EX-99.4 5 dex994.htm TECHNICAL REPORT: LEBOWA PLATINUM MINE, LIMPOPO PROVINCE, SOUTH AFRICA

Exhibit 99.4

Deloitte.

Technical Report: Lebowa Platinum Mine Limpopo Province, South Africa

on behalf of

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Table of Contents

1 Summary 15
1.1 Summary of geology and mineralisation 15
1.2 Summary of exploration concept 16
1.3 Summary status of development projects and operations 16
2008 Lebowa operations 17
Current production 17
Phase 1a Expansion - Approved Projects 18
Phase 1b Expansion - Proposed Projects 19
1.4 Qualified Persons conclusions and recommendations 19
2 Introduction 22
3 Reliance on other experts 24
4 Property description and location 25
4.1 Location of Lebowa Platinum Mine 25
4.2 Lebowa's mining licence areas 25
4.3 Mineral tenure and the South African regulatory environment 27
4.3.1 LPM's Mining Rights 27
4.3.2 The Mining Charter 28
4.3.3 The Royalty Bill 28
4.4 Issuer's interest 29
4.5 Royalties, back-in rights, payments, agreements, encumbrances 29
4.6 Environmental 30
4.6.1 Permitting 30
4.6.2 Environmental liabilities 30
4.6.3 Rehabilitation Provision 31
4.7 Permits 32
4.8 Surface structures 32
5 Accessibility, climate, local resources, infrastructure and physiography 34
5.1 Topography, elevation and vegetation 34
5.2 Access 34
5.3 Proximity to population centre and transport 34

5.4 Climate and length of operating season 35
5.5 Infrastructure 35
5.5.1 Power 35
5.5.2 Water 36
5.5.3 Compressed air supply 37
5.5.4 Concentrators 37
5.5.5 Tailings dams 37
5.5.6 Waste rock dumps 38
5.6 Personnel 38
5.6.1 Organisational structure and compliment 38
5.6.2 Employment policy 38
5.6.3 Skills shortage and development 39
5.6.4 Skills training 39
6 History 41
6.1 Prior ownership and ownership changes 41
6.2 Previous exploration and development work 41
6.3 Historical mineral resource and mineral reserve estimates 42
6.4 Production history 42
7 Geological setting 43
7.1 Regional geology 43
7.2 Geology of the Lebowa area 46
7.3 UG2 and Merensky characteristics 49
7.3.1 Potholed UG2 50
7.3.2 Bifurcated UG2 51
7.3.3 Potholed Merensky 52
7.4 Weathering and oxidation 53
7.5 Joints and faults 53
7.6 Dykes 54
7.7 Replacement pegmatoids 54
7.8 Geological losses 55
8 Deposit types 56
9 Mineralisation 57
9.1 UG2 mineralisation 57

9.1.1 UG2 geochemistry 57
9.2 Merensky mineralisation 57
9.2.1 Merensky geochemistry 57
9.3 Mineralogy and metallurgy 58
10 Exploration 59
10.1 Mapping 59
11 Drilling 60
11.1 Diamond drilling 60
12 Sampling method and approach 61
12.1 Core logging and sampling 61
12.2 Underground sampling 62
12.2.1 Planning 62
12.2.2 Sample spacing 62
12.2.3 Sampling method 62
12.2.4 Dispatch supervision 63
13 Sample preparation, analyses, and security 64
13.1 Assay procedure 64
13.1.1 Core samples 64
13.1.2 Sample preparation 64
13.1.3 XRF-AR method for the analysis of Cu Ni 64
13.1.4 Fire Assay ICP method for the analysis of Pt, Pd, Au 65
13.1.5 Fire Assay ICP method for the analysis of Rh 65
13.1.6 Quality Assurance/Quality Control (QA/QC) 66
13.1.7 Rock Density 67
13.2 Laboratory facilities at Lebowa 67
14 Data verification 69
15 Adjacent properties 70
16 Mineral processing and metallurgical testing 71
16.1 Introduction 71
16.2 Merensky (MF3) Concentrator Process Review 72
16.2.1 Merensky concentrator process flow sheet 72
16.2.2 Merensky Ore Stockpiling and Blending 73
16.2.3 Merensky Ore Processing 74

16.2.4 Merensky plant modernization program	78
16.3 UG2 (MF2) Metallurgical Processing Plant	78
16.3.1 UG2 Ore Stockpiling and Blending	79
16.3.2 UG2 plant process flow sheet	80
16.3.3 UG2 Ore Processing	81
16.4 Concentrate Handling	84
16.5 Processing plants condition and management	85
16.5.1 Concentrator safety	85
16.5.2 Plant condition and maintenance	86
16.5.3 Process Control	87
16.5.4 Spares and Stocks	89
16.5.5 Laboratory facilities	89
16.6 Concentrator performance	90
16.6.1 Merensky Concentrator	90
16.6.2 UG2 Concentrator	90
16.6.3 Concentrator 4E metal Recoveries	92
16.6.4 Indicated individual 4E element recoveries	92
16.7 Concentrator expansions	93
16.8 Tailings management and expansion	93
16.9 Metallurgical testing	94
16.10 Sale of Concentrate Agreement	95
16.11 Prices and Charges	95
16.12 Penalties	95
16.13 Conclusions	95
17 Mineral Resource and Mineral Reserve estimates	98
17.1 The Mineral Resource and Mineral Reserve statement, December 2008	99
17.1.1 Estimation methodology	100
17.1.2 Definition of pay limits based on economic criteria	100
17.1.3 The Mineral Resource and Mineral Reserve statement, December 2008	102
17.1.3 Summary	104
17.2 Disclosure	105
17.2.1 Lebowa Competent Persons procedure	105
17.2.2 Minimum standards	105

17.2.3 Lebowa Competent Person(s) for Mineral Resource estimates 106
17.2.4 Lebowa Competent Person(s) for Mineral Reserve estimates 106
17.2.5 Technical specialists for Mineral Reserve estimates 107
17.2.6 Review process 107
17.3 Merensky Mineral Resource estimation 107
17.3.1 Modelling overview and estimation criteria 108
17.3.2 Merensky - Borehole distribution plan 110
17.3.3 Merensky Reef - Statistical analysis 111
17.3.4 Merensky Reef - Brakfontein Regional Pothole area 113
17.3.5 Merensky - Geozone definitions 114
17.3.6 Merensky Reef - Variogram analysis 115
17.3.7 Merensky – Estimated Mineral Resource model plans 115
17.3.8 Merensky Mineral Resource classification 123
17.3.9 Merensky Reef - Geological losses 123
17.3.10 Merensky Reef – Mineral Resource cut evaluation and resource reporting 124
17.3.11 Merensky Reef - Remaining Mineral Resources evaluation 127
17.3.12 Merensky Reef - Mineral Reserve Statement 130
17.4 UG2 Mineral Resource estimation 133
17.4.1 Modelling overview and estimation criteria 133
17.4.2 Flood-line and River Pillars 135
17.4.3 UG2 - Data sources 139
17.4.4 UG2 Mineral Resource block model 139
17.4.5 UG2 - Borehole distribution plan 139
17.4.6 UG2 - Statistical analysis 141
17.4.7 UG2 - Capping strategy 142
17.4.8 UG2 - Geozone definitions 142
17.4.9 UG2 – Estimated Mineral Resource Model Results 144
17.4.10 UG2 Mineral Resource classification 154
17.4.11 UG2 - Resource mining cut evaluation and reporting 159
17.4.13 UG2 - Mineral Reserve Statement 162
17.4.14 Risk assessment results 164
17.5 Known issues that materially affect mineral resources and mineral reserves 164
18 Other relevant data and information 167

18.1 Grade-tonnage curves 167
18.2 LPM Depletion Strategy 176
18.2.1 Mining blocks 176
18.2.2 Access strategy 179
18.3 Mining methods 182
18.3.1 Decline shaft system (Phase 1) 182
18.3.2 Main vertical shaft system (Phases 2 and 3) 182
18.3.3 Primary development 183
18.3.4 Secondary development 183
18.3.5 Conventional stoping 183
18.3.6 Hybrid stoping 186
18.4 Mining Production and Projects 188
18.4.1 Current production 188
18.4.2 Phase 1a Expansion - Approved Projects 189
18.4.3 Phase 1b Expansion - Proposed Projects 190
18.4.4 Life-of-Mine production profile 190
18.5 Capital Expenditure 191
18.5.1 Capital expenditure budgeting and categories 191
18.5.2 Previously approved Project capital expenditure 193
18.5.3 Previously proposed projects capital expenditure estimates 195
18.5.4 Mining 'Stay-in-Business' capital expenditure estimates 195
18.5.5 Summary of capital expenditure estimates 195
18.5.6 Concentrator capital expenditure estimate 197
18.5.7 Concentrator 'Stay-in-Business' capital expenditure estimate 197
18.5.8 Working capital expenditure 197
18.6 Operating Costs 197
18.7 Markets 199
18.8 Economic analysis 203
18.8.1 Commodity prices 203
18.8.2 Company tax 204
18.8.3 Financial indicators 205
18.8.4 Payback and Internal rate of return 214
18.8.5 DCF sensitivity analysis 214

