays vs Repeat Assays



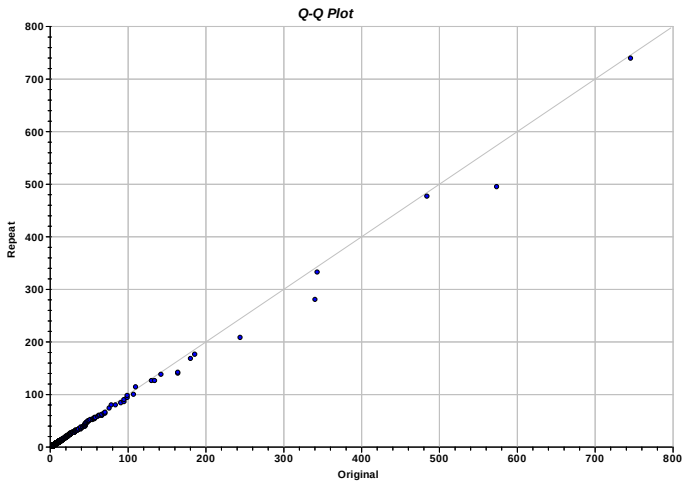
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	0.788	0.766
varnc:	52.018	46.777
coefvn:	9.156	8.925
min:	0.001	0.001
q1:	0.004	0.004
median:	0.058	0.057
q3:	0.441	0.434
max:	745.000	740.000
iqr:	0.437	0.430

covarnc: 49.128
Pearson: 0.996
Spearman: 0.991
no. data: 36441 / 36442

(data is sub-setted)

Figure 14.26
QQ Plot, Original Assays vs Repeat Assays



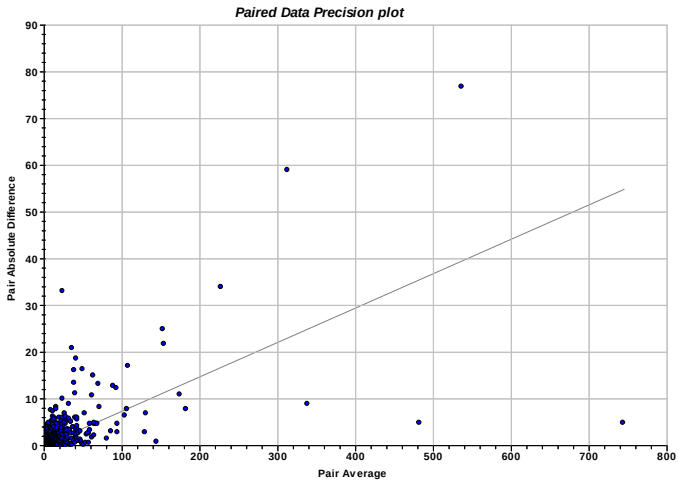
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	0.788	0.766
varnc:	52.018	46.777
coefvn:	9.156	8.925
min:	0.001	0.001
q1:	0.004	0.004
median:	0.058	0.057
q3:	0.441	0.434
max:	745.000	740.000
iqr:	0.437	0.430

covarnc: 49.128
Pearson: 0.996
Spearman: 0.991
no. data: 36441 / 36442

(data is sub-setted)

Figure 14.27
Precision Plot, Original Assays vs Repeat Assays



Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	0.788	0.766
varnc:	52.018	46.777
coefvm:	9.156	8.925
min:	0.001	0.001
q1:	0.004	0.004
median:	0.058	0.057
q3:	0.441	0.434
max:	745.000	740.000
iqr:	0.437	0.430

covarnc:	49.128
Pearson:	0.996
Spearman:	0.991
precision:	+/-7% @ 40%CI
no. data:	36441 / 36442

(data is sub-setted)

14.4.2 Routine Fire Assay Checks

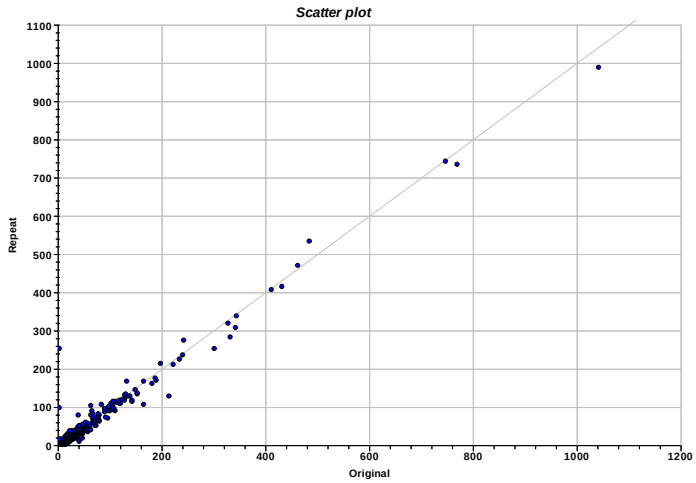
Of samples that return assays more than 1g/t Au, a selection are subsequently analysed by 30g fire assay with AAS finish. There are 23,925 check fire assays recorded for the period under investigation, amounting to approximately 2 in every 3 samples assaying over 1g/t Au.

Figures 14.28 and 14.29 show scatter and quantile-quantile plots comparing original gold assays with fire assay repeats. Fire assays correlate well, however, there is a slight tendency for original assays to be higher at higher grades resulting in a global bias in mean grades towards the AR assaying of less than 1%. This is an unexpected result; normally it would be expect the AR assaying to give, on average, lower grades when compared to FA.

14.4.3 Blanks

Assays of blank material inserted within sample batches undertaken during the review period are shown in Figure 14.30. The blank material is inserted after a drill sample and before a standard to check for cross sample contamination at the pulverisation stage of sample preparation. The blank material is sourced from a known barren outcrop (gabbro composition) within the Sukari project area. On the figure the group of blanks with variable gold grades early in the period (<50 on the X axis) are noted to be assays of the quartz wash material previously used but now have been discontinued. No significant issue are identified from observations on the plot of blanks over time after the introduction of the barren gabbro material.

Figure 14.28
Scatter Plot, Original Assays vs FA Repeats, all Data



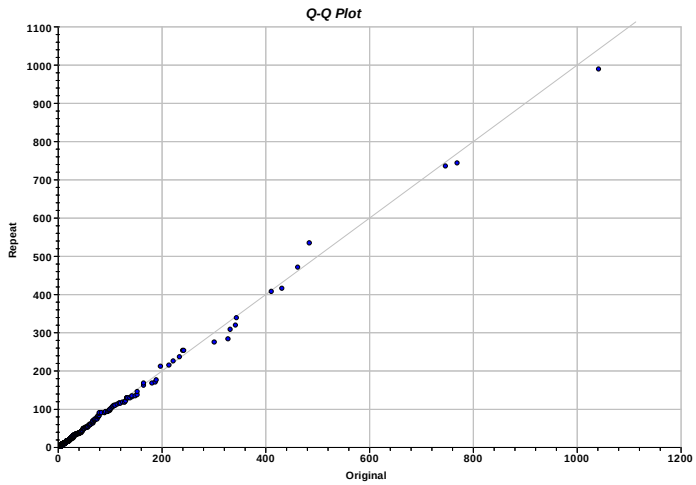
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	2.792	2.771
varnc:	203.289	196.976
coefvm:	5.107	5.065
min:	0.001	0.001
q1:	0.483	0.469
median:	1.420	1.410
q3:	2.470	2.450
max:	1040.000	990.000
iqr:	1.987	1.981

covarnc: 197.422
Pearson: 0.987
Spearman: 0.979
no. data: 23924 / 23925

(data is sub-setted)

Figure 14.29
QQ Plot, Original Assays vs FA Repeats



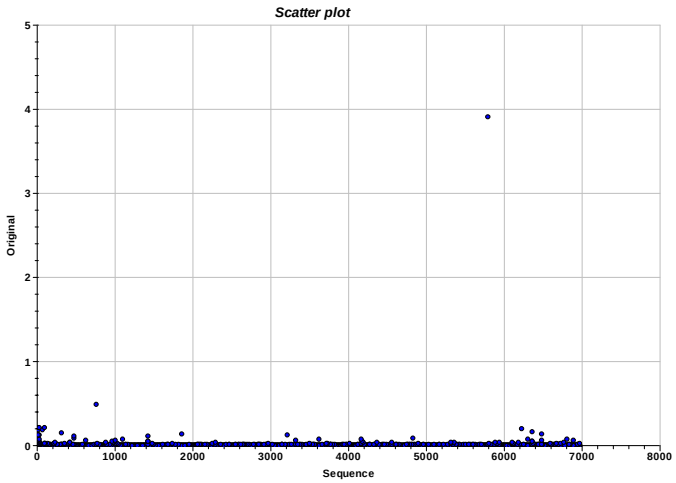
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	2.792	2.771
varnc:	203.289	196.976
coefvm:	5.107	5.065
min:	0.001	0.001
q1:	0.483	0.469
median:	1.420	1.410
q3:	2.470	2.450
max:	1040.000	990.000
iqr:	1.987	1.981

covarnc: 197.422
Pearson: 0.987
Spearman: 0.979
no. data: 23924 / 23925

(data is sub-setted)

Figure 14.30
Blanks assayed with Drill Samples



Data Statistics

variable:	Sequence	Original
weight by:	--	--
mean:	3483.416	0.004
varnc:	4044384.500	0.002
coefvm:	0.577	11.396
min:	1.000	0.001
q1:	1741.000	0.001
median:	3483.000	0.002
q3:	5224.000	0.003
max:	6966.000	3.910
iqr:	3483.000	0.002
covarnc:	0.324	
Pearson:	0.003	
Spearman:	0.057	
no. data:	6966	

(data set at full limits)

14.4.4 Certified Reference Samples

Eight certified reference samples, sourced from Gannet Laboratories, have routinely inserted within the assaying batches at Sukari during the 2005 and 2006 drilling from 2005 - 2009. The standards used are listed below together with their recommended value;

- ST353, with a recommended gold assay of 0.51g/t Au
- ST17, with a recommended gold assay of 0.78g/t Au
- ST01, with a recommended gold assay of 1.44g/t Au
- ST09_0285, with a recommended gold assay of 1.98g/t Au
- ST09_3320, with a recommended gold assay of 1.90g/t Au
- ST355, with a recommended gold assay of 2.34g/t Au
- ST274, with a recommended gold assay of 6.08g/t Au
- ST279, with a recommended gold assay of 7.17g/t Au

Figures 14.31 through 14.38 show the control charts for each of the standards used (with ± 3 standard deviation control lines plotted either side of the recommended grade). General observations that can be made from viewing these control charts are;

- For ST353 the average grade returned for the 99 assays available is 0.518g/t Au and is a very good result.
- For ST17 the average grade returned for the 1688 assays available is 0.764g/t Au and is a very good result.
- For ST01 the average grade returned for the 1137 assays available is 1.452g/t Au and is a very good result.

- For ST09_0285 the average grade returned for the 556 assays available is 2.03g/t Au and is an acceptable result (discontinued, replaced by ST09_3320). For ST09_3320 the average grade returned for the 1119 assays available is 1.954g/t Au and is also an acceptable result.
- For ST355 the average grade returned for the 548 assays available is 2.365g/t Au and is a very good result.
- For ST274 the average grade returned for the 284 assays available is 6.45g/t Au which is about 6% higher than the recommended value. Quite a few assays for this standard lie above the +10% control line, no assays full below the -10% control line. The assaying of this standard pre-2005 (Figure 60 in the previous section) shows that there was a marked step in the control graph, previous to this step the assaying conformed to the recommended grade. Since that time the assays received for this standard have consistently been 5-10% higher than the recommended value.

  Possible reasons maybe that this rise in grade corresponds to a change in Standard batch or a change in analytical procedure at the assay Laboratory. The later is unlikely to be a reasonable explanation due to this jump in grade first occurred when assaying was being undertaken by Minesite Reference Laboratories and has continued after the change to Ultra-trace Analytical Laboratories. Discussions with site personnel indicate that a change in the Standard batch is the most likely explanation. This standard has been discontinued and replaced by STD279.

- For ST279 the average grade returned for the 1151 assays available is 7.278g/t Au and is an acceptable result, albeit, slightly biased high by about 2%

.

Figure 14.31
Control Chart for ST353 (Au ppm AR)

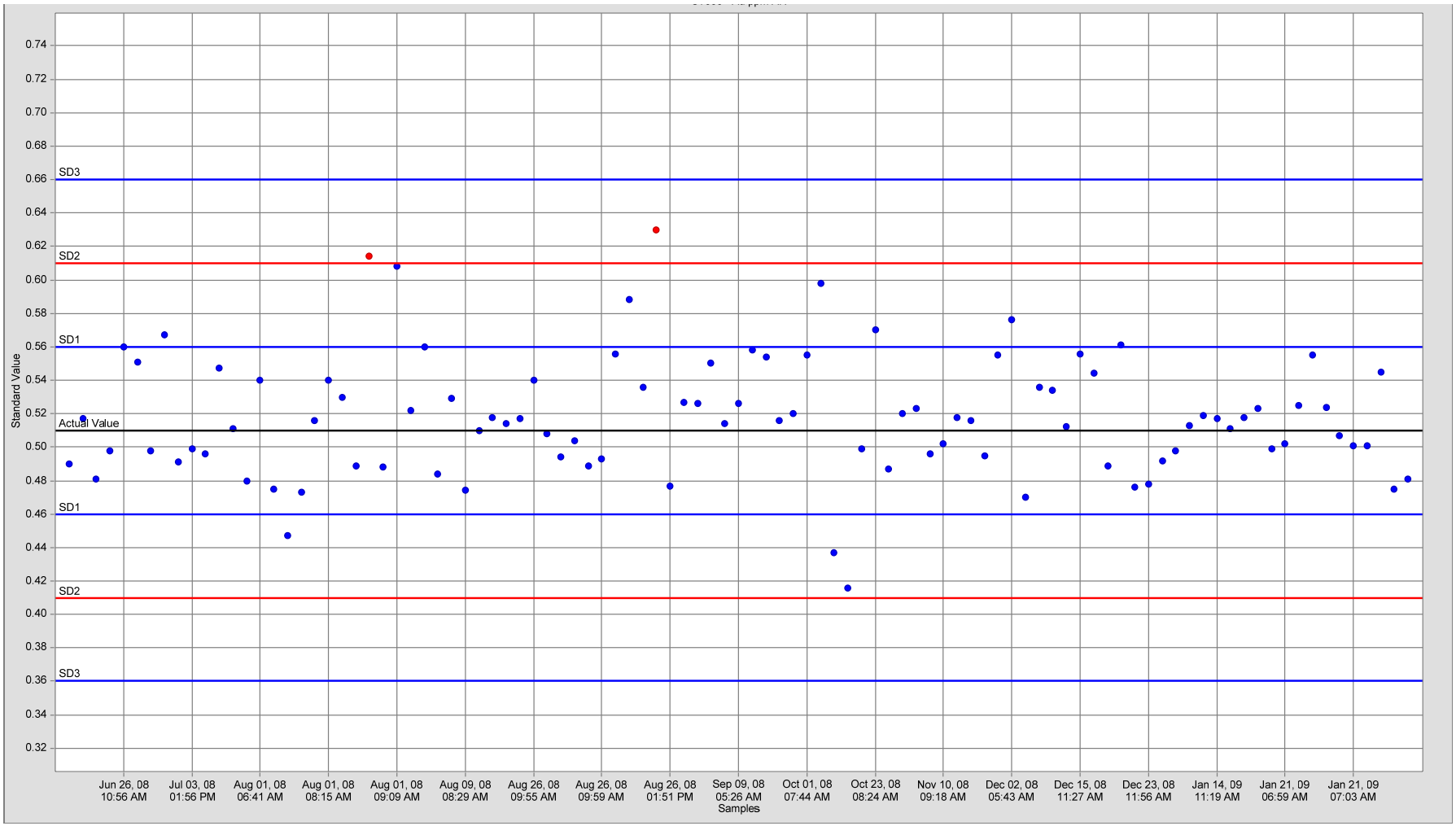


Figure 14.32
Control Chart for ST17 (Au ppm AR)

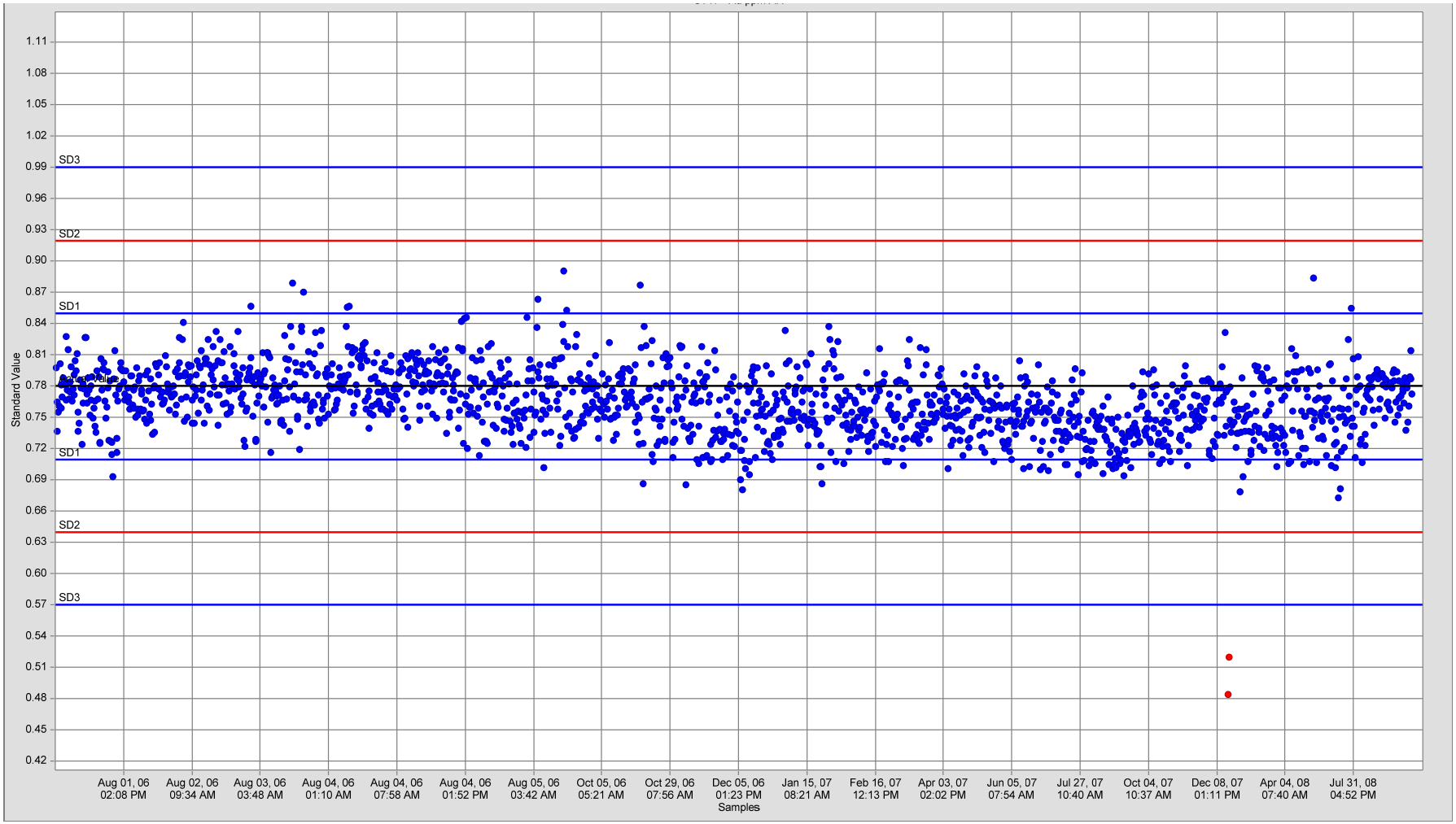


Figure 14.33
Control Chart for ST01 (Au ppm AR)

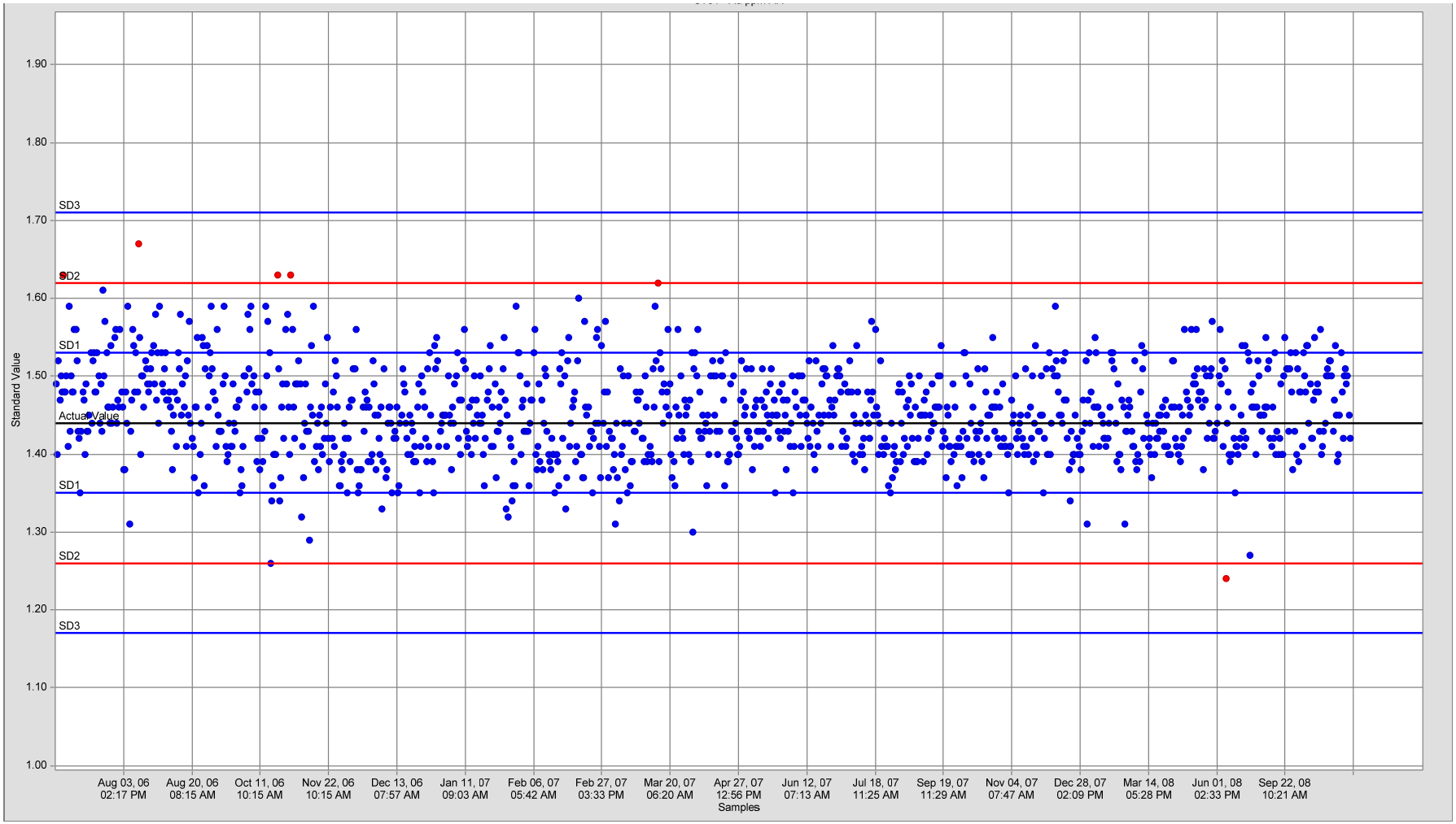


Figure 14.34
Control Chart for ST09_0285 (Au ppm AR)

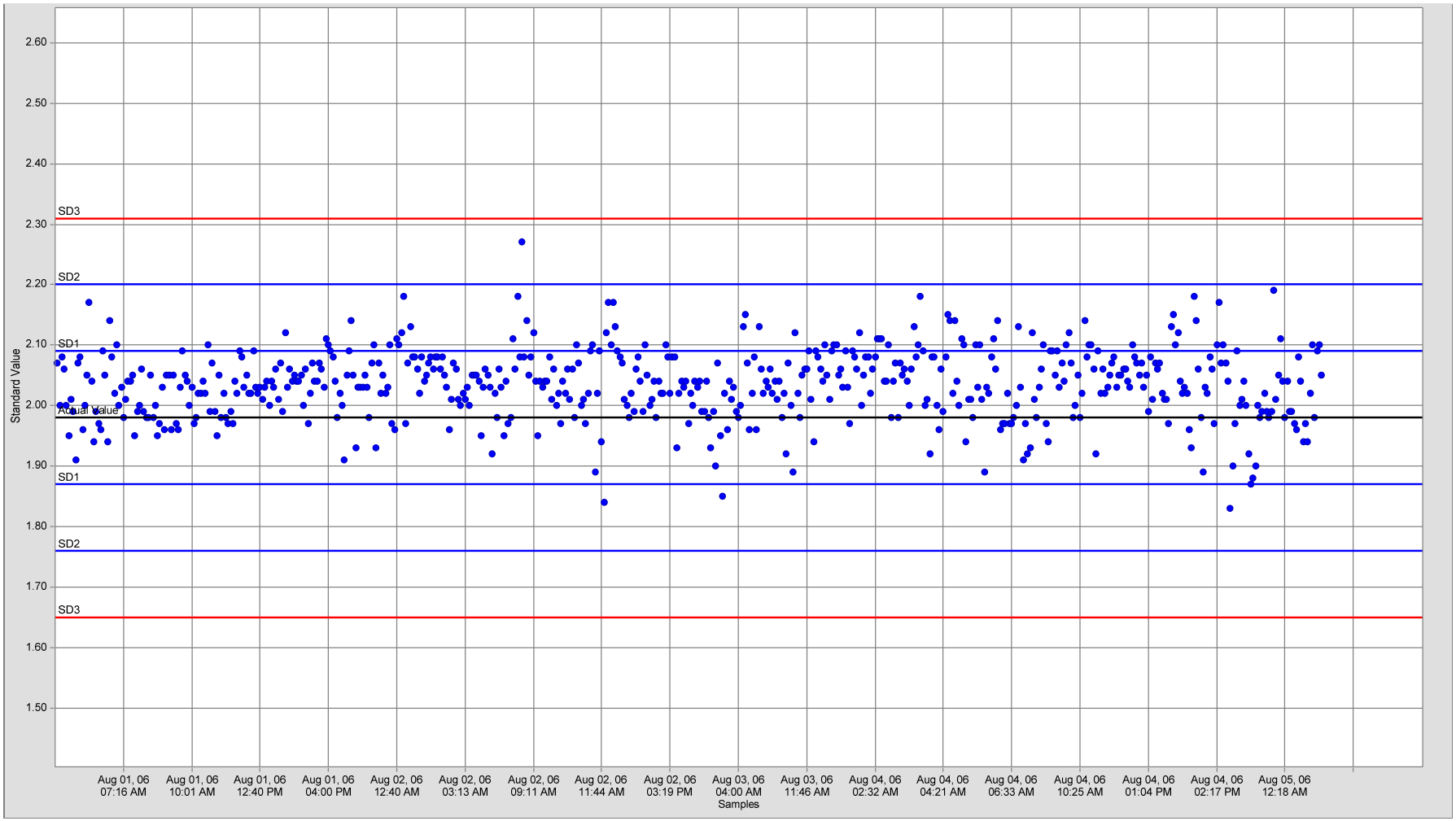


Figure 14.35
Control Chart for ST09_3320 (Au ppm AR)

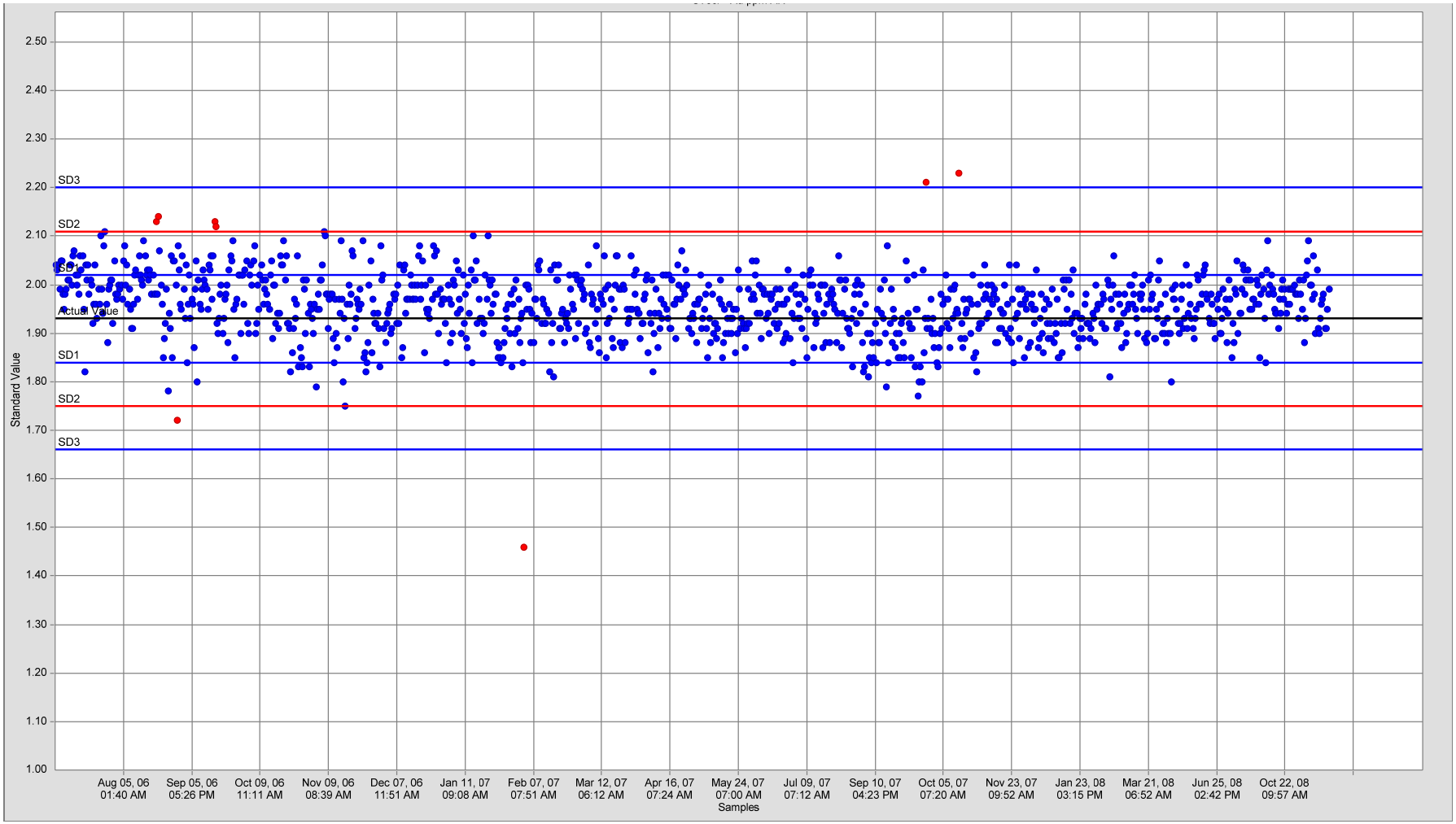


Figure 14.36
Control Chart for ST355 (Au ppm AR)

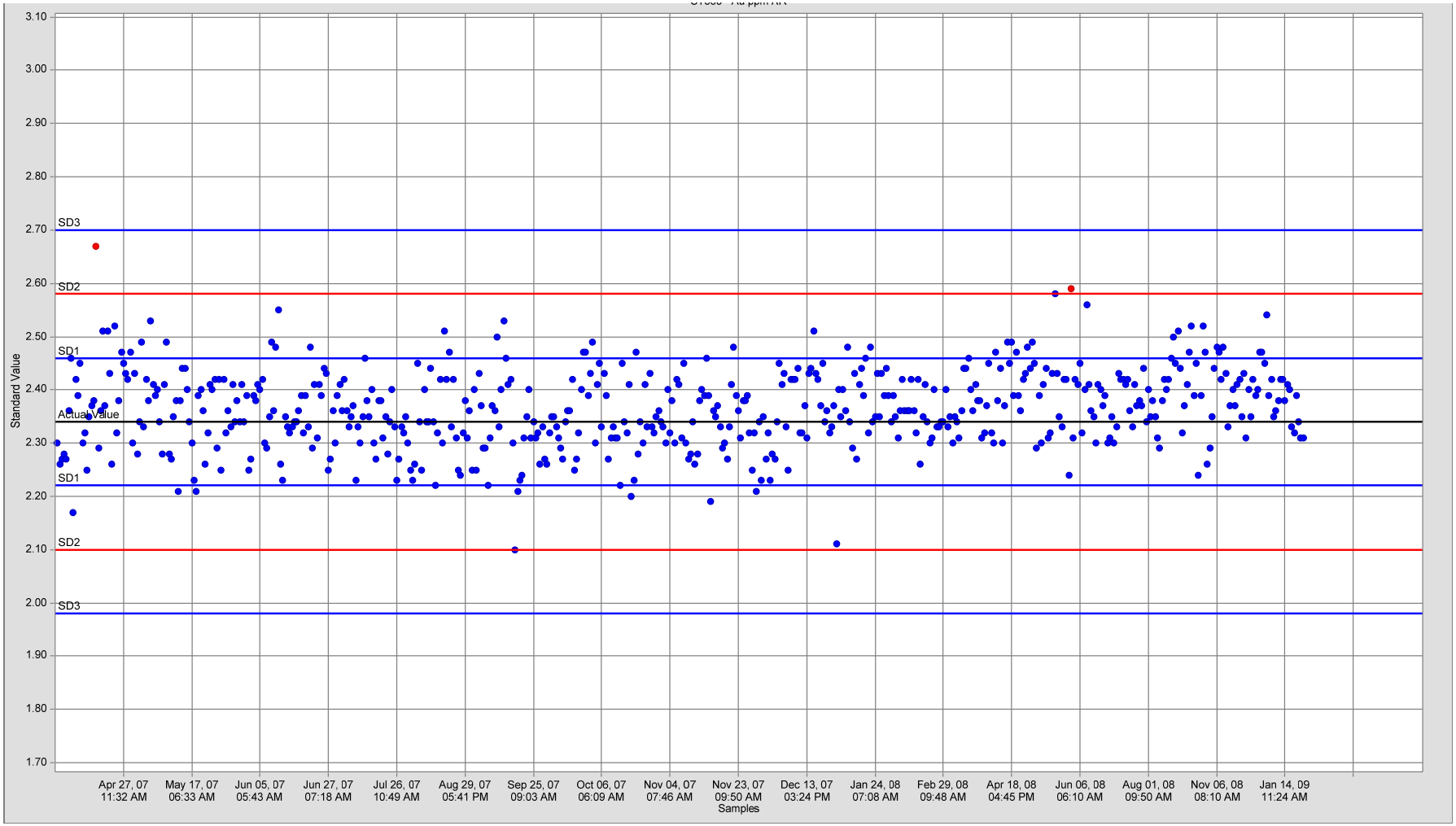


Figure 14.37
Control Chart for ST274 (Au ppm AR)

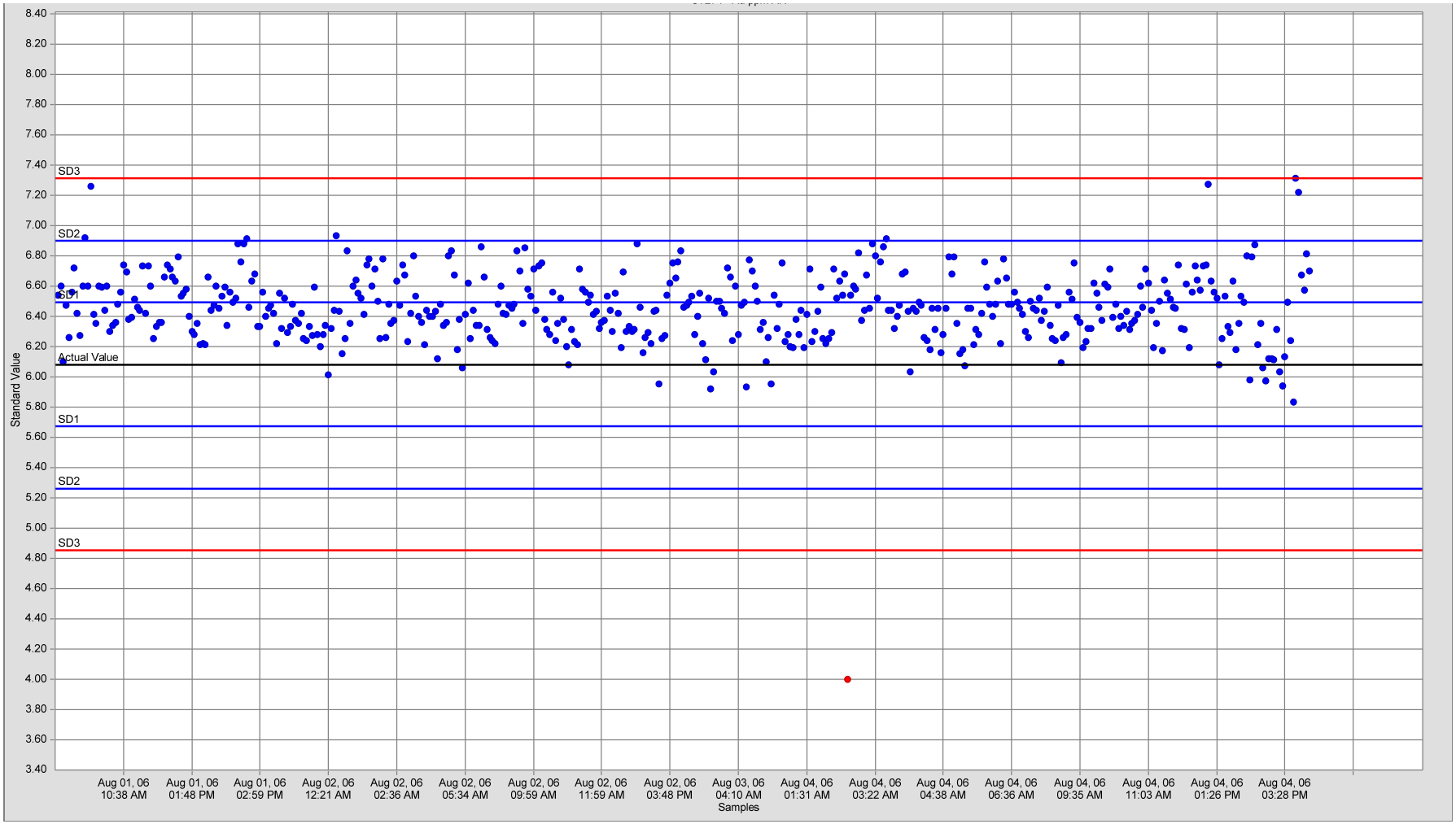
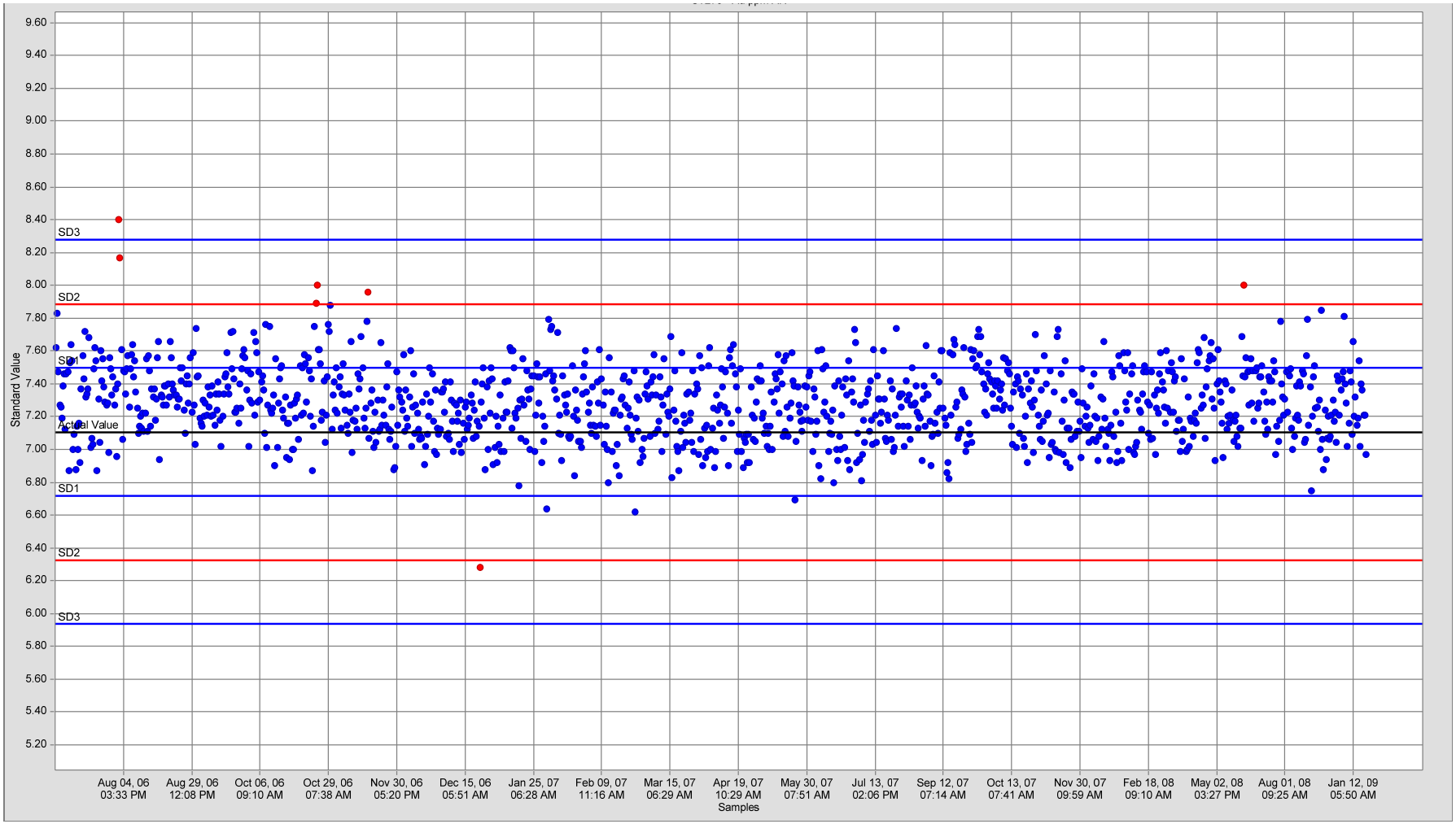


Figure 14.38
Control Chart for ST279 (Au ppm AR)



14.4.5 Field Duplicates, Diamond

Figures 14.39, 14.40 and 14.4.1 show scatter, quantile-quantile and precision plots for all available diamond core duplicate sampling. There are 3,869 pairs of data available. The means of the two data are similar; 2.305g/t Au versus 2.295g/t Au, Original and Duplicate, respectively. The quantile-quantile plot shows that the marginal distributions are comparable with only a slight departure from the 45º line for the few highest grade pairs. The overall result is considered acceptable by H&S given the type of comparison and also given that the precision is measured at ±33%.

14.4.6 Field Duplicates, RC

Figures 14.42, 14.43 and 14.44 show the scatter, quantile-quantile and precision plots for all available RC duplicate sampling. There are 4,016 pairs of data available. The means of the two data are the same (at 0.297g/t Au versus 0.298g/t Au, original and duplicate, respectively). The quantile-quantile plot shows that there is no bias towards either sample type, although there is one particularly poor comparison pair (Original 6.54g/t Au paired with a repeat assay grading 33g/t Au) which impacts the comparison statistics. Excluding the effect of this single data pair the result is considered good by H&S with the precision measured at ±20%, which is better than measured for the core duplicate sampling.