18.9 Pothole Risk 217
18.9.1 Bifurcated UG2 Reef 223
19 Interpretation and Conclusions 225
20 Recommendations 228
21 Glossary 230
22 Bibliography 241
23 Dates and signatures 245
24 Certificates 246

List of Figures

Figure 1.1: Lebowa Platinum Mine workings (Anglo Platinum Annual Report, 2008) 17
Figure 4.1: Location of Lebowa Platinum Mine in relation to the Bushveld Igneous Complex 25
Figure 4.2: Location of properties and boundaries 26
Figure 4.3: Simplified ownership structure after the Lebowa Transaction 29
Figure 4.4: Lebowa Platinum Mine surface infrastructure plan 33
Figure 7.1: Geological map of South Africa 43
Figure 7.2: Geographic relationship in between the granites, volcanic rocks, granophyres and the mafic rocks of the Rustenburg Layered Suite 44
Figure 7.3: The various zones of the RLS 45
Figure 7.4: Type 1, 2 and 3 chromitite potholes as associated with the UG2 chromitite layer 45
Figure 7.5: Lithological variations as observed at a pothole edge as associated with the UG2 46
Figure 7.6: Regional Geological Setting - Eastern Limb of the Bushveld Complex 47
Figure 7.7: Typical stratigraphic column for Lebowa Platinum Mine 48
Figure 7.8: Lebowa Platinum Mine mining area and shafts 49
Figure 7.9: Middelpunt Hill UG2 showing pothole distribution 51
Figure 7.10: Distribution of boreholes that have intersected bifurcated UG2 52
Figure 7.11: Merensky Reef pothole distribution 53
Figure 7.12: Graphic representation of the aeromagnetic survey 54
Figure 11.1: Merensky and UG2 borehole distribution 60
Figure 12.1: The top and bottom UG2 contact samples 61
Figure 16.1: General view of the concentrators 71
Figure 16.2: Merensky Plant MF3 process circuit 73
Figure 16.2: Merensky Plant MF3 process circuit 73
Figure 16.3: Ore removal chutes under 4200t silo 74
Figure 16.4: Merensky ore Accuweigh weightometer 75
Figure 16.5: Primary FAG mill with the primary rougher smart cells in the background 75
Figure 16.6: Secondary ball mill 76
Figure 16.7: General view of flotation cells in Merensky plant 77
Figure 16.8: Larox filter 77

Figure 16.9: Removed oversize rock and support timber 79
Figure 16.10: Twickenham ore drying pad 80
Figure 16.11: UG2 plant process circuit 81
Figure 16.12: General view of the UG2 crushing facility 82
Figure 16.13: Primary UG2 ball mill 83
Figure 16.14: Mill product screen and the primary rougher flotation cells 83
Figure 16.15: Two stage primary rougher tail classification 84
Figure 16.16: Merensky and UG2 filtered concentrate storage facilities 85
Figure 16.17: Cable reinforcing around the 4200 silo 87
Figure 16.18: Process control room 88
Figure 16.19: Automatic sample cutter in the Merensky plant tailings stream 89
Figure 17.1: Footwall thickness distribution used for modelling 109
Figure 17.2: Additional boreholes drilled since 2006 are shown in green 110
Figure 17.3: Hangingwall PGE cut grade distribution within the 10 cm cut 112
Figure 17.4: The relationship between the Merensky Reef PGE grade and thickness 113
Figure 17.5: Merensky Reef - Brakfontein regional pothole area 114
Figure 17.6: Merensky Reef East and West Geozones 115
Figure 17.7: Merensky Reef thickness distribution 116
Figure 17.8: Merensky Reef PGE grade (g/t) distribution 117
Figure 17.9: Merensky Reef density distribution (t/m^3) 118
Figure 17.10: Merensky Reef Resource cut width distribution (m) 119
Figure 17.11: Merensky Reef –Resource cut PGE grade distribution (g/t) 120
Figure 17.12: Merensky Reef - 4E PGM kriging variance 121
Figure 17.13: Merensky Reef - 4E PGM kriging variance 122
Figure 17.14: Merensky Reef - Mineral Resource classification 123
Figure 17.15: Merensky Reef - Lebowa mining lease area polygons 129
Figure 17.16: Stratigraphic description of bifurcated UG2 136
Figure 17.17: Areas with abnormal UG2 thickness 137
Figure 17.18: Areas with high dilution (internal waste) within the UG2 138
Figure 17.19: UG2 borehole data with new (2007) boreholes in green 140
Figure 17.20: Two Cu geochemical geozones were separated into a West and East area either side of the X -112 600 coordinate 143
Figure 17.21: Chromitite leaders relative to the mining cut 143

Figure 17.22: UG2 thickness distribution 145
Figure 17.23: UG2 4E PGE grade distribution 146
Figure 17.24: UG2 Reef density distribution 147
Figure 17.25: UG2 internal dilution distribution 148
Figure 17.26: UG2 Reef Resource mining cut thickness distribution 149
Figure 17.27: UG2 Reef Resource mining cut 4E PGM grade distribution 150
Figure 17.28: UG2 Reef Pt kriging variance distribution 151
Figure 17.29: UG2 Reef kriging efficiency distribution 152
Figure 17.30: UG2 Reef Resource mining cut thickness distribution 153
Figure 17.31: UG2 Resource mining cut 4E PGE grade distribution 154
Figure 17.32: UG2 Mineral Resource classification as defined in 2007 155
Figure 17.33: UG2 - Mineral Resource evaluation polygons 157
Figure 18.1: Grade-tonnage curves for UG2 168
Figure 18.2: Grade versus stoping width of UG2 kriged data 168
Figure 18.3: Stoping width of UG2 across Lebowa Platinum 169
Figure 18.4: Stoping width distribution of UG2 169
Figure 18.5: Grade-tonnage curves for Merensky Reef 170
Figure 18.6: Grade vs. stoping width of Merensky Reef kriged data 170
Figure 18.7: Stoping width of Merensky Reef across Lebowa Platinum 171
Figure 18.8: Stoping width distribution of Merensky Reef 171
Figure 18.9: Grade–tonnage curves for Measured, Indicated and Inferred Resources in UG2 173
Figure 18.10: Grade–tonnage curves for Measured, Indicated and Inferred Resources in Merensky Reef 174
Figure 18.11: LPM mining rights area 177
Figure 18.12: Merensky mining blocks 178
Figure 18.13: UG2 mining blocks 178
Figure 18.14: Schematic diagram of Phases 1, 2 and 3 development and associated shafts 181
Figure 18.15: Half level design parameters for Middelpunt Shaft (schematic view) 185
Figure 18.16: Cost and timing of the capital estimates (2005 terms), of postponed projects 195
Figure 18.17: Capital expenditure plan for the LOM 196

Figure 18.18: Average Platinum spot prices until January 28, 2009 200
Figure 18.19: Average Palladium spot prices until January 28, 2009 200
Figure 18.20: Average Rhodium spot prices until January 28, 2009 201
Figure 18.21: Average Gold spot prices until January 28, 2009 201
Figure 18.22: LOM real term cash-flow (2009MT) 209
Figure 18.23: LOM production profile – tonnes 210
Figure 18.24: 4E grade distribution 210
Figure 18.25: LOM real revenue by PGM and BM (2009MT) 211
Figure 18.26: LOM BM contribution by value 211
Figure 18.27: LOM real revenue by 4E (2009MT) 212
Figure 18.28: LOM revenue contribution by 4E (2009MT) 212
Figure 18.29: LOM real term operating unit cost (2009MT) 213
Figure 18.30: LOM real term operating unit cost (2009MT) 213
Figure 18.31: DCF valuation sensitivity to revenue 214
Figure 18.32: DCF valuation sensitivity to capital costs 215
Figure 18.33: DCF valuation sensitivity to operating costs 215
Figure 18.34: UG2 pothole prediction through extrapolation of known pothole density statistics, Bleskop Shaft 218
Figure 18.35: Subdivision of 1st derivative of the slope of the Merensky Reef into nine visually semi-homogeneous colour domains 219
Figure 18.36: Upper 95% confidence limits for prediction derived from the domain data 220
Figure 18.37: Merensky Reef potholes showing high risk potential of large losses due to potholes 221
Figure 18.38: A portion of the Merensky Reef pothole plan illustrating the potential for higher pothole losses in the unmined area 222
Figure 18.39: UG2 Chromitite potholes subdivided into domains of large and small potholes 223
Figure 18.40: High risk of losses from bifurcated UG2 Reef (blue dots) 224

List of Tables

Table 2.1: Responsibilities of each co-author 23
Table 4.1: Lebowa Platinum Mine's Mining Licence 26
Table 5.1: Lebowa Platinum Mine employee complement as at December 2008 38
Table 5.2: LPM critical skills shortages as at December 2008 39
Table 6.1: Production summary for 2005 to 2008 42
Table 13.1: Average densities for Merensky 67
Table 13.2: Average densities for UG2 67
Table 16.1: Historical concentrator performance data 91
Table 16.2: Average 4E metal recoveries 2004 to 2008 92
Table 16.3: Individual 4E element recoveries 93
Table 16.4: Nickel and Copper recoveries 93
Table 17.1: December 2007 Mineral Resources 98
Table 17.2: December 2007 Mineral Reserve Statement 99
Table 17.3: Global assumptions as per third quarter such as commodity prices and exchange rates 101
Table 17.4: Pay limits for UG2 4E 101
Table 17.5: Pay limits for Merensky 4E 102
Table 17.6: December 2008 Mineral Resources 103
Table 17.7: December 2008 Mineral Reserves 104
Table 17.8: Merensky Borehole and MRM data 111
Table 17.9: Statistical parameters Merensky Reef, hangingwall and footwall cuts. Footwall cuts are measured from the base of the Merensky Reef 112
Table 17.10: Merensky Reef - LPM Mineral Resources and classifications 125
Table 17.11: Merensky Reef - Brakfontein Project Mineral Resources and classifications 126
Table 17.12: Merensky Reef – Remaining Mineral Resources and classification 128
Table 17.13: Modifying factors employed for the Merensky Reef Mineral Reserve calculations 130
Table 17.14: Merensky Reef - Mineral Reserve Statement – December 2008 132
Table 17.15: UG2 borehole and MRM data 140
Table 17.16: UG2 statistical parameters for hangingwall and footwall cuts 141
Table 17.17: UG2 Reef - Lebowa Mine Mineral Resource cut evaluation 158

Table 17.18: UG2 Reef - Lebowa Mine Mineral Resource cut evaluation (exclusive of oxidised zones) 158
Table 17.19: UG2 Reef – Middelpunt Hill Project Mineral Resource cut (0-8 Level, excluding oxidised zones) 159
Table 17.20: Middelpunt Hill Project – UG2 Resource Classification 160
Table 17.21: UG2 Reef Mineral Resources 161
Table 17.22: UG2 modifying factors 162
Table 17.23: UG2 – Mineral Reserve Statement – December 2008 164
Table 18.1: Sample sizes considered for grade tonnage curves 172
Table 18.2: Metal prices and exchange rate for period under consideration 175
Table 18.3: Relative percentage loss of value of in situ ore (4PGE only) due to deterioration in metal prices and ZAR/USD exchange rates 175
Table 18.4: LPM mining blocks 180
Table 18.5: Parameters used in the mining design and scheduling for Vertical and UM2 Shafts 184
Table 18.6: Parameters used in the mining plan and scheduling for the Middelpunt Shaft 187
Table 18.7: Parameters used in the mining plan and scheduling for the Brakfontein Shaft 188
Table 18.8: BRK Merensky Project CBE in 2004 terms 193
Table 18.9: MPH UG2 Project CBE in 2006 terms 194
Table 18.10: LOM capital expenditure estimates 196
Table 18.11: Operating unit costs during the production build-up (first 13 years) 198
Table 18.12: Operating unit costs at various production stages throughout the LOM 198
Table 18.13: Steady state operating costs of the projects 198
Table 18.14: Annual PGM demand 199
Table 18.15: Metal prices used in the cashflow model (nominal terms) 203
Table 18.16: Weighted 4E metal prices for the first four years of the original cash flow model (real (2009MT) and nominal terms) 203
Table 18.17: Financial indicators (in real terms -2009MT) 205
Table 18.18: LOM real term cash-flow (abridged) 206
Table 18.19: Sensitivity matrix, OPEX and revenue 216

1 Summary

This technical report (TR), which is compliant with National Instrument 43-101 and the CIM Definition Standards, describes the Lebowa mineral exploration, development and mining production area located in the Sekhukhuneland District in the Limpopo Province of South Africa (Lebowa). This report was compiled by Deloitte at the request of Anooraq Resources Corporation.

Anooraq entered into a conditional agreement with Anglo Platinum, whereby Anglo Platinum will sell to Anooraq an effective 51% of Lebowa Platinum Mine (LPM) and an additional 1% interest in each of the Ga-Phasha, Boikgantsho and Kwanda Joint Venture Projects, which will result in Anooraq holding a controlling interest in each of these projects. This report concerns Lebowa only.

This update to the Lebowa Platinum Mine Technical Report was necessitated by Anooraq's obligation to disclose material changes in terms of the Lebowa Transaction, mine plans and financial analysis.

This TR is largely a technical review of work performed by others. The Qualified Persons (QPs) of this TR provide their comments and opinion on the applicability of this work.

1.1 Summary of geology and mineralisation

LPM is located in the geological area of South Africa commonly referred to as the Bushveld Complex (BC). The BC is situated in the northern half of South Africa and exists as an ellipse-shaped body consisting of five lobes. The BC is the world's largest known mafic igneous intrusion, and extends approximately 450 km east to west and approximately 250 km north to south. It underlies parts of the Limpopo, North-West, Gauteng and Mpumalanga Provinces. The BC is estimated to have been formed approximately 2,054 million years ago (Ma). The Rustenburg Layered Suite of the BC is host to Platinum Group Metal (PGM) mineralisation as well as chrome (Cr), vanadium (V), nickel (Ni), copper (Cu) and cobalt (Co). The BC is known for its continuity of layering and associated PGM mineralisation.

The PGM and base metal mineralisation of the eastern limb of the Bushveld is contained primarily in two reefs, the UG2 Chromitite Reef (UG2).and the Merensky Pyroxenite Reef (Merensky). The PGM and base metal mineralisation of the Merensky predominantly occurs between an upper and lower chromitite stringer. Mineralisation also extends to wider intervals above and below these stringers. PGM mineralisation is often associated with visible base metal sulphides.

The UG2 contains PGM mineralisation within a chromitite layer, together with variable occurrences in the immediate hangingwall and footwall rocks. A 95 cm cut horizon, perpendicular to the plane of the UG2, accounts for most of the mineral content.

1.2 Summary of exploration concept

PGM mineralisation includes platinum (Pt), palladium (Pd), rhodium (Rh), ruthenium (Ru) osmium (Os) and iridium (Ir). The precious metals gold (Au) and silver (Ag) and the base metals chrome (Cr), iron (Fe,) cobalt (Co), nickel (Ni) and copper (Cu) are often associated with PGM mineralisation.

The geological exploration and evaluation process of potentially mineralised magmatic horizons involves reconnaissance, planning, diamond drilling, core logging and sampling, trenching and sampling, soil sampling, aero-magnetic surveys, ground magnetic surveys, underground and surface mapping, data processing and interpretation and modelling.

Activities at LPM have centred historically on the Merensky horizon and in recent years considerable focus has been directed to the UG2.

The metals of economic interest at LPM are Pt, Pd, Rh, Au, Ni and Cu. The more significant precious metals (Pt, Pd, Rh, Au) are referred to as “4E” for analytical and resource reporting. The South African PGM mining industry has traditionally stated its mineral resources and mineral reserves in terms of 4E grade (4E g/t) rather than metal equivalents. 4E grades do not reflect metal prices or recoveries.

1.3 Summary status of development projects and operations

LPM has been in production since 1969. Originally ore was produced from the Merensky Reef via Vertical Shaft located on the Middelpunt Hill property and various inclined shafts (Figure 1.1). In addition, since 2001, LPM has been producing ore from the UG2 via a series of adits at Middelpunt Hill.

In 2007 LPM produced 94,300 ounces of refined platinum from a milled tonnage of 1,333,000 (Anglo Platinum Annual Report, 2007). This dropped to 72,600 ounces of refined platinum from a milled tonnage of 1,098,000 in 2008 (Anglo Platinum Annual Report, 2008).

In its long-term planning in respect of LPM, Anglo Platinum determined that the production rate should be increased in order to reflect the true quality and scale of the mineral resource. Accordingly, Anglo Platinum compiled the LPM 2007 life-of-mine (LOM) plan, which comprised a two phase expansion of existing production from 130,000 tpm to 245,000 tpm (nominal 410,000 4E PGM Oz pa - Phase 1a Expansion) by 2012 and 375,000 tpm (nominal 630,000 4E PGM Oz pa) (Phase 1b Expansion) by 2019.

In view of the recent decline of the commodity market, Anglo Platinum and Anooraq Resources have embarked on a revised LOM plan. The revised production and cost schedule at LPM was effected primarily as a result of the global economic slowdown witnessed from June 2008, which had an adverse impact on financial and commodity markets, as well as the proposed financing strategy for implementation of the LPM acquisition, which necessitated the deferral of the UG2 Middelpunt Hill expansion project (Delta 80) in order to accommodate senior debt financing requirements.