Figure 14.39
Scatter Plot, Original vs Duplicate Samples, Diamond Core

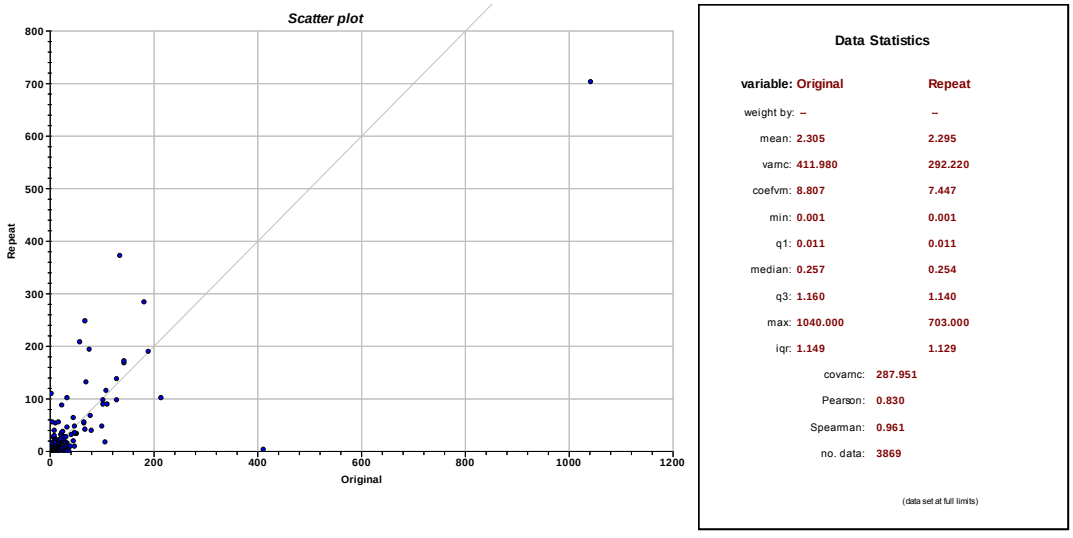
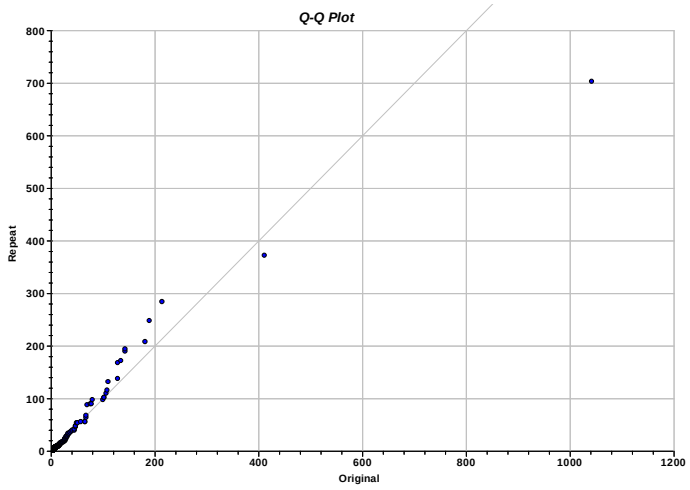


Figure 14.40
QQ Plot, Original vs Duplicate Samples, Diamond Core



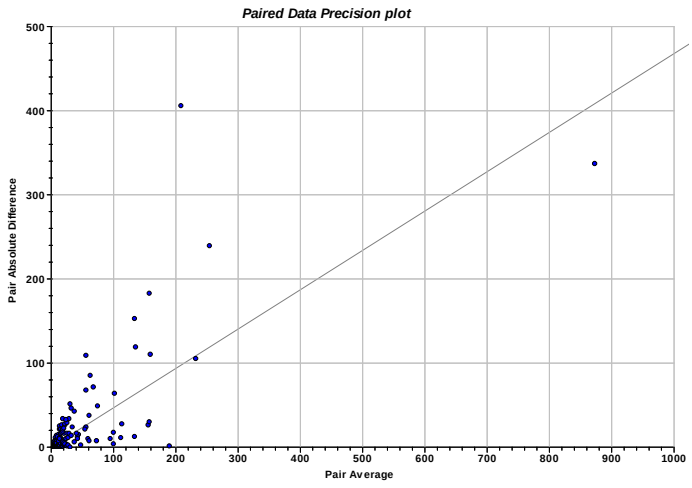
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	2.305	2.295
varnc:	411.980	292.220
coefvm:	8.807	7.447
min:	0.001	0.001
q1:	0.011	0.011
median:	0.257	0.254
q3:	1.160	1.140
max:	1040.000	703.000
iqr:	1.149	1.129

covarnc: 287.951
Pearson: 0.830
Spearman: 0.961
no. data: 3869

(data set at full limits)

Figure 14.41
Precision Plot, Original vs Duplicate Samples, Diamond Core



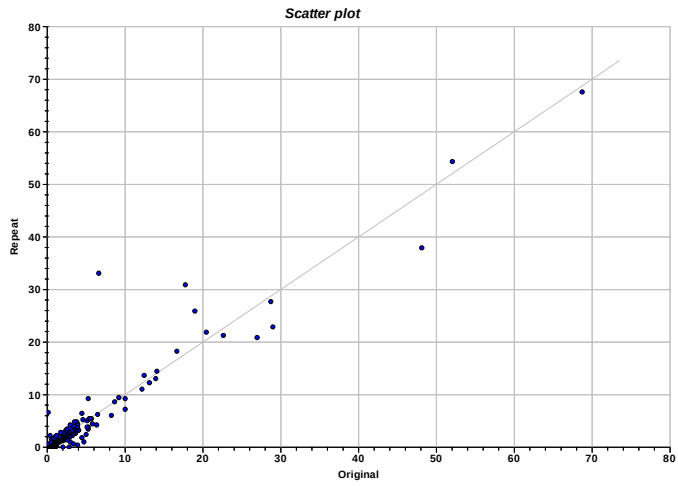
Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	2.305	2.295
varnc:	411.980	292.220
coefvm:	8.807	7.447
min:	0.001	0.001
q1:	0.011	0.011
median:	0.257	0.254
q3:	1.160	1.140
max:	1040.000	703.000
iqr:	1.149	1.129

covarnc: 287.951
Pearson: 0.830
Spearman: 0.961
precision: +/-47% @ 62%CI
no. data: 3869

(data set at full limits)

Figure 14.42
Scatter Plot, Original vs Duplicate Samples, RC



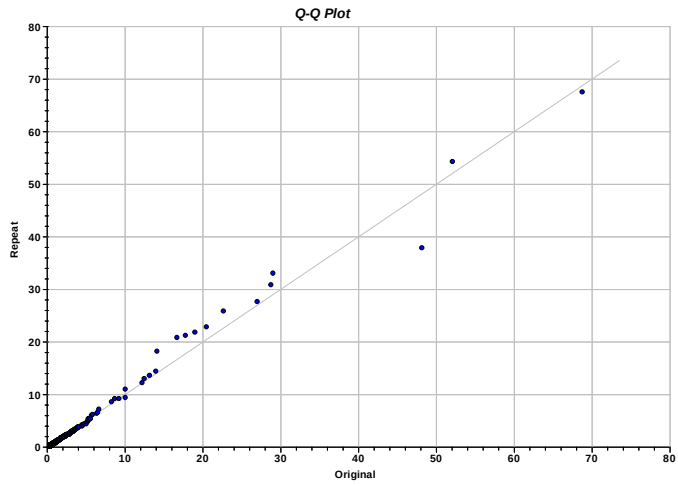
Data Statistics

variable: Original	Repeat
weight by: --	--
mean: 0.297	0.298
varnc: 4.065	4.181
coefvm: 6.792	6.860
min: 0.001	0.001
q1: 0.001	0.002
median: 0.010	0.011
q3: 0.079	0.078
max: 68.700	67.600
iqr: 0.078	0.076

covarnc: 3.959
Pearson: 0.960
Spearman: 0.946
no. data: 4016

(data set at full limits)

Figure 14.43
QQ Plot, Original vs Duplicate Samples, RC



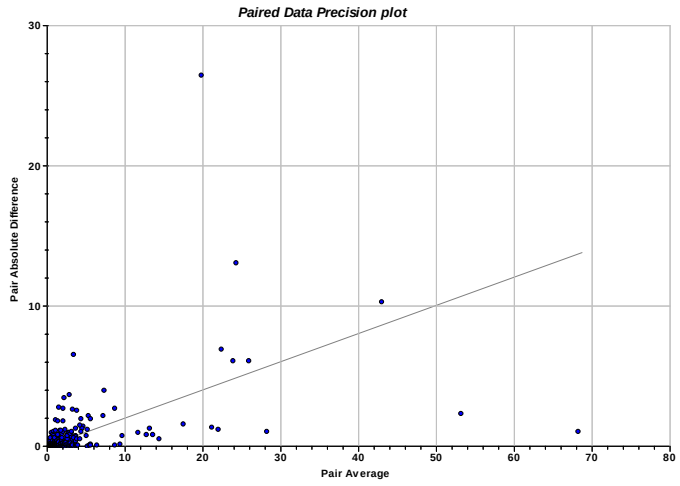
Data Statistics

variable: Original	Repeat
weight by: --	--
mean: 0.297	0.298
varnc: 4.065	4.181
coefvm: 6.792	6.860
min: 0.001	0.001
q1: 0.001	0.002
median: 0.010	0.011
q3: 0.079	0.078
max: 68.700	67.600
iqr: 0.078	0.076

covarnc: 3.959
Pearson: 0.960
Spearman: 0.946
no. data: 4016

(data set at full limits)

Figure 14.44
Precision Plot, Original vs Duplicate Samples, RC



Data Statistics

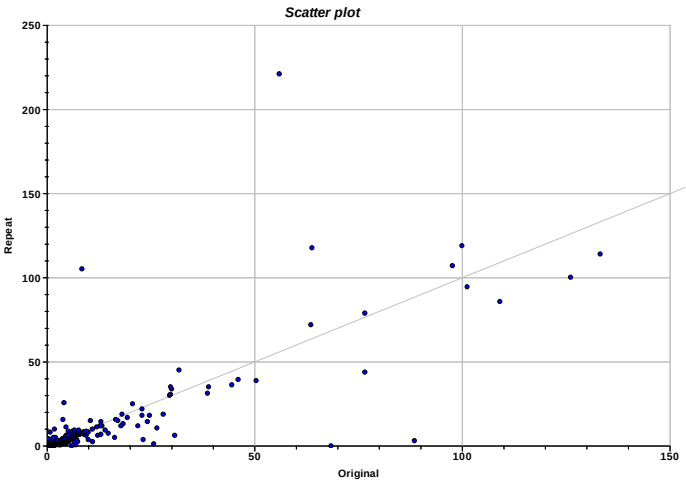
variable:	Original	Repeat
weight by:	--	--
mean:	0.297	0.298
varnc:	4.065	4.181
coefvm:	6.792	6.860
min:	0.001	0.001
q1:	0.001	0.002
median:	0.010	0.011
q3:	0.079	0.078
max:	68.700	67.600
iqr:	0.078	0.076
covarnc:	3.959	
Pearson:	0.960	
Spearman:	0.946	
precision:	+/-20% @ 52%CI	
no. data:	4016	

(data set at full limits)

14.4.7 Re-assaying of Crushed Material

As a check of the repeatability of samples split at the first stage of the sample preparation, a suite of second splits from the crushed material has been assayed and are able to be compared to the original assay. Figures 14.45 and 14.46 show the scatter and quantile-quantile plot of these data pairs. The mean grades of the two datasets are 1.632g/t Au and 1.592g/t Au, original and crushed grade, respectively. The means are within 2% and the quantile-quantile plot shows that there is no bias towards either dataset. This result indicates that the crushing and splitting stage is performing well at the sample preparation facility at Sukari.

Figure 14.45
Scatter Plot, Original vs Resplit crushed Material

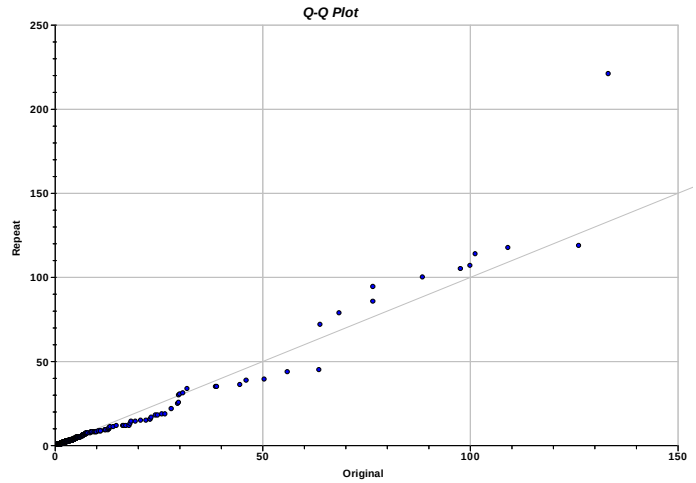


Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	1.632	1.592
varnc:	63.975	81.255
coefvm:	4.902	5.664
min:	0.001	0.001
q1:	0.004	0.008
median:	0.084	0.088
q3:	0.747	0.700
max:	133.000	221.000
iqr:	0.743	0.692
covarnc:	58.565	
Pearson:	0.812	
Spearman:	0.964	
no. data:	2099	

(data set at full limits)

Figure 14.46
QQ Plot, Original vs Resplit crushed Material



Data Statistics

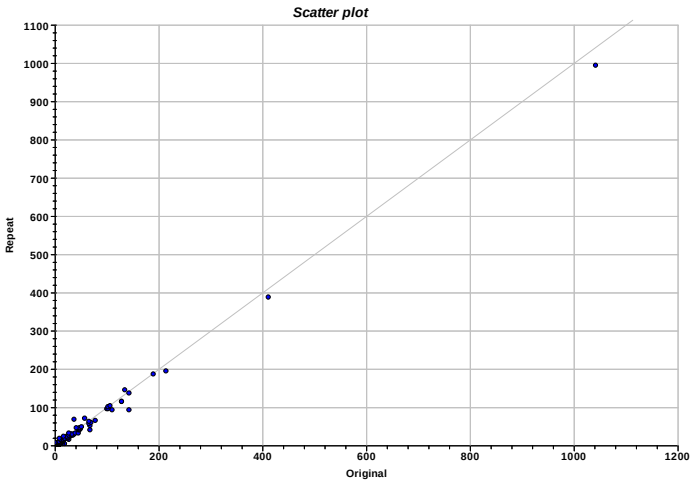
variable:	Original	Repeat
weight by:	--	--
mean:	1.632	1.592
varnc:	63.975	81.255
coefvm:	4.902	5.664
min:	0.001	0.001
q1:	0.004	0.008
median:	0.084	0.088
q3:	0.747	0.700
max:	133.000	221.000
iqr:	0.743	0.692
covarnc:	58.565	
Pearson:	0.812	
Spearman:	0.964	
no. data:	2099	

(data set at full limits)

14.4.8 Duplicate Assaying of Pulverised Material

As a check of the repeatability of the pulp samples produced by the site sample preparation facility, a suite of duplicates from the pulverised sample have been assayed and are able to be compared to the original assay. Figures 14.47 and 14.48 show the scatter and quantile-quantile plot of these data pairs. The mean grades of the original and duplicate pulverised samples are similar (1.917g/t Au and 1.868g/t Au, respectively) and the quantile-quantile plot shows that there is no bias towards either assay. The result indicates that pulverising and splitting stage is performing well at Sukari.

Figure 14.47
Scatter Plot, Original vs Resplit pulverised Material

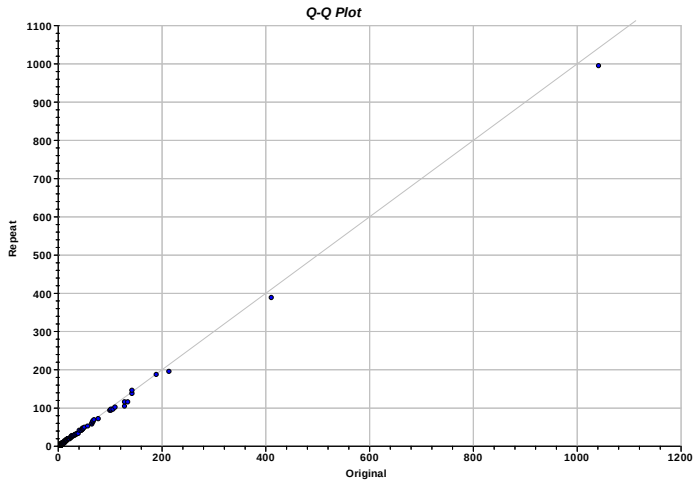


Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	1.917	1.868
varnc:	488.394	445.559
coefvm:	11.529	11.297
min:	0.001	0.001
q1:	0.001	0.003
median:	0.016	0.019
q3:	0.316	0.317
max:	1040.000	994.000
iqr:	0.315	0.314
covarnc:	465.483	
Pearson:	0.998	
Spearman:	0.943	
no. data:	3136	

(data set at full limits)

Figure 14.48
QQ Plot, Original vs Resplit pulverised Material



Data Statistics

variable:	Original	Repeat
weight by:	--	--
mean:	1.917	1.868
varnc:	488.394	445.559
coefvm:	11.529	11.297
min:	0.001	0.001
q1:	0.001	0.003
median:	0.016	0.019
q3:	0.316	0.317
max:	1040.000	994.000
iqr:	0.315	0.314

covarnc: 465.483
Pearson: 0.998
Spearman: 0.943
no. data: 3136

(data set at full limits)

14.5 Sampling and Assaying, 25th January 2009 to 31st May 2010

The QAQC programmes to verify the sampling and assaying conducted during 2009 to 31st May 2010 are exhaustive and include:-

- Routine laboratory repeat assaying, routine Fire Assay ('FA') checks of Aqua Regia ('AR') original assaying, consistent with the previous drilling periods except that the assay laboratory had been changed to Ultratrace Analytical Laboratories, Perth, with some changes to the AR procedure and charge sizes (discussed earlier in Section 13.2).
- Insertion of blanks into the sample preparation and assaying batches.
- Insertion of certified standards into the assaying batches.
- Particle size analysis of the pulverised sample (presented in Section 13.1.2).
- Duplicate sampling of both diamond core and RC samples (field duplicates).
- Sample preparation duplicates i.e. a number of second split obtained at the sample preparation facility after initial crushing but before pulverisation. These samples are dispatched to Ultratrace for pulverisation and assaying.
- Sample preparation pulp duplicates, commenced in August 2006. These samples are a second split from the pulverised material obtained at the sample preparation facility. These samples are dispatched to Ultratrace for assaying.

The insertion of QAQC samples into the assaying batches was augmented in August 2006 with the inclusion of duplicate pulp assaying. The current sequencing of the QAQC samples proceeds at frequency of 1:25 for RC samples and for the diamond samples the QAQC samples are inserted at a ratio of at least 1:50 however, this is increased in the mineralised zones. The

sequencing of the QAQC samples is shown below for both the diamond and RC sampling;

- Diamond Drilling; SAMPLE – BLANK – STANDARD – PULP SAMPLE – DUPLICATE (1/4 Core) - SAMPLE
- RC drilling; SAMPLE – BLANK – STANDARD – PULP SAMPLE – DUPLICATE (Field re-split) - SAMPLE.

The following section summaries the results of the above QAQC programmes.

14.5.1 Routine Laboratory Repeat Analyses

At least three duplicate assays are performed in every batch of samples, the samples for duplicate analyses being selected at random by the laboratory information management system (LIMS). Samples returning high assays are also selected for subsequent repeat analyses. During the period there were duplicate analyses for 11,158 samples. Figures 14.49 and 14.50 show scatter and quantile-quantile plots comparing original gold assays with assay repeats. The means of the two assay datasets are equivalent, suggesting that sample extraction and analytical errors are very low. This is supported by the precision plot (Figure 14.51) that shows the assaying precision to be ±8%.

Figure 14.49
Scatter Plot, Original Assays vs Repeat Assays

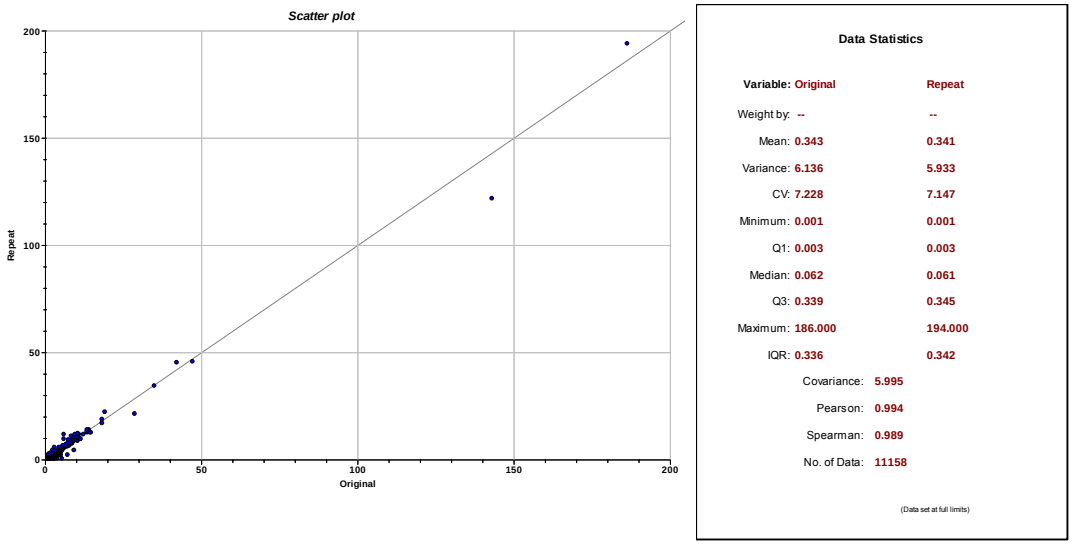
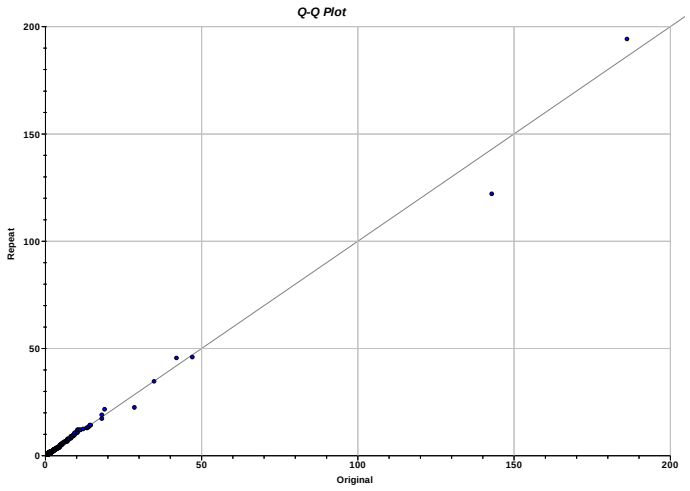


Figure 14.50
QQ Plot, Original Assays vs Repeat Assays

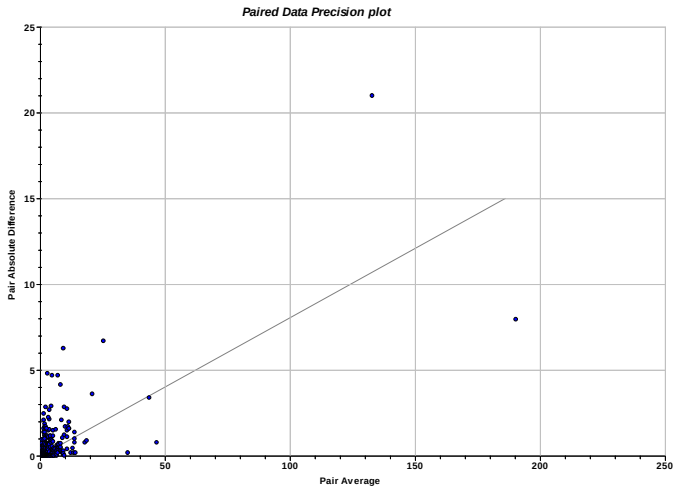


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	0.343	0.341
Variance:	6.136	5.933
CV:	7.228	7.147
Minimum:	0.001	0.001
Q1:	0.003	0.003
Median:	0.062	0.061
Q3:	0.339	0.345
Maximum:	186.000	194.000
IQR:	0.336	0.342
Covariance:	5.995	
Pearson:	0.994	
Spearman:	0.989	
No. of Data:	11158	

(Data set at full limits)

Figure 14.51
Precision Plot, Original Assays vs Repeat Assays



Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	0.343	0.341
Variance:	6.136	5.933
CV:	7.228	7.147
Minimum:	0.001	0.001
Q1:	0.003	0.003
Median:	0.062	0.061
Q3:	0.339	0.345
Maximum:	186.000	194.000
IQR:	0.336	0.342

Covariance: 5.995
Pearson: 0.994
Spearman: 0.989
Precision: +/-8% @ 42%CI
No. of Data: 11158
(Data set at full limits)

14.5.2 Routine Fire Assay Checks

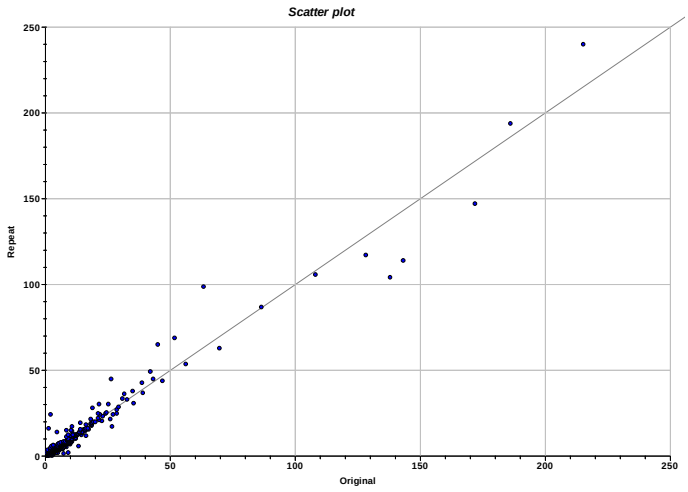
Of samples that return assays more than 1g/t Au, a selection are subsequently analysed by 30g fire assay with AAS finish. There are 5058 check fire assays recorded for the period under investigation.

Figures 14.52 and 14.53 show scatter and quantile-quantile plots comparing original gold assays with fire assay repeats. Fire assays correlate well. AR assaying has a slightly higher global mean (2.30g/t Au versus 2.29g/t Au) This is an unexpected result; normally it would be expect the AR assaying to give, on average, lower grades when compared to FA.

14.5.3 Blanks

Assays of blank material inserted within sample batches undertaken during the review period are shown in Figure 14.54. The blank material is inserted after a drill sample and before a standard to check for cross sample contamination at the pulverisation stage of sample preparation. The blank material is sourced from a known barren outcrop (gabbro composition) within the Sukari project area. No significant issue are identified from observations on the plot of blanks over the period.

Figure 14.52
Scatter Plot, Original Assays vs FA Repeats, all Data

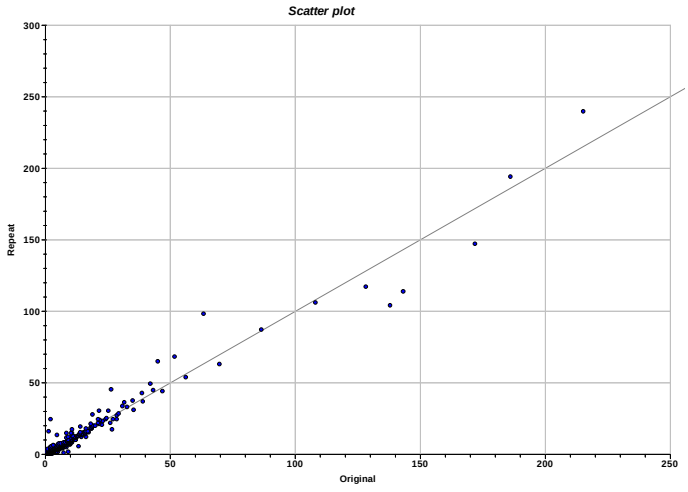


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	2.302	2.293
Variance:	47.990	48.007
CV:	3.010	3.021
Minimum:	0.001	0.001
Q1:	0.247	0.269
Median:	1.350	1.340
Q3:	2.390	2.350
Maximum:	215.000	240.000
IQR:	2.143	2.081
Covariance:	47.207	
Pearson:	0.984	
Spearman:	0.978	
No. of Data:	5058	

(Data set at full limits)

Figure 14.53
QQ Plot, Original Assays vs FA Repeats

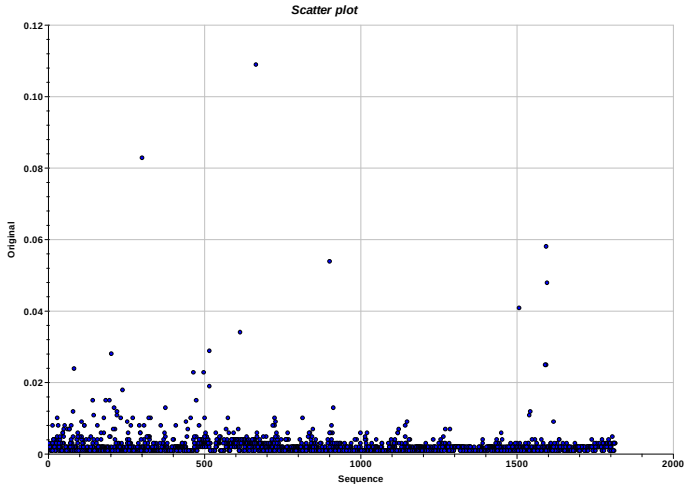


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	2.302	2.293
Variance:	47.990	48.007
CV:	3.010	3.021
Minimum:	0.001	0.001
Q1:	0.247	0.269
Median:	1.350	1.340
Q3:	2.390	2.350
Maximum:	215.000	240.000
IQR:	2.143	2.081
Covariance:	47.207	
Pearson:	0.984	
Spearman:	0.978	
No. of Data:	5058	

(Data set at full limits)

Figure 14.54
Blanks assayed with Drill Samples



Data Statistics

Variable:	Sequence	Original
Weight by:	--	--
Mean:	907.500	0.002
Variance:	274216.688	0.000
CV:	0.577	1.833
Minimum:	1.000	0.001
Q1:	453.000	0.001
Median:	907.000	0.002
Q3:	1360.000	0.002
Maximum:	1814.000	0.109
IQR:	907.000	0.001
Covariance:	-0.236	
Pearson:	-0.098	
Spearman:	-0.127	
No. of Data:	1814	

(Data set at full limits)

14.5.4 Certified Reference Samples

Eight certified reference samples, sourced from Gannet Laboratories, have routinely inserted within the assaying batches at Sukari during the 2005 and 2006 drilling from 2005 – 2010

. The standards used are listed below together with their recommended value;

- ST353, with a recommended gold assay of 0.51g/t Au
- ST17, with a recommended gold assay of 0.78g/t Au
- ST01, with a recommended gold assay of 1.44g/t Au
- ST09_0285, with a recommended gold assay of 1.98g/t Au
- ST09_6382, with a recommended gold assay of 1.90g/t Au
- ST355, with a recommended gold assay of 2.34g/t Au
- ST279, with a recommended gold assay of 7.17g/t Au
- ST383, with a recommended gold assay of 7.26g/t Au

Figures 14.55 through 14.62 show the control charts for each of the standards used (with ± 3 standard deviation control lines plotted either side of the recommended grade). General observations that can be made from viewing these control charts are;

- For ST353 the average grade returned for the 248 assays available is 0.512g/t Au and is a very good result.
- For ST17 the average grade returned for the 124 assays available is 0.776g/t Au and is a very good result. For ST17_7413 (actual grade 0.77g/t Au) the average grade returned for the 103 assays available is 0.785/t Au and is also very good result.

- For ST01 the average grade returned for the 236 assays available is 1.467g/t Au and is a very good result.
- For ST09_0285 the average grade returned for the 147 assays available is 1.950g/t Au and is a very good result.
- For ST09_6382 the average grade returned for the 83 assays available is 1.936g/t Au and is also a very good result.
- For ST355 the average grade returned for the 227 assays available is 2.386g/t Au and is an acceptable result.
- For ST279 the average grade returned for the 146 assays available is 7.279g/t Au and is an acceptable result, albeit, slightly biased high by about 2%. This standard has been discontinued and replaced by STD383
- For ST383 the average grade returned for the 106 assays available is 7.308g/t Au and is a very good result.

Figure 14.55
Control Chart for ST353 (Au ppm AR)

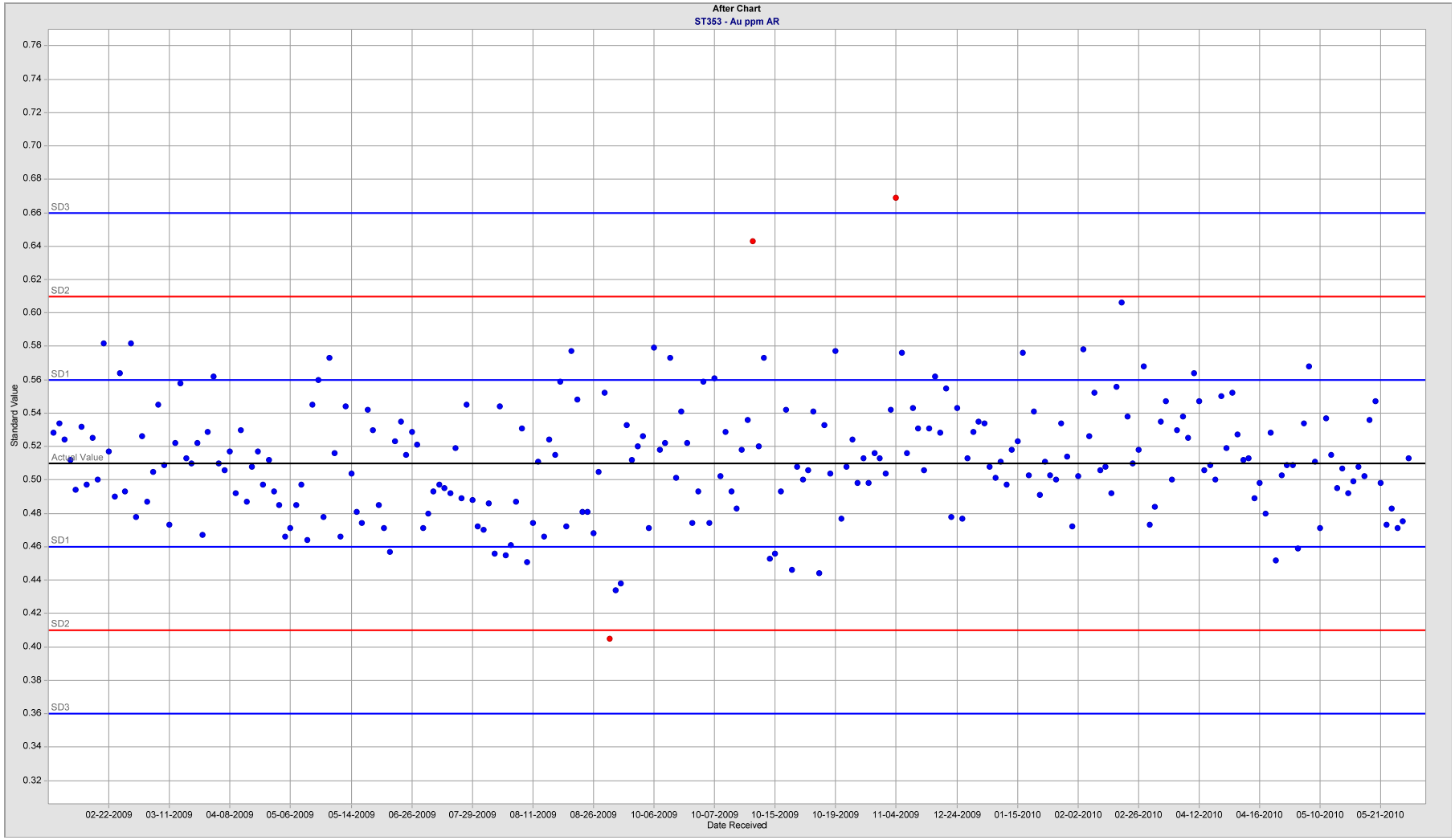


Figure 14.56
Control Chart for ST17 (Au ppm AR)

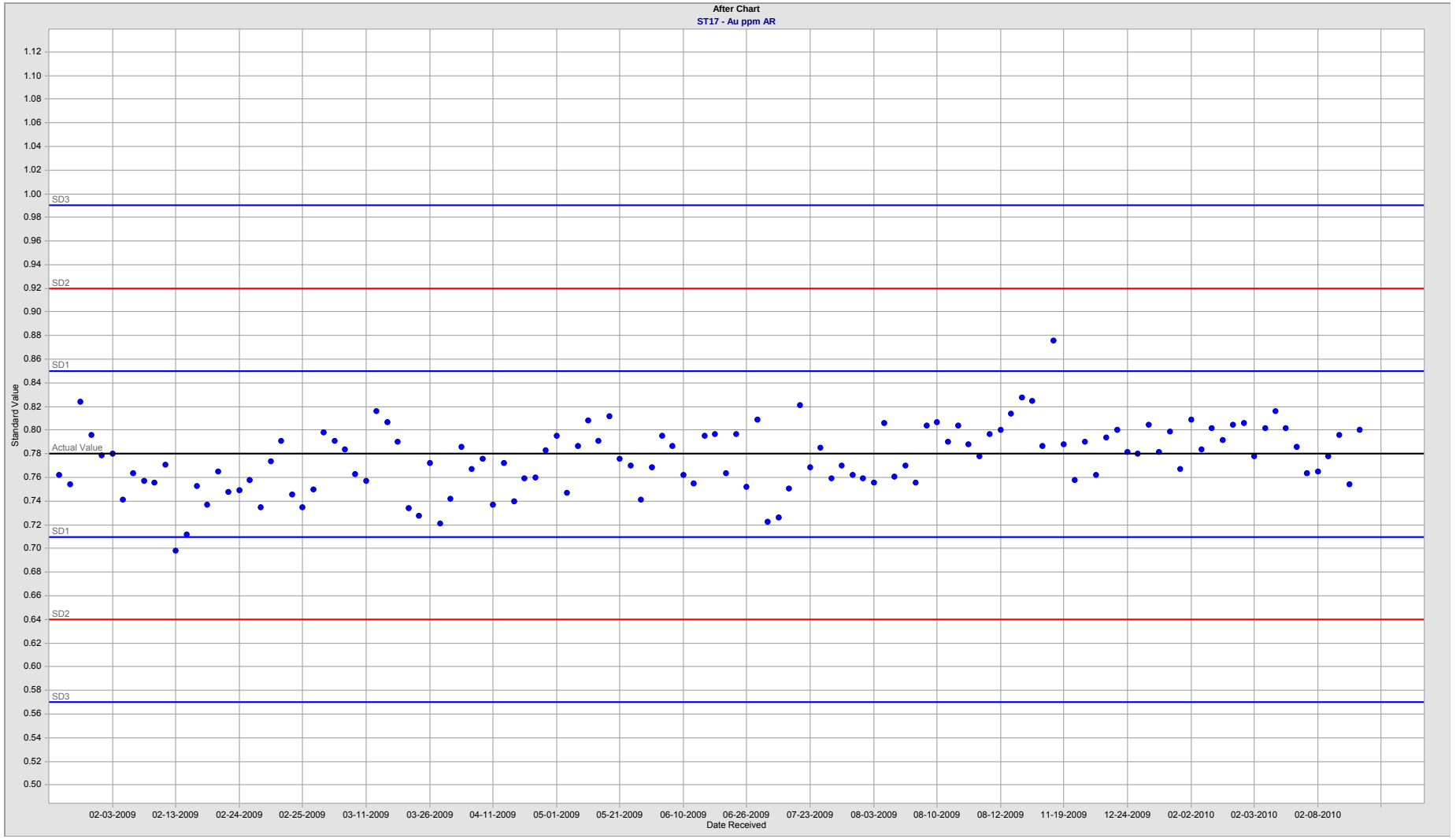


Figure 14.57
Control Chart for ST01 (Au ppm AR)

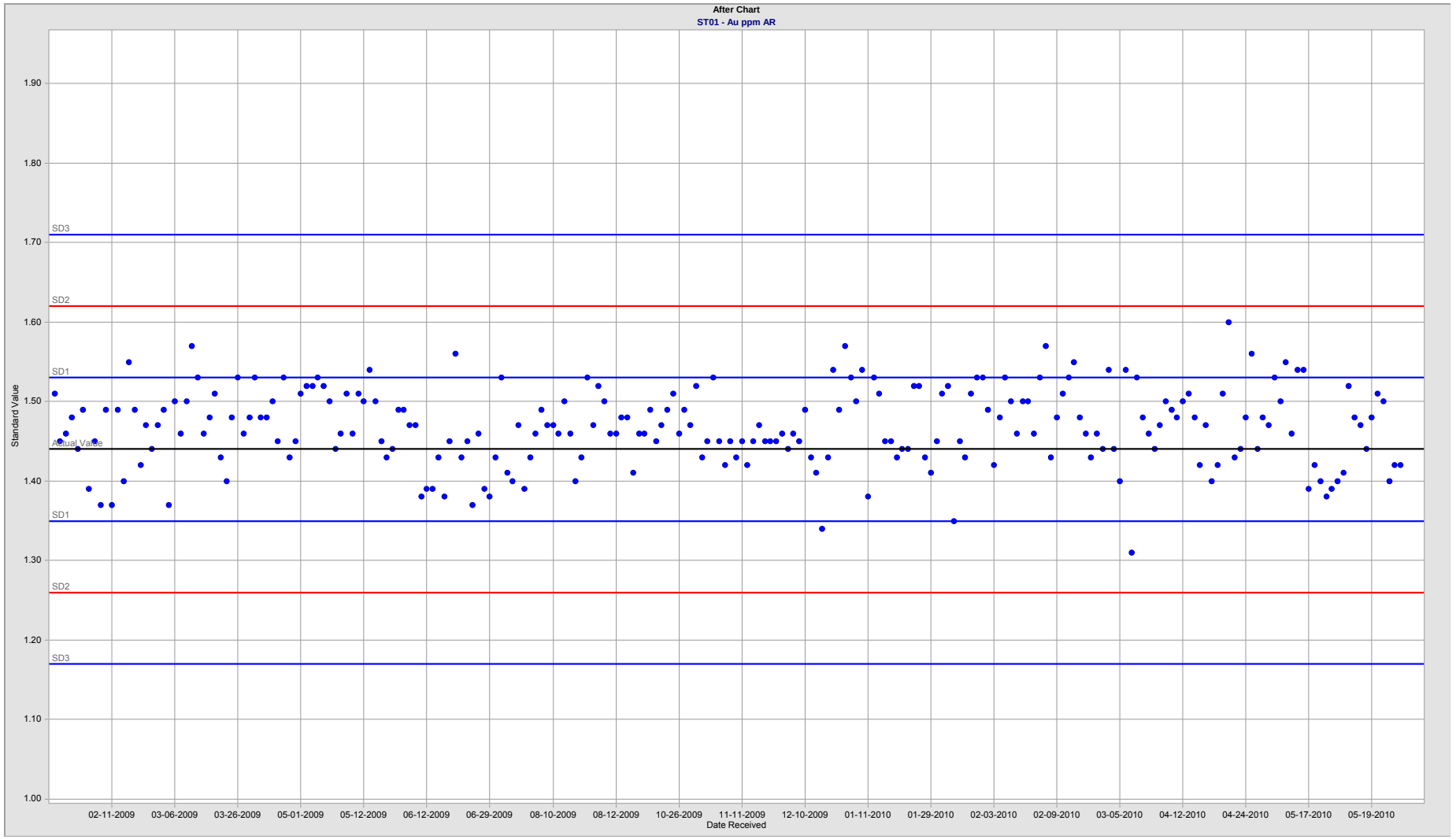


Figure 14.58
Control Chart for ST09_0285 (Au ppm AR)

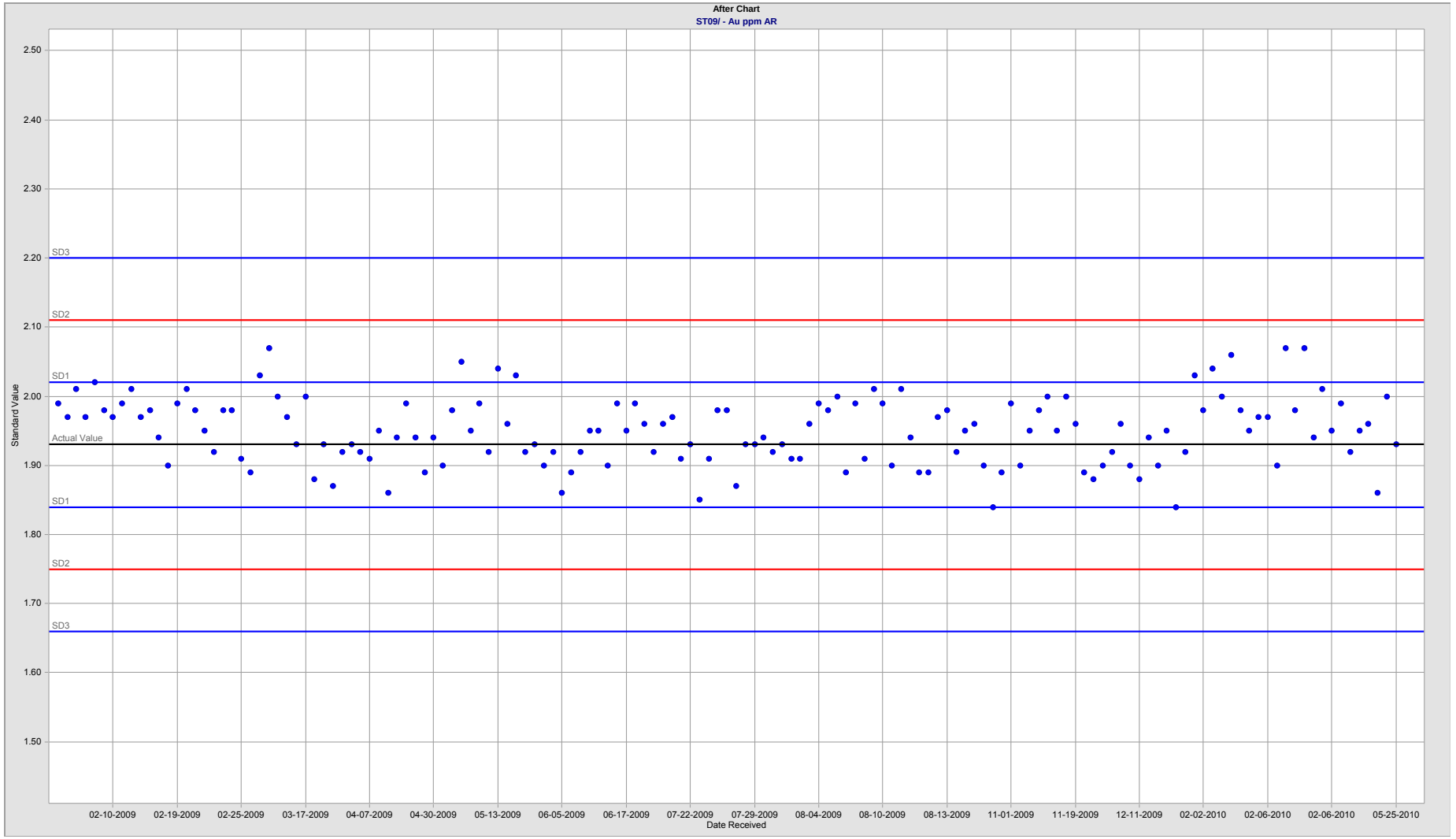


Figure 14.59
Control Chart for ST09_6382(Au ppm AR)

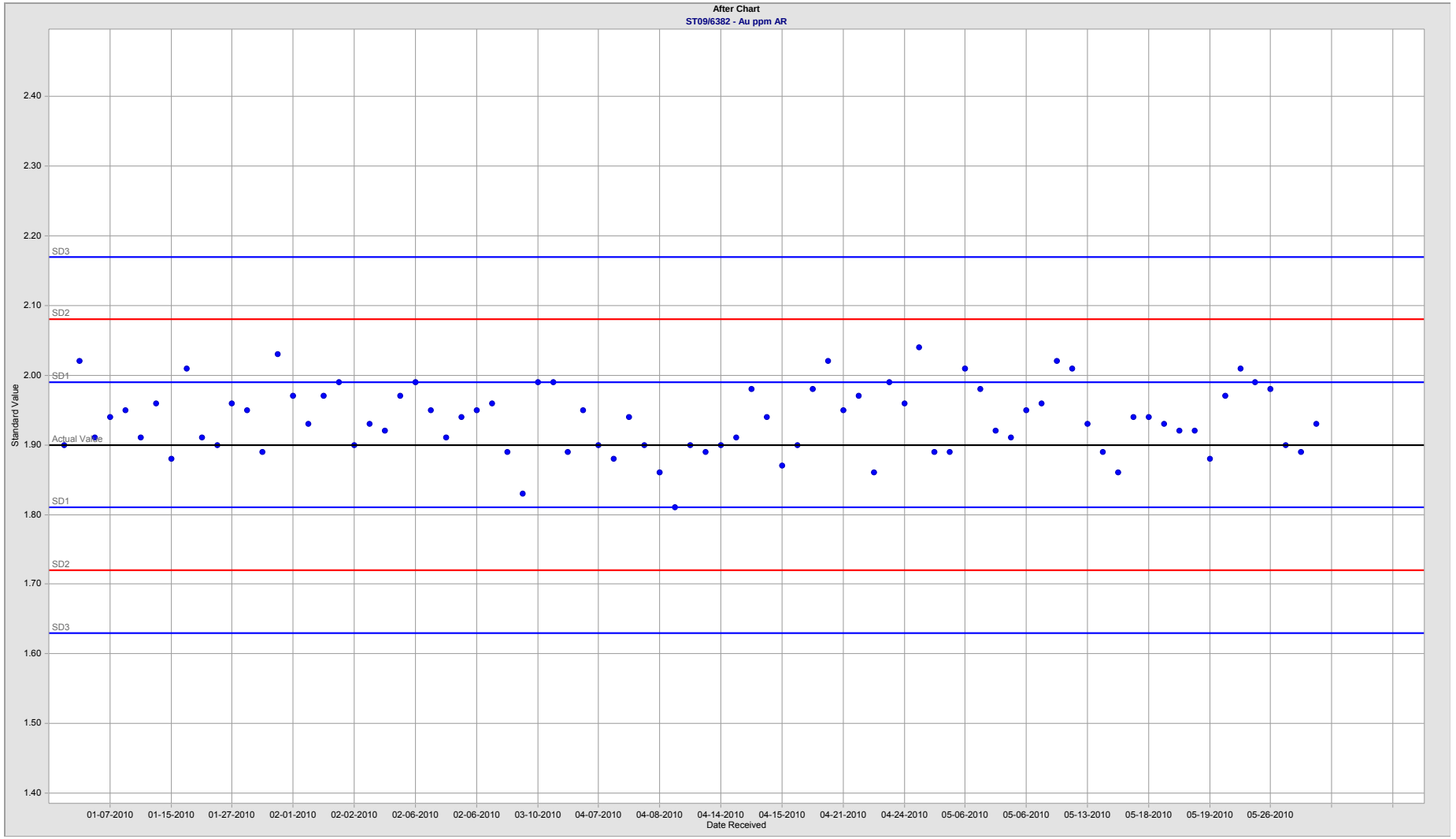


Figure 14.60
Control Chart for ST355 (Au ppm AR)