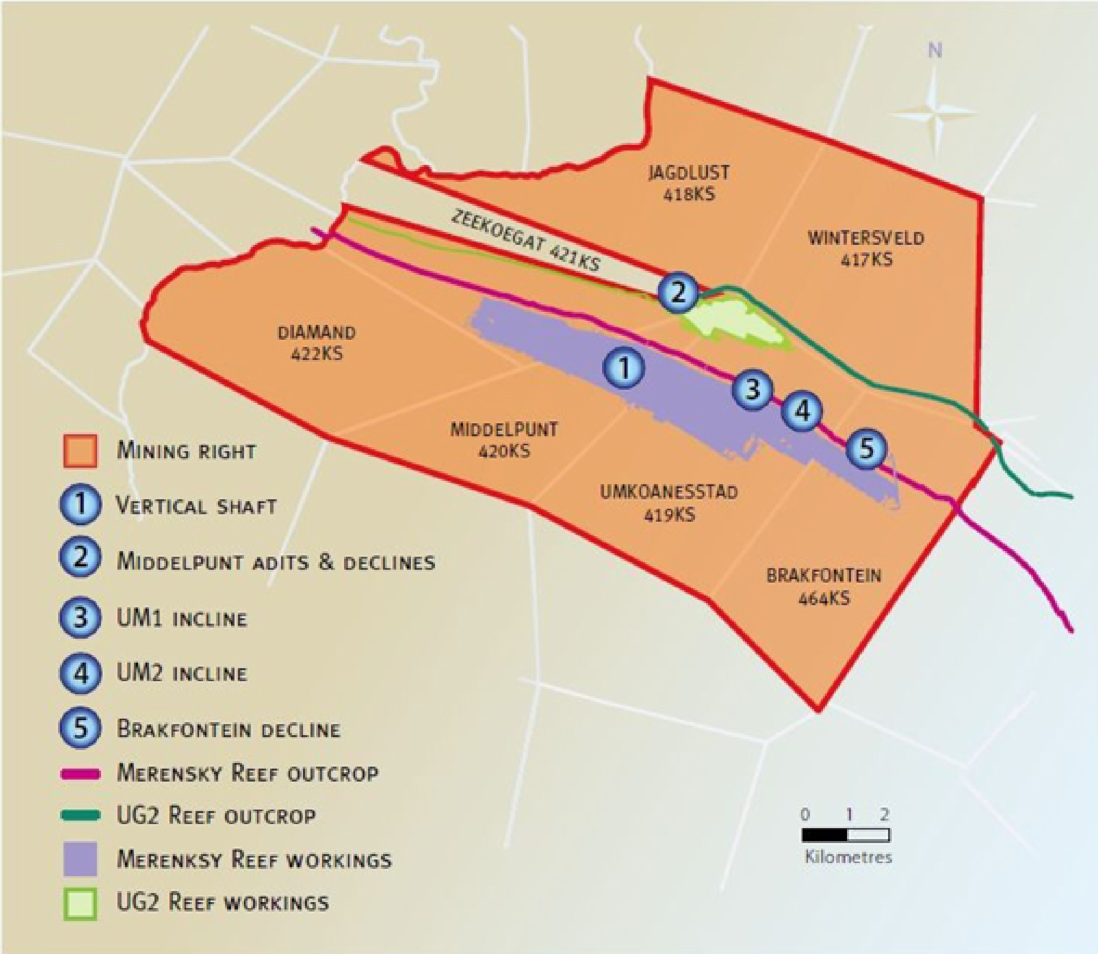


Figure 1.1: Lebowa Platinum Mine workings (Anglo Platinum Annual Report, 2008)

A LOM mining plan considering the revised parameters was under construction at the time of writing of this report. The first 5 years of this plan is being planned and scheduled in detail.

2008 Lebowa operations

Current production

Anglo Platinum indicates that current production of Merensky ore is 35 ktpm to 40 ktpm from the Vertical Shaft workings and 13 ktpm to 15 ktpm from the UM2 Inclined Shaft. UG2 production from the Middelpunt Hill Adits is 45 ktpm. This conforms to the production of 1 100 kt per annum as reported in Anglo Platinum's Annual Report (2008), which represents an average total monthly production of approximately 92 ktpm.

Merensky

The Vertical Shaft services the working places on the north-west extremity on the Zeekoegat property and on the lowest levels of the Vertical Shaft on the Middelpunt and Umkoanesstad properties while the UM2 Inclined Shaft services the workings on the south-east extremity within the Umkoanesstad and Brakfontein properties.

The current workings on Zeekoegat and Brakfontein and on portions of Umkoanesstad are remote from the shafts and as such productivity is less than optimal. Similarly, some working places on the Middelpunt property and on the nearer portions of Umkoanesstad, are accessed by winzes i.e. are below the lowest working level, and are particularly difficult to access.

UG2

Until 2006, UG2 ore production from the Middelpunt Hill Adits was steady at 55 ktpm. Since 2006 UG2 ore production from these adits has declined and has to some extent been supplemented by UG2 ore production from Level 0 of the Middelpunt Hill Decline. UG2 ore production from the Middelpunt Hill Adits is planned to ramp up from 35 ktpm to 60 ktpm until 2014.

Phase 1a Expansion - Approved Projects

There were two approved projects (at feasibility level) in Phase 1a expansion of the original plan, the Brakfontein Merensky Project at 120 ktpm (BRK Merensky Project) and the Middelpunt Hill UG2 Expansion Project at 125 ktpm (MPH UG2 Project).

Prior to Anglo Platinum's 'Investment Proposal' (IP) process, Anglo Platinum project teams – with support from selected mining engineering consulting companies, prepared Feasibility Studies including detailed capital budget estimates and mine plans.

Both of these projects have been through the IP process and funding is therefore approved. However, due to the currently subdued financial market, these expansion projects have been slowed down (BRK Merensky) and postponed (MPH UG2).

Merensky

The BRK Merensky Project, approved by Anglo Platinum in 2004, is planned to produce 120 ktpm of Merensky ore from the area below the oxidised zone and above 650 m below surface, via a Decline Shaft System located on the Brakfontein property.

In the new mining plan it is indicated that Brakfontein Merensky ore production will replace Vertical/UM2 Shaft production in 2014 at 1 113 ktpa and steady-state production of 1 440 ktpa is expected from 2016 through till 2027.

UG2

The MPH UG2 Project (Delta 80), approved by Anglo Platinum in 2007 was planned to produce 125 ktpm of UG2 ore from below the oxidised zone to 650 m below surface, within the Middelpunt property. The project included the continued sinking of the Middelpunt Hill Decline and the sinking of a new Decline Shaft System, from near the site of the current UM1 Shaft, through the hanging-wall and the UG2, to intersect the Middelpunt Hill Decline on No.2 Level. Below No.2 Level the Decline Shaft System will continue as a conventional footwall decline shaft system as described in Section 18.2.2.

This project has been postponed to ± 2017. Therefore, UG2 reef production is expected to commence in 2013 and maintained until 1 525 ktpa is achieved in 2021, which will be maintained until 2043.

Phase 1b Expansion - Proposed Projects

There were three proposed projects in the Phase 1b expansion:

- The Brakfontein UG2 Project (BRK UG2),
- The Zeekoegat Merensky Project (ZKG Merensky), and
- The Zeekoegat UG2 Project (ZKG UG2)

The Brakfontein UG2 Project was at a Pre-Feasibility Study level and the Zeekoegat Projects were at Conceptual Study level. The former included a Preliminary Cost Estimate and the latter an Order of Magnitude Cost Estimate. All these projects were postponed.

1.4 Qualified Persons conclusions and recommendations

Deloitte has reviewed the relevant information provided by Anglo Platinum and is satisfied that the quality and quantity of geological information, the interpretations thereof and the estimation methodology are sufficient for the declaration of representative Measured, Indicated and Inferred Mineral Resources. Deloitte considers that the Mineral Resources are classified in accordance with the July 2007 South African Code for Reporting of Mineral Resources and Mineral Reserves (SAMREC) under the auspices of the South African Institute of Mining and Metallurgy (SAIMM), as amended and the Definition Standards on Mineral Resources and Mineral Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) adopted by CIM Council (CIM Definition Standards).

The Mineral Resource (Table 17.6) and Mineral Reserve (Table 17.7) figures reflect the position as at 31 December 2008 by allowing for depletion, losses and any additions which have occurred following the last update of the Mineral Resource and Mineral Reserve estimates in December 2007.

The 2008 Resource estimate is based on the 2007 geological model, as drilling in 2008 did not result in material changes to the geological model.

LPM has significant additional Inferred Mineral Resources, both Merensky and UG2, for future classification to Measured and Indicated status with the appropriate follow-up drilling, development and sampling. Both economic horizons extend from outcrop and are continuous to depths beyond 2,000 m.

The UG2 and Merensky Reef 2007 and 2008 Reserve estimates were determined by applying modifying factors based on historical insights and experience, independent from the mining plan and scheduling process. This approach, according to Deloitte’s interpretation of the SAMREC Code, may invalidate code compliancy, as the SAMREC Code requires Reserves for operating mines to be based on a Life of Mine plan. National Instrument 43-101, which follows the CIM Definition Standards, requires a pre-feasibility study as a minimum basis on which to base reserve estimation.

The total 2008 declared UG2 Reserve is 41.2 Mt, however the mining schedule generates a 44.9 Mt reserve. The total 2008 declared Merensky Reserve is 27.1 Mt, as compared to a scheduled output of 25.8 Mt. These discrepancies can be attributed to reserve estimation being conducted independently from mine planning and scheduling.

The 2008 Resource and Reserve estimates, as published in the Anglo Platinum Annual Report (2008), were not externally audited. Deloitte has not been tasked with auditing the 2008 reserve and its code compliancy.

Mineral Resource and Mineral Reserve estimates are reported (see Tables 17.6 and 17.7, respectively) for the LPM properties including the existing Lebowa Mine, and planned project expansions at Brakfontein and Middelpunt. Mineral Resources (Remaining Resources) are also reported for the regions outside of the existing mine plans and project expansion plans but within the LPM properties.