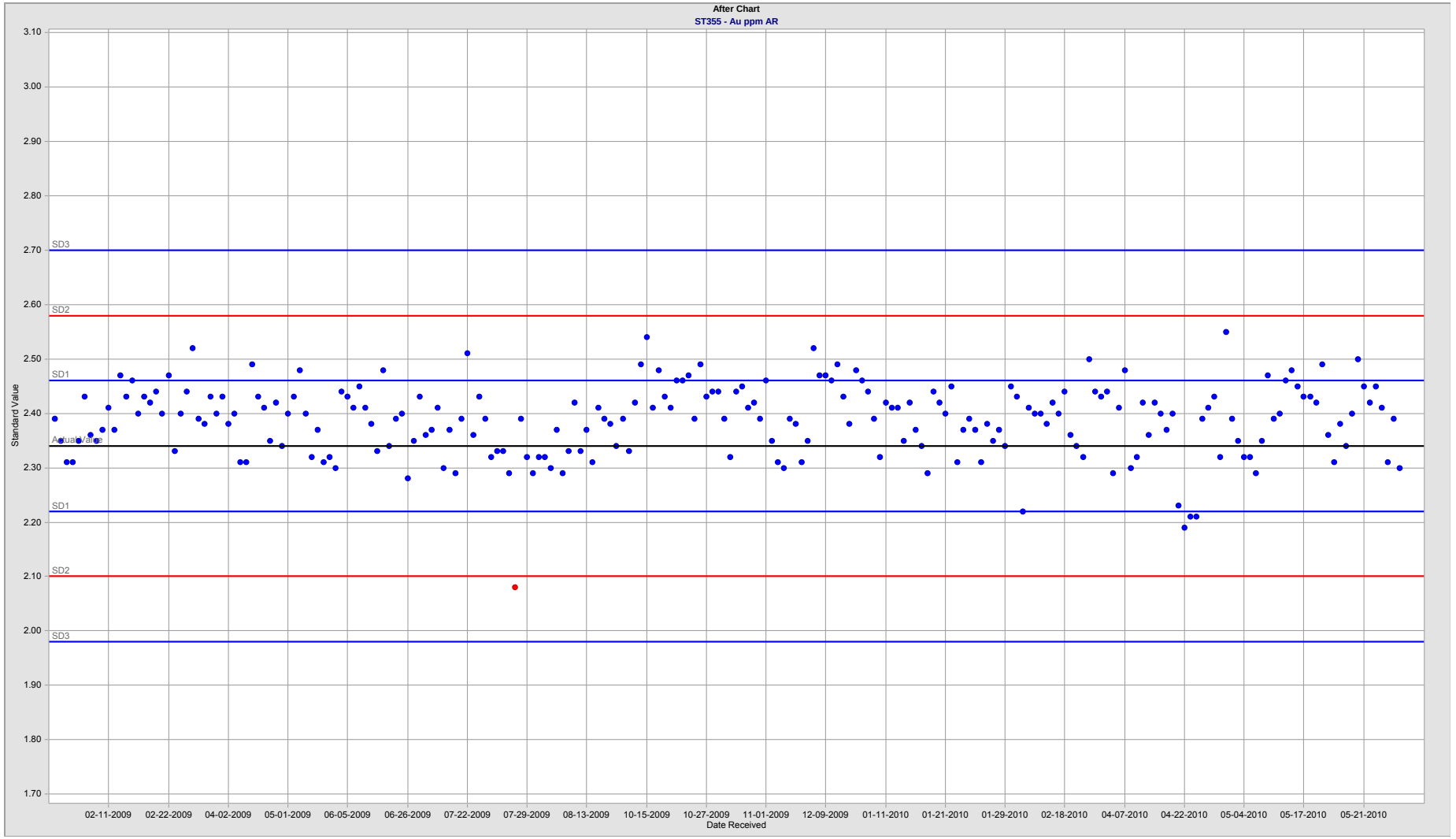


Figure 14.61
Control Chart for ST383 (Au ppm AR)

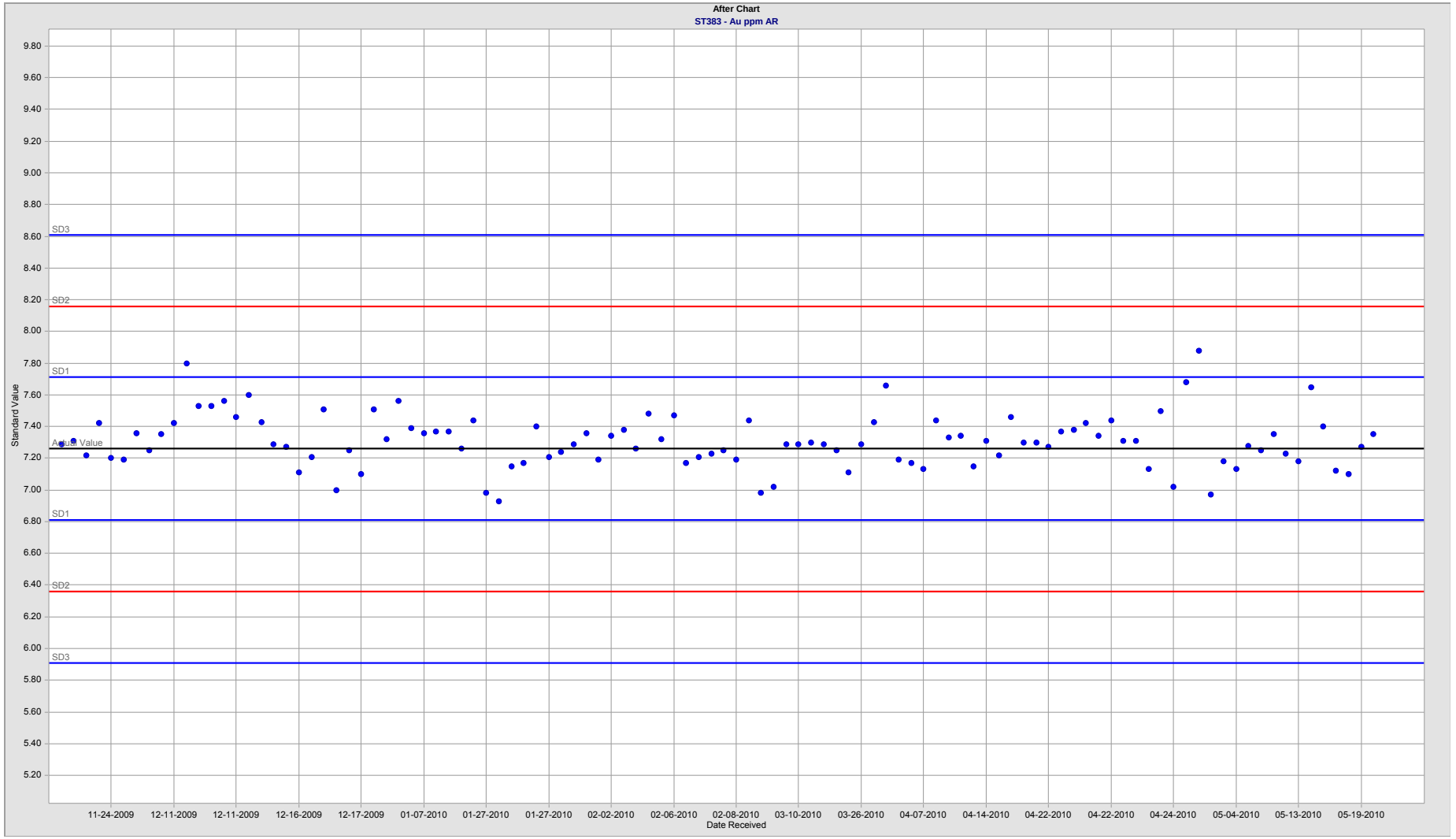
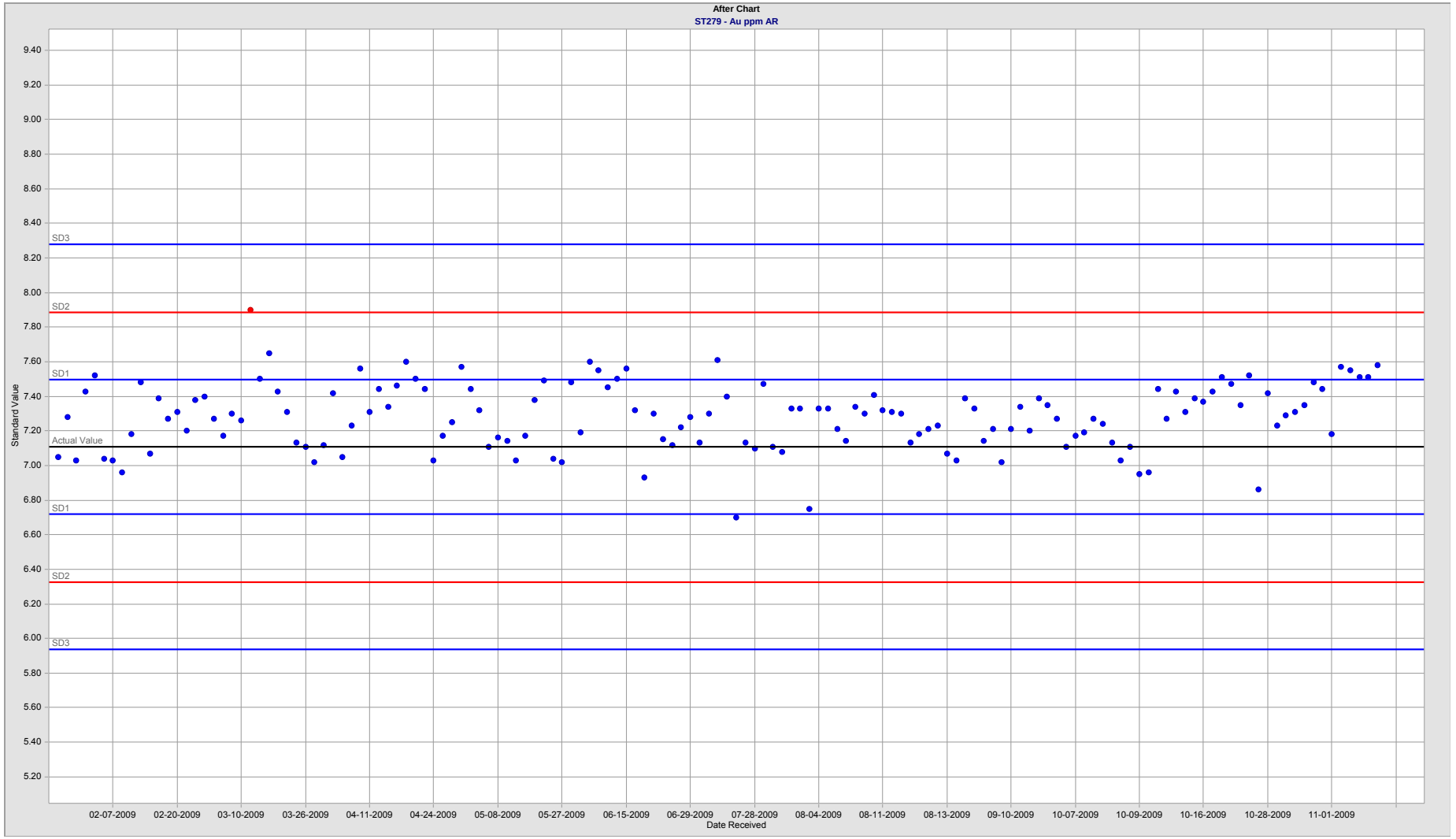


Figure 14.62
Control Chart for ST279 (Au ppm AR)



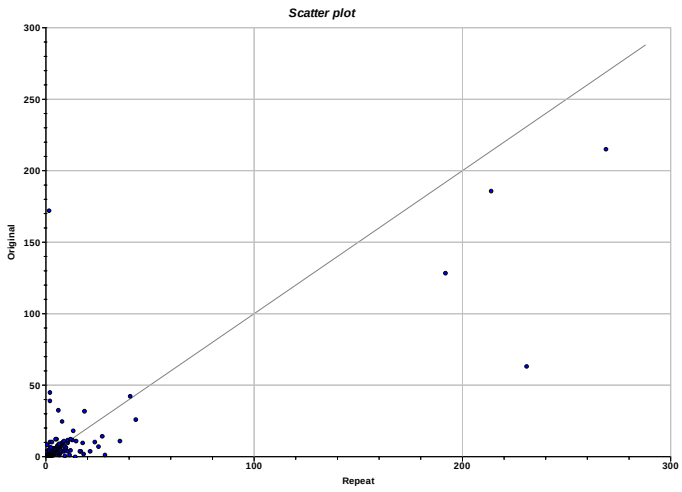
14.5.5 Field Duplicates, Diamond

Figures 14.63, 14.64 and 14.65 show scatter, quantile-quantile and precision plots diamond core duplicate sampling. There are 1694 pairs of data available. The means of the two datasets are within 8%; 1.315g/t Au versus 1.42g/t Au, Original and Duplicate, respectively, is affected by the variance of between high grade samples. The quantile-quantile plot shows that the marginal distributions are comparable with only a slight departure from the 45º line for the few highest grade pairs. The overall result is acceptable

14.5.6 Field Duplicates, RC

Figures 14.66, 14.67 and 14.68 show the scatter, quantile-quantile and precision plots for all available RC duplicate sampling. There are 108 pairs of data available. The means of the two data are the same (at 0.075g/t Au versus 0.065g/t Au, original and duplicate, respectively). The quantile-quantile plot shows that there is no bias towards either sample type, The overall result is acceptable, which is better than measured for the core duplicate sampling.

Figure 14.63
Scatter Plot, Original vs Duplicate Samples, Diamond Core

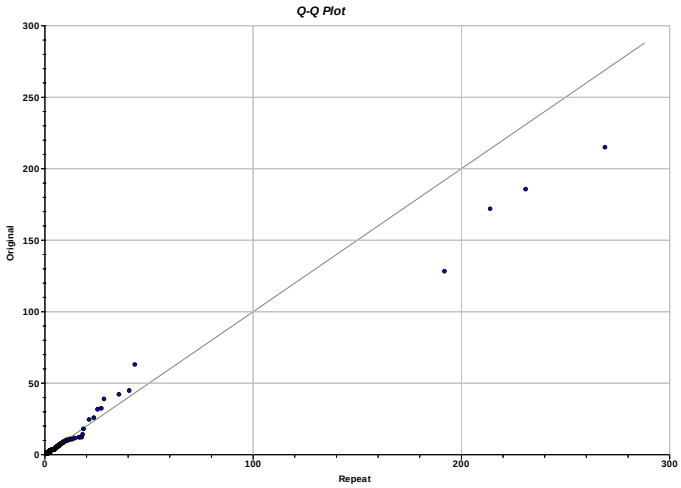


Data Statistics

Variable:	Repeat	Original
Weight by:	--	--
Mean:	1.420	1.315
Variance:	129.712	83.861
CV:	8.018	6.963
Minimum:	0.001	0.001
Q1:	0.006	0.005
Median:	0.099	0.110
Q3:	0.626	0.673
Maximum:	269.000	215.000
IQR:	0.620	0.668
Covariance:	84.930	
Pearson:	0.814	
Spearman:	0.963	
No. of Data:	1693 / 1694	

(Data are sub-setted)

Figure 14.64
QQ Plot, Original vs Duplicate Samples, Diamond Core

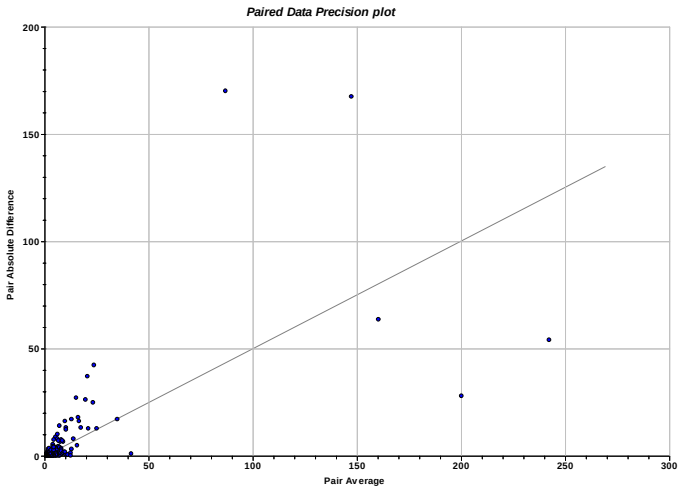


Data Statistics

Variable:	Repeat	Original
Weight by:	--	--
Mean:	1.420	1.315
Variance:	129.712	83.861
CV:	8.018	6.963
Minimum:	0.001	0.001
Q1:	0.006	0.005
Median:	0.099	0.110
Q3:	0.626	0.673
Maximum:	269.000	215.000
IQR:	0.620	0.668
Covariance:	84.930	
Pearson:	0.814	
Spearman:	0.963	
No. of Data:	1693 / 1694	

(Data are sub-setted)

Figure 14.65
Precision Plot, Original vs Duplicate Samples, Diamond Core

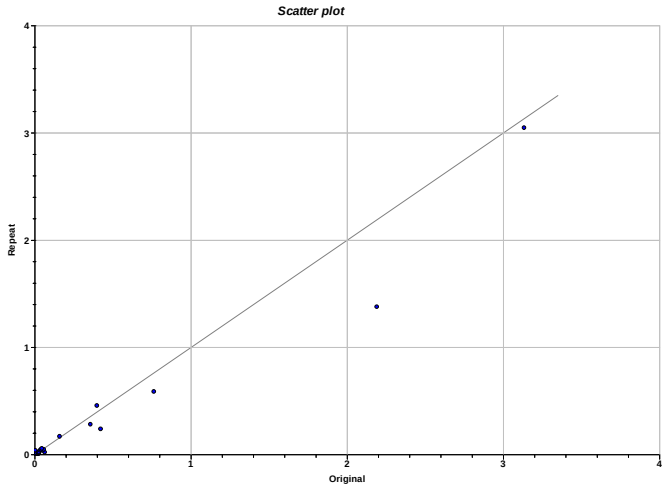


Data Statistics

Variable:	Repeat	Original
Weight by:	--	--
Mean:	1.420	1.315
Variance:	129.712	83.861
CV:	8.018	6.963
Minimum:	0.001	0.001
Q1:	0.006	0.005
Median:	0.099	0.110
Q3:	0.626	0.673
Maximum:	269.000	215.000
IQR:	0.620	0.668
Covariance:	84.930	
Pearson:	0.814	
Spearman:	0.963	
Precision:	+/-50%@ 67%CI	
No. of Data:	1693 / 1694	

(Data are sub-setted)

Figure 14.66
Scatter Plot, Original vs Duplicate Samples, RC

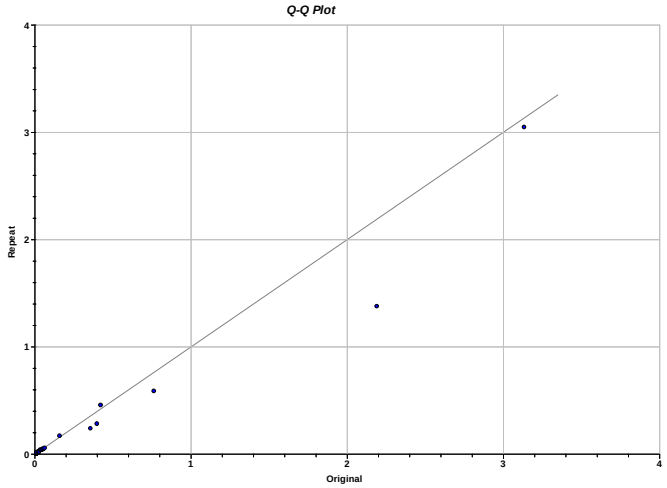


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	0.075	0.065
Variance:	0.140	0.107
CV:	4.961	5.058
Minimum:	0.001	0.001
Q1:	0.001	0.002
Median:	0.003	0.004
Q3:	0.007	0.008
Maximum:	3.130	3.050
IQR:	0.006	0.006
Covariance:	0.120	
Pearson:	0.982	
Spearman:	0.804	
No. of Data:	108	

(Data set at full limits)

Figure 14.67
QQ Plot, Original vs Duplicate Samples, RC

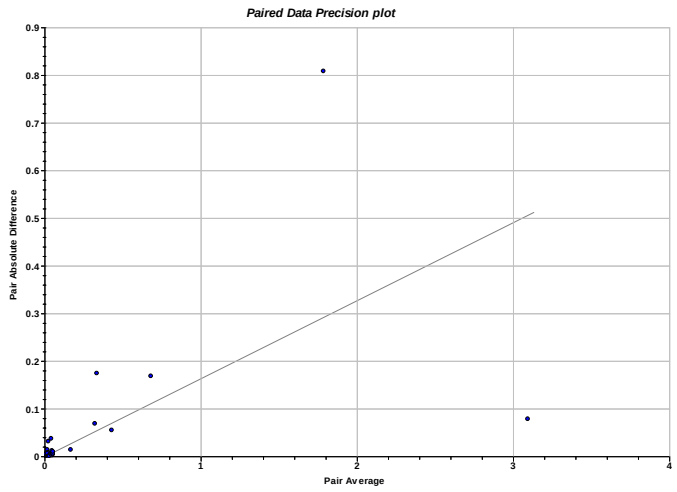


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	0.075	0.065
Variance:	0.140	0.107
CV:	4.961	5.058
Minimum:	0.001	0.001
Q1:	0.001	0.002
Median:	0.003	0.004
Q3:	0.007	0.008
Maximum:	3.130	3.050
IQR:	0.006	0.006
Covariance:	0.120	
Pearson:	0.982	
Spearman:	0.804	
No. of Data:	108	

(Data set at full limits)

Figure 14.68
Precision Plot, Original vs Duplicate Samples, RC



Data Statistics

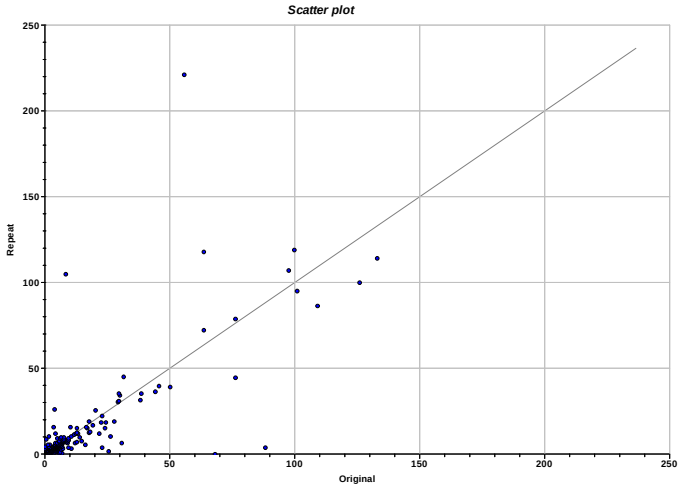
Variable:	Original	Repeat
Weight by:	--	--
Mean:	0.075	0.065
Variance:	0.140	0.107
CV:	4.961	5.058
Minimum:	0.001	0.001
Q1:	0.001	0.002
Median:	0.003	0.004
Q3:	0.007	0.008
Maximum:	3.130	3.050
IQR:	0.006	0.006
Covariance:	0.120	
Pearson:	0.982	
Spearman:	0.804	
Precision:	+/-16% @ 35%CI	
No. of Data:	108	

(Data set at full limits)

14.5.7 Re-assaying of Crushed Material

As a check of the repeatability of samples split at the first stage of the sample preparation, a suite of second splits from the crushed material has been assayed and are able to be compared to the original assay. Figures 14.69 and 14.70 show the scatter and quantile-quantile plot of these data pairs. The mean grades of the two datasets are 1.632g/t Au and 1.592g/t Au, original and crushed grade, respectively. The means are within 2% and the scatter shows there is no bias towards either dataset and the quantile-quantile plot shows a minor global bias to the original dataset below 70g/t Au and a minor global bias to the repeat data set above 70g/t Au. This result indicates that the crushing and splitting stage is performing well at the sample preparation facility at Sukari.

Figure 14.69
Scatter Plot, Original vs Resplit crushed Material

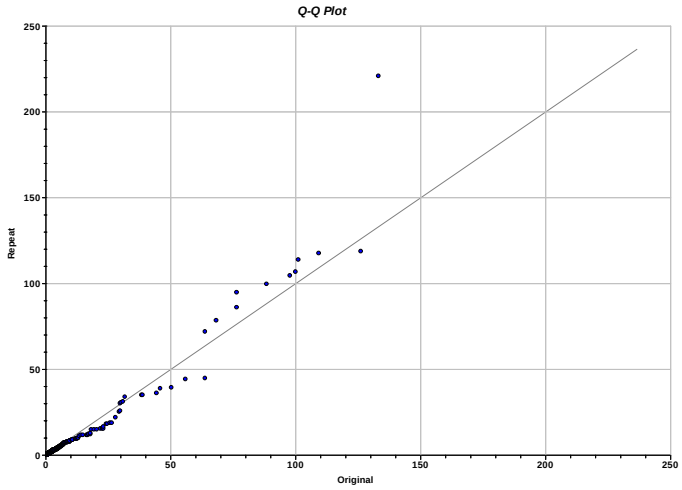


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	1.632	1.592
Variance:	63.975	81.255
CV:	4.902	5.664
Minimum:	0.001	0.001
Q1:	0.004	0.008
Median:	0.084	0.088
Q3:	0.747	0.700
Maximum:	133.000	221.000
IQR:	0.743	0.692
Covariance:	58.565	
Pearson:	0.812	
Spearman:	0.964	
No. of Data:	2099	

(Data set at full limits)

Figure 14.70
QQ Plot, Original vs Resplit crushed Material



Data Statistics

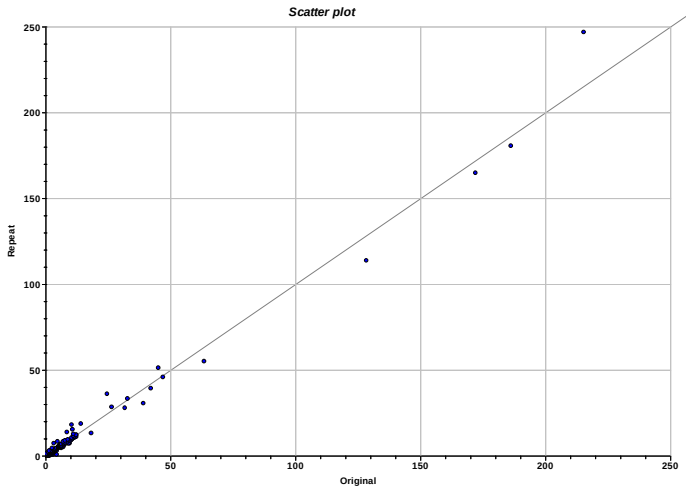
Variable:	Original	Repeat
Weight by:	--	--
Mean:	1.632	1.592
Variance:	63.975	81.255
CV:	4.902	5.664
Minimum:	0.001	0.001
Q1:	0.004	0.008
Median:	0.084	0.088
Q3:	0.747	0.700
Maximum:	133.000	221.000
IQR:	0.743	0.692
Covariance:	58.565	
Pearson:	0.812	
Spearman:	0.964	
No. of Data:	2099	

(Data set at full limits)

14.5.8 Duplicate Assaying of Pulverised Material

As a check of the repeatability of the pulp samples produced by the site sample preparation facility, a suite of duplicates from the pulverised sample have been assayed and are able to be compared to the original assay. Figures 14.71 and 14.72 show the scatter and quantile-quantile plot of these data pairs. The mean grades of the original and duplicate pulverised samples are similar (1.261g/t Au and 1.278g/t Au, respectively) and the quantile-quantile plot shows that there is no bias towards either assay. The result indicates that pulverising and splitting stage is performing well at Sukari.

Figure 14.71
Scatter Plot, Original vs Resplit pulverised Material

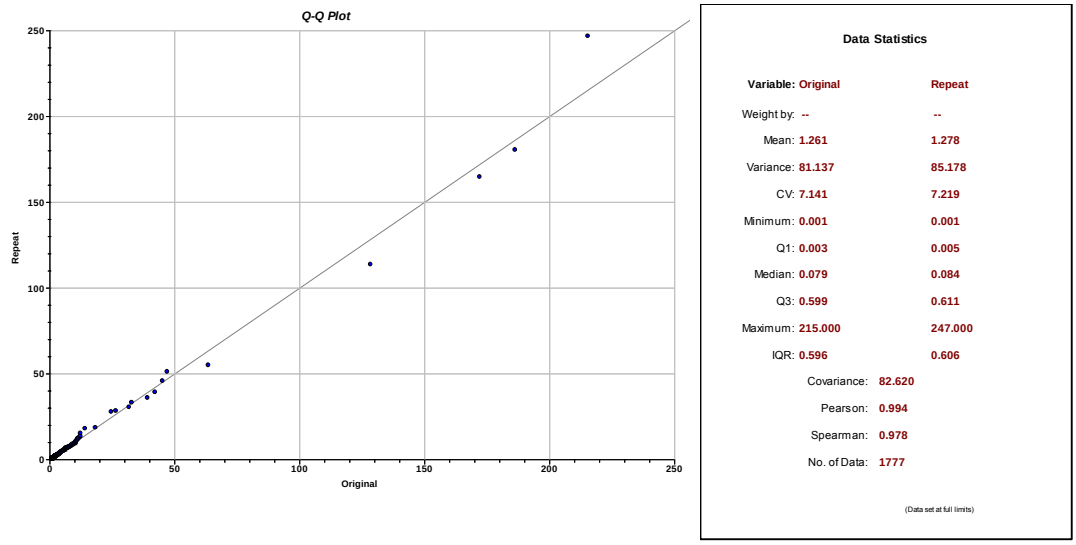


Data Statistics

Variable:	Original	Repeat
Weight by:	--	--
Mean:	1.261	1.278
Variance:	81.137	85.178
CV:	7.141	7.219
Minimum:	0.001	0.001
Q1:	0.003	0.005
Median:	0.079	0.084
Q3:	0.599	0.611
Maximum:	215.000	247.000
IQR:	0.596	0.606
Covariance:	82.620	
Pearson:	0.994	
Spearman:	0.978	
No. of Data:	1777	

(Data set at full limits)

Figure 14.72
QQ Plot, Original vs Resplit pulverised Material



14.6 Conclusions

The current QAQC systems in place at Sukari to monitor the precision and accuracy of the sampling and assaying are very good. Some conclusions from the analysis of the available QAQC data are:-

- Sample recoveries of the diamond and reverse circulation ('RC') drilling are very good, being 96.8% and 89.3%, respectively. The result for RC samples is particularly noteworthy as this is higher than would normally be expected for this type of sampling method.
- Repeat analyses have continued to confirm that the precision of sampling and assaying is within acceptable limits for sampling of gold deposits.
- Assaying by an alternative method has continued to confirm that the principal assay method is accurate.

15 ADJACENT PROPERTIES

No significant mining or exploration activity surrounds the Sukari Gold Project.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Introduction

The metallurgical behaviour of Sukari ores has been established through various test work programmes dating back to 2000 performed variously by Lakefield Oretest Pty Ltd, Independent Metallurgical Testing Laboratories and AMMTEC. Definitive testwork was conducted, under Ausenco supervision, at the AMMTEC laboratory in Perth during 2006. Comminution, flotation and cyanidation of flotation concentrate and tailings were the dominant processes tested.

The ore is relatively hard and competent being hosted in porphyry and is suitable for SAG milling. The gold is fine and associated with pyrite which is readily floated at coarse grind size, to yield a sulphide preconcentrate at high gold recovery and ultra fine grinding renders the gold in concentrates amenable to cyanidation.

Oxidised material from near surface yields acceptable gold recovery at coarse grind sizes, via direct cyanidation.

16.2 Metallurgical Sampling

Classification of ore types for the definitive Ammtec program was based on the degree of weathering (that correlated with the degree of oxidation of sulphide mineralisation), with highly weathered ore referred to as “Oxide ore” (M5) and primary, fresh ore as “Sulphide ore” (M1). This method of classification provided a suitable basis for the investigation and prediction of gold recovery.

The ore is hosted in porphyritic rock and gold tends to occur as fine inclusions in the sulphide minerals – mostly pyrite. Where weathering has occurred, the sulphide minerals have generally been broken down allowing access of cyanide solution to the contained gold and rendering the ore relatively free milling. The sulphur grade decreases from M1 to M5, consistent with the increasing proportion of oxide mineralisation. The gold grade also decreases from M1 to M5, which reflects a general trend of lower grade ore closer to surface. Approximately 90% of the ore deposit has been classified as M1 or M2.

The definition and collection of samples for the Independent Metallurgical laboratory metallurgical testwork program was carried out in April and May 2004. Ausenco provided the requirements for the samples and the six sample sets were collected by PGM as summarised in Table 16.2_1.

Table 16.1

Metallurgical Testwork Samples

Set	Description	Weight (kg)	Grade(g/t)	Core Metres
A	Floatation and Cyanidation	736.4	1.73	1148.42
B	Comminution	149	N/A	N/A
C1	Weathered	66	1.57	156
C2	Hi Silica	158.5	1.63	224
C3	Brecciated	57	1.69	100
D1	Shore Seawater	50 litres	N/A	N/A
D2	Near Shelf Seawater	50 litres	N/A	N/A

Care was taken to ensure the samples taken were representative and over the full strike length, as is illustrated in Figure 16.1 of a long section of the optimised pit as it was then.

Additional sample was obtained from site for the definitive AMMTEC program:

Weathering Type

- M1 – Sulphide Composite 13.5 Kg
- M2 – Mixed Composite 59.2 Kg
- M3 – Mixed Composite 27.1 Kg
- M4 – Mixed Composite 6.0 Kg
- M5 – Oxide Composite 182.7 Kg

Variability Samples

- By sulphur grade
- By Gold grade
- By Location in orebody

The sample classifications and head assays are summarised in Table 16.2.

Figure 16.1
Long Section Displaying the metallurgical sample intervals in relation to the optimised pit shell (blue) and topography (green).

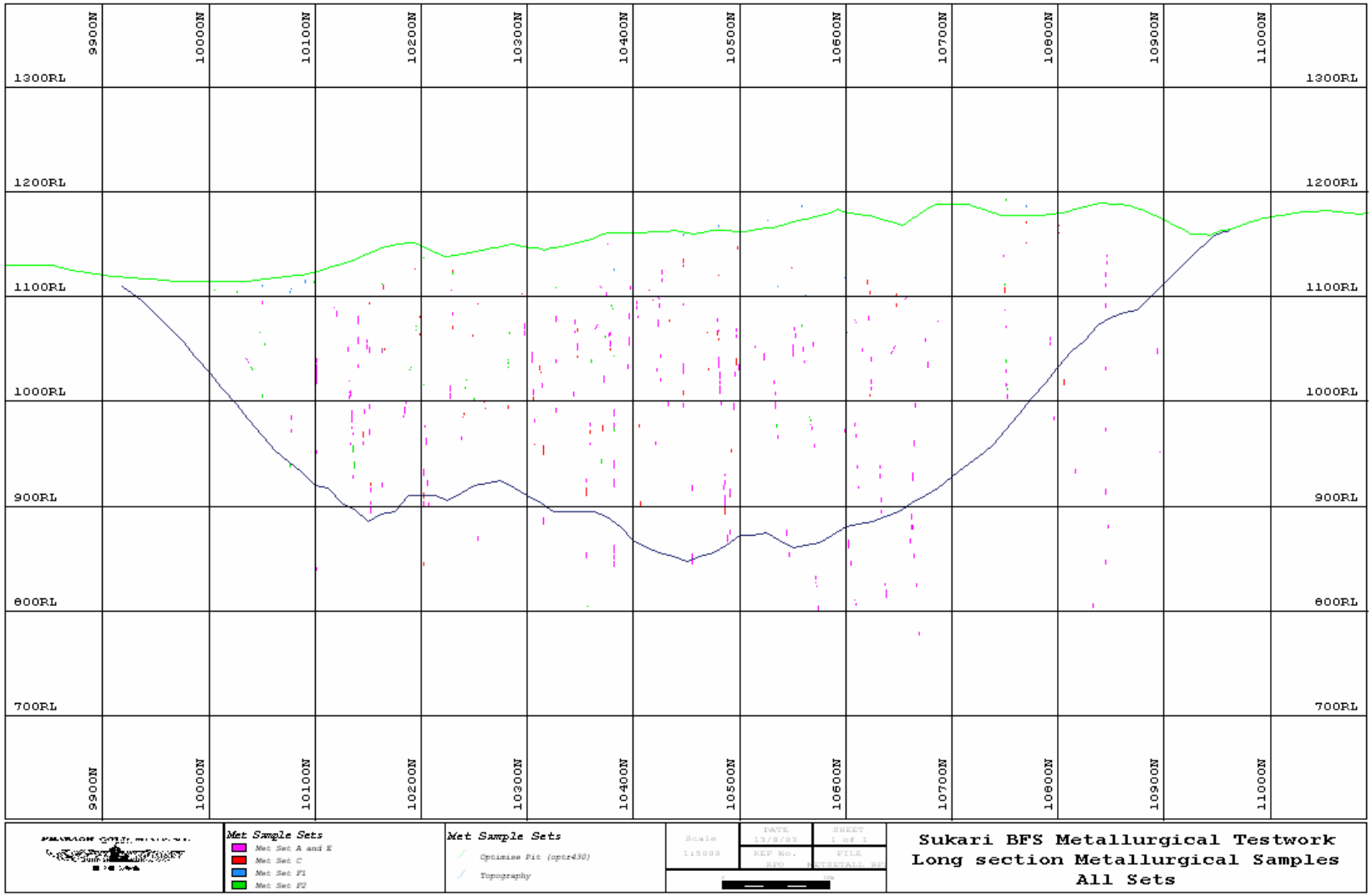


Table 16.2

Mineralisation Composite Head Assays

Sample Identification	Description	Laboratory	Gold Au g/t	Silver Ag g/t	Arsenic As ppm	Total Sulphur %S	Sulphide Sulphur %S	Total Carbon %C	Organic Carbon %C	Copper Cu g/t	Mercury Hg g/t	Iron % Fe
Met Set A	Met Composite (Fresh + Slightly Weathered)	IML	1.68	<0.5	988	1.10	1.08	0.79	0.067	10	<20	2.9
B	Comminution Testwork	IML	>0.5									
C1	Weathered (Variability)	IML	1.57									
C2	Highly Siliceous (Mostly Fresh)	IML	1.70	0.5	968	1.10						2.9
C3	Brecciated (Fresh & Weathered)	IML	1.69									
D1 & 2	Rea Sea Water	IML										
E1	Veined Mineralisation (Fresh & Weathered)	IML	1.72									
E2	Stockwork Mineralisation (Fresh & Weathered)	IML	1.71									
M1	Fresh, Sulphide	AMMTEC	1.74	<1	1080	0.91	0.85	0.39	0.04		2.63	
M2	Mixed (>75% Sulphide)	AMMTEC	1.46	<1	544	0.62	0.57	0.34	<0.03		2.80	
M3	Mixed (25% - 75% Sulphide)	AMMTEC	1.22	<1	376	0.43	0.35	0.19	<0.03		2.70	
M4	Mixed (<25% Sulphide)	AMMTEC	1.28	<1	260	0.22	0.12	0.04	<0.03		2.70	
M5	Oxidised	AMMTEC	1.13	<1	292	0.08	0.05	0.17	<0.03		2.70	
Comp 1	Lakefield Oretest #8363	Lakefield	1.81	<5	2470	0.97						
Comp 1		Lakefield	1.61	<2		1.27						
Comp 2A		Lakefield	2.43	<2		1.56						
Comp 2B		Lakefield	2.45	<2		1.24						
Comp 3	Comminution	Lakefield				0.73						
Comp 3B		Lakefield	2.15	<2		1.01						

16.3 Metallurgical Testwork

16.3.1 Process Water

Due to the very low rainfall and the lack of reliable replenishment of ground water, water will be pumped from the Red Sea, 25km away, for use in the process plant. Perth sea water was used for the testwork programme after an analysis concluded that there was little difference in metallurgical performance. Perth sea water has lower Total Dissolved Solids ('TDS'), chloride, and calcium concentration compared to Red Sea water. Buffering tests on the Perth and Red Sea water samples indicate that there is little difference in the buffering behaviour of Perth sea water compared to Red Sea water, as would be expected from the similar magnesium levels. Buffering tends to occur at a pH of 10.3. It was recommended that cyanidation be conducted at a pH of 9.5 to 10 to avoid excessive lime consumption.

16.3.2 Comminution

The results of comminution testwork are summarised in Table 16.3 and indicate that the ore is competent, resistant to breakage by impact and is abrasive. An assessment of the behaviour of the Sukari ore in the comminution circuit from Kori Kollo was undertaken by Dr Steve Morrell of SMCC Pty Ltd. It was concluded that the SAG mill was the throughput limiting item of equipment and that a pebble crusher was required. The ball mills have sufficient capacity to produce a finer grind if required.

Table 16.3
Comminution Test Results

Description	UCS MPa	Bond Rod Mill Work Index kWh/t	Bond Ball Mill Work Index kWh/t	Abrasion Index	Drop Weight Test
July 2000	16.2 – 93.1	-	19.7	0.42	-
October 2000	142 - 163	-	18.9	-	-
October 2001	64 – 105	18.1	17.4	0.49	-
Met Set B	-	22.4	17.6	0.683	A=77.26 b=0.49 t_a=0.26
Weathered C1	-	19.9	16.2	0.525	-
Siliceous 1 C2A	-	23.2	18.5	0.541	-
Siliceous 2 C2B	-	23.2	18.7	0.508	-
Brecciated C3	-	22.1	18.2	0.660	-
Barren Host Rock C4	-	23.9	19.3	0.566	-

The 85th percentile Bond ball mill work index of 19.1kWh/t and 70th percentile Bond rod mill work index of 23.2kWh/t were used as the basis for calculation of grinding circuit throughput. For oxide ore treatment, the weathered ore ball mill work index of 16.2kWh/t was used. Note also that the results are highly consistent across the ore types and indicate that the ore body has a low variation in hardness and abrasion characteristics.

Desktop reviews and modelling of the comminution circuit efficiency were completed in 2010 by Ausenco Pty Ltd (in conjunction with Dr. Steve Morrell of SMCC Pty Ltd), Orway Mineral Consultants and Senet Projects Ltd to identify the potential improvement in the grinding circuit throughput with the introduction of secondary crushing.

The reviews consistently concluded that the existing SAG circuit is capable of processing 5 Mt/y (625 tph) when provided with a secondary crushed feed (F_{80}) of 50 mm.

Description	Circuit ID	Existing circuit capacity 2nd crush Stage3
Throughput	Mt/y	5
Throughput	t/h	625
Crusher/s	#	1
SAG mill/s	#	1
HPGR	#	
Ball mill/s	#	2
Bond baseline energy	kWh/t	14.5
Circuit eff factor	**factor**	**1.22**
Ausenco method (eff. Fact)	kWh/t	17.7
Morrell's Wt method	kWh/t	17.4
JKSimMet - method	kWh/t	18.1
Ave. circuit specific energy	**kWh/t**	**17.7**
Standard deviation		0.35
Required circuit pinion power	MW	11.1

16.3.3 Gravity Separation

Gravity gold recovery using centrifugal concentrators in the laboratory on Met Set A (IML) yielded gold recoveries of 11.2% to 18.6%. A sample of ore containing coarse gold and with a head grade of 20g/t earlier gave a gravity recovery of 87% for gold and 62% for sulphur with a mass pull of 3.8%. This indicates that some of the gold is free and liberated in very high grade areas, however the occurrence of high grade ore is rare and will occur at depth.

It was therefore concluded that space should be allowed for retrofit of a gravity section within the grind circuit as the pit develops to depth.

16.3.4 Whole of Ore Cyanidation

A summary of whole of ore cyanidation gold recoveries is given in Table 16.4.

It was apparent from early testwork at IML that Sulphide ore types (Met Set A) would require fine grinding to achieve acceptable gold recovery. This led to the route of preconcentration of gold by flotation. However, direct cyanidation of Oxide ore (Weathered C1) yielded acceptable recovery at coarse grind sizes. Optimisation work at AMMTEC on M5 Oxide material defined cyanide extraction at 92.9 % Au at the design grind size.

Reagent consumptions were 1.7kg/t NaCN and 3.3 k/t lime.

Table 16.4

Comparison of CIL and Direct Cyanidation Test Results

Sample	Laboratory	Test	Grind Size P_{80} µm	Calculated Head g/t Au	Gold Extraction %
Met Set A	IML	Direct CN	90	-	73.1
Met Set A	IML	Direct CN	75	-	79.5
Met Set A	IML	Direct CN	53	-	81.5
Met Set A	IML	CIL	75	1.67	76.6
Met Set A	IML	CIL	63	1.50	74.0
Met Set A	IML	CIL	53	1.69	75.5
Met Set A	AMMTEC	CIL	63	2.08	88.4
Met Set A	AMMTEC	Direct CN	63	2.02	87.3
Weathered C1	IML	CIL	150	1.19	85.7
Weathered C1	IML	CIL	75	1.18	83.9
Weathered C1	AMMTEC	Heap Leach	3350		88.6
M3	AMMTEC	Direct CN	106		85.6
M3	AMMTEC	Direct CN	150		86.5
M3	AMMTEC	Direct CN	250		84.7
M3	AMMTEC	Heap Leach	3350		66.3
M5 Oxide	AMMTEC	Direct CN	106		93.6
M5 Oxide	AMMTEC	Direct CN	150		92.9
M5 Oxide	AMMTEC	Direct CN	250		90.7
M5 Oxide	AMMTEC	Heap Leach	3350		83.9

16.3.5 Flotation

Flotation testwork results consistently showed a high gold recovery to flotation concentrate, at low concentrate mass pull, and at natural pH. Table 16.5 summarises the test details and results. Gold recovery was consistent with mineralisation type, with the highest recovery achieved for samples of type M1 (fresh sulphide) and the lowest recovery for type M5 (oxidised).

Table 16.5
Flotation Test Summary

Sample	Grind Size P_{80} µm	Head Grade g/t Au	Mass Pull %	Gold Recovery to Concentrate %	Sulphur Recovery %
IML Met Set A T5	212	1.68	7.2	96.8	
IML Met Set A T2	175	1.68	7.1	96.3	
IML Met Set A T3	150	1.68	7.8	97.4	
IML Met Set A T4	125	1.68	7.5	97.8	
IML C1A Highly Weathered		1.55	5.3	72.5	
AMMTEC M1 Fresh	150	1.95	5.9	96.6	97.9
AMMTEC M2	150	1.16	4.1	80.6	93.4
AMMTEC M1+M2 50:50	150	1.39	4.2	76.6	95.5
AMMTEC M3	150	0.89	3.7	74.6	92.4
AMMTEC M4	150	1.06	3.1	66.5	68.5
AMMTEC M5 Oxide	150	0.96	3.6	65.4	34.9

The average mass pull for the tests was 5.0%. Ausenco determined that the 90th percentile value for the sulphur head grade based on variability samples was 1.8% sulphur. The plant's concentrate circuit will therefore be designed for 8% mass pull, to allow for short term increases in the sulphur feed grade to the plant.

Ausenco conducted an economic evaluation of the anticipated revenues and selected an optimum grind of P_{80} = 175 µm for flotation. A grind size P_{80} of 150 µm was subsequently used for flotation testwork at AMMTEC and as the design basis as it was a standard size fraction.

A test was carried out to compare a combined gravity then flotation flowsheet, with flotation only. The calculated overall gravity concentrate plus flotation concentrate gold recovery was 96% indicating that there was no apparent benefit by including the gravity step prior to flotation, compared with a flotation only process route.

The flotation testwork indicated that there was no benefit in addition of copper sulphate as an activator for the sulphides, however, given that residual cyanide may contaminate the process water, provision will be made to retrofit a copper sulphate mixing, storage and distribution system. The optimum reagent additions were found to be 75g/t potassium amyl xanthate (collector), 50g/t A404 (collector/promoter particularly for gold), 20g/t Interfroth IF20 (or MIBC) frother.

Sulphur recovery for all flotation optimisation tests was consistently high at >96%. However, a low gold recovery of 88.8% was obtained from flotation of the lowest grade (0.5%) sulphur variability sample. Arsenic recovery ranged from 95% to 96%, while iron recovery ranged from 39% to 48%. These values support the recovery of gold associated predominantly with pyrite into flotation concentrate, together with arsenopyrite.

A lower gold recovery of 91.7% was obtained from flotation of the lowest grade (0.75g Au/t) gold sample. Other low grade gold samples, had average gold recoveries averaging 95.1%.

16.3.6 Cyanidation - Flotation Tailing

The gold assay of flotation tailings for primary Sulphide samples averaged less than 0.1g/t Au. Consequently, for Sulphide ore (M1), the flotation tail can be discarded directly to final tailing without incurring the costs associated with cyanidation. However, flotation tests on the Weathered (M2, M3 & M4) composite samples gave lower flotation recoveries with a flotation tailing grade of approximately 0.4g Au/t, which was considered sufficiently high to consider tailings cyanidation. The Oxidised (M5) composite gave superior gold recovery through direct cyanidation, rather than Flotation with separate cyanidation of float concentrates and float tail.

AMMTEC performed direct cyanidation tests on the flotation tailings from the mineralisation composite flotation tests. The flotation tailings gold extraction results, at 12 hours and 24 hours, are listed in Table 16.6.

Table 16.6
Flotation Tails Cyanidation Results

Test	Sample	Extraction at 12 hours, %	Extraction at 24 hours, %
RG7494	M1	67.0	67.0
RG7490	M2	73.0	77.4
RG7491	M3	72.5	72.5
RG7492	M4	61.4	64.2
RG7493	M5	70.2	73.4

The M2, M4 and M5 samples showed a slight increase in gold extraction between 12 and 24 hours. However, based on this insignificant benefit, a tails cyanidation residence time of 12 hours was selected by Ausenco.

The average lime consumption for the mineralisation composite and variability flotation tailings cyanidation tests was 0.85kg/t of hydrated lime with 84% $Ca(OH)_2$ (or 64% CaO), equivalent to 0.72kg/t for quicklime.

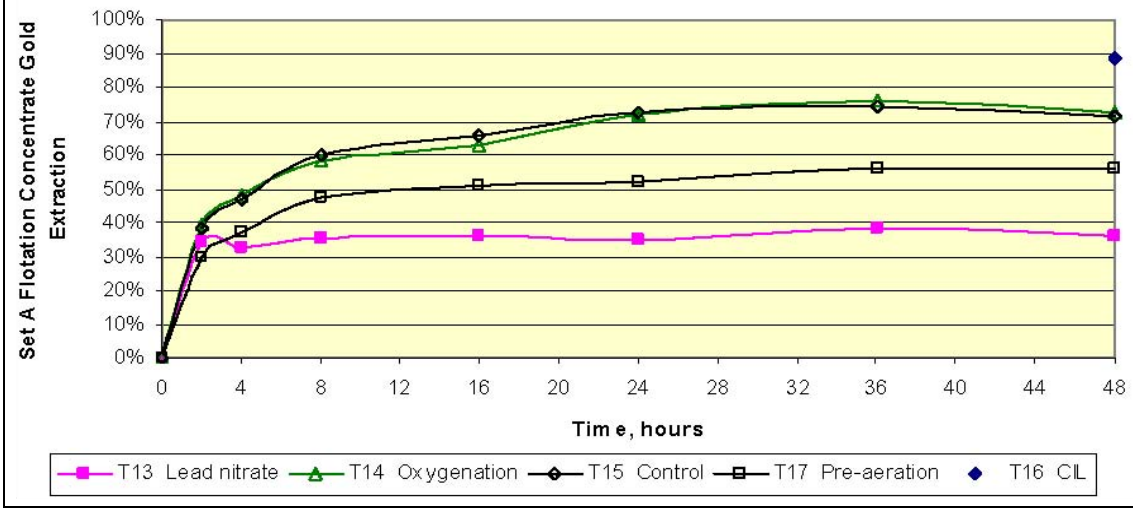
The average cyanide consumption in the flotation tailings cyanidation tests was 0.37kg/t. A residual cyanide concentration in flotation tailings leach tails was chosen as 150ppm to calculate the total cyanide addition. Overall cyanide addition was then calculated from these two values as 0.60kg NaCN/t.

16.3.7 Cyanidation - Flotation Concentrate

Most of the sulphur in the flotation concentrate is present as sulphide. Gold, silver, iron, copper, arsenic, and sulphur grades have been concentrated by flotation.

Results of the initial optimisation for flotation concentrate leaching are shown in Figure 16.2 with the CIL test having the highest gold extraction of 88.7% at 48 hours leach time.

Figure 16.2
Concentrate Cyanidation Optimisation



A slight reduction in gold extraction from 36 to 48 hours in the non CIL tests was attributed to mild "preg-robbing" (the re-adsorption of dissolved gold and silver onto a particular component of the leached solids) by Ausenco. This factor is also seen in the ultimate extraction of + 90% Au only being achieved in the CIL tests (c/f T15 versus T16 above). In practice, any tendency to Preg Robbing will be negated by the use of CIL leaching.

The results of grind optimisation tests using CIL showed a strong trend toward higher gold extraction at finer grinds.

Variability sample flotation concentrate CIL gold extraction ranged from 85.5% to 99.3% with an average of 95.1% for all samples. Gold extraction was consistent with mineralisation type, with the highest gold extraction found on M5 (oxide) samples and the lowest gold extraction found on the M1 (sulphide) samples.

The average lime consumption for the IML and AMMTEC concentrate cyanidation tests was 4.77kg/(t concentrate), based on hydrated lime with 84% $Ca(OH)_2$ content, as used in the laboratory. This is equivalent to 4.04kg/(t concentrate), based on quicklime having 75% contained CaO and 0.32kg/(t feed) based on a design concentrate mass pull of 8%.

The average cyanide consumption for the IML and AMMTEC concentrate cyanidation tests was 2.79kg/(t concentrate). A residual cyanide concentration in concentrate leach tails was chosen as 300ppm to calculate the cyanide addition rate. The overall cyanide addition was then calculated from these two values as 3.35kg NaCN/(t concentrate) or 0.27kg/(t feed) based on a design concentrate mass pull of 8%.

16.4 Testwork Summary

Definitive test work by AMMTEC on the five ore classifications provided a basis for prediction of gold recovery based on flotation response and resulted in allowance for three processing routes:-

- Stage 1 - Whole ore direct cyanidation of **"oxide ore"**. (M5).
- Stage 2 – Flotation of **"mixed ore"** with separate cyanidation of reground concentrate and flotation tail (M2, M3 and M4).
- Stage 3 – Flotation of **"sulphide ore"** with cyanidation of reground concentrate and discarding flotation tail. (M1).

Gold recoveries for the three process Stages were derived from the metallurgical test work, with adjustment for plant losses, and are shown in Table 16.7.

Table 16.7
Recoveries by Ore Type and Processing Stage

Ore Type	Stage 3 Float + Conc CIL	Stage 2 Float + Conc CIL + Float Tails CIL	Stage 1 Direct CIL
M1 - Sulphide	**89.7%**	92.2%	70%
M2, M3, M4 - Mixed	69.4%	**87.4%**	81.6%
M5 - Oxide	62.8%	88.8%	**90.8%**

Block flow diagrams depicting the process routes for each Stage are shown in Figures 16.3, 16.4 and 16.5 following:-

Figure 16.3
Process Block Flow Diagram – Stage 1 – Oxide Ore

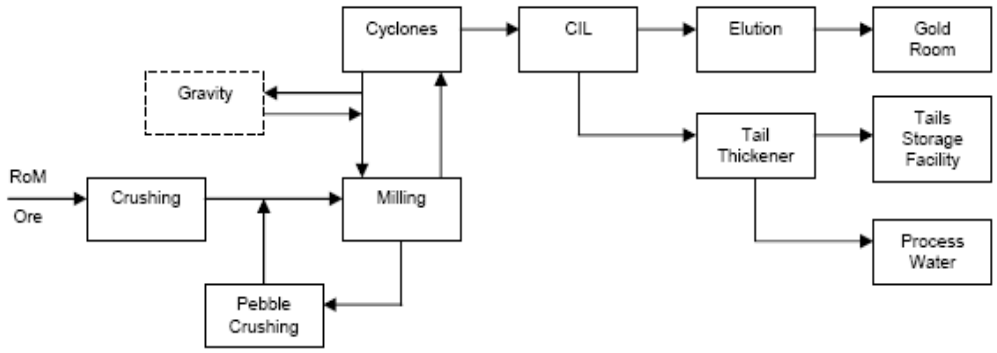


Figure 16.4
Process Block Flow Diagram – Stage 2 – Mixed Ore

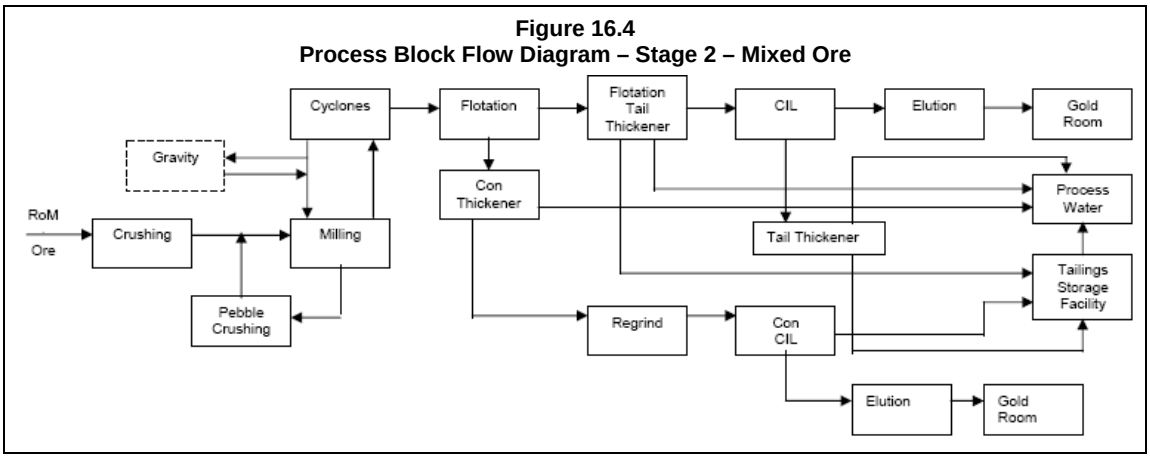
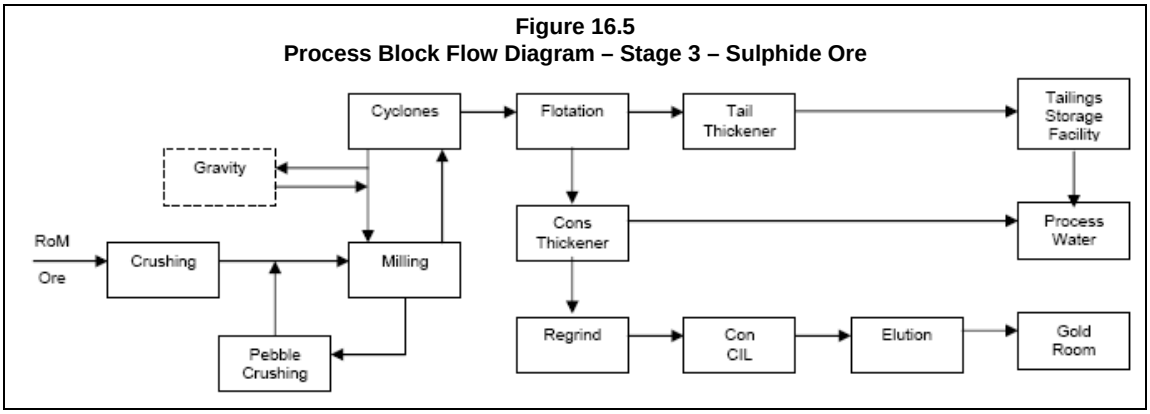


Figure 16.5
Process Block Flow Diagram – Stage 3 – Sulphide Ore



16.5 Process Plant

The process plant comprises coarse ore stockpile, milling and CIL circuits relocated from Kori Kollo combined with a new primary crusher, a new flotation, regrind and concentrate CIL circuit and new elution/gold room facilities for each of the CIL circuits.

The crushing and milling circuits are common to all three process Stages. The Kori Kollo CIL circuit is used to treat Stage 1 oxide ore and the Stage 2 flotation tail while the new flotation, regrind and concentrate CIL circuit will treat Stage 2 mixed ore concentrate and Stage 3 sulphide ore. The plant circuits are arranged and interconnected such that processing can be easily switched from one Stage route to another without the need for lengthy shutdown.

Because the plant throughput is limited by SAG milling capacity, SGM has initiated the installation of a secondary crushing circuit due for commissioning end Q1 2011. The secondary crushing circuit is designed to reduce the ore feed size to the SAG mill from the initial design of P_{80} of 105mm to a P_{80} of 50mm whereby improving the efficiency of the SAG mill and thus allowing an increase in the plant throughput rate from the initial design of 500tph to a new nominal 625tph. An additional allowance has been made in the design for possible future inclusion of a second comminution and flotation circuit. Allowance has also been made for future retro fit of a gravity circuit.

Table 16.8 provides a summary of the design criteria for each Stage from which equipment sizing and operating costs are derived.

Table 16.8

Summary of Process Plant Design Criteria

Criteria		Units	Stage 1 Value	Stage 2 Value	Stage 3 Value	Secondary Crushing	Source
Ore Throughput		Mt/y	4.6	4.0	4.0	5.0	PGM
		t/h	580	500	500	625	Calculated
Plant Availability	Design	%	91.3	91.3	91.3	91.3	Engineer
Gold Grade	Design	g Au/t	1.55	1.76	1.83	1.83	PGM
Physical Characteristics	BWI	kWh/t	16.2	19.1	19.1	19.1	Testwork
	RWI	kWh/t	23.2	23.2	23.2	23.2	Testwork
	CWI	kWh/t	6.9-35.4	6.9-35.4	6.9-35.4	6.9-35.4	Testwork
Flotation Grind Size	P_{80}	µm	150	150	150	150	Testwork
Concentrate Regrind Size	P_{80}	µm	N/A	12	12	12	Testwork
Overall Plant Recovery	Gold	%	90.8	87.4	89.7	89.7	Calculated
Flotation Tails CIL Residence Time		hours	N/A	12	N/A	N/A	Calculated
Flotation Conc CIL Residence Time		hours	N/A	33.0	33.2	26.6	Calculated
Collector Addition		g/t	N/A	75	75	75	Calculated
Promoter Addition		g/t	N/A	50	50	50	Testwork
Frother Addition		g/t	N/A	20	20	20	Testwork
Flocculant Addition		g/t	18	18	18	18	Engineer
Max. Cyanide Consumption		kg/t	1.74	2.96	2.8	2.8	Testwork
Max. Lime Consumption		kg/t	3.3	3.36	4.13	4.13	Testwork

Major equipment items selected for the plant unit operations are listed in Table 16.9.

Table 16.9

Major Equipment Items

Description	Type	Source
Primary crusher	Gyratory, 54" – 74"	New
Secondary crusher	2 x Cone crusher CH870C	New
Milling circuit	SABC	Kori Kollo
SAG mill	8.53m Ø x 4.27m EGL, 5.6 MW	Kori Kollo
Ball mill	5.03m Ø x 9.29m EGL, 4.1 MW	Kori Kollo
Recycle crusher	Cone crusher HP500	New
Flotation cells (Stage 2 & 3)	6 x 100m^3 tank cells	New
Regrind mill (Stage 2 & 3)	1 x Vertimill 1250HP, 2 x SMD 355kW	New
Concentrate thickener	14m diameter	New
Leach feed thickener	23m diameter	Kori Kollo
Tails thickener	23m diameter	New
Concentrate CIL tanks: (Stage 2 & 3)	9 x 205m^3	New
Elution circuit	1 x 8t batch capacity pressure Zadra + 1 x 2t batch capacity pressure Zadra	New New
Oxide and Conc. Tail CIL Tanks	8 x 3250m^3	Kori Kollo

There are two separate gold room facilities associated with each Zadra elution circuit each comprising electrowinning cells and smelting furnace. The reasons for separating these facilities are the significant differential in required carbon handling capacity through each circuit and the security risk in transporting loaded carbon cathodes.

Because the plant throughput is limited by SAG milling capacity, allowance has been made in the design for possible future inclusion of a second SAG mill. Allowance has also been made for future retro fit of a gravity circuit.

Due to the lack of a local source of fresh water in the Sukari area, seawater will be used for process water and will be pumped a distance of 26km from the Red Sea. Storage capacity at the plant is provided in a lined pond of 30,500m3 capacity equivalent to two days at maximum demand. A Reverse Osmosis plant is provided to produce fresh water for the elution process and for domestic uses on site. The reject stream from the RO plant reports back to the seawater storage pond.

Since cyanide suppresses pyrite flotation, process water recovered from the CIL tailings thickener is kept separate to preclude its use in the milling and flotation circuits. A separate cyanide process water tank is provided for distribution within cyanide areas of the plant only.

Conventional facilities for storage and mixing of reagents including lime, sodium cyanide, flotation agents, flocculant, carbon and acid are provided.

Air compressors are included to provide low and high pressure air for flotation, CIL air lifts, plant utility and instrument air.

Plant controls will be through a new PLC and SCADA system. Belt weighers and magnetic flowmeters together with process samplers are provided for metallurgical accounting.

Power is distributed to area sub-stations at 11 kV where it is transformed to 4160v for the mill drives and 400v for the balance of drives. Electrical power systems are standard at 400v 50 Hz which coincides with the electrical system standards of the Kori Kollo plant.

17. MINERAL RESOURCES AND MINERAL RESERVES

17.1. Mineral Resources

17.1.1. Introduction

Stockwork gold deposits such as Sukari do not readily lend themselves to resource estimation by polygonal methods, nor by interpolation of block grades by methods such as ordinary kriging or inverse distance weighting into polygons (or wire-frames) interpreted around drill-hole assays. Experience has demonstrated that such methods normally under-estimate recoverable ore tonnages and overestimate recoverable grades.

H&S has for some years used a method of Multiple Indicator Kriging ('MIK') to estimate recoverable resources in gold and polymetallic deposits, with very good results. The method is particularly suited to estimation of resources in deposits such as Sukari.

It must be emphasised that recovery of resources modelled in the approach taken for Sukari relies upon the use of an optimum method of ore selection during mining. Use of a polygonal method to select ore and waste in mining at Sukari will almost certainly result in a shortfall of ore tonnes against the MIK model. H&S recommends the use of the MP© method for ore selection in mining (Schofield & Rolley, 1997).

17.1.2. Indicator Kriging for Recoverable Resource Estimation

The MIK method was developed in the early 1980's with a view toward addressing some of the problems associated with estimation of resources in mineral deposits. These problems arise where sample grades show the property of extreme variation and consequently where estimates of grade show extreme sensitivity to a small number of very high grades. These characteristics are typical of many lode gold deposits, where the coefficient of variation in samples normally exceeds 2.0. MIK is one of a number of methods which can be used to provide better estimates than the more traditional methods such as ordinary kriging and inverse distance weighting.

It is fundamental to the estimation of resources that the estimation error is inversely related to the size of the volume being estimated. To take the extreme case, the estimate of the average grade of a deposit generated from a weighted average grade of the entire sample dataset is much more reliable than the estimate of the average grade of a small block of material within the deposit generated from a local neighbourhood of data.

Another fundamental notion relevant to the optimisation of resources to develop an open pit mine and schedule is that the optimisation algorithm does not require the resource be defined on extremely small blocks relative to data spacing. Small blocks cannot provide the basis for reliable estimates of recoverable resources.

The basic unit of an MIK block model is a panel that normally has the dimensions of the average drillhole spacing in the horizontal plane. At Sukari, the average drillhole spacing is 20m in the grid east direction and 25m in the grid north direction. The panel should be large enough to contain a reasonable number of blocks, or Selective Mining Units (SMUs; about 15). The SMU is the smallest volume of rock that can be mined separately as ore or waste and is usually defined by a minimum mining width. At Sukari, the dimensions of this block are assumed to be in the order of 4mE x 8mN x 3mRL.

The goal of MIK is to estimate the tonnage and grade of ore that would be recovered from each panel if the panel were mined using the SMU as the minimum selection criteria to distinguish between ore and waste. To achieve this goal, the following steps are performed:-

1. Estimate the proportion of each domain within each panel. This estimation can be achieved by kriging of indicators of domain classifications of sample data points or by passing the model panels through wire-frame interpretations of domains and calculating domain proportions directly from the wireframes. Wire-frames have been used for the assigning domain proportions into panels for the Sukari resource model.
2. Estimate the histogram of grades of sample-sized units within each domain within each panel using MIK. MIK actually estimates the probability of the grade within each panel being less than a series of indicator threshold grades. These probabilities are interpreted as panel proportions.
3. For each domain, and for each panel that receives an estimated grade greater than 0.0g/t Au, implement a block support correction (variance adjustment) on the estimated histogram of sample grades in order to achieve a histogram of grades for SMU-sized blocks. This step incorporates an explicit adjustment for Information Effect.
4. Calculate the proportion of each panel estimated to exceed a set of selected cutoff grades, and the grades of those proportions.
5. Apply to each panel, or portion of a panel below surface, a bulk density to achieve estimates of recoverable tonnages and grades for each panel.

Apart from considerations of resource confidence classification (Section 17.1.11), Step 5 completes construction of the resource model. The estimates of recoverable resources for each panel may be combined to provide an estimate of global recoverable resources for the deposit.

17.1.3. The Resource Dataset

The resource dataset has been addressed in Section 14.1. In summary, assays were composited to 2m down-hole lengths with residual intervals less than 2.0m length being discarded. The resulting file contained 189,649 sample composites (including rock chip samples) with gold grades after trimming to the resource study area.

17.1.4. Geological Interpretation and Domaining of Data

Throughout the area drilled to date, the limits of the main Sukari porphyry body effectively delineate the limits of gold mineralisation. Samples in footwall and hangingwall rocks rarely assay greater than detection limit (0.02g/t Au) whereas most samples within the porphyry report some detectable level of gold. An exception is gold mineralisation in porphyry dykes to the east of the main porphyry body, intersected by drilling on sections between 10,275mN to 10,650mN and 10,900mN to 11,025mN. Drill coverage is insufficient to define the trend of these subsidiary porphyries but surface mapping indicates them to be discrete dykes, up to 20 metres wide, striking grid east-west.

For the purpose of resource estimation, six domains were interpreted:-

- ***Domain 1:*** A footwall waste domain, west of the main porphyry body.

- ***Domain 2:*** Oxide zone of the Southern Sukari porphyry, hosting the bulk of mineralisation defined by drilling. The western and eastern domain limits were normally digitised at logged geological contacts however, where mineralisation extends for short distances (rarely more than 2m to 3m) into footwall and hangingwall rocks, domain limits were expanded to include this mineralisation. The northern limit to the domain is defined by an approximately 50 degree dipping plane extending north from 10 950mN.
- ***Domain 3:*** Sulphide zone of the Southern Sukari porphyry.
- ***Domain 4:*** Oxide zone of the Northern Sukari porphyry, lower grade, domain in the Sukari porphyry, north of section 10,950mN. The justification for separating the main porphyry body into northern and southern modelling domains is based on a combination of observation made when viewing sections and plans through the resource data and differences seen in the univariate statistics (Section 17.1.6) between the two domains.
- ***Domain 5:*** Sulphide zone of the Northern Sukari porphyry,
- ***Domain 6:*** A domain encompassing mineralisation interpreted to relate to the Horus Zone.
- ***Domain 7:*** A domain encompassing mineralised intercepts in subsidiary porphyry dykes positioned in the hanging wall of the main porphyry body.
- ***Domain 8:*** Contains all remaining resource sample data not contained in domains 1 to 7.

Interpretation of domain boundaries utilised surface mapping and drillhole information. Domain boundaries were digitised on cross-sections, snapped to drillhole traces where appropriate, then wire-framed into three-dimensional solids. Views of domains are shown in Figures 17.1.4_1 to 17.1.4_7.

Figure 17.1.4_1
Footwall Waste

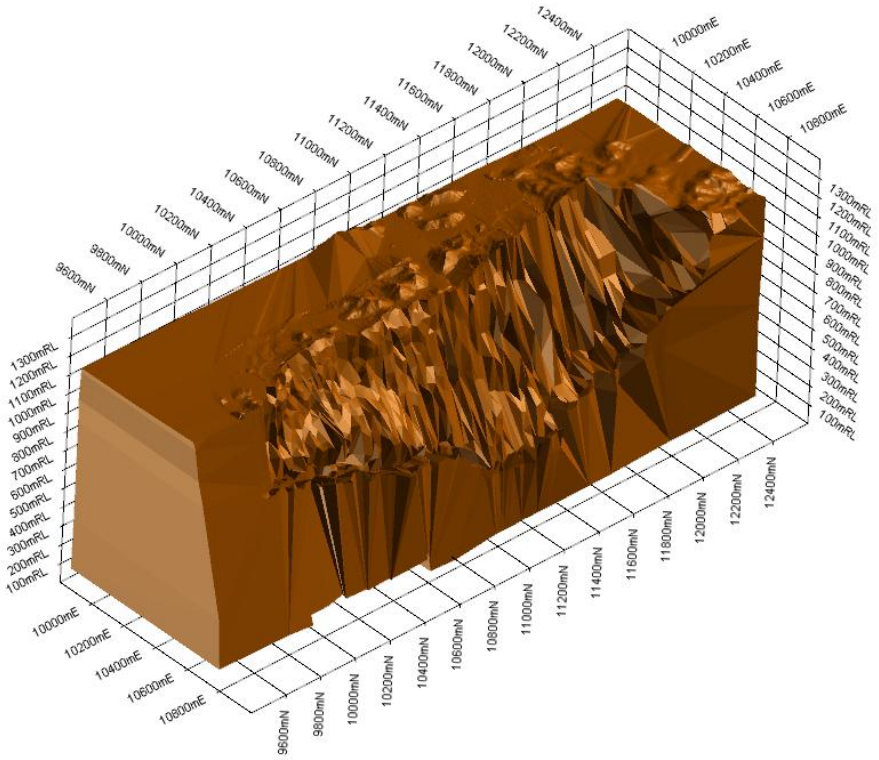


Figure 17.1.4_2
Domain 2, Oxide Zone Sukari Porphyry South

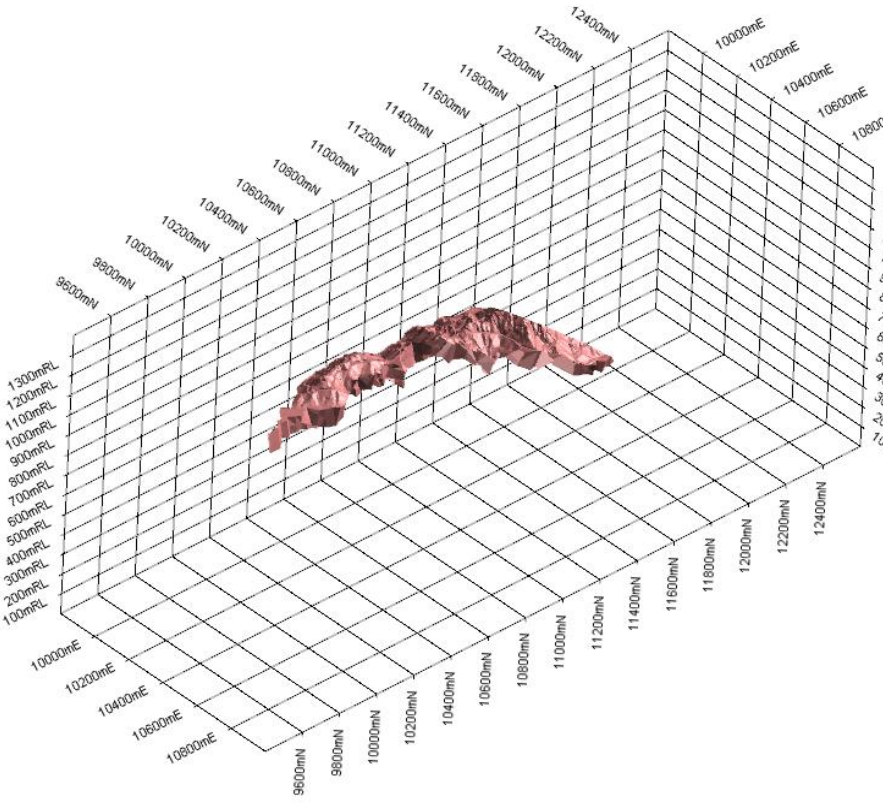


Figure 17.1.4_3
Domain 3, Sulphide Zone Sukari Porphyry South

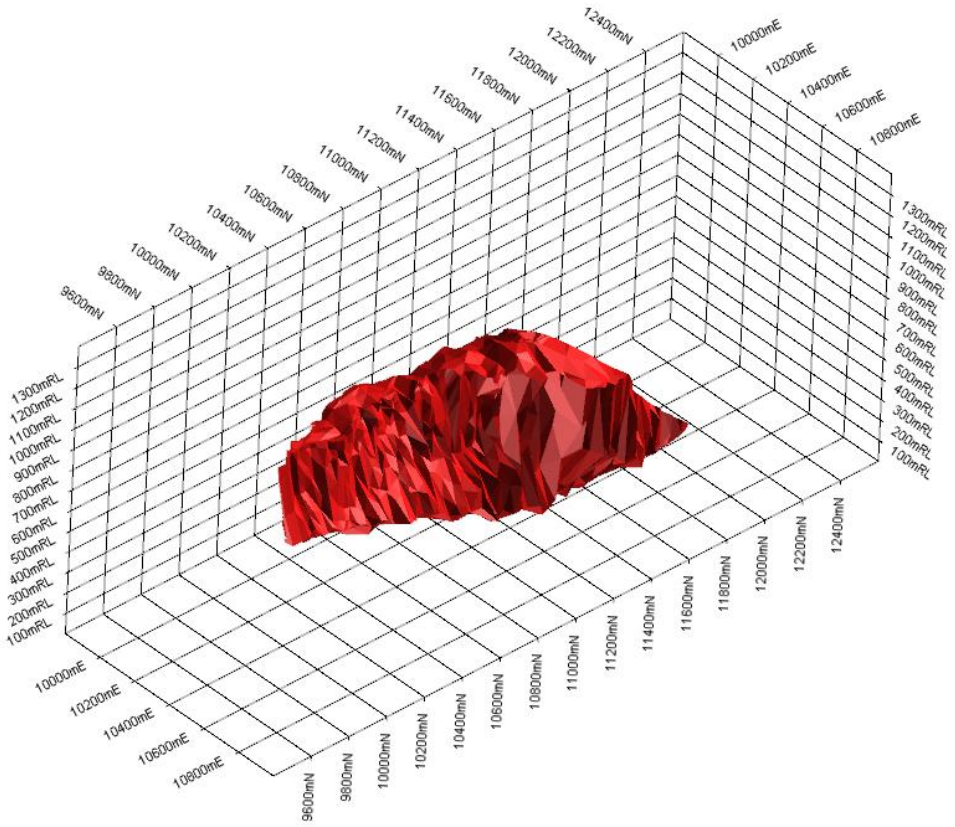


Figure 17.1.4_4
Domain 4, Oxide Zone Sukari Porphyry North

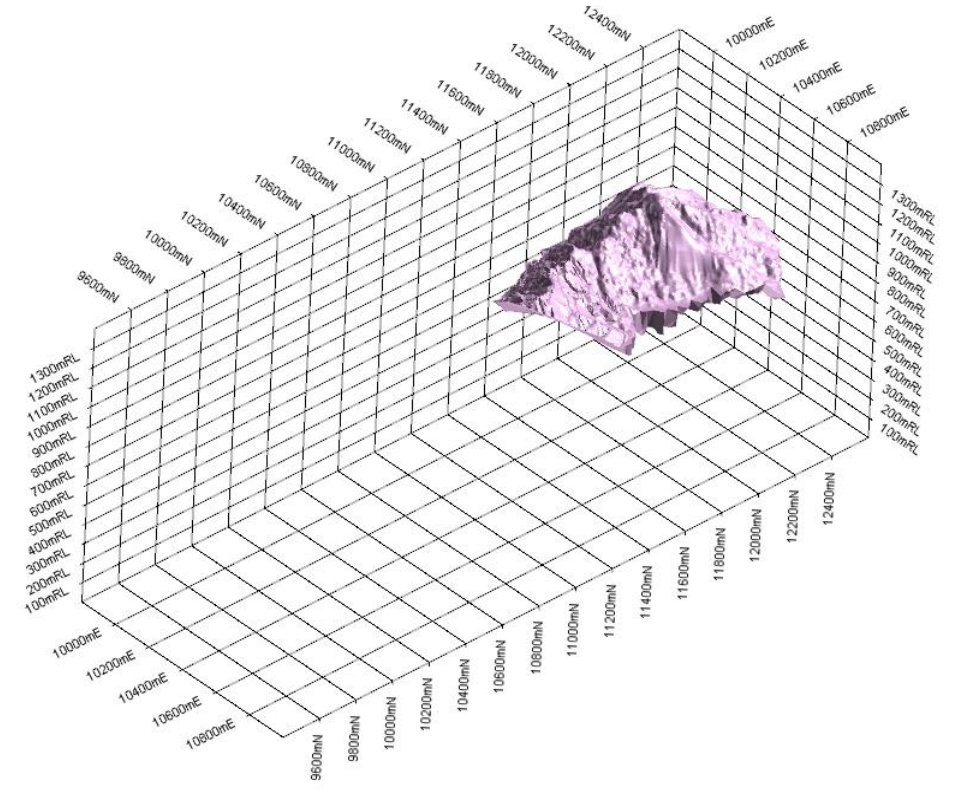


Figure 17.1.4_5
Domain 5, Sulphide Zone Sukari Porphyry North

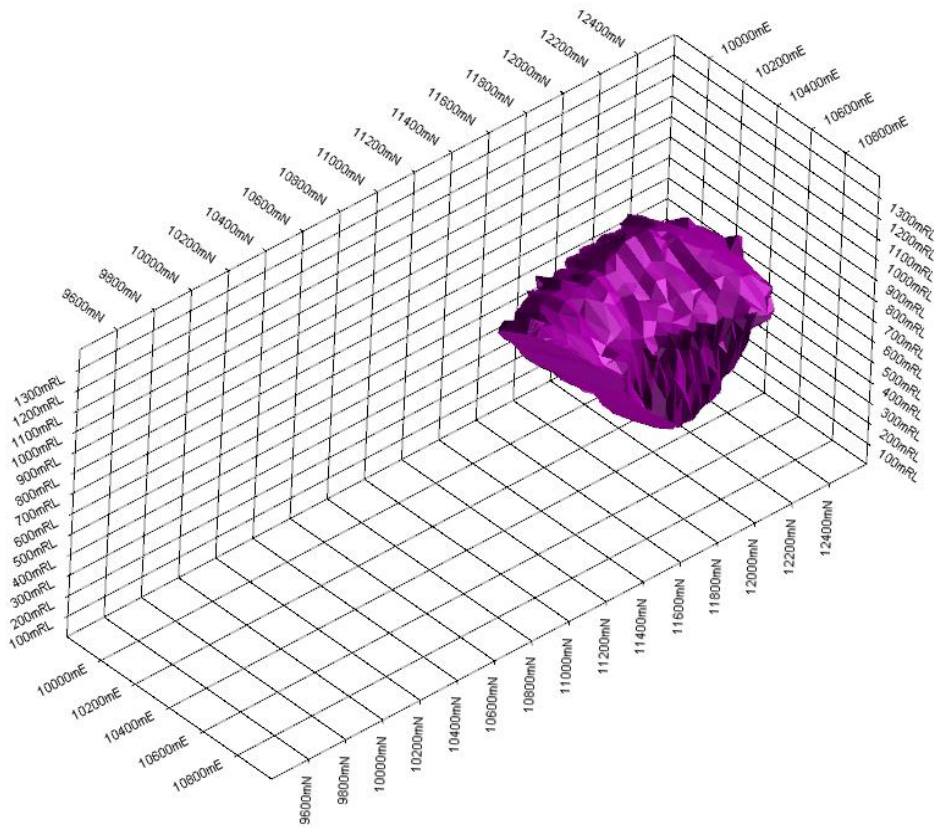


Figure 17.1.4_6
Domain 6, Horus Zone

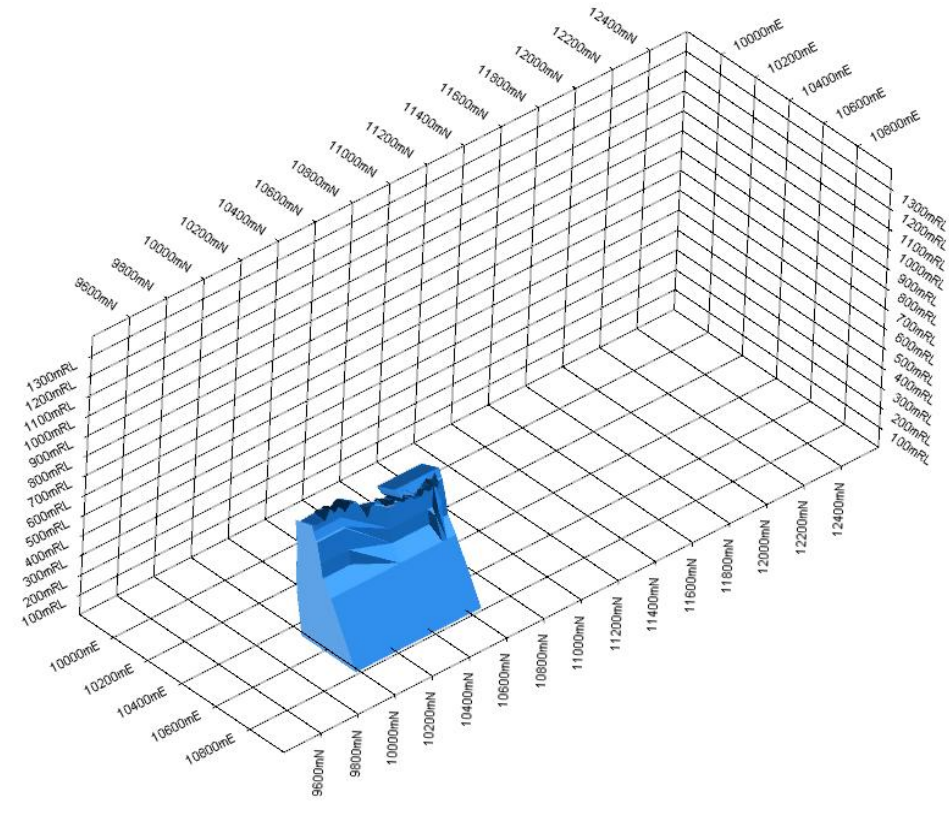
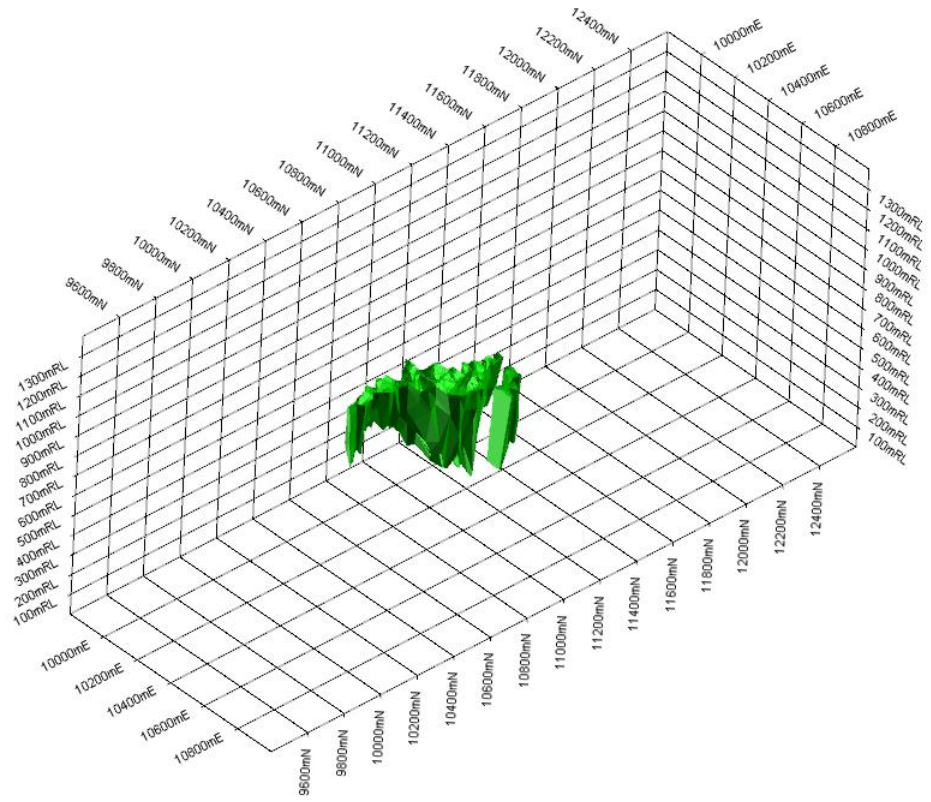


Figure 17.1.4_7
Domain 7, Hangingwall Porphyry Mineralised Zones



Drillhole composites and surface rock chip samples were flagged by domain code and combined into one file of located samples.

A panel template model was established, proportions of panels within each domain being calculated by passing the model through the domain wire-frames.

17.1.5. Oxidation Interpretation

17.1.5.1. Oxidation

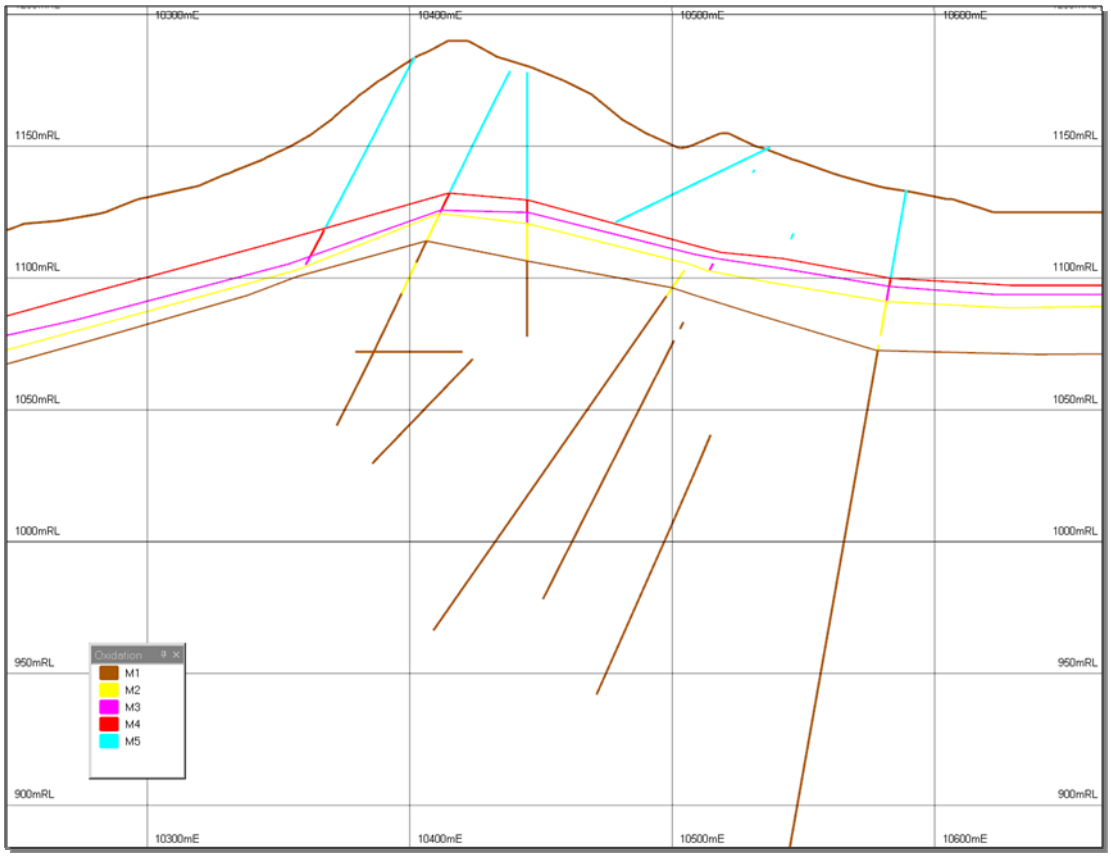
Drill sections at 50m intervals, extending from 9,900mN through to 11,200mN, and for the most part at 50m intervals to 12,200mN, have been logged using the oxidation criteria shown in Table 17.1.5.1_1. The sectional interpretations developed from the oxidation logging produce four transitions of M1-M2, M2-M3, M3M4 and M4-M5 on sections south of 11,200mN, while north of this section, due to the majority of drilling are RC, only transitions of M1-M2 and M4-M5 were able to be interpreted. The sectional interpretations were used to develop DTMs of the transitions. The DTMs have been used to provide the basis for partitioning the final resources into the oxidation types (M1 through to M5). Only the M1 surface have been used in the grade modelling process (Figure 17.1.5.1_1) and has been used to create the wire-frames defining the domains representing the oxide and sulphide primary domains of the main Sukari porphyry and assigning the sub-domain coding (oxide or sulphide) of resource composites in the remaining model domains.