LOM Plan

The revised production and cost schedule at LPM was effected primarily as a result of the global economic slowdown witnessed from June 2008, which had an adverse impact on financial and commodity markets, as well as the proposed financing strategy for implementation of the LPM acquisition, which necessitated the deferral of the UG2 MPH Delta 80 project in order to accommodate senior debt financing requirements.

A revised long term mine plan, which will be finalised as the 2009 LOM plan, is therefore currently being developed. This will be incorporated into the 2009 Resource and Reserve statement for LPM.

The previous LOM plan included 5 projects, 2 of which have been excluded in the revised financial plan. It is noted that the revised plan is composed of the 3 previously approved projects, with initial commencement times and ramp-up schedules changing.

Key production and financial indicators

Anooraq’s financial model, as presented here, is not based on the current Life of Mine plan, but on a statement of strategic intent. According to this financial model, considering LPM’s anticipated 35-year LOM some 92.7 Mt of ore (66.9 Mt UG2, 25.8 Mt Merensky) will be processed, producing some 13.68 MOz’s of 4E PGM’s including 6.5 MOz’s of platinum, 5.8 MOz’s of palladium, 1.0 MOz’s of rhodium and 320 kOz’s of gold plus 50 kt of copper, 73.7 kt of nickel and 1.6 kt of cobalt.

The planned tonnage of UG2 in the current financial model (66.9 Mt) exceeds the UG2 Proven and Probable Reserves (41.2 Mt) by 25.7 Mt. The Measured and Indicated UG2 Resources available for conversion to Reserves are 180.4 Mt.

The DCF model indicates, in real terms, revenue of ZAR126,749 M and a net profit of ZAR33,974 M at a 24.7% average margin.

The NPV at a discount rate of 7.5% is ZAR9,290 M.

An overall payback period is not applicable as Lebowa is an operating mine.

Concentrators

No major problems or flaws with the basic process flow sheets of either the Merensky or UG2 concentrators at LPM are evident. By modern technological standards, the concentrators employ older technology, but nevertheless maintain high operating efficiencies and availability. In general, the plants operate well with few major equipment breakdowns. The most common reasons for current plant stoppages are due to lack of ore from the shafts and planned maintenance. Both concentrators can operate at utilization efficiencies near 90%.

The Merensky concentrator recoveries of the 4E'S are good, due to the accommodating mineralogy of the Lebowa ore-body. Typically, PGM's are associated with base metal sulphides, and clearly, liberation is very good at the current plant grinds.

The average 4E's recovery over the past 5 year period (2004 to 2008) is 91.9%, over a range 90.6% to 93.0% (annual recoveries). Deloitte is of the opinion that the Merensky concentrator metal recovery performance is good.

Similarly, the UG2 PGM recoveries are good with a five year (2004 to 2008) average of 88.1%, over a range 86.9 to 89.3 (annual recoveries). The concentrate grades here too are acceptable for smelting. Deloitte is of the opinion that the UG2 concentrator performance is good.

In addition, it is noted that as UG2 production increases, cognisance has to be taken of the increased overall chrome content for smelting. The ratio of Merensky and UG2 concentrates, in terms of the chrome content, will be affected and may necessitate reducing the chrome content in the UG2 concentrate. This may be at the expense of 4E recovery from the UG2 plant. This aspect would have to be considered when the chrome content specifications and penalties are defined with Polokwane Smelter.

2 Introduction

This Technical Report (TR) has been prepared by Deloitte Mining Advisory Services (Deloitte) on behalf of Anooraq Resources Corporation (Anooraq) in compliance with the disclosure requirements of the Canadian National Instrument 43-101 **STANDARDS OF DISCLOSURE FOR MINERAL PROJECTS**. This report also complies with the requirements of the SAMREC Code, **THE SOUTH AFRICAN CODE FOR THE REPORTING OF EXPLORATION RESULTS, MINERAL RESOURCES AND MINERAL RESERVES**.

Anooraq entered into a conditional agreement with Anglo Platinum, whereby Anglo Platinum will sell to Anooraq an effective 51% of the Lebowa Platinum Mine (LPM) and an additional 1% interest in each of the Ga-Phasha, Boikgantsho and Kwanda Joint Venture Projects, resulting in Anooraq holding a controlling interest in each of these projects, for a total cash consideration (Lebowa Transaction). This TR specifically deals with LPM only.

This TR was prepared by Messrs J Schweitzer, G Güler, P Lambert, S de Waal, P Kramers, all associates of Deloitte, and T Naidoo, a Senior Geologist with Deloitte. A site visit was undertaken to Lebowa from 17 to 19 December 2008 by Messrs Schweitzer, Güler, Lambert and Naidoo. Messrs Güler and Naidoo visited the mine again from 20 to 21 January 2009, and Mr Güler visited the mine again on the 17th of March 2009. Mr Kramers visited the plant from 22 to 23 January 2009.

This TR was written as an update to the technical report prepared by Snowden in April 2008, and is largely a technical review of work performed by others. The Qualified Persons (QPs) of this TR provide their comments and opinion on the applicability of this work. Comments on the current Resource and Reserve Statement (Anglo Platinum Annual Report 2008) are also provided, but do not constitute an independent audit. Deloitte was not requested to conduct an independent audit.

This update to the Lebowa Platinum Mine Technical Report was necessitated by Anooraq's obligation to disclose material changes in terms of the Lebowa Transaction, mine plans and financial analysis.

The authors of the various sections of this report are listed in Table 2.1.

Table 2.1: Responsibilities of each co-author

Section	Title	Author
1	Summary	J Schweitzer, P Lambert, G Güler
2	Introduction	
3	Reliance on other experts	T Naidoo,
4	Property description and location	J Schweitzer
5	Accessibility, climate, local resources, infrastructure and physiography	
6	History	
7	Geological setting	J Schweitzer,
8	Deposit type	P Lambert,
9	Mineralisation	S de Waal
10	Exploration	
11	Drilling	
12	Sampling method and approach	T Naidoo,
13	Sample preparation, Analyses and Security	J Schweitzer,
14	Data verification	P Kramers
15	Adjacent properties	
16	Mineral processing and Metallurgical Testing	P Kramers
17	Mineral Resource and Mineral Reserve estimates	G Güler,
		S de Waal,
18	Other relevant data and information	J Schweitzer,
		P Lambert
		J Schweitzer,
19	Interpretation and conclusions	P Lambert,
20	Recommendations	G Güler

3 Reliance on other experts

Unless otherwise stated, information and data contained in this TR or used in its preparation has been provided by Anooraq and Anglo Platinum in the form of documentation located in a virtual data room maintained by Anglo Platinum, and from data obtained and observations made during the various site visits. Documentation was reviewed and verified for accuracy by Deloitte and relevant information extracted for this TR.

In addition, Deloitte was given access to a due diligence report prepared by Coffey Mining (trading as RSG Global) for Anooraq entitled **LEBOWA PLATINUM MINES, INDEPENDENT TECHNICAL REVIEW OF THE OPERATIONS AND PROJECTS** – REVISED **MARCH 2008** (RSG Global Report), as well as a technical report prepared by Snowden for Anooraq entitled **TECHNICAL REPORT: LEBOWA PLATINUM MINE,** dated April 2008 (Snowden Report).

Deloitte has not conducted an in-depth review of mineral title and ownership but accepts in good faith the legal opinion on this matter expressed by Deneys Reitz Attorneys in their **LEBOWA TITLE VERIFICATION** report. Mineral title and ownership details are provided in Section 4.

4 Property description and location

4.1 Location of Lebowa Platinum Mine

Lebowa Platinum Mine is situated in the Sekhukhuneland District of the Limpopo Province, the northernmost province of South Africa. It is located approximately 80 km southeast of Polokwane, the provincial capital, and approximately 330 km northeast of Johannesburg (Figure 1). The area is serviced by a tarred road between Polokwane and Lydenburg, the R37. There is direct access along a tarred service road from main tarred road to the Lebowa Platinum Mine.