Table 17.1.5.1_1

Sukari
Oxidation Logging Codes

M1	Fresh rock, comprises 100% sulphide mineralisation
M2	Mixed sulphide and oxide, comprises >75% sulphide mineralisation
M3	Mixed sulphide and oxide, comprises >25% to <75% sulphide mineralisation
M4	Mixed sulphide and oxide, comprises <25% sulphide mineralisation, >75% secondary hematite
M5	Fully oxidized, comprises 100% oxide (secondary hematite) mineralisation

Figure 17.1.5.1_1
Section 10,350mN Drillholes by Oxidation Code and interpreted Oxidation Surfaces



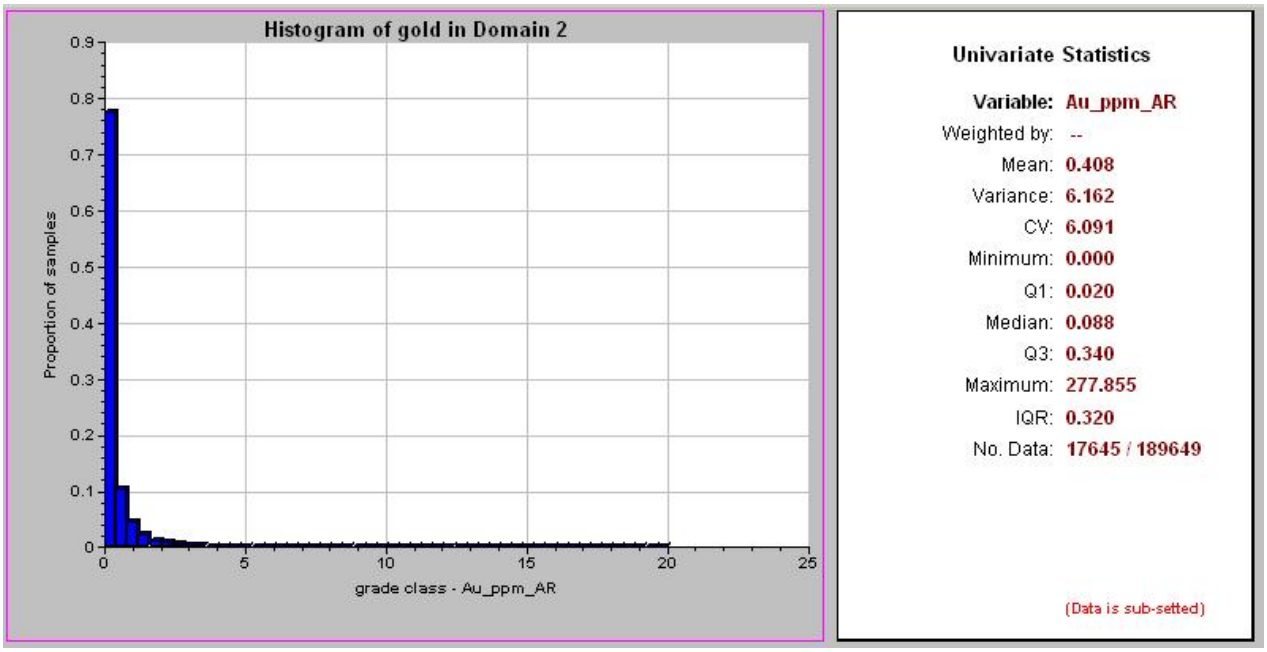
17.1.6. Exploratory Data Analysis

Figure 17.1.6_1 and 17.1.6_2 show histograms of gold grades for samples coded as oxide and primary in Domains 2 through 5, the main Sukari porphyry mineralisation. Note that on these figures the maximum gold grade shown on the x-axis is 20g/t Au and is for presentation purposes only. The summary univariate statistics shown are calculated from the entire data set. Typical of many gold deposits, all populations of gold grades show strong positive skewness with coefficient of variation ('CV') ranging from 3.02 to 14.05.

The CV's are very high for the southern porphyry (Domains 2 and 3) indicating that it will be difficult to maintain a high degree of selectivity in mining at Sukari. The very high CV's shown for Domain 2 and 3, being 6.09 and 14.05, respectively, are effected by single very high gold grades, 277.85g/t Au and 2749.50g/t Au, located in the oxide and primary data sets, respectively. Excluding these gold grades from the CV calculation reduces the CV to 3.42 and 6.610, Domain 2 and 3, respectively. Given the effect of these individual grades on the domain statistics, particularly the mean grade of the highest indicator bin, H&S is of the opinion that it is appropriate to exclude these sample grades when calculating the indicator statistics for use in the MIK model.

The CV of 3.02 and 5.35, oxide and primary, respectively, for gold in the north porphyry domains (Domain 4 and 5) are lower compared to the southern porphyry domains. The difference in the CV's, together with the differences seen in the mean grades and the presence of a significant population of high grades in Domain 2 and 3, support the separation of the main porphyry into the north and south modelling regions.

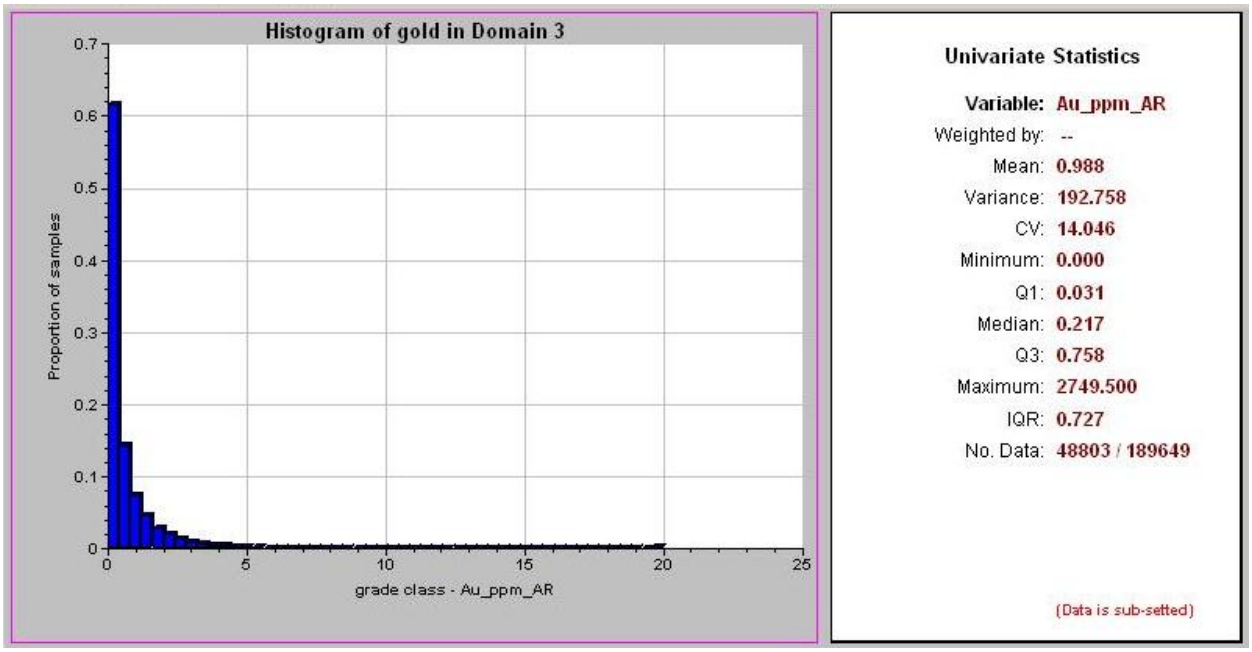
Figure 17.1.6_1
Histograms Au grade Domain 2 and 3



Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.408**
Variance:	**6.162**
CV:	**6.091**
Minimum:	**0.000**
Q1:	**0.020**
Median:	**0.088**
Q3:	**0.340**
Maximum:	**277.855**
IQR:	**0.320**
No. Data:	**17645 / 189649**

(Data is sub-setted)

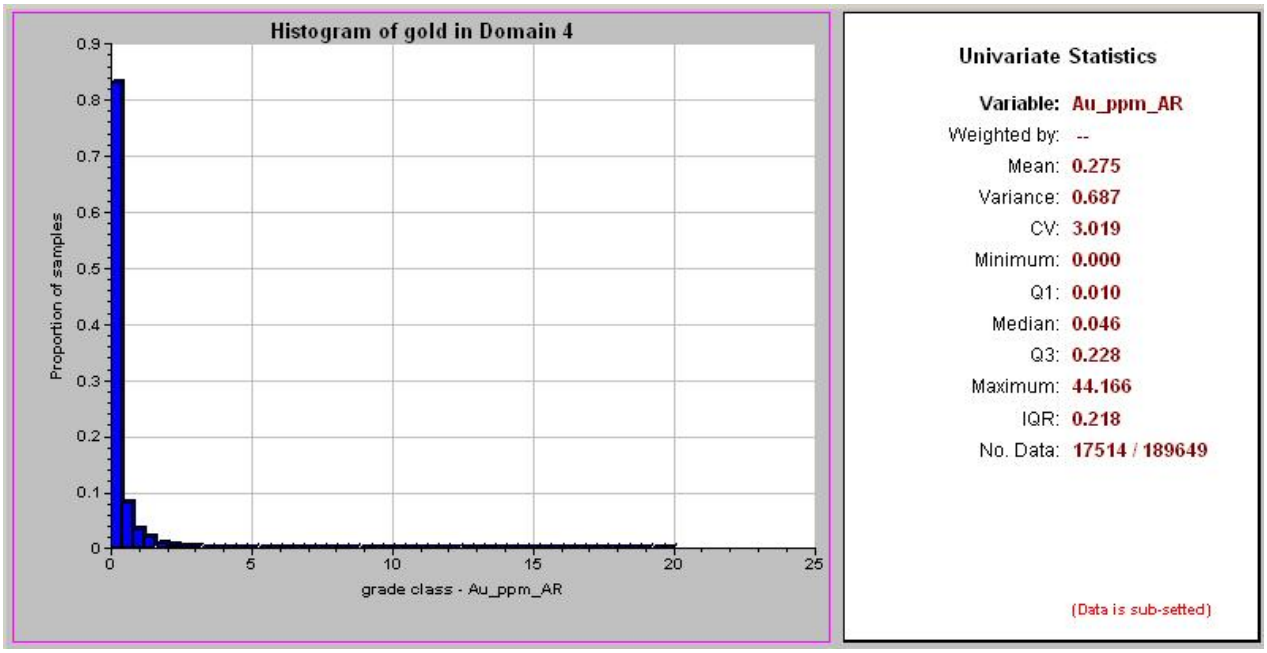


Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.988**
Variance:	**192.758**
CV:	**14.046**
Minimum:	**0.000**
Q1:	**0.031**
Median:	**0.217**
Q3:	**0.758**
Maximum:	**2749.500**
IQR:	**0.727**
No. Data:	**48803 / 189649**

(Data is sub-setted)

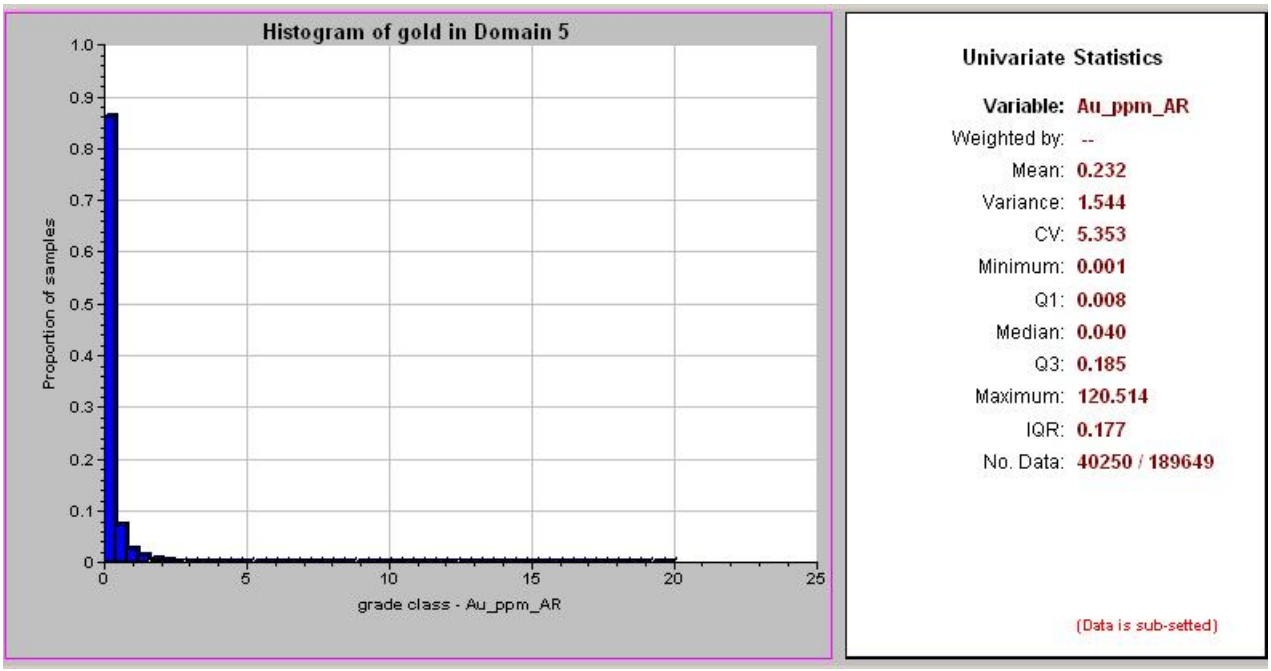
Figure 17.1.6_2
Histograms Au grade Domain 4 and 5



Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.275**
Variance:	**0.687**
CV:	**3.019**
Minimum:	**0.000**
Q1:	**0.010**
Median:	**0.046**
Q3:	**0.228**
Maximum:	**44.166**
IQR:	**0.218**
No. Data:	**17514 / 189649**

(Data is sub-setted)



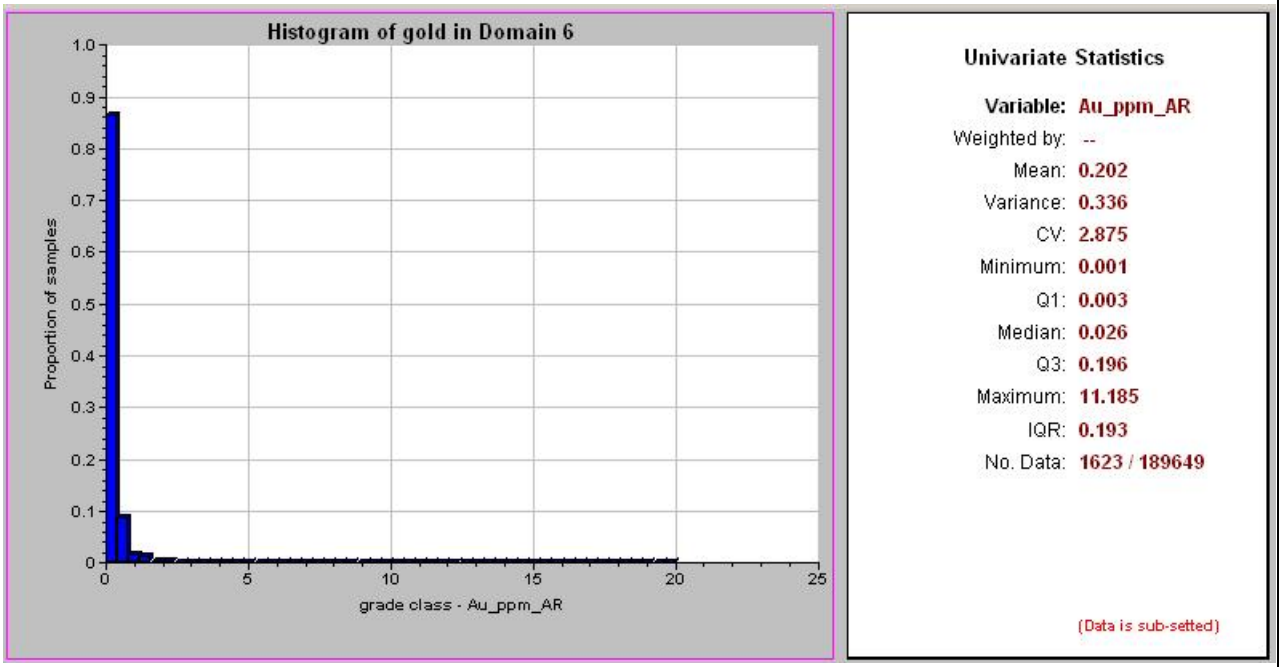
Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.232**
Variance:	**1.544**
CV:	**5.353**
Minimum:	**0.001**
Q1:	**0.008**
Median:	**0.040**
Q3:	**0.185**
Maximum:	**120.514**
IQR:	**0.177**
No. Data:	**40250 / 189649**

(Data is sub-setted)

The histogram of gold in samples in Domain 6, the domain capturing the interpretable mineralised intercepts in Horus Zone, is shown in Figure 17.1.6_3. This domain only exists at depth, beneath Domain 3, so only the primary (sulphide) sub-domain is modelled for this domain. The mean grade and CV for this domain are similar to those seen for the north Sukari porphyry sulphide domain (Domain 5).

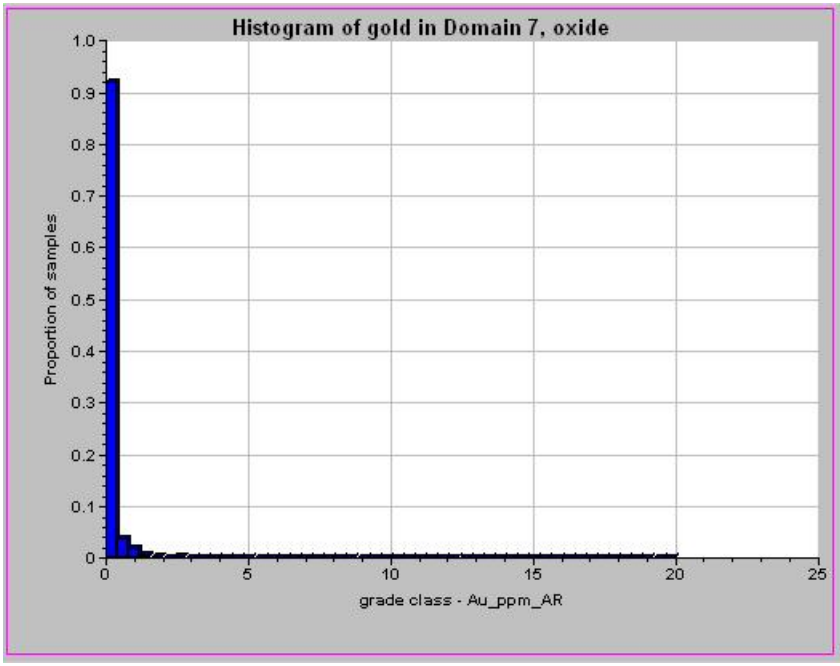
Figure 17.1.6_3
Histogram Au grade Domain 6



The histogram of gold in samples in Domain 7, the domain capturing the interpretable mineralised intercepts in subsidiary porphyry dykes in the hanging wall, is shown in Figure 17.1.6_4. Figures 17.1.6_5 and 17.1.6_6 show the histograms of gold in samples for the, essentially barren, footwall and hanging wall domains. The weathered and fresh coded composites are grouped into a single histogram for the later domains as there is little difference in the statistics of the two material types.

Conditional statistics for data within each domain are listed in Tables 17.1.8_2, 17.1.8_3, 17.1.8_4 and 17.1.8_5 shown later in this section. In Domain 1, the footwall position to the main porphyry, 99 per cent of samples grade less than 0.33g/t Au and 0.225g/t Au, oxide and primary, respectively. Clearly this domain will make very little, if any, contribution to potentially economic gold resources.

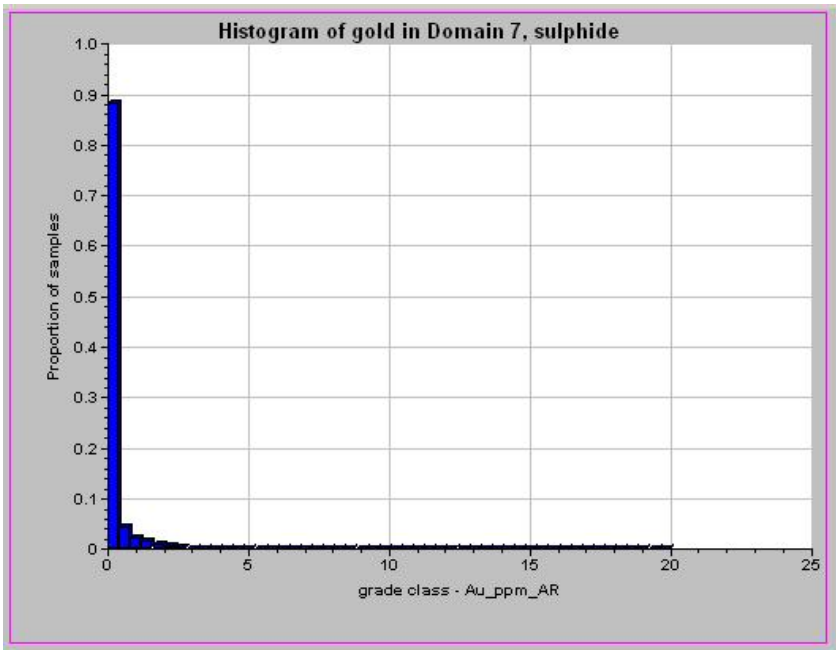
Figure 17.1.6_4
Histograms Au grade Domain 7



Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.115**
Variance:	**0.149**
CV:	**3.347**
Minimum:	**0.000**
Q1:	**0.002**
Median:	**0.009**
Q3:	**0.035**
Maximum:	**6.560**
IQR:	**0.033**
No. Data:	**3188 / 189649**

(Data is sub-setted)



Univariate Statistics

Variable:	**Au_ppm_AR**
Weighted by:	--
Mean:	**0.194**
Variance:	**0.462**
CV:	**3.495**
Minimum:	**0.000**
Q1:	**0.001**
Median:	**0.005**
Q3:	**0.043**
Maximum:	**16.850**
IQR:	**0.042**
No. Data:	**5531 / 189649**

(Data is sub-setted)

Figure 17.1.6_5
Histogram Au grade Domain 1 – Footwall

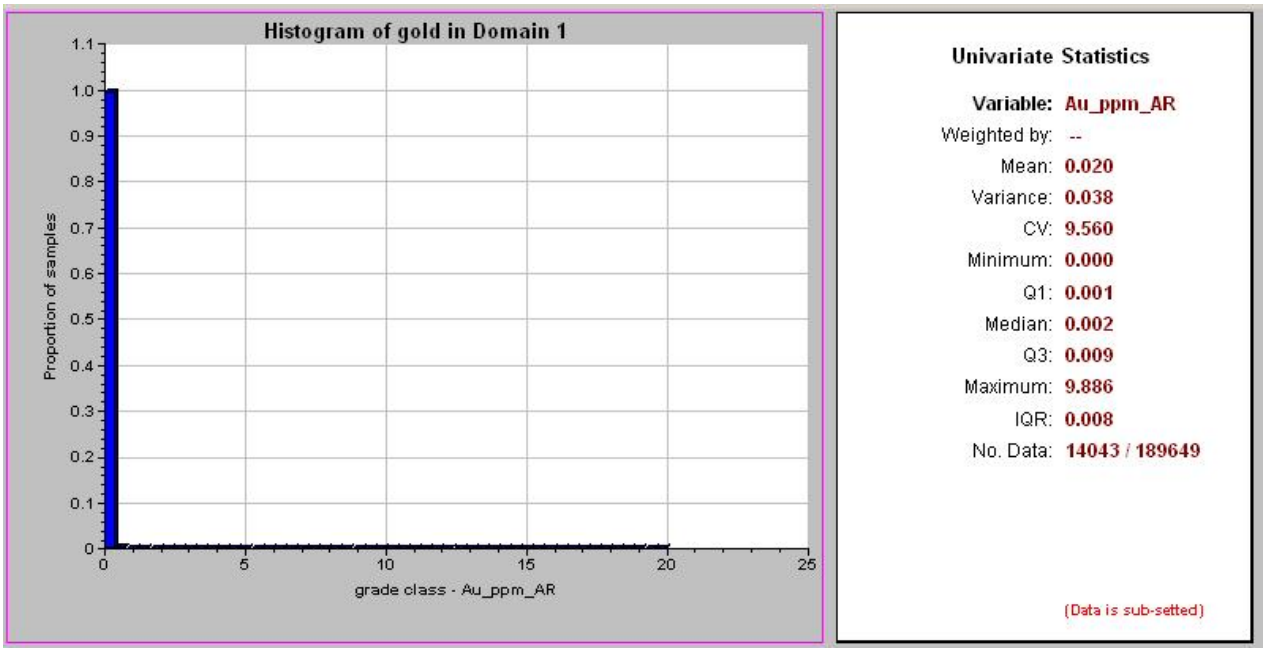
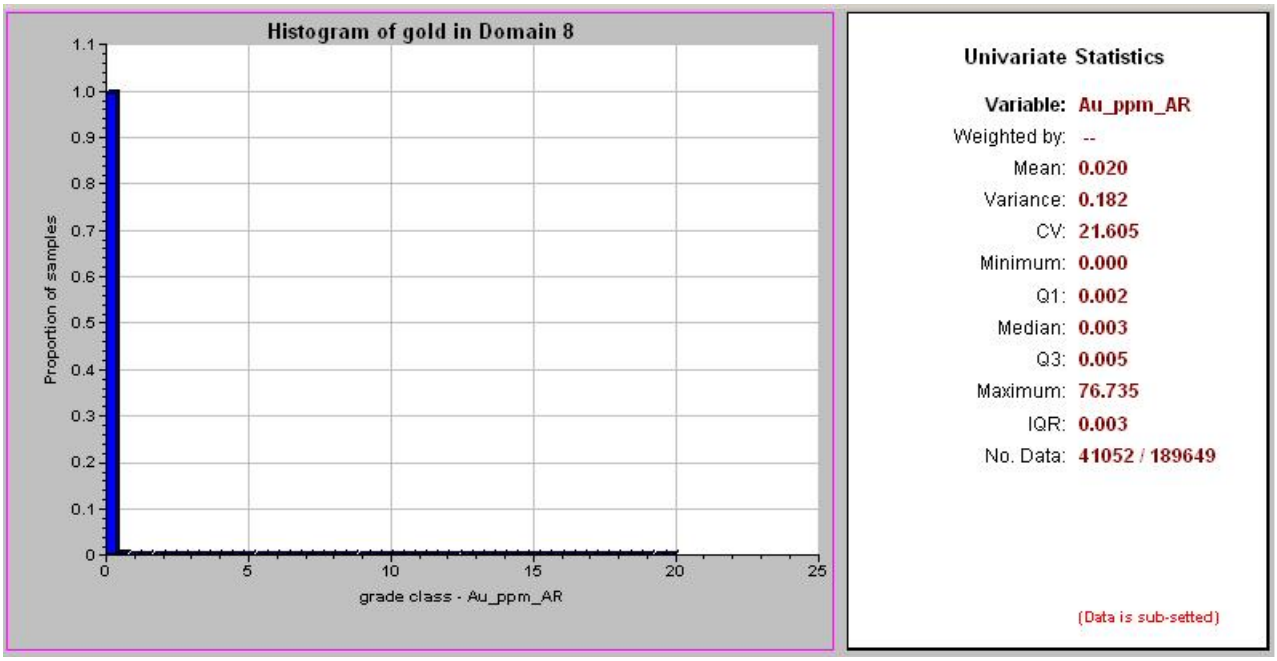


Figure 17.1.6_6
Histogram Au grade Domain 8 - Hangingwall



17.1.7. Spatial Continuity Analysis

17.1.7.1. Measures of Spatial Continuity

Most resource estimation methods use a measure of spatial continuity to estimate the grade of blocks in a resource model. In some methods, the measure is implicit; for example a polygonal method assumes that the grade is perfectly continuous from the sample to its surrounding polygon boundary. Geostatistical methods like Ordinary Kriging and Indicator Kriging are among those methods for which the continuity measure is explicit and is customised to the dataset being studied. This measure in its many forms is usually called the variogram.

Geostatistics provides several measures for describing spatial continuity: the variogram, the covariance, the correlogram and many others. All are valid descriptions but not all provide a basis for constructing kriging and simulation models of mineralisation. Whatever the method of description used, it is common to use the term variogram in a generic sense to describe contour plots and directional plots of spatial continuity measures. Throughout the present work, the maps and directional variograms used are all based on the correlogram measure. Directional correlograms are displayed inverted so as to resemble familiar variogram plots. The use of the correlogram as a robust and reliable measure of spatial continuity is proposed by Srivastava & Parker (1988) and Isaaks & Srivastava (1989). The correlogram measure has the advantages of being standardised to a sill of 1.0 and being robust with respect to clustering in the sample data. Models of the sample correlogram can be used directly in Ordinary Kriging and Indicator Kriging.

The various parameters of the variogram model, such as the nugget effect and ranges in different directions, describe properties of the statistical continuity of metal grades. For example, a variogram with high nugget may indicate that there is a high level of error in the sample grades being used to construct the variograms or that there is a high degree of variability in the grade over very short distances in the mineralisation. A different range in one direction compared to another is likely to be indicating that grade is more continuous in one direction than another.

17.1.7.2. Sukari Indicator Variogram Models

Directional variograms were calculated for each indicator threshold, using Domain 1, 2, 3 and 4 sample composites only, for a series of directions as dictated by the average drillhole orientations. These domains contain the bulk of gold mineralisation at Sukari and are the only domains containing sufficient mineralised samples, at reasonable spacing, to allow calculation of meaningful gold and indicator variograms. Indicator transforms were undertaken with probability thresholds 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99 for data in each data subset.

The indictor and gold variograms for Domain 2 have been used for modelling the gold mineralisation within the oxide sub-domain for Domain 1 and 7. Domain 3 indictor and gold variograms have been used for modelling the gold mineralisation within the remaining domains and sub-domains.

The number of variograms modelled in the present study was very large (75 for each domain). The spatial continuity observed in the variograms is consistent with the known geological interpretation. The fitted models generally have a fairly large short range

structure and a smaller long range structure consistent with the strike and plunge of dominant mineralised vein orientation. Tables showing a complete listing of the modelled variograms used in the resource model and shown in Figures 17.1.8_1 and 17.1.8_4 are contained in Section 17.1.8.

17.1.8. Indicator Kriging Parameters

The input parameters to Indicator Kriging of the Sukari mineralisation include:-

- Indicator variogram models describing the spatial continuity of indicator variables within each domain at each indicator threshold.
- Variograms describing the spatial continuity of gold grades within each domain.
- Mean gold grades of each of the indicator classes within each domain.

Table 17.1.8_1 shows the grid framework and kriging parameters used in the indicator kriging model. A description of the categorisation of resources is described in Section 17.1.11.

Table 17.1.8_1
Model Framework and Kriging Parameters

Panel Model Extents			
	m East	**m North**	**m elevation**
Panel origin (centroid)	10,200	9,750	5
Panel Dimensions	20	25	6
No. of panels	43	100	220
Panel Discretisation	6	6	3
Kriging Parameters (all domains)			
Criteria	**Measured**	**Indicated**	**Inferred**
Min no. of data	16	16	8
Max no. of data per octant	8	8	8
Min no. of octants with data	4	4	2
X (east) search radius (metres)	30	45	45
Y (north) search radius (metres)	30	45	45
Z (rl) search radius (metres)	15	22.5	22.5
Search Rotations			
	Rotation axis	**Rotation**	**Grid Direction**
All Domains 1	x (east)	-20	-20/360
2	y (north)	20	-20/090

Figures 17.1.8_1 and 17.1.8_2 show the indicator variograms used for modelling the gold grades in Domain 2 and 4 and Domains 1, 3, 5 and 6, respectively. Note that the variogram model shown on the bottom line of each figure is for the variogram of gold and are an input to the calculation of the block variance correction used to produce the recoverable resource estimate.

Figure 17.1.8_1
Variogram Models Domain 2 (Domain 1 Oxide, Domain 7 Oxide)

Ind Thres.		1st Structure					2nd Structure					3rd Structure				
	C0	C1	Type	x	y	z	C2	Type	x	y	z	C3	Type	x	y	z
1	0.14	0.57	exp	13.5	8.5	7.0	0.21	sph	30.0	260.0	71.5	0.08	sph	542.0	451.0	216.0
2	0.06	0.55	exp	9.0	9.0	7.5	0.19	sph	32.5	18.5	60.0	0.19	sph	103.0	760.0	77.0
3	0.06	0.58	exp	11.0	9.0	8.0	0.14	sph	33.0	17.5	53.5	0.22	sph	103.0	118.0	97.0
4	0.06	0.61	exp	7.5	7.5	7.0	0.11	sph	31.5	18.0	53.0	0.21	sph	98.0	107.0	138.0
5	0.06	0.61	exp	7.0	10.0	7.0	0.08	sph	10.0	46.0	32.5	0.25	sph	112.0	90.0	84.0
6	0.10	0.61	exp	69.0	7.0	7.0	0.08	sph	160.5	16.0	34.5	0.21	sph	172.0	95.0	95.0
7	0.15	0.61	exp	64.5	21.0	7.0	0.08	sph	66.0	21.5	91.0	0.16	sph	158.0	111.0	125.0
8	0.17	0.61	exp	71.0	17.0	7.0	0.08	sph	79.0	17.5	120.5	0.14	sph	143.0	98.0	121.0
9	0.21	0.61	exp	10.5	33.0	7.0	0.08	sph	237.0	41.5	141.5	0.10	sph	238.0	121.0	142.0
10	0.25	0.61	exp	7.0	19.5	7.5	0.08	sph	279.0	60.0	292.5	0.06	sph	316.0	143.0	293.0
11	0.25	0.63	exp	16.0	15.5	7.5	0.10	sph	377.5	52.5	110.0	0.02	sph	411.0	54.0	123.0
12	0.40	0.52	exp	27.0	23.0	8.5	0.05	sph	268.5	46.5	108.5	0.03	sph	297.0	091.0	109.0
13	0.51	0.41	exp	70.5	9.0	7.0	0.05	sph	415.5	41.5	63.5	0.03	sph	451.0	115.0	64.0
14	0.55	0.41	exp	27.5	63.5	7.0	0.02	sph	36.0	74.5	7.5	0.02	sph	102.0	472.0	50.0
Gold	0.33	0.47	exp	7.0	7.0	7.0	0.08	sph	79.5	79.5	8.0	0.12	sph	111.0	114.0	44.0
	Rotation: z 59, y 21, x21															

Figure 17.1.8_2
Variogram Models Domain 3 (Domain 1 Sulphide, Domain 3 Sulphide and Domain 8)

Ind Thres.		1st Structure					2nd Structure					3rd Structure				
	C0	C1	Type	x	y	z	C2	Type	x	y	z	C3	Type	x	y	z
1	0.14	0.36	exp	8.5	20.0	7.5	0.34	sph	18.5	35.0	157.0	0.16	sph	168.0	513.0	613.0
2	0.14	0.41	exp	8.0	24.0	8.5	0.21	sph	18.5	36.5	131.0	0.24	sph	92.0	278.0	173.0
3	0.15	0.42	exp	9.0	53.5	8.0	0.20	sph	18.5	54.0	167.5	0.23	sph	109.0	533.0	168.0
4	0.15	0.42	exp	20.0	32.5	7.0	0.20	sph	20.5	67.5	127.5	0.23	sph	125.0	032.0	128.0
5	0.17	0.42	exp	21.5	23.0	7.0	0.20	sph	22.0	212.0	81.5	0.21	sph	176.0	184.0	118.0
6	0.21	0.42	exp	23.0	22.0	8.0	0.16	sph	23.5	123.5	90.5	0.21	sph	201.0	091.0	117.0
7	0.27	0.42	exp	12.5	17.5	9.5	0.10	sph	22.5	149.5	123.5	0.21	sph	250.0	487.0	124.0
8	0.32	0.42	exp	8.5	85.0	10.5	0.10	sph	63.5	112.5	131.5	0.16	sph	410.0	294.0	132.0
9	0.32	0.42	exp	9.5	9.5	9.5	0.10	sph	33.5	203.5	65.0	0.16	sph	802.0	805.0	80.0
10	0.36	0.42	exp	8.5	12.5	9.5	0.09	sph	36.0	143.5	82.5	0.13	sph	829.0	403.0	83.0
11	0.40	0.42	exp	10.0	11.0	8.0	0.09	sph	39.0	300.0	51.5	0.09	sph	741.0	741.0	74.0
12	0.45	0.42	exp	9.5	7.0	7.0	0.08	sph	47.5	415.5	41.5	0.05	sph	364.0	421.0	42.0
13	0.50	0.35	exp	7.5	7.0	7.0	0.09	sph	76.0	38.0	7.5	0.06	sph	82.0	748.0	92.0
14	0.61	0.35	exp	7.0	7.0	7.0	0.01	sph	43.0	71.5	7.5	0.03	sph	47.0	282.0	33.0
Gold	0.53	0.38	exp	7.0	7.0	7.0	0.01	sph	82.0	60.5	8.0	0.08	sph	301.0	200.0	30.0
	Rotation: z -22, y 18, x-30															

Figure 17.1.8_3
Variogram Models Domain 4

Ind Thres.		1st Structure					2nd Structure					3rd Structure				
	C0	C1	Type	x	y	z	C2	Type	x	y	z	C3	Type	x	y	z
1	0.25	0.51	exp	26.0	34.5	19.5	0.12	sph	61.0	45.5	211.0	0.12	sph	218.0	174.0	788.0
2	0.15	0.51	exp	30.0	12.0	14.0	0.12	sph	77.5	472.5	745.0	0.22	sph	100.0	877.0	768.0
3	0.15	0.49	exp	35.0	120.5	12.0	0.33	sph	88.0	830.5	537.5	0.03	sph	99.0	985.0	551.0
4	0.15	0.49	exp	60.5	59.0	11.5	0.01	sph	85.5	838.0	248.0	0.35	sph	86.0	861.0	256.0
5	0.15	0.50	exp	71.0	56.5	11.0	0.08	sph	81.5	362.0	216.0	0.27	sph	87.0	866.0	221.0
6	0.26	0.30	exp	66.5	34.5	7.0	0.37	sph	91.5	537.5	61.0	0.07	sph	191.0	886.0	301.0
7	0.28	0.31	exp	30.5	68.0	7.0	0.18	sph	78.0	292.5	31.0	0.23	sph	79.0	775.0	439.0
8	0.28	0.31	exp	7.0	7.0	7.0	0.18	sph	91.5	232.0	23.0	0.23	sph	92.0	922.0	923.0
9	0.34	0.30	exp	7.0	53.5	7.0	0.15	sph	83.0	316.0	33.0	0.21	sph	84.0	827.0	787.0
10	0.34	0.32	exp	7.0	22.0	7.0	0.13	sph	77.5	135.5	13.5	0.21	sph	78.0	775.0	782.0
11	0.34	0.35	exp	7.0	24.0	7.5	0.10	sph	59.5	80.0	8.0	0.21	sph	60.0	586.0	79.0
12	0.34	0.39	exp	7.5	7.0	7.0	0.12	sph	8.0	81.0	36.5	0.15	sph	76.0	374.0	37.0
13	0.34	0.44	exp	7.5	7.0	7.0	0.12	sph	8.0	62.0	30.0	0.10	sph	53.0	285.0	31.0
14	0.47	0.41	exp	7.0	24.5	7.0	0.03	sph	7.5	38.0	22.5	0.09	sph	76.0	92.0	23.0
Gold	0.34	0.45	exp	7.0	7.0	7.0	0.06	sph	7.5	70.0	65.5	0.15	sph	81.0	283.0	66.0
	Rotation: z 35, y 26, x3															

Figure 17.1.8_4
Variogram Models Domain 5

Ind Thres.		1st Structure					2nd Structure					3rd Structure				
	C0	C1	Type	x	y	z	C2	Type	x	y	z	C3	Type	x	y	z
1	0.30	0.30	exp	38.5	24.5	7.5	0.10	sph	39.0	25.0	14.5	0.30	sph	40.0	93.0	69.0
2	0.25	0.35	exp	50.5	77.5	8.0	0.07	sph	51.5	97.0	10.0	0.33	sph	52.0	122.0	157.0
3	0.25	0.28	exp	49.5	68.0	7.0	0.12	sph	50.0	97.5	10.0	0.35	sph	51.0	136.0	174.0
4	0.26	0.29	exp	52.0	54.0	7.0	0.09	sph	53.5	60.5	8.0	0.36	sph	55.0	466.0	119.0
5	0.29	0.25	exp	9.5	7.0	7.0	0.12	sph	57.5	92.0	9.0	0.34	sph	58.0	523.0	125.0
6	0.32	0.25	exp	9.0	9.0	7.0	0.09	sph	54.0	72.0	8.0	0.34	sph	58.0	575.0	130.0
7	0.37	0.32	exp	51.5	75.5	7.5	0.05	sph	58.5	100.0	10.0	0.26	sph	62.0	620.0	172.0
8	0.38	0.37	exp	57.5	76.0	7.5	0.01	sph	58.0	82.5	8.0	0.24	sph	59.0	581.0	168.0
9	0.38	0.37	exp	46.5	35.5	7.0	0.03	sph	54.5	80.0	9.5	0.22	sph	56.0	497.0	214.0
10	0.41	0.37	exp	39.0	71.0	7.0	0.03	sph	51.0	142.0	14.0	0.19	sph	58.0	582.0	182.0
11	0.47	0.29	exp	7.0	7.5	7.0	0.07	sph	58.5	99.0	10.0	0.17	sph	59.0	391.0	169.0
12	0.47	0.39	exp	8.0	7.5	7.0	0.01	sph	10.0	80.5	101.0	0.13	sph	42.0	342.0	152.0
13	0.47	0.42	exp	7.0	7.0	7.0	0.05	sph	13.0	56.5	117.0	0.06	sph	55.0	320.0	257.0
14	0.56	0.35	exp	8.0	7.0	7.0	0.03	sph	11.0	28.0	7.5	0.06	sph	20.0	168.0	168.0
Gold	0.50	0.28	exp	37.0	63.0	7.0	0.05	sph	38.5	70.0	11.0	0.17	sph	43.0	387.0	431.0
	Rotation: z -37, y 37, x7															

Tables 17.1.8_2, 17.1.8_3, 17.1.8_4 and 17.1.8_5 show the indicator thresholds and class means for each domain used in modelling gold at Sukari.

Table 17.1.8_2
Sukari
Indicator Class Means Domain 1

Probability Threshold	Domain 1			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.0010	0.0010	0.0011	0.0010
0.2	0.0010	0.0010	0.0012	0.0010
0.3	0.0010	0.0010	0.0014	0.0010
0.4	0.0030	0.0020	0.0015	0.0010
0.5	0.0080	0.0050	0.0016	0.0020
0.6	0.0100	0.0100	0.0021	0.0020
0.7	0.0110	0.0100	0.0040	0.0030
0.75	0.0150	0.0140	0.0060	0.0050
0.8	0.0250	0.0180	0.0100	0.0080
0.85	0.0380	0.0300	0.0110	0.0100
0.9	0.0700	0.0510	0.0210	0.0160
0.95	0.1500	0.1020	0.0580	0.0360
0.97	0.2150	0.1820	0.0970	0.0740
0.99	0.3150	0.2640	0.2600	0.1550
Max	9.6000	1.8350	9.8860	0.7520

Table 17.1.8_3
Sukari
Indicator Class Means Domain 2 and 3

Probability Threshold				
	Domain 2		Domain 3	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.0100	0.0060	0.0060	0.0030
0.2	0.0120	0.0100	0.0180	0.0110
0.3	0.0280	0.0190	0.0500	0.0330
0.4	0.0500	0.0380	0.1150	0.0800
0.5	0.0880	0.0680	0.2170	0.1620
0.6	0.1500	0.1160	0.3690	0.2880
0.7	0.2550	0.1980	0.5940	0.4710
0.75	0.3400	0.2950	0.7580	0.6720
0.8	0.4560	0.3950	0.9810	0.8620
0.85	0.6350	0.5420	1.3040	1.1330
0.9	0.9450	0.7730	1.8400	1.5480
0.95	1.6000	1.2150	3.0160	2.3250
0.97	2.2500	1.8940	4.1800	3.5300
0.99	4.1500	2.9490	8.8350	5.7020
Max	37.7250	8.2700	520.6450	31.5090

Table 17.1.8_4

Sukari

Indicator Class Means Domains 4 & 5

Probability Threshold	Domain 4		Domain 5	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.0030	0.0010	0.0031	0.0020
0.2	0.0080	0.0050	0.0060	0.0040
0.3	0.0130	0.0100	0.0110	0.0080
0.4	0.0240	0.0180	0.0220	0.0160
0.5	0.0460	0.0340	0.0400	0.0300
0.6	0.0880	0.0650	0.0740	0.0560
0.7	0.1640	0.1210	0.1360	0.1020
0.75	0.2280	0.1940	0.1850	0.1590
0.8	0.3200	0.2720	0.2560	0.2180
0.85	0.4630	0.3850	0.3620	0.3050
0.9	0.7120	0.5750	0.5480	0.4460
0.95	1.2560	0.9450	0.9600	0.7180
0.97	1.7550	1.4670	1.3480	1.1290
0.99	3.0980	2.3020	2.6350	1.8170
Max	44.1660	5.4900	28.0230	4.7780

Table 17.1.8_5

Sukari

Indicator Class Means Domain 6

Probability Threshold	Domain 6	
	Grade Threshold	Class Mean
0.1	0.0010	0.0010
0.2	0.0011	0.0010
0.3	0.0020	0.0020
0.4	0.0040	0.0030
0.5	0.0090	0.0060
0.6	0.0100	0.0100
0.7	0.0200	0.0140
0.75	0.0350	0.0270
0.8	0.0600	0.0450
0.85	0.1290	0.0890
0.9	0.2650	0.1850
0.95	0.6140	0.4180
0.97	1.0300	0.8190
0.99	1.9190	1.3570
Max	6.5600	2.9610

Table 17.1.8_6

Sukari

Indicator Class Means Domain 7

Probability Threshold	Domain 7			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.0010	0.0010	0.0010	0.0010
0.2	0.0011	0.0010	0.0011	0.0010
0.3	0.0020	0.0020	0.0020	0.0010
0.4	0.0040	0.0030	0.0030	0.0020
0.5	0.0090	0.0060	0.0050	0.0040
0.6	0.0100	0.0100	0.0100	0.0080
0.7	0.0200	0.0140	0.0210	0.0140
0.75	0.0350	0.0270	0.0430	0.0300
0.8	0.0600	0.0450	0.1060	0.0700
0.85	0.1290	0.0890	0.2300	0.1620
0.9	0.2650	0.1850	0.5150	0.3560
0.95	0.6140	0.4180	1.1350	0.7780
0.97	1.0300	0.8190	1.6690	1.3840
0.99	1.9190	1.3570	2.9500	2.1930
Max	6.5600	2.9610	16.8500	4.9490

Table 17.1.8_7

Sukari

Indicator Class Means Domain 8

Probability Threshold	Domain 8			
	Oxide		Sulphide	
	Grade Threshold	Class Mean	Grade Threshold	Class Mean
0.1	0.0011	0.0010	0.0013	0.0010
0.2	0.0012	0.0010	0.0016	0.0010
0.3	0.0013	0.0010	0.0019	0.0020
0.4	0.0015	0.0010	0.0021	0.0020
0.5	0.0020	0.0020	0.0022	0.0020
0.6	0.0031	0.0020	0.0025	0.0020
0.7	0.0060	0.0040	0.0028	0.0030
0.75	0.0100	0.0080	0.0031	0.0030
0.8	0.0101	0.0100	0.0050	0.0040
0.85	0.0150	0.0110	0.0080	0.0060
0.9	0.0270	0.0200	0.0120	0.0100
0.95	0.0610	0.0400	0.0330	0.0190
0.97	0.1080	0.0810	0.0670	0.0470
0.99	0.3330	0.1870	0.2240	0.1190
Max	11.5200	0.8430	15.4550	0.9930

In addition to reviewing the effect of high grades on the indicator statistics, further checks have been made to test the effect on the estimated resources to the use of hard or soft boundaries (particularly contact between the mineralised porphyry and barren footwall domain) and this showed that the calculated resources are insensitive to the choice of boundary option. It is however geologically sensible for the footwall contact between the porphyry (Domains 2, 3, 4 and 5) and footwall domain (Domain 1) to be treated as hard for grade estimation. All other domain transitions have been treated as soft boundaries.

17.1.9. Block Support Adjustment (Variance Adjustment)

17.1.9.1. General

The block support adjustment is one of the most important properties of a recoverable resource model based on non-linear estimation methods like MIK. It is an essential part of the model and involves important assumptions about the nature of the block grade distribution within each panel of the model.

Indicator Kriging provides a direct and reliable estimate of the histogram of grades of sample-sized units within each panel of the model provided the panel dimensions are of an appropriate size. However, ore is not selected on sample-sized units during mining; it is selected by shovels that have a minimum mining width and loaded into trucks that are despatched to either ore or waste. The flexibility of digging equipment and the size of the trucking equipment provide an indication of the size of the smallest block of rock that will be mined as ore or waste. To estimate with some accuracy the resources in a deposit that will be recovered with a certain set of mining equipment, the histogram of grades of sample-sized units in a panel provided by MIK must be adjusted to account for the size of the mining block.

There are a number of adjustment methods that can be used and most of these are described well in Journel & Huijbregts (1978) or Isaaks & Srivastava (1989). These methods make three reasonable assumptions:

- The average grade of sample-sized units and blocks within the panel is the same and is equal to the estimated average grade of the panel.
- The variance, or spread, of the block grades within the panel is less than the variance of grades of sample-sized units within the panel and the change of variance from sample-sized units to blocks can be calculated from the variogram of gold grades.
- The approximate shape of the histogram of block grades can be reasonably predicted by some appropriate assumptions.

17.1.9.2. The Variance Adjustment

The size of the variance adjustment needed to obtain the variance of the block grade distribution within the panel can be calculated using the rule of additivity of variances, which in the case of block support adjustment is often called Krige's Relationship:-

Var(samples in a panel) = Var(samples in a block) + Var (blocks in a panel)

The variance of sample grades in a panel and the variance of samples within a block can be directly calculated from the variogram of gold grade for the particular domain. The ratio of **Var(blocks in panel) to Var(samples in panel)** is that required to implement the block support adjustment. The variograms of gold grade are shown in Figures 17.1.8.2_1 and 17.1.8.2_4.

17.1.9.3. Shape of the Block grade Distribution

There are a number of rules of thumb that are useful when making judgements about the shape of the block grade distribution within each panel and they relate to the size of the variance adjustment ratio:-

- If the variance adjustment ratio is greater than 0.7, it may be useful to assume that the shape of the histogram of block grades is similar to that of the histogram of grades of sample-sized units. This is known as the Affine Correction method. Its application to gold deposits is usually inappropriate.
- If the variance adjustment ratio is between 0.3 and 0.7 and the information adjustment is negligible, then the Indirect Lognormal Correction method of Isaaks & Srivastava (1989) can be useful. This is a rule of thumb based on the experience of the authors.
- If the variance adjustment ratio is less than 0.3, a high degree of symmetrisation in the block grade histogram will occur and a direct lognormal assumption (Journel & Huijbregts, 1978, page 481) for the shape of the block histogram is an appropriate choice. This model is well supported by reconciliation studies of resource and grade control models.

17.1.9.4. The Information Effect

The variance adjustment described above is only part of the adjustment required in many gold deposits because the short scale variation in gold grades is extreme, as is the case at Sukari. This variance adjustment provides an estimate of the variance of *true* block grades under the assumption that grade control selection will operate with knowledge of the true block grades. While this assumption is never absolutely true, it can be a reasonable assumption in some deposits where the short scale variability is small and the grade control sampling density is high. In many deposits, however, an additional variance adjustment must be undertaken to account for the “Information Effect”.

In the absence of production information or grade control sampling, the Information Effect ratio is based on the variograms of gold grade and on the grade control sample spacing expected to be used during mining.

17.1.9.5. Variance Adjustments Applied to the Sukari Model

For Sukari, a total variance adjustment ratio ranging from 0.042 or 0.140 have been applied in estimating recoverable gold resources (Table 17.1.9.5_1). These ratios have been applied using the Direct Lognormal Correction method (i.e. incorporating symmetrisation of block grade distributions). Selective mining (SMU) dimensions of 4mE x 8mN x 3.0mRL have been assumed.

Table 17.1.9.5_1 Variance Adjustments				
	Domains 1, 2 and 7	**Domains 3, 6 and 8**	**Domain 4**	**Domain 5**
Panel to block adjustment Information effect Total ratio	0.099	0.042	0.096	0.140

17.1.10. Bulk Densities

The bulk densities ('BD') used in calculating tonnages in the Sukari resource model were obtained from the site generated BD data base and are shown in Table 17.1.10_1.

Table 17.1.10_1 Bulk Densities		
Rock Type	**Modelling Domain**	**Bulk Density**
Porphyry, Weathered	Domains 2, 4 and 6, 7, 8 Oxide sub-domains	2.63
Porphyry, Primary	Domains 3, 5, and 6, 7, 8 Sulphide sub-domains	2.66
Footwall, Weathered	Domains 1, Oxide	2.74
Footwall, Primary	Domains 1, Sulphide	2.78

17.1.11. Resource Classification

Panels in the resource model have been allocated a confidence category based on the number and location of samples used to estimate proportions and grade of each panel. The approach is based on the principle that larger numbers of samples, which are more evenly distributed throughout the search neighbourhood, will provide a more reliable estimate. The number of samples and the particular geographic configurations that may qualify the panel as Measured rather than Indicated or Inferred are essentially the domain of the Competent Person. The search parameters used to decide the classification of a panel resource in this study are:

- Minimum number of samples found in the search neighbourhood.

  For Measured and Indicated categories, this parameter is set to sixteen. For Inferred category, a minimum of eight samples is required. This parameter ensures that the panel estimate is generated from a reasonable number of sample data.

- Minimum number of spatial octants informed.

  The space around the centre of a panel being estimated is divided into eight octants by the axial planes of the data search ellipsoid. This parameter ensures that the samples informing an estimate are relatively evenly spread around the panel and do not all come from one drillhole. For Measured and Indicated categories, at least four octants must contain at least one sample. For Inferred panels, at least two octants must contain data.

- The distance to informing data.

  The search radii define how far the kriging program may look in any direction to find samples to include in the estimation of resources in a panel. Panel dimensions and

the sampling density in various directions usually influence the length of these radii. It is essential that the search radii be kept as short as possible while still achieving the degree of resolution required in the model. For Measured category, the easting, northing and elevation search radii are set to 30m, 30m and 15m respectively. For Indicated and Inferred categories, these radii are each expanded by fifty percent.

The number of samples and the particular geographic configurations that may qualify the panel as Measured rather than Indicated or Inferred may be a somewhat subjective decision. The confidence classification is essentially the domain of the Qualified Person and, in H&S's experience, the strategy adopted here results in a geologically sensible classification whereby Measured and Indicated panels are surrounded by data in close proximity. Inferred panels may occur on the peripheries of drilling but are still related to drilling data within reasonable distances. Downgrading of these confidence categories may result from a consideration of other factors such as QA/QC or areas of inadequate drillhole sampling, etc.

H&S are of the view that the categorisation used in the current study is appropriate for resource classification for the resources modelled at Sukari.

17.1.12. Resource Estimates

Table 17.1.12_1 lists estimated mineral resources by confidence categories as described in the previous section.

All reported resources have been adjusted to the present land surface and extend to 2mRL (to a maximum depth of 1,350m from surface).

Allowance has been made for ore extracted in historic workings. The volume of ore extracted by past mining is not significant when compared to volume of gold mineralisation estimated in the current resource model. Sample data used in resource modelling do not include any samples of the mined material i.e. there is no drilling that pre-dates the underground mining. For this reason the material removed from the resource model has been assumed to have a grade equivalent to the resource grade estimated for the panels intersected by the underground wireframes i.e. at each cut-off grade reported, the resources for the panels that are intersected by the underground wireframes have had their estimated recoverable proportion reduced by the mined proportion with no change to the estimated panel recoverable grade.

Table 17.1.12_1

Estimated Mineral Resources

Cutoff	Measured		Indicated		Measured plus Indicated		Inferred	
	Mt	g/t Au	Mt	g/t Au	Mt	g/t Au	Mt	g/t Au
0.5	84.01	1.42	151.72	1.47	235.73	1.45	68.9	1.6
0.6	71.29	1.57	130.31	1.62	201.60	1.60	58.4	1.8
0.7	61.23	1.72	112.85	1.77	174.08	1.75	50.1	2.0
0.8	53.05	1.87	98.40	1.92	151.45	1.90	43.7	2.1
0.9	46.27	2.02	86.25	2.07	132.52	2.05	38.3	2.3
1.0	40.54	2.17	75.95	2.22	116.49	2.20	33.8	2.5
1.2	31.44	2.48	59.73	2.53	91.18	2.51	26.7	2.9

This Mineral Resource estimate has been classified and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in November 2004. Furthermore, the resource classifications are also consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of 2004 ("the Code") as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia ("JORC").

The reporting of resource classification under the JORC Code and NI 43-101 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 Qualified Person responsible for the Mineral Resources is Nic Johnson who is a full time employee of Hellman & Schofield Pty Ltd and a member of the Australian Institute of Geoscientists. He visited the Sukari project site in April 2009.

17.1.13. Plots of the Sukari Resource Model

Figures 17.1.13_1, 17.1.13_2, 17.1.13_5 and 17.1.13_6 show a representative cross-section and two plans through the resource composites and Sukari resource model. Each figure shows resource panels scaled and coloured by the estimated resource tonnes above 0.7g/t Au cutoff grade (Note: the estimated proportion at 0.7g/t cut-off grade has been used to scale the east dimensions of the panels). Figures 17.1.13_3, 17.1.13_4, 17.1.13_7 and 17.1.13_8 show the same cross-section and plans, however, in these figures the resource model panels are coloured by confidence category (panels plotted at their full dimension). The model blocks in all figures are only shown below the end of May 2010 mined surface.

It should be noted that when viewing these plots through the resource model there are situations where the panels appear to be un-correlated to the mineralised intercepts in the neighbouring drillholes. This is occurring because of the way the resource panels have been presented. The panels plotted are only those that contain an estimated resource above 0.7g/t Au Cut-off and the proportion of ore has been used to scale the east dimension of the panel for presentation purposes. The scaling occurs about the panel centroid co-ordinate and therefore introduces the apparent miss-match between data and the resource panels.

Figure 17.1.13_1
Section 10,325mN Panel Estimated Proportion and Grades @0.7g.t Au Cutoff

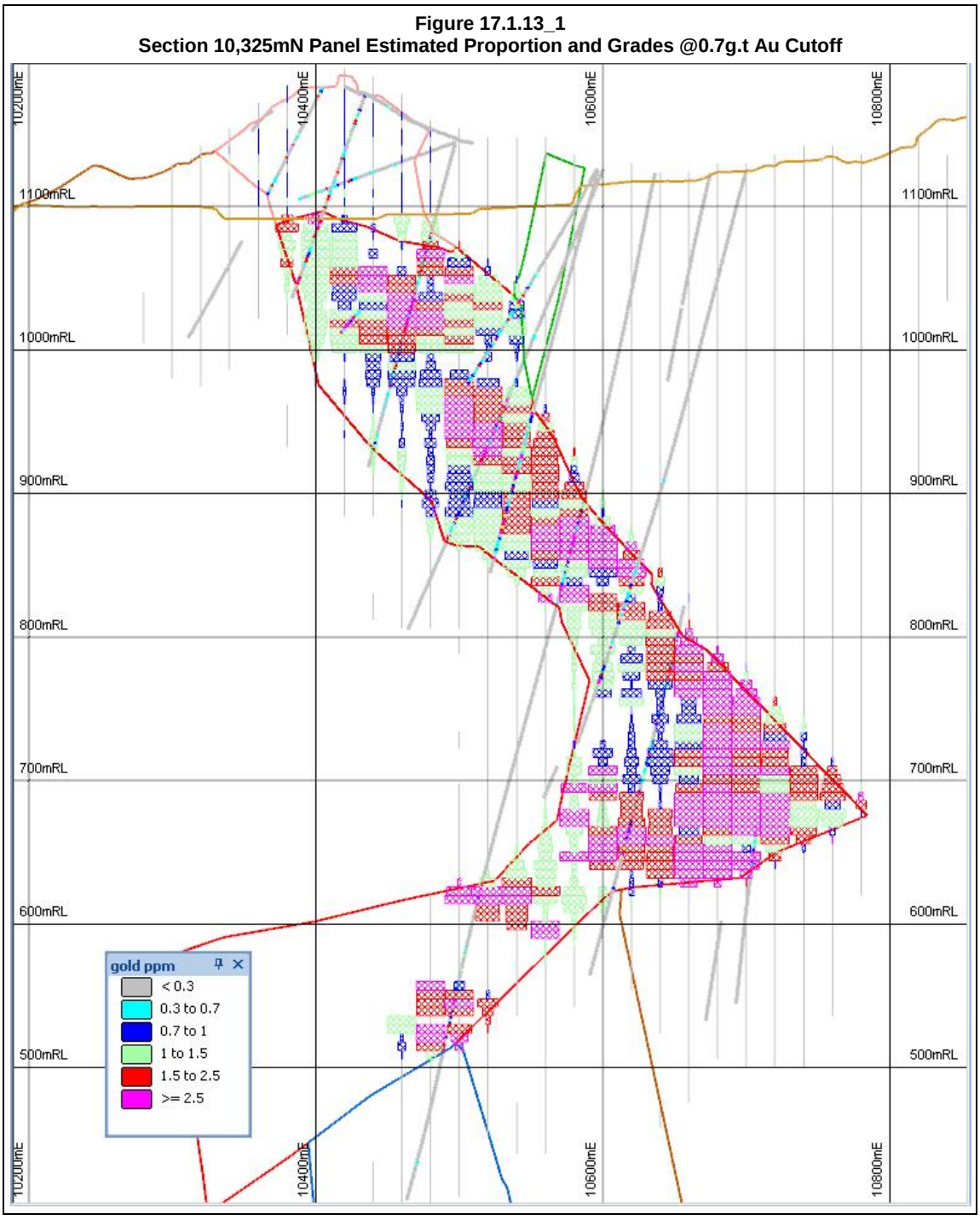


Figure 17.1.13_2
Section 11,125mN Panel Estimated Proportion and Grades @0.7g.t Au Cutoff

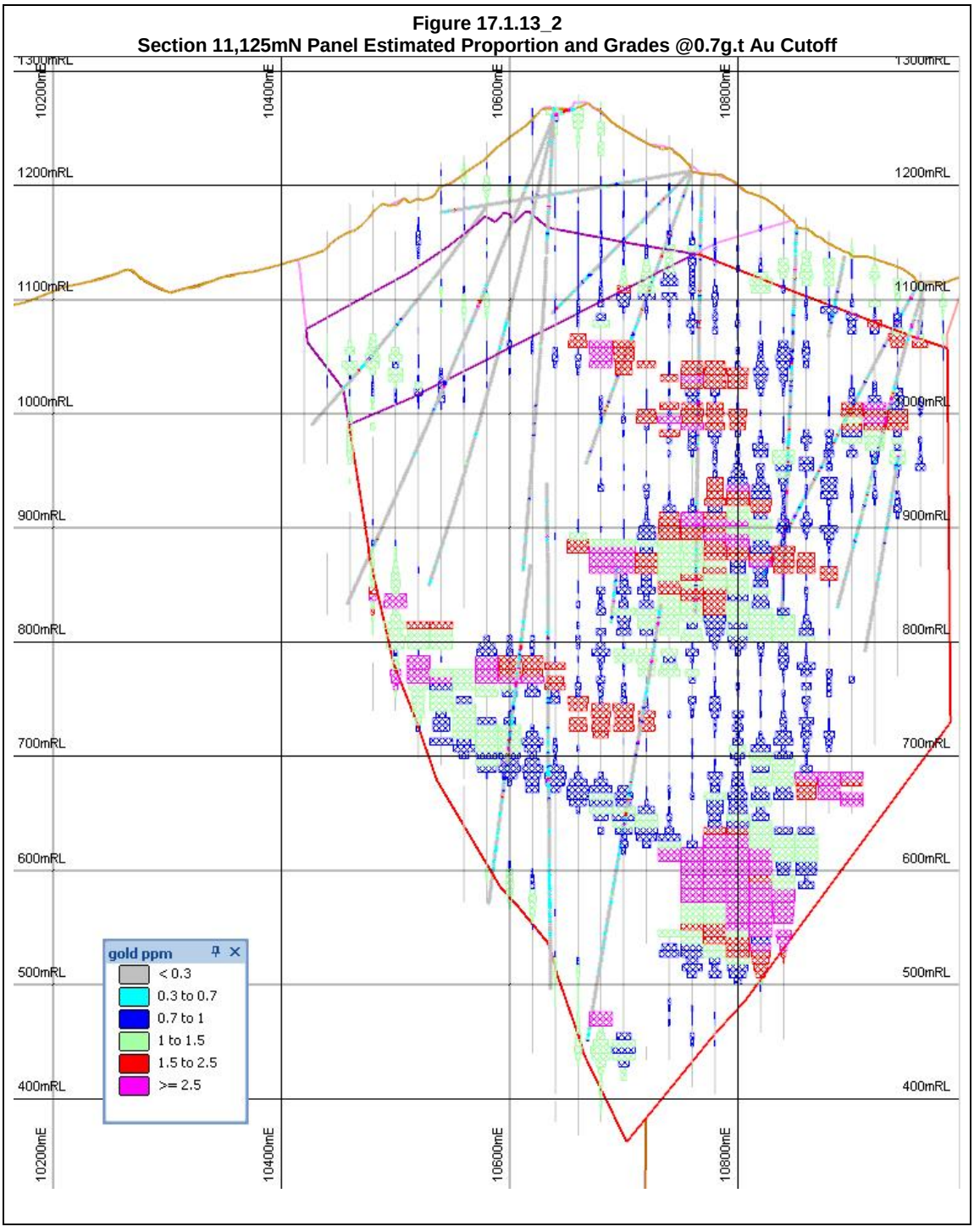


Figure 17.1.13_3
Section 10,325mN Panel Confidence Categories

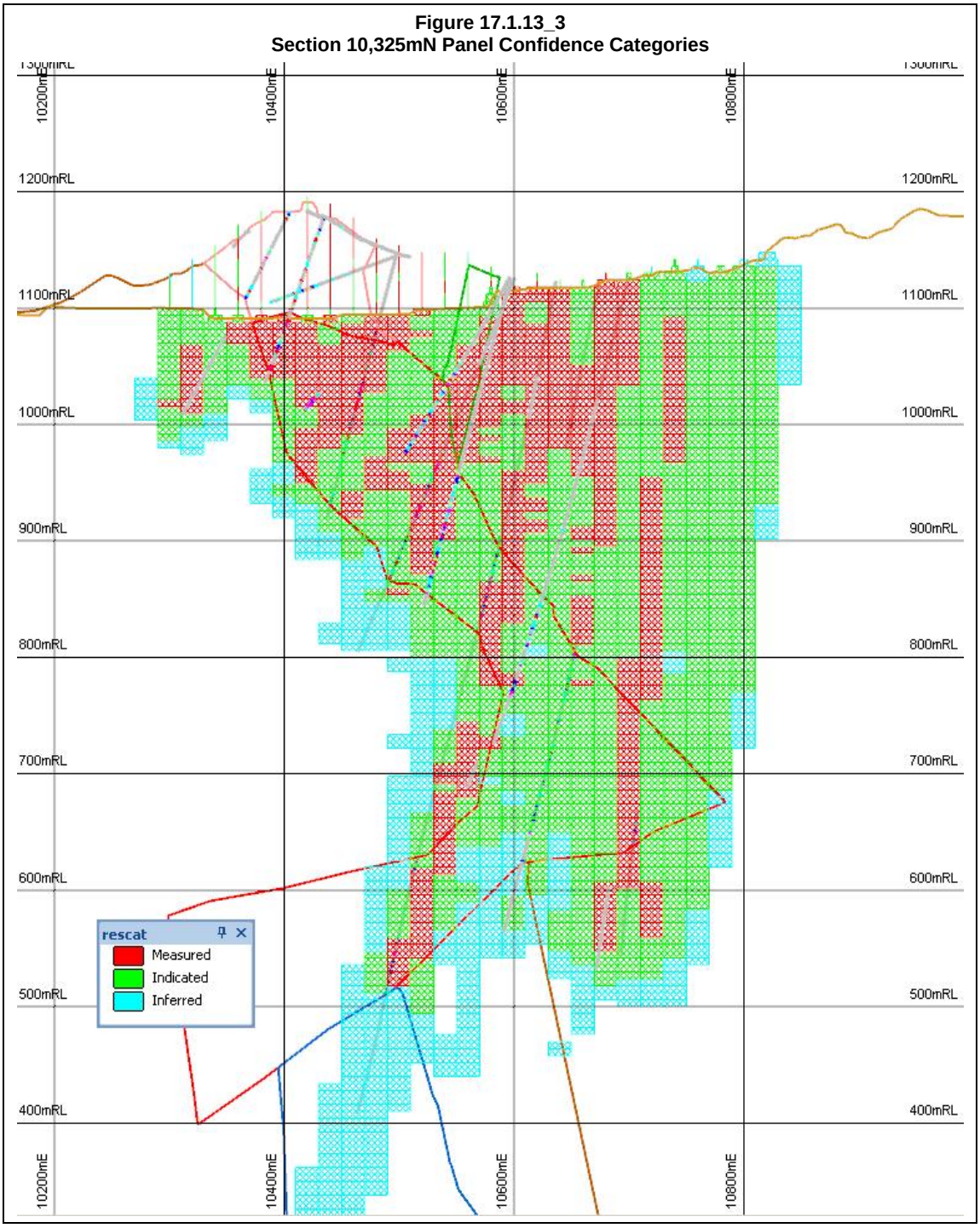


Figure 17.1.13_4
Section 11,125mN Panel Confidence Categories

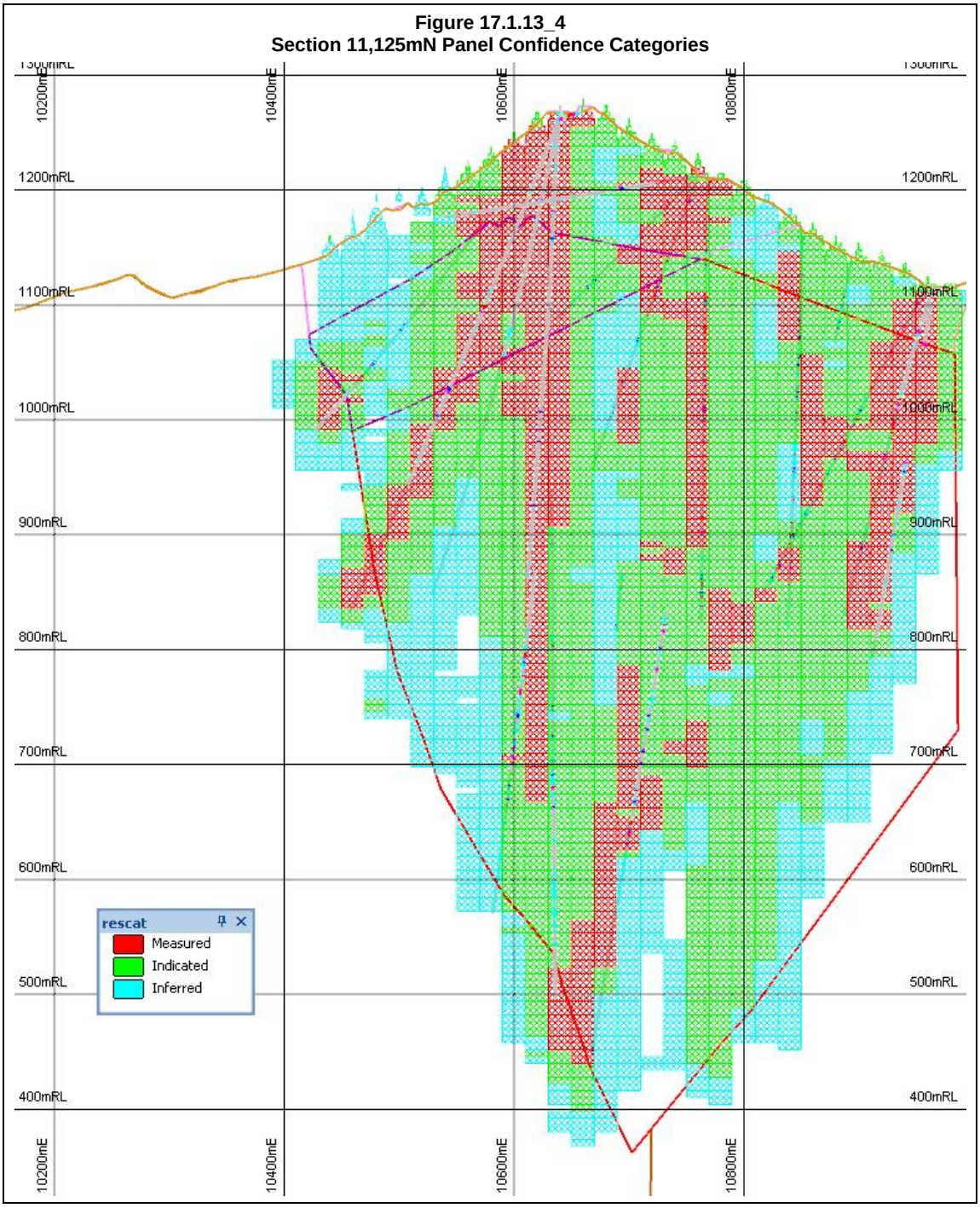


Figure 17.1.13_5
Panel Estimated Grades @ 0.7g/t cutoff 1103mRL

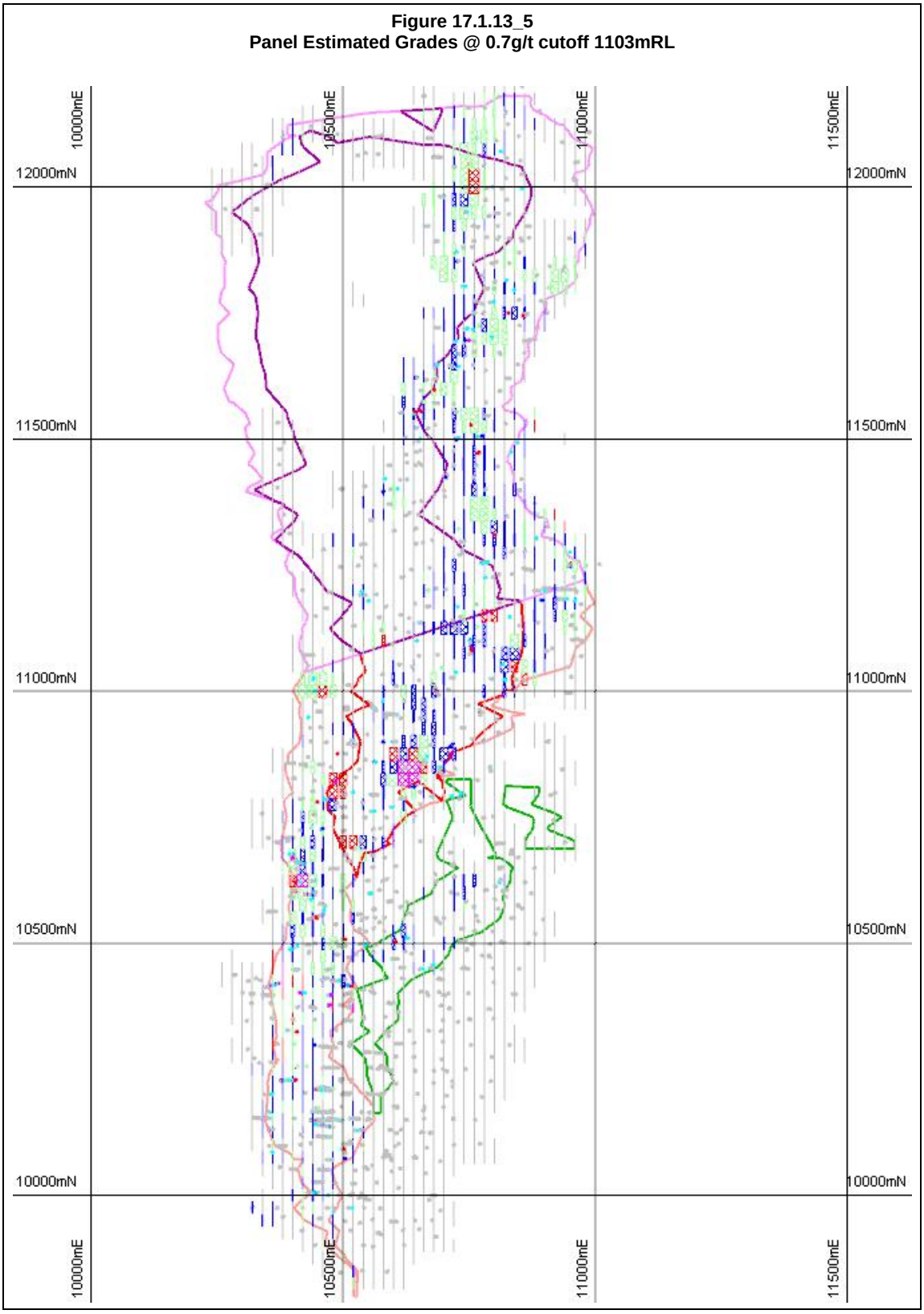


Figure 17.1.13_6
Panel Estimated Grades @ 0.7g/t cutoff 1001mRL

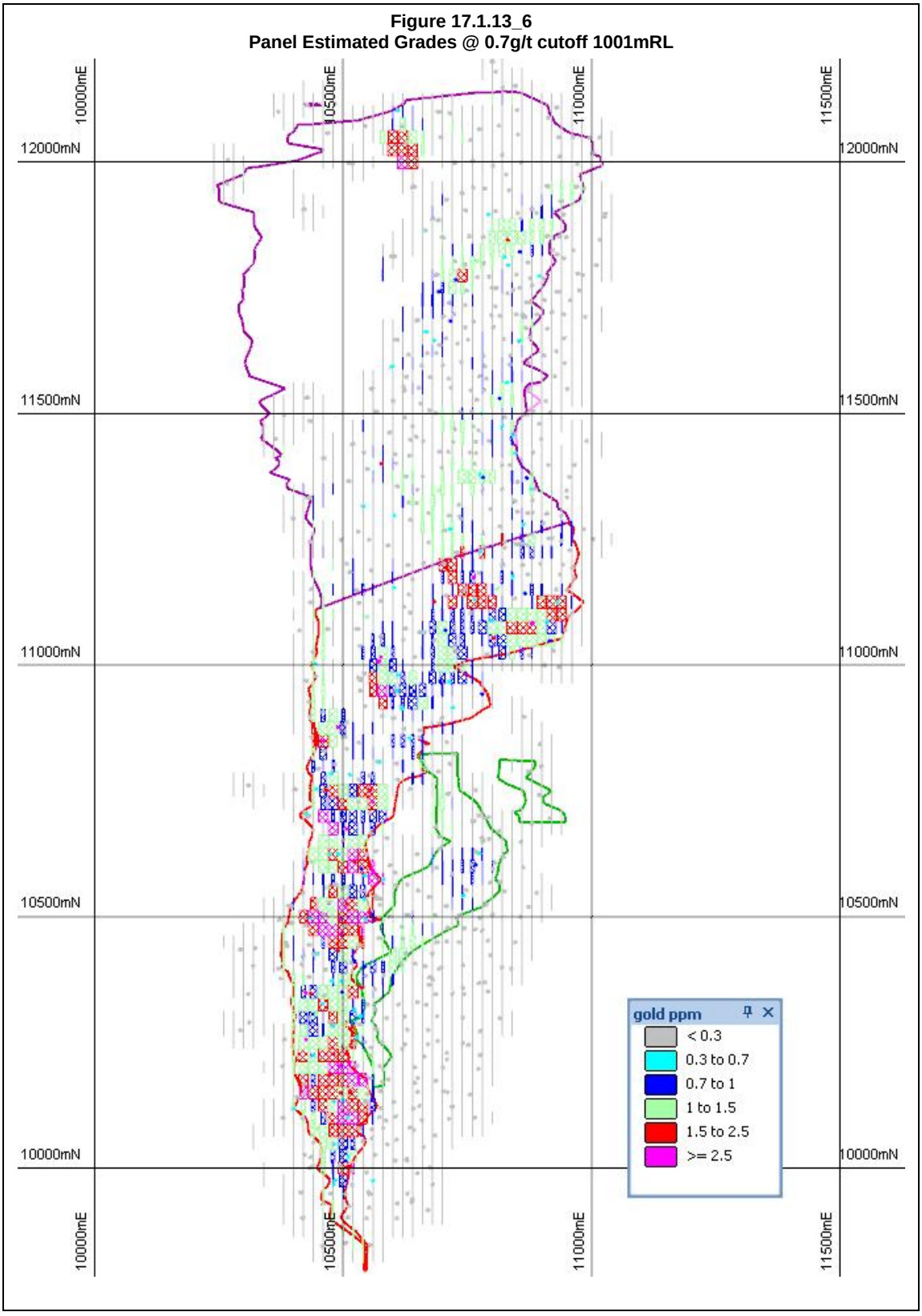


Figure 17.1.13_7
Panel Confidence Categories 1103mRL

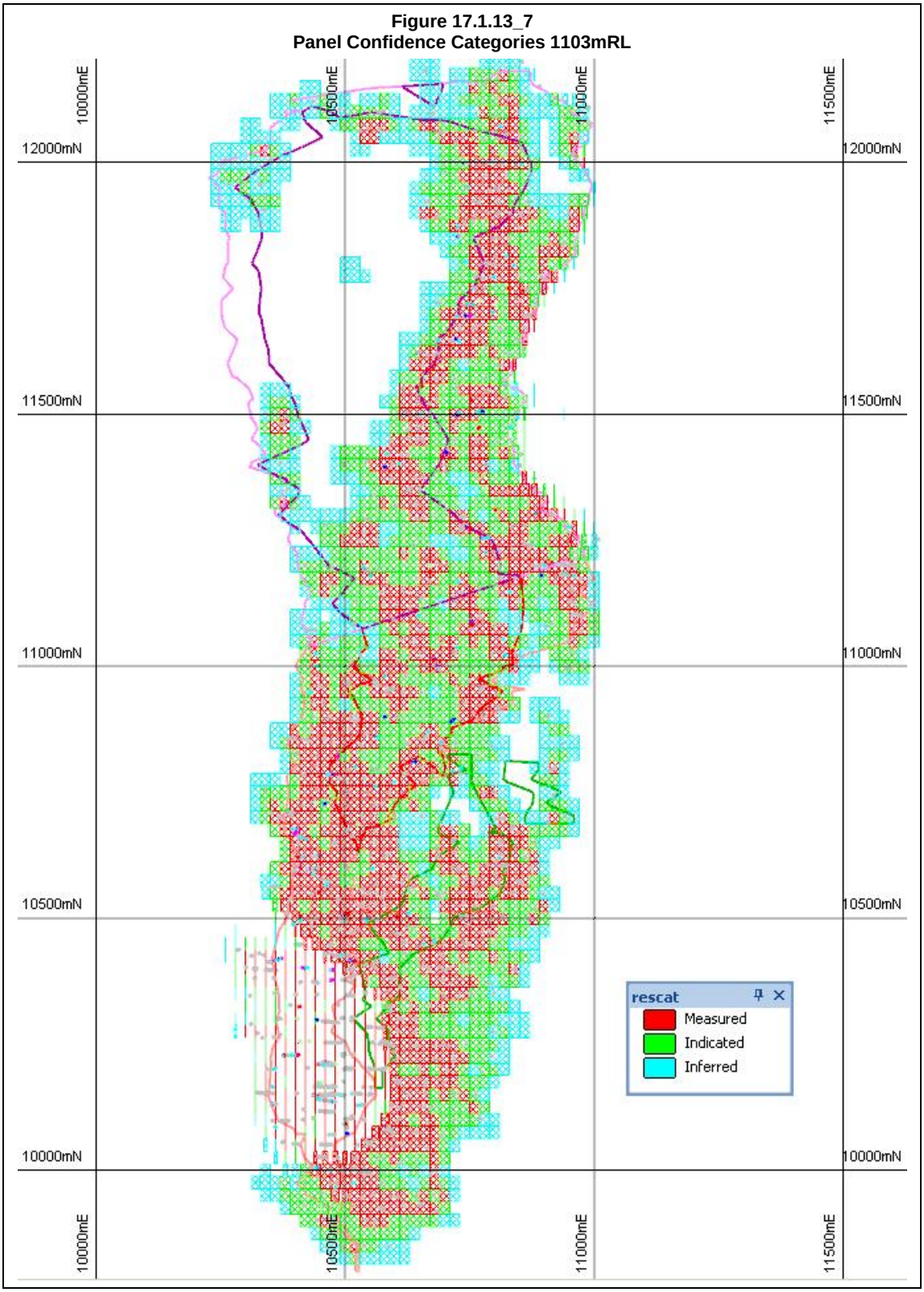
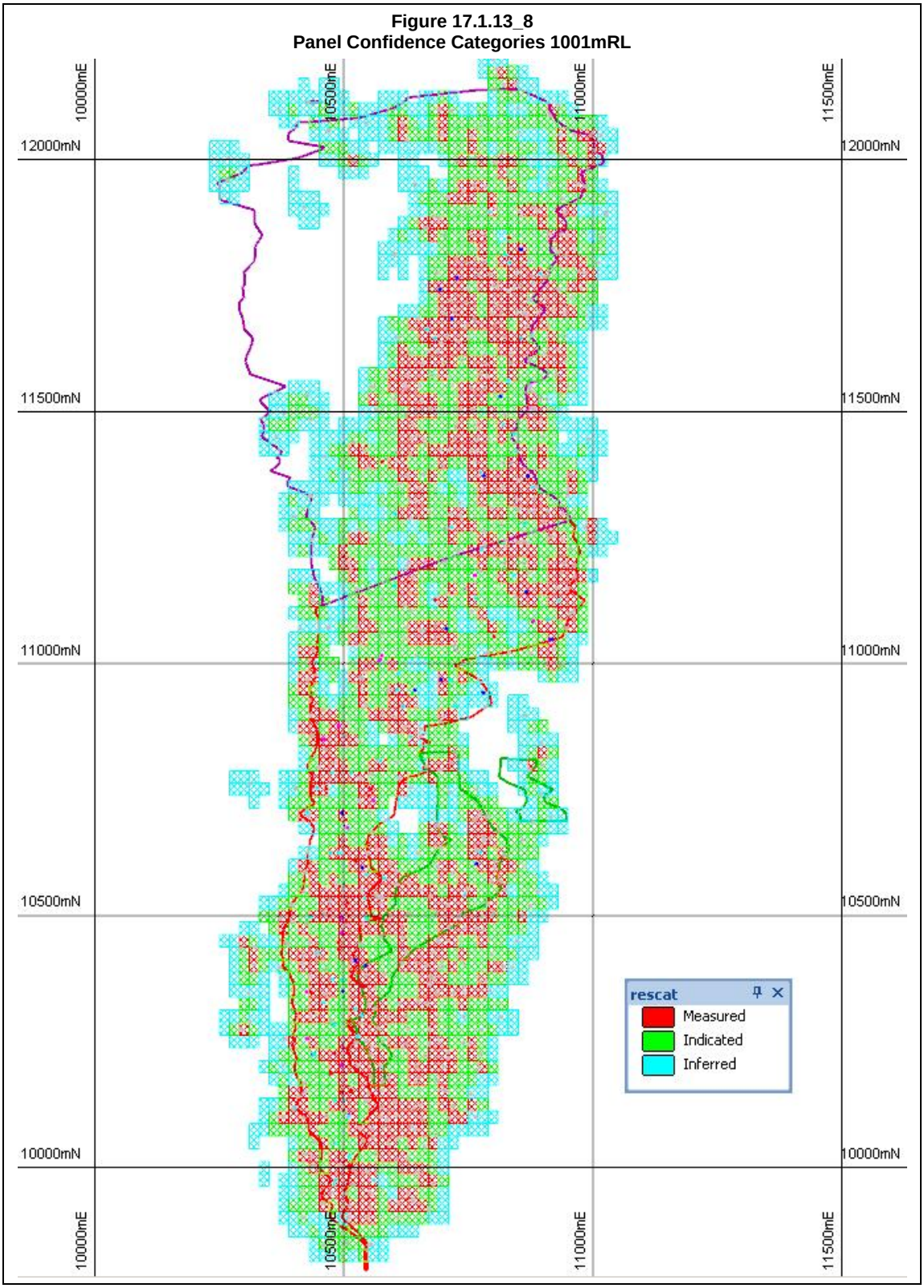


Figure 17.1.13_8
Panel Confidence Categories 1001mRL



17.2. Mineral Reserves

The Mineral Reserves for the Sukari Gold Project were estimated by Centamin as at September 2010. The Mineral Reserve estimate was based upon a Mineral Resource estimate prepared by H&S in June 2010.

The Sukari pit will produce ores of varying stages of weathering. The processing costs are expected to vary with the extent of the weathering and hence the cut- off grade applied also varies with the weathering state.

The grade estimates provided in the Mineral Resource model incorporates dilution for the appropriate selective mining unit (SMU) size for open pit mining equipment and no additional dilution was added. Ore recovery was assumed to be 100%.

The Mineral Reserves that have been declared are based on a gold price of US$900/oz and a gold cut-off grade of 0.4g/t Au for oxide, transitional & sulphide material.

The Sukari pit will be developed in a number of Stages and the Mineral Reserve is tabulated by mining stage in Table 17.2_1.

Table 17.2_1
Sukari Open Pit Mineral Reserve [2] – By Design Stages
(reported at a cut-off grade of 0.4g/t Au for oxide, transitional and sulphide material)

Stage	Oxide Ore		Sulphide Ore		Transitional Ore		Total Ore[3]		Waste	Strip Ratio
	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	(t/t)
1	0.03	0.67	1.93	2.65	1.23	1.75	4.43	1.65	3.33	0.75
2	2.20	0.85	11.47	1.42	3.26	1.17	16.94	1.30	34.88	2.06
3	2.19	0.76	15.34	1.59	0.94	0.79	18.47	1.45	66.18	3.58
Rem	12.96	0.82	139.28	1.37	11.10	0.75	163.34	1.28	1,025.70	6.28
Total	**17.38**	**0.82**	**168.02**	**1.41**	**16.53**	**0.91**	**201.93[1]**	**1.31**	**1,130.09**	**5.60**

Notes to Table: (1) Ore within design pit shells at 0.4g/t cut-off for all types
(2) Based on as mined surfaced as at 01 August 2010
(3) Excludes 45Mt of low grade oxide ore stockpiled for end of mine life feed

The total Mineral Reserves at Sukari were estimated at 245Mt of ore at an average grade of 1.15g/t Au for 9.1M contained ounces of gold.

The mining and treatment schedule developed through the Sukari Feasibility Study ("FS") uses an elevated cut-off grade through the early years to increase the head grade to the treatment facility. However material between this elevated cut-off grade and the cut-off grade used for the Mineral Reserve estimate is stockpiled and treated at the end of the project life.

The Mineral Reserves are contained within designed and scheduled open pits which were based upon the results of Lerchs Grossman pit optimisations of Measured and Indicated

Resources. The Inferred Resources which occur within the pit design are treated as waste in the production schedule and project economic evaluation.

This Mineral Reserve estimate has been classified and reported in accordance NI 43-101 and the classifications adopted by CIM Council in November 2004. Furthermore, the reserve classifications are also consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of 2004 ("the Code") as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia ("JORC"), with the minor exception that the Code refers to Ore Reserves while NI 43-101 refers to Mineral Reserves.

The Qualified Person who was responsible for the Mineral Reserve estimate was Andrew Pardey, a full time employee of PGM. Mr Pardey has the appropriate relevant qualifications and experience to be considered a Competent Person as defined in the Australasian JORC Code and a Qualified Person as defined in NI 43-101.