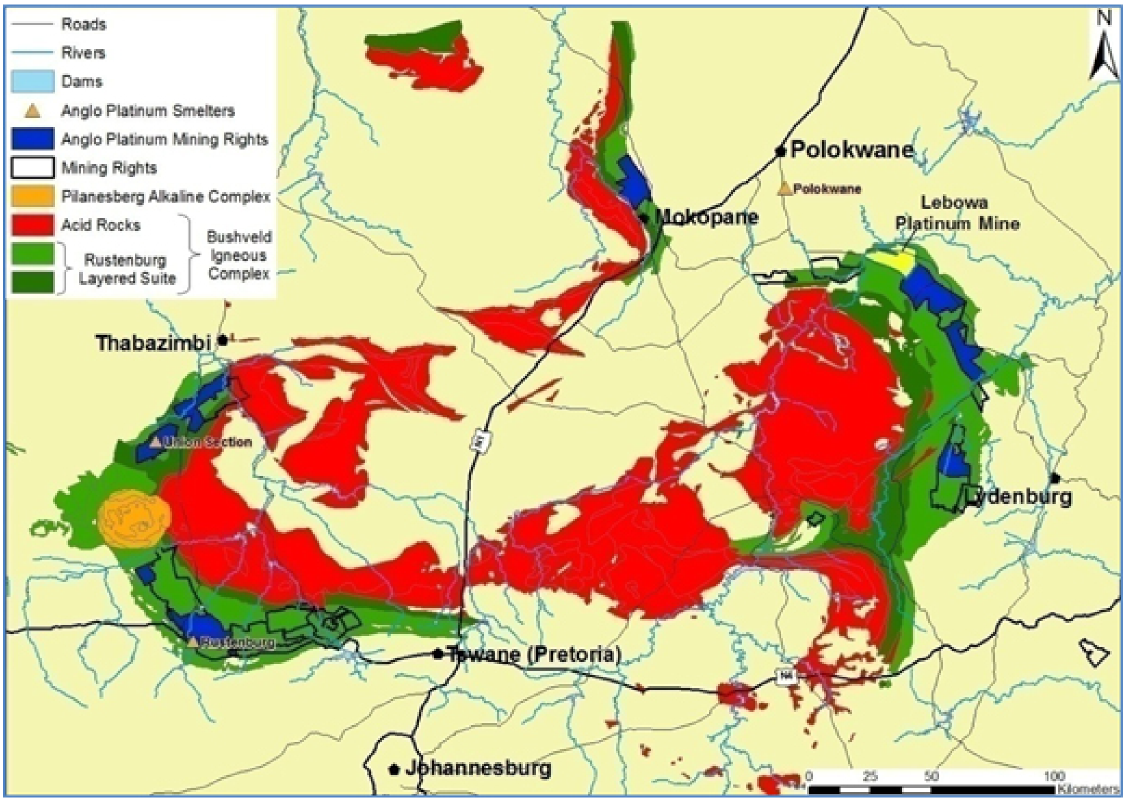


Figure 4.1: Location of Lebowa Platinum Mine in relation to the Bushveld Igneous Complex.

4.2 Lebowa's mining licence areas

Lebowa Platinum Mines Limited (LPM) holds mining licences over several adjacent properties that overlie a portion of the northeastern limb of the Bushveld Complex. The properties comprising the LPM mining area, together with licence numbers and expiry dates are presented in Table 4.1.

Table 4.1: Lebowa Platinum Mine's Mining Licence Areas.

Property	Portion	Area (ha)	Old Order Licence No.	Original Expiry Date	Date Conversion Granted	New Order Licence Number	Valid for
Middelpunt 420 KS	farm	1 544.91					
Diamand 422 KS	farm	2 238.65					Up to 30 years
Umkoanesstad 419 KS	farm	2 635.10	06/2003	17/12/2025	12/05/2008	**LP 30/5/1/2/59MR**	
Zeekoegat 421 KS	portion	2 127.69					
Brakfontein 464 KS	farm	2 391.04					
Wintersveld 417 KS	farm	2 459.75					Up to 30 years
Jagdlust 418 KS	farm	2 062.63	23/2003	26/11/2013	12/05/2008	**LP 30/5/1/2/65MR**	
Total		**15 459.77**					

The Mineral Resources and Reserves estimated for the properties in question are located within the Merensky Reef and the UG2 mineralised horizons, which outcrop on or underlie these properties, and dip from the northeast towards the southwest. The Measured and Indicated Resources are primarily located in the relatively shallow areas above 650 m below surface, while the balance of the Mineral Resource is located in the deeper areas below 650 m. The majority of the Proven and Probable Reserves are located less than 650 m below the surface.

Figure 4.2 illustrates the locations of the areas covered by the mining licences according to South African Surveyor General's plans. Traditionally, South African mining rights are issued over complete properties (farms) or portions thereof.

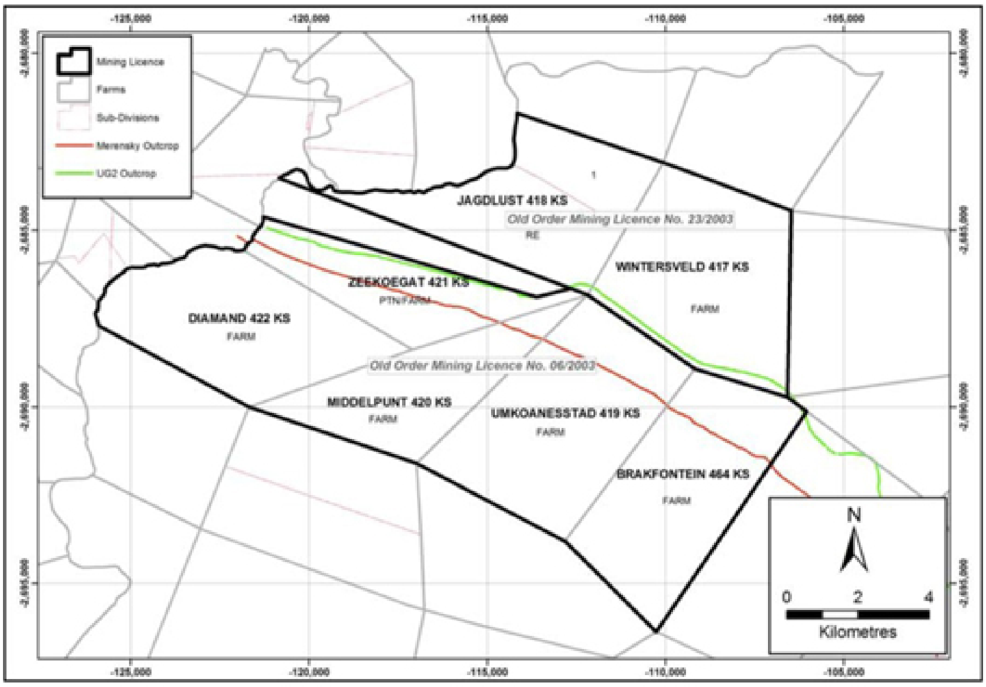


Figure 4.2: Location of properties and boundaries.

4.3 Mineral tenure and the South African regulatory environment

Prior to the promulgation of the Minerals and Petroleum Resources Development Act, Act No.28 of 2002 (MPRDA), mineral rights in South Africa, including the right to prospect and mine, were held privately, or in some instances owned by the State. Ownership of these mineral rights was effected through common law, whereby the mineral rights were vested with the surface owner of the land. The common law recognised the principle that mineral rights could be separated from the title to the land. Therefore it was possible for ownership of the surface rights, the rights to precious metals and the rights to base minerals to belong to different persons. Registration of the title to the mineral rights in the Deeds Registry Office ensured that real rights were constituted.

The MPRDA became effective legislation on 1 May 2004, replacing the Minerals Act (Act No. 50 of 1991 - Minerals Act). The objectives of the MPRDA are to adopt the internationally accepted right of the State to exercise sovereignty over the mineral and petroleum resources within South Africa and to give effect to the principle of the State's custodianship of the nation's mineral and petroleum resources. In addition, the MPRDA seeks to improve opportunities for Historically Disadvantaged South Africans (HDSA's) to become involved in the country's mineral and petroleum resources, whilst at the same time promoting development and economic growth.

The acceptance into law of the MPRDA has resulted in the transfer of South Africa's mineral rights and their administration to the State, subject to a number of transitional provisions. Three categories of rights are given recognition in terms of these transitional provisions, namely 'old order' mining rights, 'old order' prospecting rights and unused 'old order' rights. The 'use it or lose it' principle was also adopted by the State that forced companies to relinquish non-core mineral rights.

The MPRDA also provides that a Mining Right is valid for up to 30 years and can be renewed for periods of up to 30 years.

Regulations under the MPRDA set out the procedures for undertaking environmental impact assessments (EIA) and for developing environmental management programmes (EMPRs) for the construction, operation and closure of mines. The EMPR contains the environmental conditions of authorisation for the development, operation and closure of a mine. Currently operating mines must convert old order mining rights to 'new order' mining rights by 30 April 2009. A key requirement for a new mine or for the conversion process is the need for a social and labour plan, a mine works plan, proof of technical and financial competence as well as an approved EMPR.

An approved EMPR certifies that all the legislative requirements at the date when a prospecting or mining right is granted have been met or adequately provided for, and that ongoing compliance with the approved EMPR will be monitored.

4.3.1 LPM's Mining Rights

In September 2004, Anglo Platinum submitted applications for new order mining rights to the South African Department of Minerals and Energy (DME), including an application for the conversion of old order mining rights held by LPM. A grant letter for the conversion of old order Mining Rights to New Order Mining Rights was issued on the 12th of May 2008.

Deloitte understands that the Lebowa Transaction forms an integral component of Anglo Platinum's application to the DME for the conversion of LPM's old order mining rights, as it increases Lebowa's HDSA's equity ownership interest, one of the key outstanding requirements for conversion as contemplated in the Broad Based Socio-Economic Empowerment Charter for the South African Mining Industry (Mining Charter).

Consent for the transfer of the Mining Rights pursuant to the Lebowa Transaction, has been obtained (5 May 2008) from the DME in terms of Section 11(1) of the MPRDA.