18 Additional Requirements for Reports on Development Properties and Production Properties

18.1 Mining

PGM are conducting the mining operations at the Sukari Gold Project themselves rather than utilising mining contractors. PGM adopted this approach in the absence of a local mining contracting industry in Egypt and in recognition of the long project life that would allow full utilisation of the mining fleet and the development of in house capabilities.

18.1.1 Mining Method

18.1.1.1 Load and Haul

All ore and waste is being mined using open pit gold mining methods consisting of four (4) O&KRH120 Excavators and currently fourteen (14) CAT785C haul trucks to carry out the bulk of the ore and waste mining.

The operation would be selective in terms of separating ore and waste. The degree of selectivity upon which the dilution and ore loss allowances are based reflects the scale of mining equipment, the proposed grade control method and the nature of the mineralisation. A selective mining unit ("SMU") of 4m wide by 8m long and 3m high was adopted by H&S for the block support adjustment employed in developing the Multiple Indicator Kriged ("MIK") resource model.

The mining fleet includes the requisite ancillary equipment (track & wheel dozers, motor graders, projects fleet for Tailings Storage Facility (TSF) and pioneering work, service trucks and water trucks) to construct and maintain the pit haul roads, ROM pad and waste dump.

Ore is being hauled to the run-of-mine ("ROM") pad adjacent to the primary crusher. The majority of ore is being direct tipped into the crusher with provision for ore to be stockpiled for reclaim by a front-end-loader operated as part of the crushing and milling operation. Ore is not expected to always be visually distinct from waste and hence ore and waste segregation would rely on reverse circulation ("RC") drilling, sampling and assays for grade control definition of ore blocks. Ore would be preferentially, but not exclusively, mined on dayshift.

Waste will be used for construction or hauled to the waste dumps on the west and east sides of the pit. Waste will be used to construct ramps to provide access on the hillside and to construct provide fill for the TSF. The haul roads from the pit to the ROM pad and waste dumps will also be constructed with waste rock sourced from the pit.

Acid generation from waste is not anticipated to be an issue and hence no consideration of acid forming waste has been included in the waste dump design or schedule. Low grade (sub grade) ore, if selectively mined and stockpiled, would be placed adjacent to or near the ROM pad and encapsulated under a waste dump should economics not allow treatment of this material towards the end of the mine life.

The load and haul costs are based upon actual mining cost and cost inputs developed by PGM personnel using haul profiles derived from pit designs. Mining benches are named according to the bench toe elevation in metres above a normal datum point ("mRL"), where the datum is often defined as sea level.

18.1.1.2 Grade Control

Grade control ("GC") samples will be collected through RC drilling campaigns. The proposed drilling pattern, appropriate for this style of gold mineralisation, is based on recommendations from H&S. GC holes will be drilled 12m or 24m deep, 5m apart on east-west orientated sections with the holes vertical. The drilling sections would be on 8m northing's.

The pattern would target the main structures and the associated Sukari stock work. The hanging wall porphyries would be drilled on the same pattern size (5m x 8m x 120) though with the holes orientated to azimuth 320° (mine grid) and angled -60° to obtain a representative coverage of the mineralised units.

The annual RC drilling program is estimated at 140,000m. Drill penetration rates are estimated to average 16m/hr based on a 5½ inch diameter hole. Drilling would be conducted on 11 hour shifts giving an average rate of 180m/shift. Two RC rigs would normally be utilised on day and night shift.

Personnel in the PGM Mining-Geology department will design the GC patterns. The GC holes would also be utilised as advance probe drilling for voids around the old Sukari underground mine. Personnel under the direction PGM operations staff would sample drill hole cuttings. Two samplers would be allocated to each RC drill rig. Samples will be collected as 1.5m composites down hole.

Sample preparation and analysis will occur on site with approximately 300 samples including QAQC samples being processed daily at the mine laboratory. The Fire Assay technique will be used. Quality control samples would also be submitted within the sequence of GC samples. These quality control samples would consist of blanks, standards, field resplits, pulps. Sample recoveries would be recorded.

Geological and positional data would be electronically recorded into a digital data logger at the rig and imported directly into the GC database, being a subset of the central geological database. The recorded geological and metallurgical properties for each hole would be used for ore type interpretation and delineation. Conditional Simulation has been judged suitable for the style of mineralisation at Sukari and would be used for ore estimation.

18.1.1.3 Mining Dilution and Ore Loss

The MIK resource model has parent cell dimensions that are 20m x 25m x 6m in the x, y and z directions respectively. When creating the resource model, an SMU size of 4m x 8m x 3.0m (x, y, and z directions) was assumed to predict the grade and tonnage at the elevated cut-off grades. As such, each parent cell can be considered to represent 30 SMU.

An MIK resource model is designed to incorporate an allowance for ore dilution during mining, based on the SMU dimensions. However, this allowance may not be applicable in situations where ore presents as a small percentage of the parent cell. For example if ore represents only 3% of the parent cell volume, equivalent to a single SMU or one and a half truck loads, the recovery of that ore would be difficult unless it was adjacent to ore within another parent cell.

In order to assess this ore dilution risk the total resource above a 0.6g/t Au was compared with the proportion of that resource occurring as less than 3% of a parent cell volume. Only

0.5% of the resource tonnage was located in parent cells with such a low ore proportion and, as might be expected, the grade of this material was considerably below the average grade, resulting in less than 0.3% of contained gold within the resource being constrained in very low proportion blocks. Given that this material would not have a material effect on the resource estimate no adjustment was made to eliminate it from the model.

18.1.1.4 Drill and Blast

It has been designed and costed that all ore and waste requires blasting with no "free dig" material. There may be free dig material at the foot of the slopes of the hill, however the tonnage of this transported material is expected to be small. A powder factor of 0.86kg/bcm is currently being used in Sulphide/Transitional material and 0.76kg/bcm in oxide. This powder factor is the weighted average powder factor of a study that considered the rock types at Sukari and the primary crusher specifications.

All drilling is conducted on 12m benches with predominantly 140mm holes drilled on a nominal 3.8m by 4.3m pattern with an average powder factor of 0.86kg/bcm however variations to pattern size, hole diameter and Pf are also used depending on rock type, structure and lithology.

18.1.1.5 Other Mining Activities

Haul roads are constructed and maintained by PGM personnel.

Waste dump slopes will be progressively battered down to their final profiles during construction.

All production mining activities at Sukari operate on both 12 hour day and night shifts. Permanent lighting for night time operation is installed at fixed and moving locations close to mains power such as the crusher feed area and haul road intersections on and near the ROM pad.

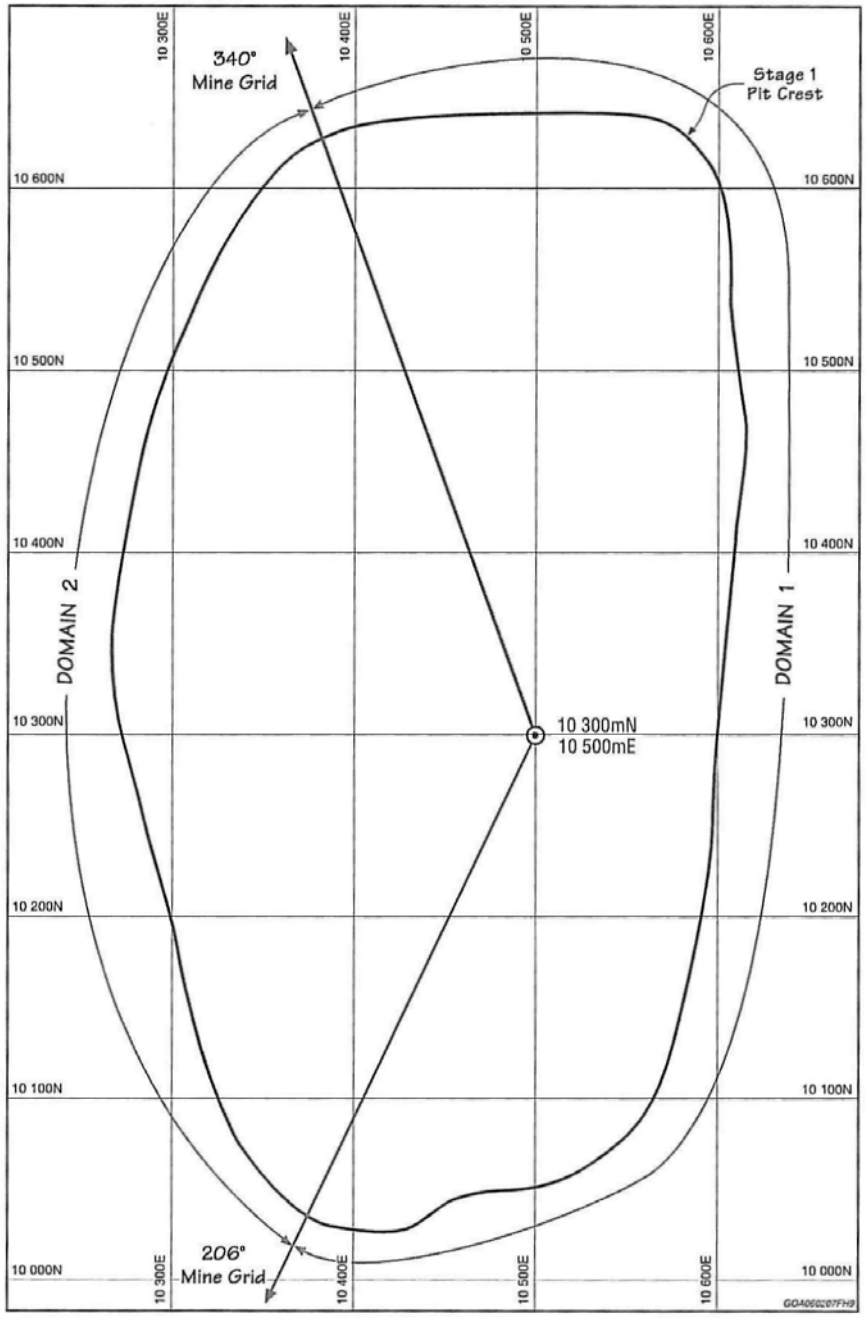
Portable lighting towers, trailer mounted diesel generator sets with banks of halogen floodlights mounted on an easily erected tower, are be used to illuminate the remaining night time working areas. Typically, lighting towers are used at the excavating face, dumping face, other locations around the pit perimeter to give overall illumination of working areas and ramp intersections. Lighting towers are also required for night shift drilling crews.

18.1.2 Geotechnical Considerations

The pit slope designs were based upon advice from George, Orr and Associates (Australia) Pty Ltd ("GOA").

The geotechnical recommendations divide the pit into two domains in plan. The West wall being categorised as Domain 2 and the North, East and South walls categorised as Domain 1 as shown in Figure 18.1.2_1.

Figure 18.1.2_1
Pit Slope Regions (GOA)



GOA makes recommendations for the pit GOA makes recommendations for the pit slopes in the initial stages of the pit and recommends additional data collection and observations through the mining of pit Stage 1 to be used to refine the slope designs for later pit stages. For the purposes of the DFS, the slope designs recommended for the initial stages have been applied to all stages.

The latest slope design specifications are summarised in Table 18.1.2_1.

Table 18.1.2_1

Slope Design Specifications

	Domain 1(H/W)	Domain 2 (F/W)
Batter Angle	80 degrees	80 degrees
Normal Berm Width	7.3m (berm width calculated based on face angle and overall slope angle)	10m
Wide Berm Width	None	None
Normal Berm Interval	12m	12m
Upper North Wall - Wide Berm Interval (north wall, 1130mRL and 1080mRL)	15m	15m

GOA further recommended that no haul roads be developed on the West wall of the pit.

18.1.3 Hydrogeology and Hydrology Input

The Sukari region is generally quite dry. Rainfall events are very infrequent but of high intensity and subsequently there is a risk of high volumes of water shedding from the Wadi into the pit. In-pit water inflows are expected to be low volume and localised.

Knight Piesold ("KP") developed a surface water management plan for the Sukari Gold Project, in consultation with the stakeholders.

Historical records of rainfall events were limited, however KP used the records of a high intensity rainfall event in November 2001 to develop the plan. The exact watershed flow path during a heavy rainfall event was difficult to establish due to the flat undefined nature of the Wadi valleys. KP inferred the natural watershed, based on topographical information supplied by PGM. Conservative assumptions were made to determine the critical flow paths.

The KP plan sets out a system to control surface water using drainage channels and diversion bunds. Topographical features and gravity flow were utilised such that pumping would not be required for the surface water system. KP recommends that three major diversion bunds be built in addition to the pit diversion system. Diversion bunds would be constructed to protect the plant site, pit, waste dump and haul road network.

The pit diversion system would comprise a series of small steep catchments and bunds, constructed concurrently with pit development that would collect surface flows and discharge the water away from the pit. The diversion drains would be excavated in competent rock such that the water is discharged away from the pit with minimal water seeping back towards the pit, potentially energising an in-pit structure.

In-pit water inflows and rainfall captured within the pit would be managed with in-pit sumps and pumps. Booster pumps would be required with increasing pit depth. In-pit water collected in sumps, would be pumped to a storage dam on the surface and be used as process water in the plant or haul road dust suppression.

18.1.4 Pit Optimisation

18.1.4.1 Model Preparation

The geological resource model was imported into Mine Sight – Meds software and a mine planning model was built.

The mine planning model uses the information in the resource model to create material types that are used for the optimisation. The process used to build the model is detailed in the PGM manual "Building a Mine Planning Model"

The material types are divided into Ore and Waste types.

Ore types include

- Oxide ore,
- Transitional ore
- Sulphide ore.
- Inferred ore

Oxide, transitional and sulphide ore carry a resource classification of measured or indicated.

An inferred ore material type is also carried to optimisation but is not used in the pit shell selection process. It is used to investigate the impact of converting this resource to measured and indicated on a potential pit

All ore types are associated with the porphyry zone or associated shears.

Waste types include footwall waste, porphyry waste (this is internal waste within the porphyry) hanging wall waste and fill which is material that has been used to create access haul roads within the pit in the last 18 months.

18.1.4.2 Mining Costs

Mining costs were derived from

- Actual production and cost performance data incurred to date
- Costs detailed in completed and active contracts (e.g. explosive supply and tyre management contracts)
- Assumptions on productivities and consumable and fluid usage collated from actual data derived during the last 12 months.
- Consumption and productivity rates are based upon actual performance to date.

The base case mining costs used for pit optimisations are detailed in Tables 18.1.4.2_1 and 18.1.4.2_2.

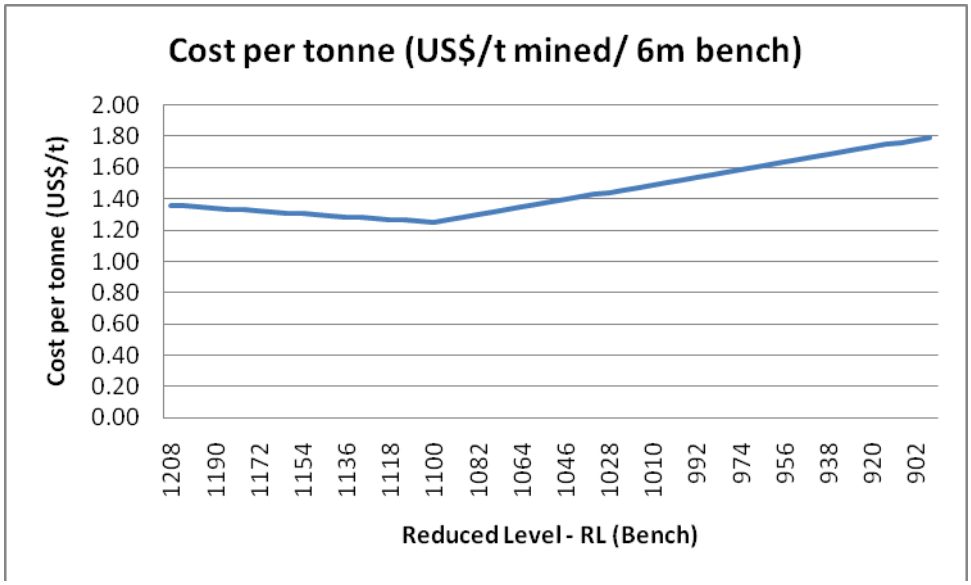
Table 18.1.4.2_1 Base Case Mining Unit Costs used for Pit Optimisations	
BASE Mining Unit Cost Parameters	
Drill ($/t)	$0.25
Grade Control ($/t)	$0.15
Blast($/t)	$0.25
Load ($/t)	$0.40
Haul (variable with mining depth)	*$0.00*
Ancillary	$0.20
Input to Whittle 4X plus the variable haul cost	$1.25

The variable haulage costs were estimated according to the following steps:

1. Calculating the hourly operating cost of a 785C dump truck. The 785C dump truck is the main haulage unit at the Sukari Gold Project.
2. Completing a preliminary optimisation to determine the potential size of an open pit.
3. Developing a waste dump that can contain the volume of the potential pit
4. Generating simplified haul profiles representative of the pit extremities, pit centroid and hauling to the dump developed at step 3.
5. Calculating haul costs using Talpac and extrapolating between levels to give the average change in haulage cost with depth.

The haulage cost by relative level in the model is given in Figure 18.1.4.4_1.

Figure 18.1.4.2_2
Haulage Cost Estimate with Depth



18.1.4.3 Processing Costs

Processing costs used in optimisation were taken from actual operating data and budgeted estimates for 2010.These costs were assumed to be appropriate for the nominated mill throughput rate of 4Mtpa increasing to 5Mtpa in 2011.

18.1.4.4 Administration Costs

Administration costs were taken from actual operating data and 2010 estimates.

18.1.4.5 Metallurgical Recoveries

Recoveries used for optimisation were taken from 2010 estimates based on feasibility work.

Table 18.1.4.5_1
Pit Optimisation Metallurgical Recoveries and Processing Costs

Ore Type	Processing Cost US$/t	Total Recovery
Oxide	1.36	50.0%
Transitional	8.50	93.4%
Sulphide	8.50	93.4%

18.1.4.6 Optimisation Results

Using the costs, metallurgical and geotechnical parameters a pit optimisation of the resource model was conducted. Only measured and indicated resources were used to generate the shells and

The option to limit the size of the open pit by considering underground mining methods was not used. The assumptions used for a potential underground operation were

The optimisation results excluding Inferred Resources using the reference gold price of $900/oz are summarised in Figures 18.1.4.6_1 and 18.1.4.6_2. These figures show the increasing number of recovered ounces contained within pit shells of increasing size. For each pit shell the undiscounted operating surplus is estimated using the base case costs and gold price.

Two indicative schedules are then developed. The best case schedule assumes that each incremental optimisation shell is mined out before commencing the next. This approach reflects the staging of the pit to an extent that is impractical. The worst case schedule assumes that there is no staging of the pit and that each bench is mined to the final limits and completed prior to commencing the next.

In practice, a designed schedule is likely to fall between the two cases. These schedules were used to develop discounted operating surplus estimates using an 8% discount rate.

The individual optimisation results are tabulated in Table 18.1.4.6_1

Figure 18.1.4.6_1
Optimisation Results

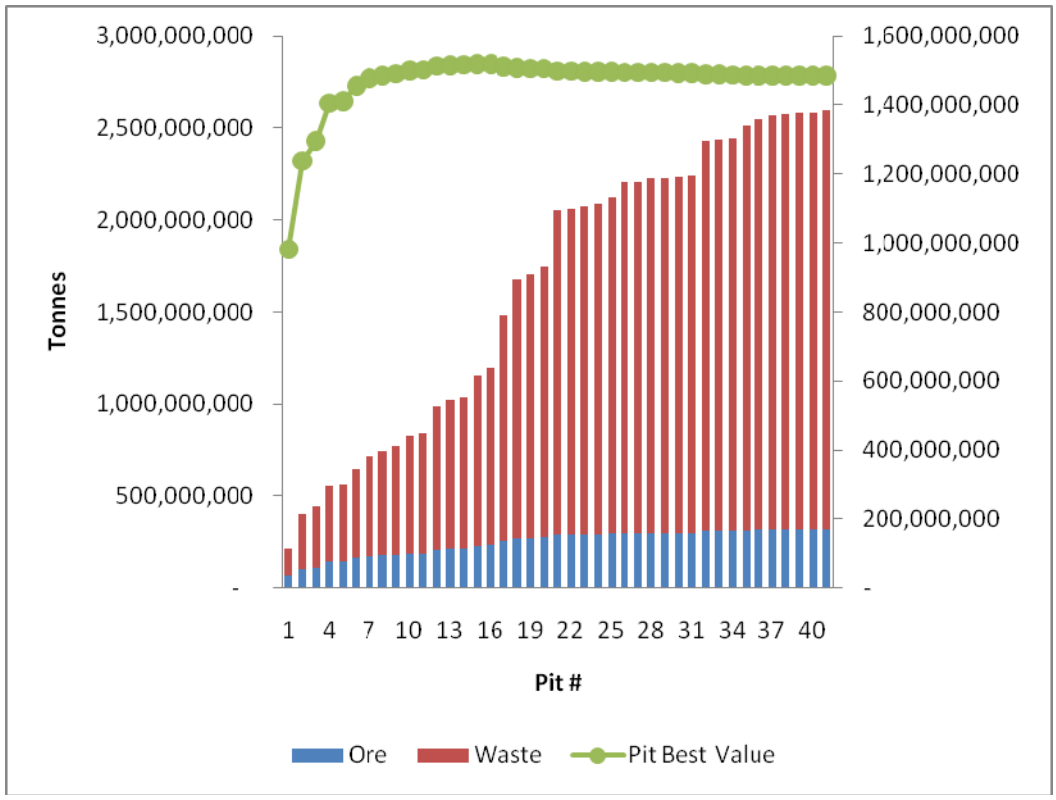


Figure 18.1.4.6_2
Optimisation Results

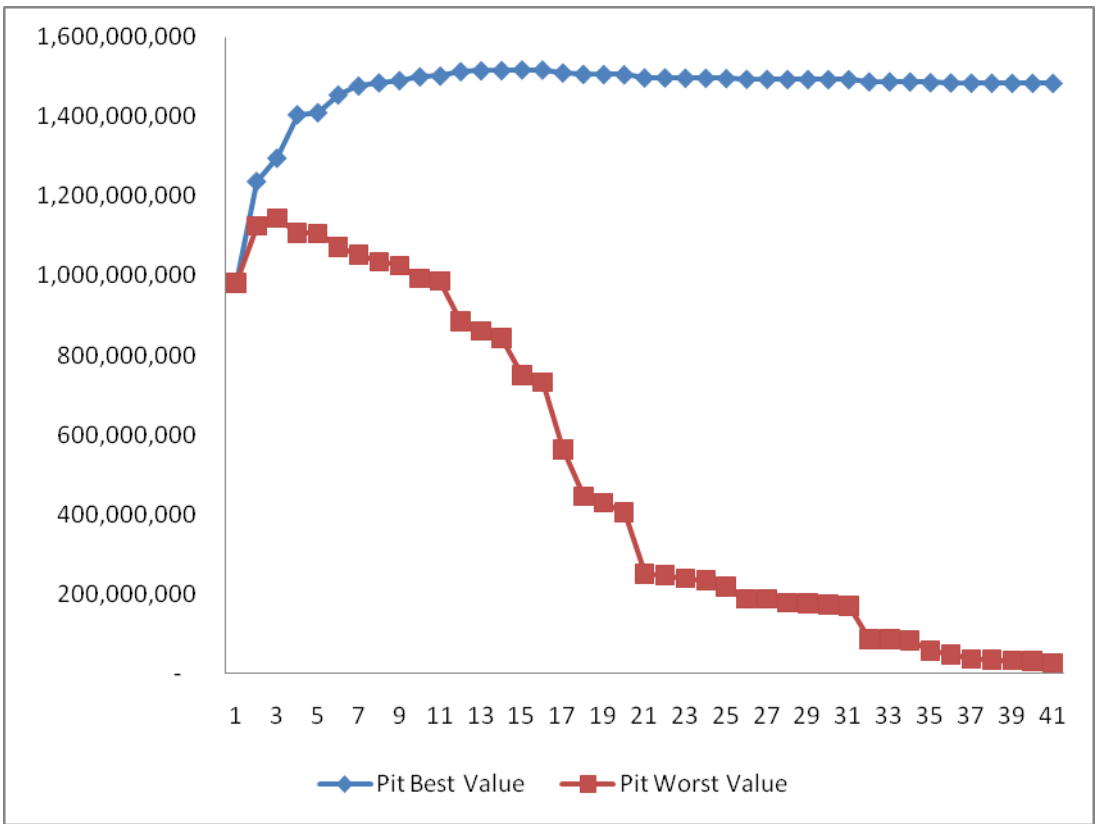


Table 18.1.4.6_1

Sukari Pit Optimisation Results

Final pit	Total Rock tonne	Total Waste tonne	Total Ore Process Tonnes tonne	Strip Ratio tonnes	Grade (g/t) best	Units output AU best	Units output AU (grams) specified	Units output MILL AU best	Units output LEACH AU best	Open pit cashflow best $ disc	Open pit cashflow specified $ disc	Open pit cashflow worst $ disc	Mine life years best	Mine life years specified	Mine life years worst
1	216,754,426	147,869,910	68,884,516	2.1	1.44	90,867,101	90,867,101	88,563,930	2,303,171	981,720,363	981,720,363	981,720,363	9.17	9.17	9.17
2	398,519,476	297,869,575	100,649,901	3.0	1.44	133,430,614	133,430,614	130,656,870	2,773,744	1,237,461,905	1,124,547,848	1,124,547,848	12.73	12.68	12.68
3	445,089,475	332,683,749	112,405,726	3.0	1.41	145,478,681	145,478,681	142,272,958	3,205,723	1,296,084,437	1,143,884,607	1,143,884,607	13.78	13.76	13.76
4	553,293,001	408,720,222	144,572,779	2.8	1.32	174,177,118	174,177,118	170,078,275	4,098,843	1,405,319,690	1,107,527,464	1,107,527,464	16.70	16.90	16.90
5	561,121,620	414,610,465	146,511,155	2.8	1.31	175,833,657	175,833,657	171,583,512	4,250,145	1,410,515,196	1,106,230,778	1,106,230,778	16.85	17.08	17.08
6	646,888,646	482,899,623	163,989,023	2.9	1.28	191,958,971	191,958,971	187,278,174	4,680,797	1,455,138,618	1,072,080,182	1,072,080,182	18.45	18.82	18.82
7	716,402,165	544,036,038	172,366,127	3.2	1.28	201,856,879	201,856,879	197,127,893	4,728,986	1,477,623,986	1,053,273,431	1,053,273,431	19.44	19.78	19.78
8	745,325,366	567,899,326	177,426,040	3.2	1.27	206,289,326	206,289,326	201,439,861	4,849,465	1,485,485,961	1,035,175,087	1,035,175,087	19.90	20.33	20.33
9	771,460,076	591,008,515	180,451,561	3.3	1.27	209,458,063	209,458,063	204,518,333	4,939,730	1,490,978,054	1,025,370,678	1,025,370,678	20.17	20.63	20.63
10	828,930,272	641,970,017	186,960,255	3.4	1.26	216,177,572	216,177,572	211,211,248	4,966,323	1,500,532,011	992,995,033	992,995,033	20.92	21.43	21.43
11	843,512,085	654,644,493	188,867,592	3.5	1.26	217,907,251	217,907,251	212,919,916	4,987,335	1,502,570,497	986,330,715	986,330,715	21.13	21.62	21.62
12	990,271,724	782,490,829	207,780,895	3.8	1.23	234,187,333	234,187,333	229,103,196	5,084,137	1,513,662,513	884,407,816	884,407,816	23.81	23.78	23.78
13	1,023,077,387	810,508,409	212,568,978	3.8	1.22	237,956,057	237,956,057	232,859,758	5,096,300	1,515,715,136	861,271,182	861,271,182	24.28	24.30	24.30
14	1,036,603,759	821,472,778	215,130,981	3.8	1.21	239,612,861	239,612,861	234,436,585	5,176,276	1,516,395,754	844,070,584	844,070,584	24.51	24.59	24.59
15	1,157,319,624	929,033,880	228,285,744	4.1	1.20	250,580,170	250,580,170	245,355,576	5,224,595	1,518,028,257	750,324,493	750,324,493	26.33	26.27	26.27
16	1,194,542,689	961,938,298	232,604,391	4.1	1.19	253,981,037	253,981,037	248,750,414	5,230,623	1,517,832,151	732,322,442	732,322,442	26.76	26.78	26.78
17	1,484,032,988	1,230,436,955	253,596,033	4.9	1.18	275,112,793	275,112,793	269,881,878	5,230,915	1,510,447,432	563,431,875	563,431,875	31.48	30.08	30.08
18	1,675,225,347	1,406,074,839	269,150,508	5.2	1.17	289,339,035	289,339,035	284,084,738	5,254,297	1,506,867,635	445,589,306	445,589,307	33.99	32.59	32.59
19	1,704,535,125	1,433,415,768	271,119,357	5.3	1.17	291,310,871	291,310,871	286,055,453	5,255,418	1,506,462,155	428,628,256	428,628,256	34.37	32.94	32.94
20	1,748,186,202	1,474,460,973	273,725,229	5.4	1.17	294,111,762	294,111,762	288,853,919	5,257,843	1,505,765,705	405,876,097	405,876,097	34.99	33.42	33.42
21	2,053,523,438	1,762,240,467	291,282,971	6.0	1.17	312,674,993	312,674,993	307,416,834	5,258,159	1,497,922,817	250,550,959	250,550,959	39.88	36.85	36.85
22	2,057,472,276	1,766,024,730	291,447,546	6.1	1.17	312,879,734	312,879,734	307,621,575	5,258,159	1,497,847,880	247,912,259	247,912,259	39.90	36.91	36.91
23	2,077,445,983	1,785,022,037	292,423,946	6.1	1.17	313,988,964	313,988,964	308,730,365	5,258,599	1,497,380,024	238,870,709	238,870,709	39.99	37.12	37.12
24	2,086,979,668	1,794,156,735	292,822,933	6.1	1.17	314,459,485	314,459,485	309,191,031	5,268,453	1,497,184,821	234,339,358	234,339,358	40.04	37.22	37.22
25	2,120,443,981	1,825,702,604	294,741,377	6.2	1.17	316,226,305	316,226,305	310,956,349	5,269,956	1,496,387,647	217,268,795	217,268,795	40.52	37.59	37.59
26	2,204,832,288	1,906,912,750	297,919,538	6.4	1.17	320,313,242	320,313,242	315,032,945	5,280,297	1,494,414,069	187,939,179	187,939,179	41.72	38.28	38.28
27	2,207,179,134	1,909,147,305	298,031,829	6.4	1.17	320,427,440	320,427,440	315,147,143	5,280,297	1,494,356,371	186,966,074	186,966,074	41.76	38.31	38.31
28	2,226,713,671	1,928,021,106	298,692,565	6.5	1.17	321,280,107	321,280,107	315,999,809	5,280,297	1,493,908,583	177,322,737	177,322,737	42.05	38.51	38.51
29	2,229,030,549	1,930,234,574	298,795,975	6.5	1.17	321,389,009	321,389,009	316,108,712	5,280,297	1,493,845,807	176,584,217	176,584,217	42.07	38.53	38.53
30	2,234,374,987	1,935,099,071	299,275,916	6.5	1.17	321,666,636	321,666,636	316,384,390	5,282,245	1,493,674,067	172,881,502	172,881,502	42.15	38.61	38.61
31	2,238,778,230	1,939,212,777	299,565,453	6.5	1.17	321,865,206	321,865,206	316,579,194	5,286,012	1,493,540,201	170,251,069	170,251,069	42.21	38.67	38.67
32	2,432,231,793	2,121,396,729	310,835,064	6.8	1.15	330,618,753	330,618,753	325,317,864	5,300,890	1,488,042,383	86,070,767	86,070,767	44.97	40.96	40.96
33	2,433,062,809	2,122,167,474	310,895,335	6.8	1.15	330,657,622	330,657,622	325,356,732	5,300,890	1,488,017,806	85,695,101	85,695,101	44.99	40.97	40.97
34	2,443,797,741	2,132,598,462	311,199,279	6.9	1.15	331,057,700	331,057,700	325,756,811	5,300,890	1,487,764,753	82,345,568	82,345,568	45.14	41.06	41.06
35	2,511,575,951	2,197,522,104	314,053,847	7.0	1.15	333,766,691	333,766,691	328,465,802	5,300,890	1,486,083,851	57,080,402	57,080,402	46.22	41.79	41.79
36	2,544,697,048	2,229,550,264	315,146,784	7.1	1.15	334,993,885	334,993,885	329,692,995	5,300,890	1,485,313,013	46,469,104	46,469,104	46.58	42.10	42.10
37	2,571,971,323	2,256,173,892	315,797,431	7.1	1.15	335,929,970	335,929,970	330,629,080	5,300,890	1,484,719,943	36,209,339	36,209,339	46.97	42.38	42.38
38	2,575,978,577	2,260,092,079	315,886,498	7.2	1.15	336,064,059	336,064,059	330,763,169	5,300,890	1,484,632,410	34,803,367	34,803,367	47.03	42.42	42.42
39	2,581,562,544	2,265,493,766	316,068,778	7.2	1.15	336,252,043	336,252,043	330,951,153	5,300,890	1,484,501,180	32,551,407	32,551,407	47.11	42.49	42.49
40	2,583,237,366	2,267,129,396	316,107,970	7.2	1.15	336,302,071	336,302,071	331,001,182	5,300,890	1,484,465,467	31,624,000	31,624,000	47.13	42.52	42.52
41	2,595,351,858	2,279,023,531	316,328,327	7.2	1.16	336,658,687	336,658,687	331,357,797	5,300,890	1,484,203,176	26,683,852	26,683,852	47.30	42.65	42.65

18.1.5 Pit Designs

18.1.5.1 Design Basis

Optimisation Pit Shell 16 was selected by from the set of optimisations which excluded the Inferred Resources as the basis for design. This shell has a total rock content of 1,194Mt and 232.6Mt ore grading 1.09g/t

An Ultimate pit design was completed and new designs were completed for Stages 1 2 and 3, however at the time of publication the remaining stages were in process of being designed.

The four staged pit designs are shown in Figures 18.1.5.1_1 to 18.1.5.1_4.

18.1.5.2 Haul Road Designs

The design specifications of the Final Pit in-pit haul roads are set out in Table 18.1.5.2_1.

Table 18.1.5.2_1
Final Pit In-Pit Ramp Design Specifications

	Dual Lane	Single Lane
RL (Top)	1100mRL	680mRL
RL (Bottom)	680mRL	608mRL
Ramp Gradient	10%	10%
Ramp Width	32m running width	20m running width

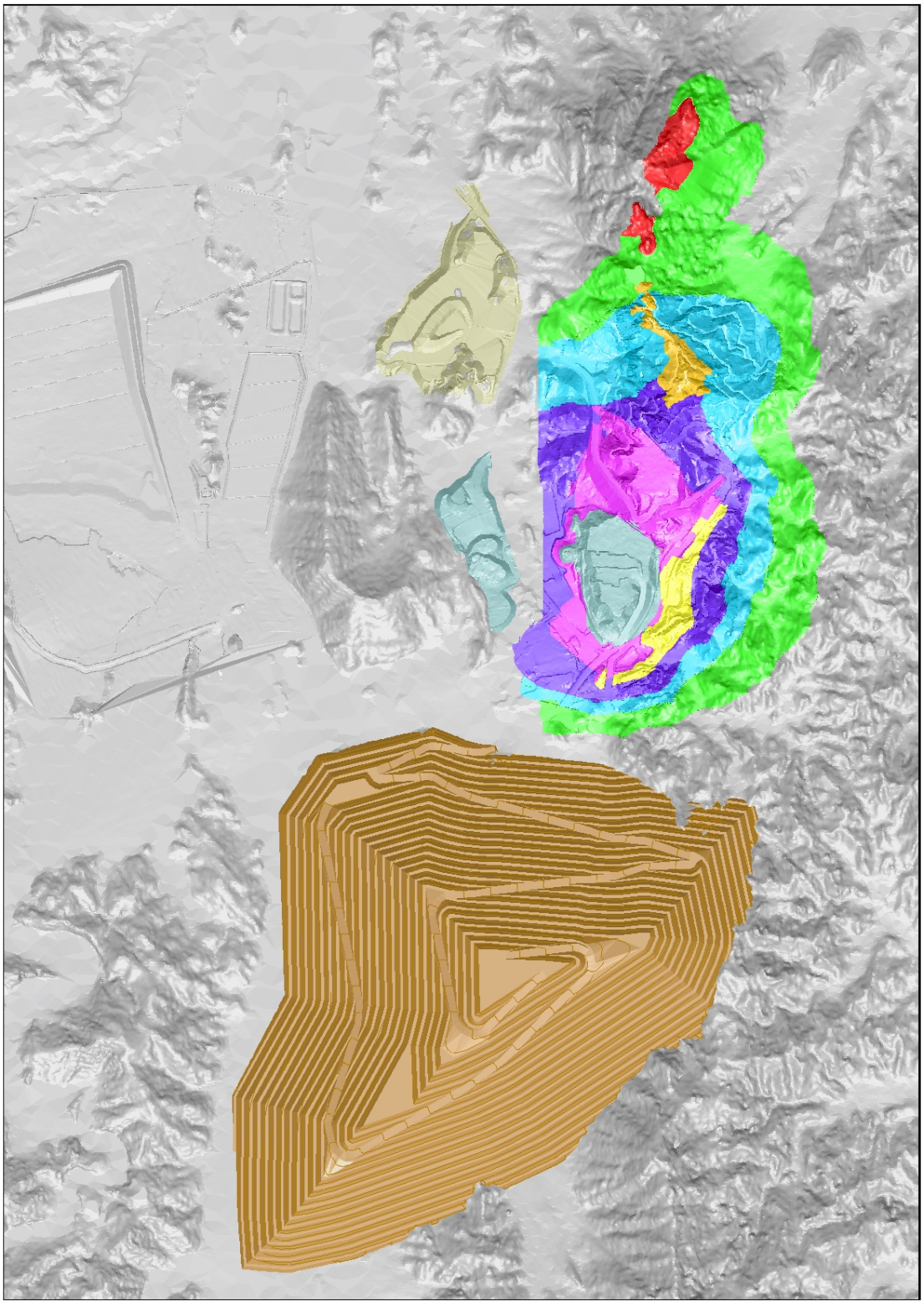
Note – In lower levels of stages width ramp reduces to 1 way haulage

18.1.5.3 Pit Slopes

The final pit design overall slope angles (from pit floor to the pit crest at natural surface) are 37 degrees and 52 degrees for the West and East walls respectively. The face angle slope is 80° for all side walls. For the Hanging wall (H/W) and the Porphyry body, the berm width is calculated to achieve both parameters, face slope and overall slope angle. Pit slope design parameters are set out in Table 18.1.5.3_1.

Table 18.1.5.3_1
Pit Slope Design Parameters

	Foot Wall	Hanging Wall and walls built in Porphyry
Batter Height	12m	12m
Berm Width	10m	7.3m (Berm width calculated based on face angle and overall slope angle)
Wide Berm Width (every 60m)	20m	None
Overall Slope	37 degrees	52 degrees



18.1.6 Waste Dumps

A total of 1000Mt or 450Mbcm of waste is to be mined from the final pit. A swell factor of 25% has been assumed equating to a waste dumping requirement of Mm3. At this time all the waste has been apportioned to the South Waste Dump. This dump was designed to hold this volume however further work is being done to develop an eastern ring road from which Eastern Dumps and any backfill opportunities within the existing pit shell with shorter haul cycles for waste will be developed. For this work to be completed a more detailed survey of the eastern hill areas needs to be completed and detailed finalisation of stage development needs to be completed.

In addition to the large waste dumps, sub grade, low grade, medium and high grade stockpiles are required. These dumps are being designed as close as practicable to the plant site.

Figure 18.1.5.3_4 shows the site layout plan including the waste dumps.

18.1.7 10 Year Mining and Treatment Schedule

A ten (10) year mining and treatment schedule was completed by PGM incorporating the new designs for Stages 1, 2 and 3. The new designs allow for higher grade ore to be brought forward in the mining and treatment schedule compared to previous schedules.

Mining has already begun in Stages 1 and 2.

The key assumptions used in the ten year mining schedule are.

- Initial 18 month schedule the same as the 2011 budget
- 5mtpa Mill throughput for 2011 onwards
- The maximum vertical advance in a one month period is 12m or the maximum fleet digger capacity whichever is the greater.
- Excavator Productivity - 1,400tph to 2013 and then 1,500tph from 2013 onwards
- Shift configuration – 2 x 12hr shifts daily on a 3 panel crew roster working 10 days/ 10 nights and 10 off.
- Excavator availability - 85% LOM
- Excavator Utilisation - 85% LOM
- Excavator Productivity – 1,300tph for 2011, 1,400tph to 2013 and then 1,500tph from 2013 onwards
- Truck Availability - 85% LOM
- Truck Utilisation - 85% LOM
- Projects fleet to undertake pioneering pre-strip work of Stage 7 & 8
- Each RH120E excavator is assigned trucks as per the loading requirement and aligned with the haul profile for the 785C dump trucks.
- The south waste dump was used as the only location for waste dumping.

- A new extended south waste dump with the capacity to contain all the waste in the new pit. This assumption is conservative as there are numerous short haul options available. These will be explored and developed prior to the completion of a full life of mine plan.
- Mining costs used in the schedule are averages developed for each bench from the existing operating costs (also used in optimisation). In practice, these costs vary as the mining schedule changes, due to the allocation of fixed type costs.

The mine scheduling strategy is to feed the highest grade possible to the mill at the maximum mill throughput within the constraint of the total period movement.

This will be achieved by:

1. Finish Mining Stage 1 at a maximum vertical advance rate of 12m per month until this pit is completed in early 2011.
2. Continue Mining Stage 2 until the ore being derived from the stage is suitable for Mill feed Sulphide and above grade on stockpiles or from Stage 1 in 2011.
3. In the latter part of 2011 begin the pioneer development Stage 3.
4. Continue using Stage 2 and Stage 3 and subsequent Stages

Ore can be processed via any one of three plant operating modes:-

- Direct CIL – this process is for oxide ore only and predominantly in the first 5 months of operation or where oxide was greater than 90% of the feed in any one scheduling period.
- Flotation with regrind of concentrate and cyanide leaching of both concentrate and tailings – this process will handle all mixed oxide and sulphide ores.
- Flotation with regrind and cyanide leaching of concentrate – this process is for sulphide ores only (assumed to be greater than 90% sulphides for scheduling.

Ore processing parameters for each process were applied in the schedules.

Plant throughput rates were fixed for all ore types at 5.0Mtpa.

The mining and treatment schedule summary by year is shown in Table 18.1.7_1.

Table 18.1.7_1
Mining and Treatment Schedule Annual Summary

	Unit	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total 10Yr
Total Mining	kt	32,766	30,920	34,524	35,552	33,038	33,128	33,038	33,038	33,038	33,128	**332,171**
Stockpile Rehandle	kt	2,161	235	2,903	3,943	2,920	1,785	1,478	1,920	1,264	1,557	**20,167**
Total Movement	**kt**	**34,927**	**31,155**	**37,428**	**39,495**	**35,958**	**34,913**	**34,516**	**34,958**	**34,302**	**34,686**	**352,337**
Waste Mined												
Waste	kt	26,756	19,081	26,942	33,736	30,287	28,387	26,105	25,737	26,459	27,368	**270,857**
Sub-grade Sulphide	kt	364	325	109	4	0	0	187	301	155	79	**1,523**
Sub-grade Oxide	kt	666	1,028	927	419	639	687	731	1,080	870	951	**7,999**
Total Waste	**kt**	**27,786**	**20,434**	**27,979**	**34,159**	**30,926**	**29,074**	**27,023**	**27,118**	**27,484**	**28,398**	**280,379**
Ore Mined												
Oxide	kt	1,075	900	305	0	0	0	359	657	423	286	**4,006**
Grade	g/t	0.91	0.76	0.79	0.67	0.00	0.00	0.82	0.76	0.84	1.00	**0.83**
Transitional	kt	957	453	293	19	0	0	111	406	206	131	**2,575**
Grade	g/t	1.04	0.85	0.91	0.63	0.00	0.00	0.88	0.72	0.91	0.85	**0.91**
Sulphide	kt	2,948	9,133	5,948	1,373	2,112	4,055	5,545	4,858	4,925	4,313	**45,210**
Grade	g/t	1.49	1.50	1.28	1.03	0.92	1.33	1.56	1.62	1.42	1.36	**1.41**
Total Ore Mined	**kt**	**4,979**	**10,486**	**6,546**	**1,393**	**2,112**	**4,055**	**6,015**	**5,920**	**5,554**	**4,730**	**51,791**
Grade	**g/t**	**1.28**	**1.41**	**1.24**	**1.02**	**0.92**	**1.33**	**1.51**	**1.46**	**1.36**	**1.32**	**1.34**
Sulphide Mill Feed												
To Mill	kt	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	**50,000**
Grade	g/t	1.64	2.04	2.10	0.75	0.67	1.16	1.67	1.63	1.43	1.25	**1.37**
Recovery	%	93.4%	93.4%	93.4%	93.4%	93.4%	93.4%	93.4%	93.4%	93.4%	93.4%	
												2,051,349
Recovered Ounces	oz	238,235	305,816	253,311	113,301	100,958	174,565	250,236	244,654	214,200	188,073	
Cost to process	US$/t	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	
Oxide Dump Leach												

To Dump Leach	kt	3,600	900	305	0	0	0	359	657	423	286	**4,006**
Grade	g/t	0.91	0.76	0.79	0.67	0.00	0.00	0.82	0.76	0.84	1.00	**0.83**
Recovery	%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	
Recovered Ounces	oz	34,687	11,013	3,884	5	0	0	4,760	7,975	5,728	4,598	**53,755**
Cost to process	US$/t	1.36	1.36	1.36	1.36	1.36	1.36	1.36	1.36	1.36	1.36	

Table 18.1.7_1
Mining and Treatment Financial Annual Summary

	Unit	2011	2012	2013	2014	2015	2016	2017	2018	2019
Total Gold Poured	**Oz**	272,923	316,828	257,194	113,306	100,958	174,565	254,996	252,628	219,928
Gold Price/Oz	**$/Oz**	900	900	900	900	900	900	900	900	900
Total Revenue	**USD(m)**	245.6	285.1	231.5	102.0	90.9	157.1	229.5	227.4	197.9
Mining Cost	**USD(m)**	53.5	56.8	56.8	56.8	56.8	56.8	56.8	56.8	56.8
Process Cost	**USD(m)**	41.2	42.0	42.032	42.0	42.0	42.0	42.0	42.0	42.0
Dump Leach Cost	**USD(m)**	4.9	1.2	0.4	0	-	-	0.5	0.9	0.6
Administration Cost	**USD(m)**	8.3	9.6	7.8	3.4	3.1	5.3	7.7	7.7	6.7
Mining Fleet and Equipment	**USD(m)**	13.0	6.3	0	0	9.3	4.8	0	11.6	4.6
Sustainable Capital	**USD(m)**	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0
Capital Expansion Project	**USD(m)**	0	0	0	0	0	0	0	0	0
Cash Flow	**USD(m)**	115.8	160.2	115.5	(9.3)	(29.3)	39.3	113.5	99.4	78.3
Cumulative Cash Flow	**USD (m)**	202.6	362.8	478.2	469.0	439.6	478.9	592.4	691.8	770.1

NPV:	**USD810,110**
Discount Rate:	**8%**

18.2 Proposed Processing Operations

18.2.1 Introduction

The processing plant and key design criteria are described in Section 16.5. The proposed process route encompasses:-

- crushing;
- stockpiling crushed ore;
- grinding;
- flotation of a bulk sulphide concentrate;
- thickening of the concentrate and the flotation tailings;
- fine milling of the concentrate;
- leaching the precious metals from the concentrate and/or flotation tailings in a dilute cyanide solution;
- adsorbing the precious metals onto activated carbon;
- eluting (or desorbing) the precious metals from the carbon;
- recovering the precious metals as gold doré, and
- combining the flotation tailings and leached concentrate and pumping to the TSF.