4.3.2 The Mining Charter

Pursuant to Section 100 of the MPRDA, the Mining Charter sets out the framework, targets and timetable for increasing the participation of HDSA's in the mining industry, as well as enhancing the benefits to HDSA's from the exploitation of mineral resources. The Mining Charter is based on seven key principles, five of which are operationally oriented and cover areas focussed on improving conditions for HDSA's. The remaining two principals are focussed on HDSA's ownership targets and beneficiation. Compliance with the Mining Charter, for the purposes of applying for the conversion of old order prospecting or mining rights under the MPRDA, is evaluated using a scorecard system.

The Mining Charter requires that HDSA's acquire 15% ownership of a mining company's South African mining assets within five years, and 26% ownership within ten years. In addition, mining companies are obliged to formulate plans for achieving employment equity at management level with a view to reaching 40% participation by HDSA's in management and 10% participation by women in the mining industry, each within five years. The State will evaluate the company's commitment to the different facets of promoting the objectives of the Mining Charter against the scorecard when considering applications for the conversion of old order mining rights to new order mining rights.

4.3.3 The Mineral Resources and Petroleum Royalty Act

The South African government released a Royalty Bill for comment in March 2003; as well as a revised version of the Royalty Bill in October 2006. In December 2007, a third (and final) draft Bill was released for public comment and parliamentary review.

The third draft of the Royalty Bill confirmed gross sales as the tax base, but took the process of beneficiation into account. This Bill moved away from a dual rates system (differential rates for refined and unrefined minerals) towards an allowance for deductions of beneficiation related expenses. The revised tax base will thus be equal to gross sales less allowable beneficiation related expenses and transport expenses between the seller and buyer of the final product.

The Royalty Bill was enacted in November 2008, as Act No. 28 of 2008, Mineral and Petroleum Resources Royalty Act, 2008. According to the Act, a royalty on the transfer of mineral or petroleum resources would be payable to the South African government from 1 May 2009. In his Budget Speech of 11 February 2009, the Minister of Finance announced that the implementation of the Royalty Act would be delayed to March 2010.

4.4 Issuer's interest

On implementation of the Lebowa Transaction, Anglo Platinum will sell to Anooraq an effective 51% ownership interest in Lebowa and an additional 1% interest in each of the Ga-Phasha, Boikgantsho and Kwanda joint venture projects, resulting in Anooraq holding a controlling interest in each of these projects.

The resultant ownership structure illustrating Anooraq and Anglo Platinum's interests are depicted in the organogram set out in Figure 4.3 below.

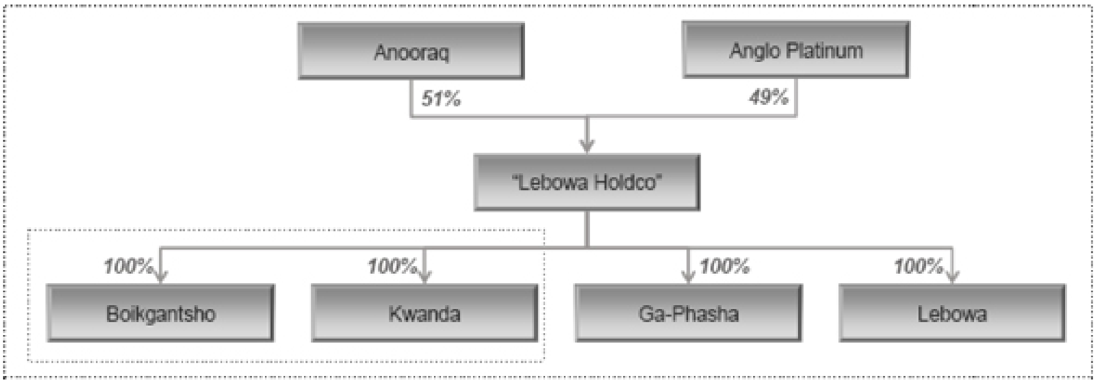


Figure 4.3: Simplified ownership structure after the Lebowa Transaction.

4.5 Royalties, back-in rights, payments, agreements, encumbrances

Previously a mining lease consideration (royalty) was payable on a mining lease area on the Diamand property. This however ceased to be payable on 31 December 1993, by virtue of section 47(1)(d) of the Minerals Act, 1991 as LPM was the holder of the common law rights to minerals.

A mining lease consideration (royalty) is payable on the Middelpunt property until 28 February 2010, where after state royalty will be payable. The property is state land and the mineral rights have historically (i.e. prior to enactment of the MPRDA) belonged to the state while LPM holds the 'old order' mining lease. Additionally pursuant to the cession whereby the mining lease was ceded to LPM, a cession consideration of 10% of the lease profits (equivalent to taxable income) is payable to the cedent which in this case is the state.

This cession consideration will remain payable after conversion to new order rights, and will be payable in addition to the state royalties discussed in section 18.7.2 of this technical report. A rent of ZAR4.20 per hectare per year is payable in respect of surface areas on Middelpunt used by LPM. The cession endures until 2017 and is therefore current.

LPM is currently a wholly-owned subsidiary of Anglo Platinum and as far as Deloitte is aware no third parties have any back-in rights and LPM has no other encumbrances.

4.6 Environmental

4.6.1 Permitting

In accordance with the MPRDA, all EMPRs must be submitted to the DME together with financial guarantees. An approved EMPR certifies that all the legislative requirements at the date when a prospecting or mining right is granted, have been met or adequately provided for, and ongoing compliance will be monitored in terms of the approved EMPR. As and when mining plans are altered, so must the existing EMPR be amended and submitted for consideration and approval. In the case of Lebowa the following approvals have been granted.

An approval under Section 39(1) of the Minerals Act, dated 23 June 1998, exists in respect of an EMPR for the mining licence area in Mining Licence 6/2003 on the Middelpunt, Diamand, Zeekoegat, and Umkoanesstad properties (initially Brakfontein was not included).

An approval under Section 39 of the Minerals Act, dated 21 June 2003, was granted to an amendment to the existing EMPR, concerning a new tailings dam on the Umkoanesstad property.

An approval under Section 39 of the Minerals Act, dated 23 September 2003, was granted to an amendment to the existing EMPR, concerning the UG2 Concentrator.

Finally, an approval under Section 39 of the Minerals Act, dated 26 November 2003, was granted to an amendment to the existing EMPR, concerning the BRK Merensky Project and its associated infrastructure on the Middelpunt, Umkoanesstad and Brakfontein properties. This last approval therefore included Brakfontein in the EMPR, therefore all EMPR amendments have been approved.

As part of the EMPR, LPM is obliged to engage a professional independent consultant to conduct an annual audit of performance against the standards set in the EMPR. As part of this audit, the independent consultant is also obliged to estimate the financial liability for closure and rehabilitation at Lebowa.

4.6.2 Environmental liabilities

Currently the most significant environmental liabilities that have been identified at LPM are dust generation from the tailings dams and seepage of contaminated water from the settling dams.

Tailings dams dust

The consolidated Merensky tailings dam at LPM has been identified as a major source of dust in this relatively arid area. At present, some remedial steps have been undertaken to allay the dust and these include partial vegetation of the slopes of the dam as well as constructing wind-screens on the top of the dam. Both are considered inadequate and in the longer term as legislation becomes stricter it is expected that the slopes and top of the tailings dams will have to be clad with rock and/or adequately vegetated.

Contaminated water run-off and seepage

Initial shallow underground mining at LPM intersected both weathered and fractured overlying aquifers. Therefore, there is an ongoing seepage of ground water into the workings from the Rapholo River. In addition, water from the decant water catchment dam below the tailings dam also seeps into the workings. Total ingress is in the order of 11,000 cubic metres per day.

Subsequently there is on-going pumping of a significant amount of water out of the mines and into surface settling dams.

Until July 2008, LPM was permitted to discharge up to 1.9 million cubic metres of water annually into the Rapholo River, but this permit expired in July 2008 after which time a zero discharge policy has been in effect. In order to comply with this policy, LPM was obliged to construct and commission a lined catchment dam which will both reduce seepage into the aquifers and ingress into the workings which will minimise LPM's water discharge into the river.

Deloitte has been advised that SRK Consulting has been appointed as the design engineer for purposes of designing the lined water-catchment dams and that funds were provided in the 2008 budget.

4.6.3 Rehabilitation Provision

Environmental liability provisioning in the South African mining industry is a requirement of the MPRDA and must be agreed upon by the relevant regulatory authorities (mainly DME and the Department of Water Affairs and Forestry (DWAF)). Annual contributions are made to the Platinum Producers Environment Trust, an environmental trust fund, which provides for the estimated costs of pollution control and rehabilitation.

The South African Revenue Service approves such annual contributions to the trust fund and requires that these contributions be estimated on the basis of the remaining liability over the remaining LOM.

If contributions to the trust fund in respect of a mining project are deficient at the start of the project, a mining company is obliged to lodge guarantees with the DME, or alternative measures such as insurance policies may be agreed between the mining company and the DME.

In 2008, Anglo Platinum stated the LOM closure cost of Lebowa to be ZAR63.139 M. The estimate for premature closure is ZAR92.205 M. LPM's contribution to the trust fund as of 31 December 2008 was approximately ZAR15.176 M. The shortfall between the estimate for premature closure and the balance in the trust fund (ZAR77.029 M) is currently provided for by bank guarantees.