As discussed in Section 16.5 each ore type requires different processing methods; therefore three separate process designs have been developed. These are Stage 1 - Oxide Ore, Stage 2 - Mixed Ore and Stage 3 - Sulphide Ore. The crushing and grinding circuits are common to all three process routes.

18.2.2 Primary Crushing and Stockpiling

The crushing plant is a single stage crushing facility designed to operate 24 hours a day, seven days a week with a utilization of 75%. The design throughput rate is 1,215tph treating run of mine (ROM) ore with a top size of 900mm and reducing it to a top size of 235mm, suitable for SAG mill feed.

ROM ore will be dumped directly into the 250t capacity primary crusher pocket by haul trucks or stockpiled on the ROM pad. The tipping station is designed to handle Caterpillar 789Cs, or equivalent haul trucks. Ore from the dump pocket will be crushed in a 54 x 74 inch gyratory crusher, fitted with a 422kW motor and operating at a nominal 152mm open side setting. Oversize will be broken using a hydraulic rock breaker.

Crushed rock (with a P_{100} of 235mm and a P_{80} of 105mm) will fall into the discharge pocket and will be withdrawn by a variable speed apron feeder which, in turn, feeds an 1800mm wide short sacrificial conveyor. Chute work under the apron feeder will direct fines spillage from the apron feeder onto the sacrificial conveyor. This conveyor will in turn discharge onto either

1. the stockpile feed conveyor to the mill feed stockpile; or,

2. the secondary crusher feed conveyor to be further reduced to a P_{80} of 50mm crushed through a single CH870C cone crusher operating in open circuit prior to discharging onto the stockpile feed conveyor to the mill feed stockpile.

Dust suppression will be provided by high pressure water sprays in the tipping area as well as at the apron feeder discharge.

Crushed ore will be stored on the 15,000t live capacity (24 hours milling at higher oxide ore throughput rates) mill feed stockpile. If the crushing circuit is off-line for an extended period, the dead portion of the pile can be reclaimed by bulldozer or FEL.

Crushed ore will be drawn from the mill feed stockpile at a controlled rate by a combination of three 1.2m wide x 6.0m long variable speed apron feeders discharging onto the mill feed conveyor. The design capacity of each of the apron feeders is 580tph, which is the design maximum feed rate to the SAG mill. Any one or more feeders may be operated at the same time.

Dust suppression will be provided in the form of high pressure water sprays on all feeder discharge points.

The mill feed conveyor will be fitted with a dual idler Weightometer to measure the plant feed rate for plant metallurgical accounting and control of the SAG mill.

18.2.3 Grinding and Classification

The grinding circuit is a two stage milling circuit with hydro-cyclone classification and a pebble crushing facility. It is designed to operate 24 hours a day, seven days a week with an utilisation of 91.3%. The initial design throughput rate was 500tph treating primary crushed ore with a P_{100} size of 235mm. The introduction of a secondary crushing circuit (to be commissioned end Q2 2011) enables an improvement in the throughput rate to 625tph by treating secondary crushed ore with a P_{80} size of 50mm.The new design throughput rate of 625tph treating secondary crushed ore F_{80} size of 50mm and producing a ground cyclone overflow with a product size P_{80} of 150µm, suitable for flotation feed or CIL leach feed.

The design throughput rate is 500tph treating primary crushed ore with a P_{100} size of 235mm and producing a ground cyclone overflow with a product size P_{80} of 150µm, suitable for flotation feed or CIL leach feed. The milling circuit may treat up to 580tph on straight oxide ore.

The SAG mill is an 8.53m diameter by 4.27m long grate discharge mill with a 5,595kW synchronous motor installed and an air clutch for starting the mill. The mill operates at 76% critical speed with a design ball charge of 15% to draw 4,540kW at the pinion (4,800kW at the motor). The mill is fitted with a steel lining system with 18mm grate apertures and 80mm pebble ports. The design make up ball size is 125mm.

A 3 axis mill liner handler is included in the plant equipment to facilitate removal and installation of SAG mill liner plates and lifter bars. The liners are positioned on the concrete work platform with a mobile crane so that the liner handler can lift them from the transport pallet and position them in the mill.

Water will be added to the SAG mill feed to attain a desired SAG mill discharge density (approximately 75% solids by weight). A flow meter and proportional flow control valve in the mill feed water line allows this to be controlled automatically to a feed rate ratio or to a set point flow rate utilising a PID loop controller in the plant control system.

Ground slurry from the SAG mill will be screened by a 1.8m diameter by 2.1m discharge trommel fitted with 15mm aperture rubber screen panels. Trommel screen undersize will report to the SAG mill discharge hopper (02-PB-01) and will be pumped by one of two (one duty, one standby) variable speed SAG mill discharge pumps to the ball mill discharge hopper. An ultrasonic level detector will be fitted over the SAG mill discharge hopper and will provide continuous level indication on the plant control system. The SAG mill discharge pump speed will be varied by a PID feedback loop controller to maintain a set point hopper level. Trommel screen oversize will report to the pebble crushing circuit for size reduction before being returned to the SAG mill for further comminution.

Allowance has been made in the design and layout to retro fit a second SAG mill of a similar size to the existing one.

Pebbles from the SAG mill will be conveyed to the pebble (cone crusher) which will operate with a closed side setting of 12mm. The design crusher feed rate is 150tph or 30% of new feed to the mill. Crushed pebbles will be returned to the SAG mill via the SAG mill feed conveyor.

Trommel screen undersize from the SAG mill will report to the ball mill discharge hopper where it will combine with the ball mill discharge streams and will be pumped by one of two (one operating, one standby) variable speed cyclone feed pumps to a cluster of 24 Krebs DBB15 (380mm diameter) cyclones. 20 cyclones will be operating with four spare. Cyclone feed pump speed will be varied by a PID feedback loop controller to maintain a set point hopper level.

Cyclone overflow is flotation or leach feed and gravitates to the trash screen feed box. Cyclone underflow is split into two boil boxes to feed each ball mill. There will be a facility to direct some cyclone underflow to the SAG mill.

Cyclone underflow will report to two 5.03m diameter by 9.29m long overflow ball mills with rubber linings and fitted with 4,100kW synchronous motors and air clutches for starting. The ball mills are designed to operate in parallel however may be operated singly whilst the other is being maintained. The ball mills are designed to be operated with a 24% ball charge to draw 3,007kW at the motor. Make up ball size is 65mm. Each mill will be fitted with a 1.8m diameter by 1.8m long trommel fitted with 10mm by 20mm aperture rubber screen panels. Trommel oversize will report to scats bins for later collection.

Water will be added to the mill feed chutes and to the common mill discharge hopper to attain the desired densities for optimum milling performance.

Allowance has been made in the design to retro fit a gravity recovery circuit.

18.2.4 Leaching and Carbon in Leach (CIL)

The CIL circuits are designed to leach gold (and silver) from the ground ore and recover it onto carbon for subsequent recovery by elution electrowinning and smelting. There are two

leaching circuits included in the design. This enables whole of ore leaching for the oxide ore and either flotation concentrate leaching and flotation tailings leaching for the mixed ore or a flotation concentrate only leach for the sulphide ore. The circuit are designed to operate 24 hours a day, seven days a week with an utilisation of 91.3%. The combined design throughput rate is 500tph treating primary crushed ore. Viscosity characteristics of the pulps allow use of conventional open tank, moderate power dual impeller agitators to achieve evenly suspended solids and sufficient shear action to achieve satisfactory carbon adsorption kinetics.

Cyclone overflow will gravitate to the trash screen feed boil box and thence to the three vibrating trash screens. Screen undersize will report to the feed pump hopper from where it will be pumped either to the leach feed distribution box or to the flotation feed conditioning tank.

18.2.4.1 Whole of Ore and Flotation Tails CIL

Tailings from the flotation circuit will be pumped to the flotation tailing thickener collection box and from there gravitate into the 23m diameter high rate flotation tailings thickener. Flocculant and dilution water will be added to assist in settling of the solid.

The flotation tailings thickener underflow lines will discharge into the flotation tailings leach feed pump box.

Thickener underflow will be pumped to the leach feed distribution box by one of two (one operating, one standby) variable speed flotation tailings leach feed pumps. Thickener overflow will gravitate to the process water tank.

Cyclone overflow (at approximately 42% solids) or thickened flotation tailings (at approximately 46% solids) report to the leach feed distribution box and then into the CIL circuit.

The CIL circuit will consist of eight mechanically agitated leach/adsorption tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. The first tank is a 3149m^3 leach tank followed by seven 3050m^3 (active volume) CIL tanks, which will provide 25 to 34 hours total residence time depending on the feed source. Eight tanks in series have been selected to maximize leach recovery. A single leach stage has been utilized to attain higher gold in solution tenor in the first stage of adsorption thereby allowing higher gold on carbon loadings. The circuit has been sized for 24 hours leach residence time on whole of ore processing. The longer leach time on flotation tail is due to the higher feed density and slightly lower dry solids feed rate.

Cyanide will be dosed into the leach feed box, CIL tank one, CIL tank two and CIL tank four. The launder system will allow individual tanks to be taken off line for maintenance without unduly compromising recovery. Average carbon concentrations of 8g/L are required in all tanks to provide high gold adsorption efficiencies and a low solution tail.

Carbon will be advanced counter current to pulp flow using airlifts with barren carbon from the elution circuit being added to CIL tank six or seven. Loaded carbon will be pumped to a loaded carbon screen. The design carbon advance rate is 7.8 tonnes of carbon per day. The airlift and carbon recovery systems are designed to achieve the carbon advance or recovery in four hours.

Each CIL tank will be fitted with two mechanically swept, wedge wire inter-tank screens to retain carbon. Each screen has an area of 12m^2 and a screen aperture of 0.1mm. Overhead hoists will be used to remove the screens for maintenance and routine cleaning.

Flotation operates at natural pH and therefore provision is made in the layout and piping runs for pH control by adding slaked lime slurry to the flotation tailing leach feed pump box.

Leach tails will gravitate to the carbon safety screen feed boil box and then to three vibrating carbon safety screens.

The CIL area, including the areas of inter-tank screen maintenance and underneath the carbon safety screen, has a concrete floor and bunds to retain overflow or spillage. The volume of the bunded area included is sufficient to hold at least one tank volume in case of tank rupture.

18.2.4.2 Flotation Concentrate CIL

Reground flotation concentrate will report to the concentrate leach feed box and then to the concentrate leach tank. The design throughput rate is 25tph of flotation concentrate.

The concentrate CIL circuit consists of nine mechanically agitated leach/adsorption tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. The first tank is a 265m^3 leach tank followed by eight 247m^3 (active) CIL tanks, which provide 33 hours total residence time.

Cyanide will be dosed into the leach feed box, CIL tank one, CIL tank three and CIL tank four. The launder system allows individual tanks to be taken off line for maintenance without unduly compromising recovery. Average carbon concentrations of 12g/L are required in all tanks to provide high gold adsorption efficiencies and a low solution tail.

Carbon will be advanced counter current to pulp flow using airlifts with barren carbon from the elution circuit being added to CIL tank seven or eight. Loaded carbon will be pumped to a vibrating loaded carbon screen. The design carbon advance rate is 1.8 tonnes per day over a 12 hour period.

Each CIL tank will be fitted with a mechanically swept, wedge wire inter-tank screens to retain carbon. Each screen will have an area of 2m^2 and a screen aperture of 0.8mm.

Flotation operates at natural pH and therefore provision is made in the layout and piping runs for pH control by adding slaked lime slurry to the regrind mill discharge pump box.

Leach tails will gravitate to the carbon safety screen. Screen oversize will be collected in drums and returned to the circuit. Screen undersize will report to the variable speed concentrate tailings pump and will then be pumped to the tailings dam.

The CIL area will have a concrete floor and bunds to retain overflow or spillage. The volume of the bunded area included is sufficient to hold at least one tank volume in case of tank rupture.

18.2.5 Flotation

The flotation circuit is designed to recover a bulk sulphide concentrate for further comminution and subsequent leaching. The circuit is designed to operate 24 hours a day, seven days a week with a utilization of 91.3% and to treat 500 dry tph.

Cyclone overflow will report to the mechanically agitated flotation feed conditioning tank. This tank will have a working volume of 90m^3, which will be sufficient for 5 minutes residence time (as used in the flotation test work) at design flow rates.

Slurry from the conditioning tank will overflow into the first rougher cell feed box. The flotation cell circuit will comprise six by 100m^3 mechanically agitated tanks cells. The total residence time in the flotation circuit will be 31.8 minutes. Six cells in series have been chosen to minimize short circuiting.

The flotation concentrate from all the cells will be combined in a launder and gravitates to the rougher concentrate pump and pumped to the concentrate thickener.

Flotation tailings will be pumped to the leach feed thickener.

If only the sulphide concentrate leach circuit is operating then flotation tail will be final tail.

18.2.6 Concentrate Thickening and Regrind

The concentrate thickening and regrind circuit is designed to thicken the diluted flotation concentrate from approximately 30% solid to 60% solids, suitable as feed to the regrind circuit. The circuit is designed to produce a ground product size P_{80} of 12 microns, sufficient to liberate the fine gold particles for subsequent leaching. The circuit is designed to operate 24 hours a day seven days a week with an utilisation of 91.3% and to treat 25tph of flotation concentrate.

Trash screen underflow will report to 2 235m^3 (working volume) mechanically agitated concentrate surge tanks. The surge tanks will have sixteen hours surge capacity to allow for process interruptions and maintenance in the regrind circuit.

Concentrate slurry is pumped from the surge tanks into the regrind cyclone feed pump feed.

The regrind circuit consists of a Metso VTM1250 Vertimill in closed circuit with 100mm cyclones. Cyclone overflow is deslimed in another cyclone cluster, with the deslime cyclone underflow being ground further in two Metso stirred media detritor mills (SMD335) in series.

The final product – at a d80 size of 12 microns, will be a combination of SMD product and deslime cyclone overflow which will then be pumped to the concentrate CIL circuit.

18.2.7 Tailings Thickening and Disposal

The tailings thickening and disposal circuit is designed to thicken the tailings stream to recover water for reuse and minimise the volume of the supernatant lake in the tailings storage facility.

The tailings thickening and disposal circuit is designed to operate 24 hours a day, seven days a week with a 91.3% availability. The design throughput rate is 500tph.

Tailings from the flotation tailings CIL plant (or whole of ore leach) will be screened and will report to the 23m diameter high rate thickener.

Flocculant and dilution water will be added to assist in settling of the solid.

Thickened tailings residue will be pumped to the tailings storage facility (TSF).

Supernatant liquor from the TSF will be recovered by a decant system and returned to the plant (via the cyanide process water tank) for re-use in the process in areas excluding the milling and flotation circuits. This water will contain a low concentration of cyanide and cannot be used in the milling and flotation circuit, as cyanide may depress sulphide minerals in flotation.

18.2.8 Gold Recovery

There are two separate elution and electrowinning circuits due to the large difference in elution plant capacity required for the different stages of the operation.

Both circuits are pressure Zadra type circuits. Pressure Zadra circuits were selected because of their lower requirement for high quality water. A two tonne carbon capacity circuit was selected for the flotation concentrate CIL carbon elution duty. An eight tonne carbon capacity circuit was selected for the flotation tail/whole of ore CIL carbon elution duty. The operation of both circuits is essentially the same with the exceptions that the smaller circuit has a separate acid wash column and an integrated pressure electro winning cell.

The following operations will be carried out in the elution and gold room areas:-

- Acid washing of carbon;
- Stripping of gold and silver from loaded carbon using a Zadra elution circuit;
- Electro-winning of gold and silver from pregnant solution, and
- Smelting of electro-won products.

The elution and gold room area will operate up to 7 days a week, with the loaded carbon recovery on night shift and the majority of the elution occurring during day shift.

18.2.9 Carbon Regeneration

Upon completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen and into the horizontal carbon regeneration kiln. The carbon will be heated to approximately 700°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched in water and pressure transferred to the carbon sizing screen. In the case of the concentrate CIL circuit the barren regenerated carbon will be pressure transferred to the concentrate barren carbon screen.

The screen oversize (regenerated, sized carbon) will return to last CIL tank, and the quench/transfer water and fine carbon will report to the tails hopper via the carbon safety screen for disposal in the TSF.

18.2.10 Reagents

The main processing reagents are:-

- Collector (Potassium Amyl Xanthate)
- Frother (MIBC)
- Promoter (Dithiophosphate A404)
- Quicklime (CaO)
- Sodium Cyanide (NaCN)
- Sodium Hydroxide (Caustic Soda, NaOH)
- Hydrochloric Acid (HCl)
- Grinding Media
- Flocculant
- Activated Carbon

18.3 Tailings Storage Facility

18.3.1 Introduction

Knight Piésold ('KP') was engaged by PGM to design the tailings storage facility (TSF) for the Sukari mine site.

Site surveys and geotechnical investigations were carried out at alternate locations and a site was chosen immediately adjacent to the proposed plant site.

To avoid tailings and strongly saline water penetrating the wadi groundwater, the TSF basin will be lined with 1.5mm thick HDPE material.

Tailings will be deposited by sub-aerial techniques and decant water will be pumped from a floating barge back to the plant process water tank.

The embankment is designed to be raised in annual increments by upstream construction methods. Although the design carried out was for a total 48Mt tailings capacity it is predicted, subject to satisfactory engineering evaluation at the time, that the ultimate capacity of 78Mt will be achieved by continuing with staged upstream construction.

The FS design for the TSF was based on the following design parameters:-

- Total storage required is 48 million tonnes (consisting of 45.6Mt of flotation tailings and 2.4Mt of leached flotation concentrate tailings)
- Throughput is 4.0 million tonnes per annum (3.8Mtpa of flotation tailings and 0.2Mtpa of leached flotation concentrate tailings).
- Tailings slurry percent solids are 45%.
- Flotation tailings and leached flotation concentrate tailings will be stored in a single facility.

18.3.2 Geotechnical Investigation

A geotechnical investigation of the site was undertaken. In general the site is characterised by two distinct surface condition types, with a well defined boundary separating the two.

Rock outcrops are found where the topography is steep. Their degree of weathering varies with the topography, making them difficult to utilise for infrastructure development. They are usually quite fractured and vary locally between low and high strength.

The other typical geotechnical features at the site are the large, broad alluvial in-filled valleys that span between the hills. These areas are sometimes described as "wadi" sands and gravels. At the TSF site there is typically ~1.7m of transported non-cohesive soil over the fractured bedrock.

18.3.3 Tailings Characteristics

Three samples of tailings were generated from the metallurgical bench scale testwork. The samples are described as follows:-

1. Flotation tailings – waste product of flotation cycle (not leached).
2. Leached flotation tailings – waste product of flotation cycle, after CIL processing.
3. Leached flotation concentrate (LFC) tailings – primary product of flotation cycle, after CIL processing.

Both physical (settling, consolidation and drying) and geochemical testing was undertaken. The physical test-work indicated:-

- The tailings settles relatively quickly (within 1 – 2 days), releasing about 60 – 70% of total water as supernatant and achieving reasonably high densities.
- The tailings dries well, achieving densities of about 1.6 to 1.7t/m^3 after evaporation losses of around 50mm.
- The permeability of the tailings was around 10^{-6} to 10^{-7} m/s.

The geochemical testwork for the tailings indicated that:-

- The flotation tailings is classified as non-acid forming.
- The leached flotation concentrate is classified as potentially acid forming.
- Both samples were enriched in known toxicants such as arsenic, molybdenum, antimony, chromium and selenium, and pose a significant environmental potential impact. Therefore lining of the tailings basin will be required to reduce seepage.

18.3.4 Site Water Management

The water balance models for the TSF indicate that the water balance for the site is strongly negative under all conditions, due to the low rainfall and high evaporation rates that are characteristic of Egypt's arid eastern coast region. A large volume of "make-up" water (typically 60 – 70% of total process water requirements) will need to be extracted via the Red Sea intake.

The TSF has been designed to contain the 1 in 100 year extreme rainfall event at every stage. Any rainfall exceeding this event will be released in a controlled manner using an emergency spillway structure.

18.3.5 Tailings Storage Facility (TSF)

The TSF consists of a "paddock" type impoundment, to be operated as a single cell. The flotation tailings (waste product from the initial flotation cycle), accounts for 95% of the total tailings solids, the other 5% of the processed solids will be collected in the flotation circuit and will then undergo CIL processing. The combined tailings product will be deposited in the tailings storage facility.

The geochemical properties of the tailings together with the relatively high permeability of the in-situ soil materials (wadi sands and gravels) mean that there is potential for contaminated seepage to occur. Given this potential and the proximity of the project area to the environmentally sensitive Red Sea, KP has incorporated an HDPE liner throughout the basin area. The design also includes an extensive underdrainage system, which will serve the dual functions of reducing the potential for seepage to occur through the bottom of the TSF basin and increasing the density that is achieved by the tailings.

Deposition of tailings into the facility will use the sub-aerial technique in order to increase the density that the tailings can achieve in storage.

It is envisaged that the TSF can be practically constructed using embankment fill from a variety of sources. During initial stages of TSF development the embankments will be built using waste rock delivered to the embankment by mining operations. Results of the geotechnical investigation indicate that it may not be possible to find in-situ materials that are suitable for use as a low permeability embankment fill. Therefore it is expected that low permeability fill required during early TSF development will be sourced from off site and blended with soil containing some fine, plastic soil material at the site. During later stages of TSF development, it is envisaged that low permeability fill will comprise tailings won from the beaches.

The embankments of the TSF are designed to utilize a combination of upstream and downstream methods, depending on the strength achieved in the tailings beaches and the location of the supernatant pond. The final embankments of the tailings facility will be up to 49m in height.

18.4 Infrastructure and Services

18.4.1 Plant Site Geotechnical Investigations

18.4.1.1 Introduction

A report has been prepared by KP (Tailings Storage Facility Design and Plant Site Foundation Assessment, Feasibility Study PE 401 – 00015/5, July 2006) that provides detail of the proposed plant site foundations at Sukari.

The process plant site is located on a flat area with a gentle downward slope to the north west of 1V:100H. The hill on which the proposed Primary Crusher will be located (at the south east end of the plant site) has a slope of approximately 1V:5H.

The ground surface at the plant site consists of alluvial wadi gravel. The wadi material generally consists of sub-angular sandy gravel that commonly contains cobbles and boulders. The fines content of the gravel is generally less than 10%, and the gravel, cobbles and boulders are composed of metavolcanics/tuff, porphyry/granitoid, quartz and metasediments lithologies. The main difference in near-surface ground conditions across the plant site is the depth of the unconsolidated wadi gravel, sand and cobbles/boulders observed in the test pits and drill holes.

The bedrock in the vicinity of the plant site is from the calc-alkaline volcanic and subvolcanic group. The drilling program at the plant site identified gabbrodiorite bedrock at the Primary Crusher location and lapilli tuff and agglomerate bedrock to the north west of the Primary Crusher. The gabbro-diorite bedrock is medium to high strength and the lapilli tuff and agglomerate is low to medium strength. Strength of both rocks generally increase with depth. Both rock types are generally distinctly weathered.

Groundwater levels were monitored in three of the five drill holes for approximately one week after the holes were drilled. Water levels continued to fall during that time with the lowest depths of the water table being between 15 m and 21 m below grade, suggesting that the water levels recorded do not represent static water levels.

18.4.1.2 Foundation Conditions

The foundation conditions at the five proposed plant locations are all suitable for the proposed use. The gravel consistency is similar at all locations. The stratigraphy observed at the drillhole locations is as follows:-

- At the proposed Tailings Thickener (DPS472) the gravel is 3m thick and underlain by medium strength rock.
- At the Flotation Plant (DPS473) the gravel is 5m thick and underlain by a low to medium strength rock.
- At the SAG and Ball mills (DPS474) the gravel is 6m thick and underlain by low strength rock.
- At the Stockpile location (DPS475) the gravel is 4m thick and underlain by 4m of dense sand and 1.3m of cobbles and boulders over medium to high strength rock.
- At the Primary Crusher location (DPS476) a medium to high strength rock was observed from surface to a depth of 20m.

18.4.1.3 Geotechnical Issues

There are no significant geotechnical issues at the plant or crusher locations. It is anticipated that for the design of shallow footings settlement criteria rather than bearing capacity will dominate.

The plant site is located in a major wadi channel. Surface runoff during the infrequent precipitation events may have the potential to mobilise and erode the sand and gravels from around footings. Therefore, footings in the plant site should be protected with surface water diversion channels and bunds.

18.4.1.4 Safe Bearing Capacity

Given the flat terrain at the plant site, significant cut/fill earthworks are not anticipated. Shallow footings may therefore be founded either within the dense alluvial gravel or in weathered bedrock. As a guide, estimates of safe bearing capacity (that is, the maximum bearing pressure to which the soil can be subjected without risk of shear failure, incorporating a factor of safety) of the foundation materials based on visual assessment in test pits, have been calculated for each potential foundation horizon, as follows:

- Safe bearing capacity of footings founded on dense or very dense alluvial gravel at 0.5m below the final grade level is 320kPa, and 485kPa at 1.0m below the final grade level.
- The safe bearing capacity of footings founded in low strength bedrock at 0.5m below the final grade level is greater than 1MPa.

These bearing capacities apply to square pad footings (1m x 1m in size) and are dependent on the site being dry and well drained such that the foundation material does not become saturated. The actual safe bearing capacity of a particular footing will be dependent on its location, geometry and founding depth, as well as the founding horizon. Hence, further assessment of specific foundation locations will be required to confirm higher recommended bearing capacities.

18.4.2 Buildings

A mine camp facility and various mine buildings were required for the Sukari Gold Plant Project. The mine buildings are located within and adjacent to the processing plant compound. The camp facility is separate and is located approximately 1 kilometre NNE of the processing plant.

Construction of the various site buildings is essentially complete. The administration office complex and all other buildings located at the mine site are modular steel clad buildings custom built to suit the particular requirements. The camp facility consists of a mix of modular steel clad buildings and traditional brick domed-roof buildings.

A maintenance/workshop building will be provided with facilities for offices, ablutions, storage, tools and equipment, light vehicle maintenance and welding. At the one end of the building, an adjoining two storey office houses the Maintenance Manager and key supervisors.

18.4.3 Communications

A communications network is established utilising satellite technology and fibre-optic communication for voice, fax, internet and PC network traffic. The communications and IT infrastructure comprises satellite link, PABX, ethernet LAN, IT servers, desktop computers, A4 & A3 printers, photo-copiers (b/w & colour), plotter, fax machine, UPS, copper and fibre cabling, and site two-way radio system.

VSAT satellite equipment comprises satellite antenna, transceiver, modem and bandwidth manager.

Fibre-optic connection is to the national communications fibre-optic network that runs along the Red Sea coast.

Ethernet LAN points are provided in all administration offices, mine office, stores and workshop.

A “trunked” repeater system provides the infrastructure to enable hand-held and mobile radio sets to communicate around the site.

All the satellite equipment and communications racks are protected by lightning protection and earthing systems.

IT system and desktop computers software programming is to be performed by the Operations IT personnel.

18.4.4 Security

To maintain controlled access to the plant areas, all persons entering the site are required to pass the continuously manned security gatehouse/boom gate.

A high security area fence is proposed for the process plant site, offices/buildings, fuel depot, power station and water ponds which incorporates a double fence. (two fences, 30m apart)

Added security in the form of secure buildings is provided for the gold room housing the electrowinning cells.

The security area fences are 2.4m high wire mesh type fencing.

A stock fence is proposed for around the Tailings Storage Facility to keep out animals and wildlife.

18.4.5 Power Supply

Power supply for both stage one and stage two operations phases will be supplied by medium speed 750/1000 RPM diesel generating sets capable of operating on heavy fuel oil (HFO) at a viscosity up to 700 centistokes, and generating at 11,000volts at 50HZ. A 35MW power station located at the plant site will consist of 7 x 4MW and 7 x 1MW engine/alternator sets with associated air coolers, fuel cleaning equipment, switchgear etc.

The power station will be designed to supply nominal output at site based climatic conditions with minimal derating.

Stage one (oxide and mixed ores) will cater for a load capacity of circa 16,000/17,000kW, stage two (mixed ores) will cater for a load of circa 22,000kW, while stage three (sulphide ore) will be only slightly lower than stage two.

The operating mode will be N+1 sets, (where N equals the number of sets required to meet the load with one additional machine on standby/ repair) operating at an 85% load factor. A smaller capacity set running on diesel fuel and generating at 400 volts will serve as a black start facility, as well as emergency power.

The power station main building will consist of:-

- an engine hall containing the generating sets

- auxiliaries annexe containing fuel treatment and engine auxiliary equipment
- switchgear annexe containing generator and main distribution switchgear
- transformer compound containing station earthing, auxiliary and 11kv/33kv step up transformers.

Fuel (HFO) will be stored in a tank farm adjacent to the power station and incorporate tank heating for the HFO storage tank(s). The power station auxiliary facilities will include air start compressors, HFO fuel cleaning and conditioning with facility for sludge removal, lubrication equipment and prime mover cooling equipment including water treatment.

The power station will operate in a semi automatic mode i.e. normally unattended except for major load change and machine maintenance selection.

Facilities will be provided for operational maintenance on the prime movers and basic spare parts will be carried (e.g. spare set of cylinder heads, turbocharger etc.).

18.4.6 Water Supply

Sukari is located in an arid desert terrain with little or no vegetation, where only a few Bedouin people live a marginal existence. There is no surface water and no regular rainfall. Rainfall is limited on average to one torrential downpour once every six or seven years.

Hydrological assessments commissioned by Centamin have determined that it will not be practical to develop ground water sources to consistently supply process plant requirements. A pipeline was required to transfer the water from the coast to the Sukari mine site a distance of approximately 25 km. Water is drawn directly from the Red Sea and if necessary from bore holes located very close to the coast line where seawater freely infiltrates into the groundwater, using submersible pumps. The seawater intake pumps will be mounted off well heads suitable for supporting the pump assemblies. The seawater will be pumped to the Sukari mine site to satisfy all process plant and mining requirements. Most of the seawater will be pumped into a raw water pond located near the processing plant, whilst around 500m^3/day will be pumped to a water treatment plant for potable and fresh water supplies and approximately 97m^3/hr will be used to top up supply to the process water tank.

Pricing information, in relation to the pipeline and pumping stations has been solicited and provided by PGM.

18.5 Personnel

PGM currently have approximately 1000 employees for the operation of the Sukari Gold Mine. Table 18.5_1 below shows the manpower build-up during the construction phase through to peak manpower which occurs in year 3.

Table 18.5_1
Total Operations Manpower and Build-up

Department	Year 1	Year 2	Year 3
Finance & Administration	**192**	**281**	**281**
Expatriate	6	6	6
Local	186	275	275

Process	**112**	**165**	**155**
Expatriate	-	13	13
Local	-	138	138
Mining	**280**	**472**	**472**
Expatriate	8	12	12
Local	272	460	460
Maintenance	**214**	**343**	**339**
Expatriate	20	34	30
Local	194	309	309
Total Manpower	**798**	**1261**	**1247**

Expatriate levels are expected to significantly reduce as training and familiarisation with the operation increase with Egyptian employees.

18.6 Markets

Gold dore bars are transported from the project site to an accredited gold refiner for smelting and refining into an LME grade gold bar on a regular basis, and the refined product credited to the company's gold account. Sale of the gold will be via the standard industry practice of delivery from this gold account into either a pre-arranged hedging contract or a spot market sale contract.

18.7 Contracts

PGM has numerous contracts with local and international companies pertaining to the operation of the mining and processing operations.

18.8 Environmental Considerations

18.8.1 Introduction

An Egyptian company, Environics, has completed an environmental and social impact assessment (ESIA) for the Sukari Project in accordance with the Equator Principles of the International Finance Corporation (IFC).

The project is categorized as a category C project (high potential impact); hence a full Environmental Impact Assessment was required to be submitted to the Egyptian Government for approval. The ESIA was completed in February 2007 and submitted to the Egyptian Environmental Affairs Agency (EEAA) through the Egyptian Mineral Resources Agency (EMRA) in March 2007. The document was submitted in both English and Arabic to comply with Egyptian Government requirements. The ESIA was approved by the EEAA in April 2007.

Public Consultation meetings were held in Marsa Alam during the course of preparation of the ESIA. All concerns raised were satisfactorily answered or mitigated and no significant objections are outstanding.

Various addenda have subsequently been submitted to the EEAA. An addendum to the ESIA for expansion of the power plant was submitted to the EEAA in March 2008 and approval was subsequently granted in May 2008. An addendum concerning seawater intake was submitted to the EEAA in January 2009 and approved in March 2009. An addendum for concerning and oil sludge incinerator was submitted in February 2009 and currently under review.

18.8.2 Legislative Background

The ESIA outlines the relevant legislation that will have to be complied with. Although the relevant articles of law are disjointed and do not fall under a single comprehensive act, the essence of such appears to be similar to that which would be applied to comparable operation in, for example, Australia.

18.8.3 Baseline Survey

Baseline information is presented in the Environics report. Baseline information captured has included:-

- Climate summary.
- Geotechnical stability (global assessment), geomorphology and regional geology.
- Hydrology (desktop level, field work limited to assessment of existing wells).
- Surface water assessment (including flooding).
- Flora, fauna and heritage assessments (limited fieldwork).
- Social and Landowner issues, including a desktop social impact study.

18.9 Taxes, Duties and Royalties

This has been covered under Section 4.4.

18.10 Operating Costs

The operating cost estimate is in United States dollars with a base date of 1 July 2010. There is no allowance for escalation or contingency in operating costs.

The exchange rates used are as shown in Table 18.10_1.

Table 18.10_1
Currency Exchange Rates

Country	Unit of Currency	Exchange Rate (to US$)
USA	Dollar (US$)	1
Australia	Dollar (AUD)	1.015
Europe	Euro (EUR)	1.307
Egypt	Pound (EGP)	0.1739

Valued added tax on imported goods and all government duties and other taxes have also been excluded based on the concession agreement with the Egyptian Government.

Costs by area have been estimated by:-

- PGM Mining, grade control, local suppliers, labour and security, general and administration
- RPE Processing, power

18.10.1 Mining Costs

Mining costs were derived from

- Actual production and cost performance data incurred to date
- Costs detailed in completed and active contracts (e.g. drilling contract, explosive and accessories supply and tyre management contracts)
- Assumptions on productivities are based on actual performance to date.
- Consumable and fluid usage collated during usages to date and predicted from MARC tendering processes in June 2007.
- Consumption and productivity rates are based upon actual performance to date.
- Benchmarking studies undertaken during the feasibility stage.

A summary of the base case mining costs is detailed in Table 18.1.4.2_1.

The variable haulage costs were estimated according to the following steps:

1. Calculating the hourly operating cost of a 785C dump truck. The 785C dump truck is the main haulage unit at the Sukari Gold Project.
2. Completing a preliminary optimisation to determine the potential size of an open pit.
3. Developing a waste dump that can contain the volume of the potential pit
4. Generating simplified haul profiles representative of the pit extremities, pit centroid and hauling to the dump developed at step 3.
5. Calculating haul costs using Talpac and extrapolating between levels to give the average change in haulage cost with depth.

The diesel fuel used for mining (and for other uses such as light vehicles) was US$0.21/l.

PGM has already purchased a significant portion of its mining fleet so mining cost was based on actual costs being achieved.

The total mining activity costs for the period 2011 to 2020 (10yrs inclusive) is approximately US$450M. The mine activity total costs include all operating costs for excavating, loading and dumping of ore and waste as well as drill and blast operating costs, grade control costs and low grade rehandling costs.

The mining activity costs do not include the labour component, which is derived from the manning levels and salary/wages data and totalled separately.

18.10.2 Processing Costs

There are five different ore type classifications, M1 through M5, described in the mine schedule. These have been grouped into three basic ore types for the purpose of the processing cost estimate; i.e. oxide ore, sulphide ore and mixed ore.

Summaries of the total annual processing costs for the three ore types are as follows:-

Table 18.10.2_1 Sulphide Ore at 5.0 Mt/y		
Cost Centre	**Total Plant Cost (Gold Price US$900/oz) @5mtpa Mill throughput**	
	US$/Year	**US$/t**
Process Cost		
Sulphide/Transitional	42,500,000	8.50
Oxide	1,100,000	1.36
Total	**43,500,000**	

The estimates shown in the Tables above are for processing cost only and were developed from actual labour costs provided by PGM. Maintenance consumable costs were derived from actuals to date.

Power necessary for the plant, sea water pipeline, tailings dam water return and the camp is provided by an on-site PGM owned and operated, heavy fuel oil (HFO) power station. Power requirements have been calculated for all areas and a continuous generating capacity of 28MW has been allowed. The diesel fuel consumption used was 210g/kWh and fuel price of US$0.15 cents per litre

18.10.3 Labour

PGM provided the base monthly remuneration rate in EGP for all local personnel according to skill level and responsibility. PGM also provided the applicable loadings to account for an annual salary increase, food allowance, public holiday allowance, annual leave, special allowances, site allowances and travel allowances. Indirect costs include elements such as social insurance and payroll taxes.

PGM provided expatriate remuneration costs, inclusive of superannuation and all on costs.

The total unit labour costs included in Table 18.10.3_1 includes all allowances, including international flights for expatriates.

Table 18.10.3_1
Total Unit Labour Costs

Title	Grade	Total Annual Cost (US$)
Expatriate		
General Manager	EP1	367,400
Department Manager	EP2	278,300
Supervisor	EP3	209,488
Senior Operator	EP4	185,728
Operator	EP5	161,968
Egyptian Senior Staff		
Manager	ES1	79,118
Superintendent	ES2	74,042
Senior Professional/Supervisor	ES3	44,501
Professional	ES4	18,024
Technical Officer	ES5	12,912
Technician/Administration	ES6	11,352
Graduate	ES7	6,816
Egyptian Junior Staff		
Leading Hand	JS1	6,060
Tradesman –Experienced	JS2	3,492
Tradesman	JS3	2,700
Operator - Experienced	JS4	2,700
Operator	JS5	2,000
Trades Assistant	JS6	1,500
Trainee	JS7	1,000

18.11 Economic Analysis

18.11.1 Project Economics

PGM has prepared a detailed five-year budget model for the purposes of evaluating the economics of the Sukari Gold Project.

The financial inputs have been calculated from the capital, mining, plant and administrative cost data generated through the feasibility study process, Owner's Cost review and actual outcomes. The economic analysis is based on the following assumptions:-

- Gold price of US$900.00 per troy ounce. No hedging of the gold price has been assumed within the analysis.
- Egyptian government royalty of 3.0%.
- No escalation has been applied to operating costs or to revenue.

The five-year budget model projects the revenue, operating costs, royalties and capital outlays over the first five years of the Sukari Gold Project. The model is based on the mining and treatment of the proven and probable reserves. Material movements are taken from the 10 year schedule. The project cash flows have been modelled on a yearly basis.

The key physical and financial outcomes for the Sukari Gold Project 10 year schedule are included in Table 18.12.1_1.

Table 18.12.1_1

10 Year Schedule Cashflow Summary

	Unit	10YS
Ore Mined – quantity	Mt	71.96
Ore Mined – grade	g/t	1.16
Ore Mined – contained Au	Moz	2.684
Total Mining	Mt	332.17
Ore Milled - Qty	Mt	50
Ore Milled – Qty	g/t	1.41
Ore Milled – contained Au	Moz	2.26
Ore Milled – recovery (sulphide)	%	93.4
Ore Milled – recovered Au	**Moz**	**2.08**
Net price received	US$/oz	900
Revenue	US$M	1,941
Total operating expenditure	US$M	1,116
Net Cashflow	US$M	735
10 Year NPV (8%)	**US$M**	**810**

It is important to note that this Financial Analysis is on a project basis and does not reflect a PGM position. The project is a joint venture between EMRA and PGM, hence Cashflow to PGM requires adjustment to match the concession agreement.

19 REFERENCES

Akaad, M.K., Abu El-Ela, A.M., & El Kamshoshy, H.I., 1994. Geology of the Region West of Marsa Alam, Eastern Desert, Egypt. *In* Annals of the Geological Survey of Egypt, vol. XIX (1993), pp 1-15.

AMC Consultants, 2006. Sukari Mining Study. Unpublished report to Centamin Egypt Limited, September 2006.

AMC Consultants, 2007. Sukari Mining Study Update. Unpublished report to Centamin Egypt Limited, January 2007.

Australasian Joint Ore Reserves Committee, 2004. Australasian Code for Reporting of Mineral Resources and Ore Reserves. Australasian Institute of Mining and Metallurgy, Melbourne.

Azzaz, S., Azab, M., Karous, S., Goubashi, M., Boutrus, N., Khalifa, K., Khalaf, I., Abdel Razik, M., El Sobky, M., Abbas, M., Kousnatov, B., & Stepnov, E., 1978. Results of Prospecting - Exploration at the Sukari Gold Ore Field in the Central Eastern Desert in 1975-1977. EGSMA & USSR Technoexport Contract 78442, Sukari Party 7A/75 & 7A/76. Unpublished EGSMA documents 11/1978 & 98/1978, EGSMA, Cairo.

Buisson, G., & LeBlanc, M., 1985. Gold in Carbonatized Ultramafic Rocks from Ophioloite Complexes. Economic Geology, vol 80, 1985, pp2028-2029.

Cavaney, R.J., 2005. Geology of Sukari Gold Mine, Eastern Desert, Egypt. Unpublished Centamin Egypt Limited Report

Coffey Mining Pty Ltd, 2009. Mineral Reserves Review. Unpublished report to Centamin Egypt Limited, April 2009.

Ghoneim, M.F., Bjoriykke, A., & Harraz, H.Z., 1999. Rb/Sr and Sm/Nd Isotopic Systems of El Sukari Granite, Eastern Desert, Egypt. Proc. First International Conference on the Geology of Africa, Assiut, Egypt, Nov 1999, vol 1, pp281-282.

Hellman & Schofield Pty Ltd, 2006. Recoverable Resource Estimation of the Sukari Deposit Egypt. Unpublished report to Centamin Egypt Limited, December 2006.

Hellman & Schofield Pty Ltd, 2007. Sukari Resource Estimate, February 2007. Unpublished report to Centamin Egypt Limited, February 2007.

Hellman & Schofield Pty Ltd, 2009. Recoverable Resource Estimation of the Sukari Deposit Egypt. Unpublished report to Centamin Egypt Limited, February 2009.

Hume, W.F., 1937. Geology of Egypt. Government Printer, Cairo.

Isaaks, E.H. & Srivastava, M.R., 1989. Applied Geostatistics. Oxford University Press, New York.

Journel, A.G., & Huijbregts, C.J., 1978. Mining Geostatistics. Academic Press Inc., London.

Khalil I K, and Arslan A.I (2006) “Petrography – Thin Section transmitted Light and Reflected light study of specimens from Sukari Gold Mine”, Internal consulting reports, July and December 2006

Knight Piesold, 2006. Tailings Storage Facility Design and Plant Site Foundation Assessment, Feasibility Study PE 401 – 00015/5, July 2006

Lakefield Oretest, 2000a. Leaching and Other Testwork on Sukari Gold Ore. Unpublished technical report prepared for Centamin Egypt Limited. Lakefield Oretest Pty Limited, Perth, Job No. 8363.

Lakefield Oretest, 2000b. Further Testwork on Ore from the Sukari Gold Mine. Unpublished technical report prepared for Centamin Egypt Limited. Lakefield Oretest Pty Limited, Perth, Job No. 8530.

Moghazi, A.M., Mohamed, F.H., El Sayed, M.M., & Kanisawa, S., 1999. Geochemistry and Petrogenesis of Late Proterozoic Rock Suites in the Homrit Waggat and El Yatima Areas, Eastern Egypt. Proc 4th International Conference on Geochemistry, Alexandria University, Egypt, 15-16 September 1999, pp1-21.

Newling, P, Orr, C, Johnson, N, Morgan, D, Staples, M, Motteram, G, 2007. Form 43-101F1 Technical Report Sukari Gold Project.

Orr, C.M, 2006. Sukari Gold Project, Amun and Ra Zones Proposed Staged Pits, Geotechnical Evaluation for Mining Feasibility Purposes. Amended draft, Unpublished report to Centamin Egypt Limited, March 2006.

Roche Process Engineering Pty Ltd, 2007. Sukari Gold Project Definitive Feasibility Study. Unpublished report to Pharaoh Gold Mines NL, February 2007.

Schofield, N.A., & Rolley, P.J., 1997. Optimisation of Ore Selection in Mining: Method and Case Studies. Proceedings of the Third International Mining Geology Conference, Launceston, Tasmania. Australasian Institute of Mining & Metallurgy Publication Series No. 6/97.

Sharara, N.A., 1999. Stable Isotopes and fluid Inclusions of the Gold Mineralisation at El Sukari District, Central Eastern Desert, Egypt: Genetic Constraints. Proc 4th International Conference on Geochemistry, Alexandria University, Egypt, 15-16 September 1999, pp317-339.

Sharara N., & Vennemann, T.W., 1999. Composition and Origin of the Fluid Responsible for Gold Mineralisation in Some Occurrences in the Eastern Desert, Egypt: Evidence from Fluid Inclusions and Stable Isotopes. Proc. First International Conference on the Geology of Africa, Assiut, Egypt, Nov 1999, vol 1, pp421-445.

Srivastava, R.M., & Parker, H., 1988. Robust Measures of Spatial Continuity. *In* Armstrong (ed.), Geostatistics, pp 295-308. Reidel, Dordrecht, Netherlands.

Stern, R.J., & Hedge, C.E., 1985. Geochronological and Isotopic Constraints on the Late Precambrian Crustal Evolution in the Eastern Desert of Egypt. American Journal of Science, vol. 285, pp 97- 127.

Vail, J.R., 1983. Pan African Crustal accretion in Northeast Africa. Journal of African Earth Science, 1, pp285-294.

20 DATE AND SIGNATURES

The effective date of this report is 6 December 2010.