4.7 Permits

Surface rights

The surface overlying the LPM orebodies is owned by the State, and tenure to the required areas is currently held through various surface right permits (SRPs) in terms of Section 90 of the Mining Rights Act of 1967, and lease agreements. Pursuant to Item 9 in Schedule II to the MPRDA, such SRPs will remain in force and attach to converted mining rights. Such SRPs have been re-registered in accordance with the requirements of Item 9.

Reconciliation is currently underway to ensure that LPM has valid SRPs over all surface areas used by Lebowa. Deloitte has been advised by Anglo Platinum that no discrepancies are expected.

During 2005, as part of the BRK Merensky Project and MPH UG2 Project, the process of rationalising, consolidating and re-applying to the Department of Land Affairs for revised and new SRPs, which include areas for the expansion projects, reached an advanced stage. Anglo Platinum states that LPM has currently received its SRPs, with the exception of the Umkoanesstad SRP. Turnaround time at the Department of Land Affairs cannot be estimated. However, there is negligible risk of the SRP not being approved, provided the surface use is in accordance with the approved EMPR and the statutory obligations referred to in Sections 37 to 46 of the MPRDA.

These surface rights and mining licences, as discussed above and depicted in Figure 4.4, are adequate for the mining that is planned.

4.8 Surface structures

In addition to the various mine shafts, LPM's surface structures include (Figure 4.4):

- The mine buildings including: offices, change-houses, hostel facilities
- Workshops, compressor houses and stores
- Concentrators
- Tailings dams and waste rock dumps

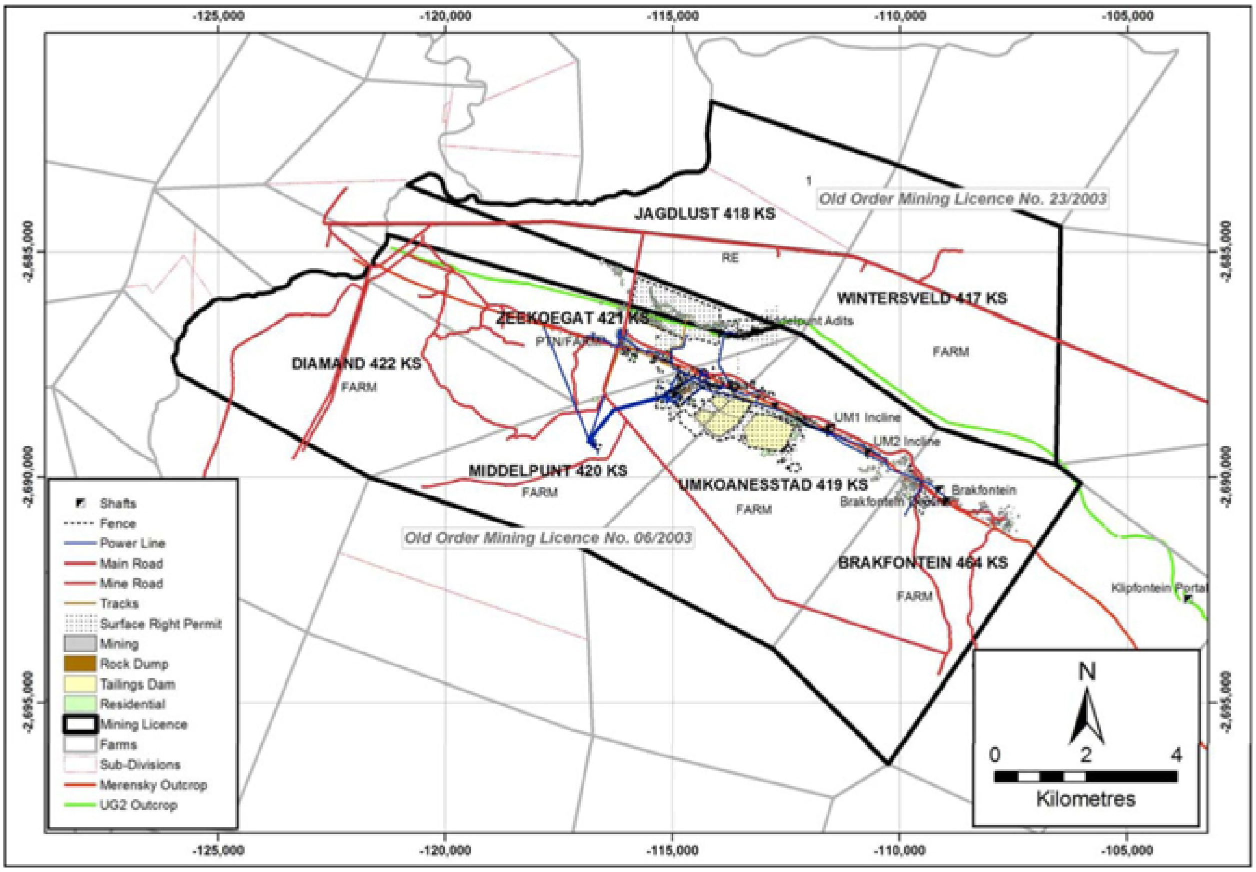


Figure 4.4: Lebowa Platinum Mine surface infrastructure plan.

5 Accessibility, climate, local resources, infrastructure and physiography

5.1 Topography, elevation and vegetation

Lebowa Platinum Mine is located on an undulating plain between a range of hills to the north and a range of low mountains to the south. The plain is bisected by the Rapholo River, a major river in the area, which joins the Olifants River further downstream (Figure 4.1).

The average altitude of the plain is 800 metres above mean sea-level (mamsl) and the average altitude of the adjacent mountains is 1,600 mamsl.

The plain, hills and mountains are sparsely vegetated with grasses, shrubs and occasional small trees with stunted growth. The vegetation is a result of both the arid climate and over-grazing by cattle and sheep.

There is some subsistence agriculture in the adjacent areas which is limited to small family-farmed maize fields.

There are notable expanses of bare soil on the surrounding properties and erosion is evident along water-courses in the area.

5.2 Access

Lebowa Platinum Mine is accessed from the R37 provincial all-weather road, which runs between Polokwane, the capital of Limpopo Province, and Lydenburg, a town to the south-east in the neighbouring province of Mpumalanga.

The nearest railway stations are at Polokwane and Steelpoort 80 km and 100 km away, respectively. However, rail is not the preferred means of transport and all stores and equipment are delivered by road-truck to LPM.

The nearest commercial (domestic) airport is at Polokwane, but LPM does have a private heliport used by visitors, which is also available for emergency evacuation if required.

5.3 Proximity to population centre and transport

The nearest large town is Polokwane, which is a modern and developing town providing housing, schooling, health care, shopping, commercial and government administrative facilities.

Many of LPM employees reside in Polokwane in company-owned or privately-owned suburban housing and commute to LPM by company bus or by private vehicle. The remaining employees are housed in a mine residential ‘village’ at LPM, while some (junior) staff, who are indigenous to the area, reside in local private dwellings in the surrounding rural area.

5.4 Climate and length of operating season

The Sekhukhuneland District of the Limpopo Province has a typical arid, temperate Southern African climate.

In the summer (September/October to March/April), day-time temperatures can reach the mid to high 30°C, cooling to slightly below 20°C overnight. Rainfall occurs between November and March and annually can be between 300 millimetres (mm) and 500 mm.

Winter temperatures can be below 10°C overnight but warming to the mid-20s in the daytime.

Winter is characterised by clear skies, and summer by clear skies with isolated clouds. In both cases the vast majority (70% or greater) of days can be classified as ‘sunny’.

Extreme weather conditions occur only a few times a year and can include mist, high wind with dust, thunderstorms and occasional hail.

The mine operates twelve months per year and is not affected by climate and weather.

5.5 Infrastructure

5.5.1 Power

Lebowa Platinum Mine’s electricity requirements are provided directly from Eskom, the South African national utility.

Electricity is distributed from the dedicated on-site ‘Middelpunt’ Eskom sub-station, which is fed with 133 MVA from two separate 33 kVA high-tension power-lines. The Middelpunt bulk intake sub-station is equipped to provide 60 MVA firm or 80 MVA non-firm supply. Currently LPM draws less than 60 MVA.

Bulk electricity reticulation from the Middelpunt sub-station to the two primary Lebowa sub-stations ‘Atok Main sub-station’ and ‘Compressor sub-station’ is at 22 kV while distribution to various installations including the Middelpunt Hill and Brakfontein substations is at 6.6 kV.

In most cases, there is a ring-feed connection between the two primary sub-stations and the various installations.

The installed capacity of 60 MVA is sufficient for the current workings at LPM plus the two approved expansion projects, the MPH UG2 Project and the BRK Merensky Project.

Regionally, Eskom has strengthened their grid by building the Lesideng Main Transmission Station which feeds the Witkop-Merensky 400 kV ring which provides a firm 500 MVA to the region. This is expected to enhance electricity supply in the longer term.

Historic Eskom supply

Until recently the ‘quality’ of electricity supply has been described as very good. However, in the second half of 2007 the mine has suffered from some power-dips and power failures which have had negligible impact on the operation.

Current Eskom supply and shut-downs

In January 2008, the mismatch between installed, available, electrical generation capacity and rapidly growing demand in South Africa came to a head, resulting in rolling blackouts and a temporary shutdown of the mining industry. After the shutdown, electricity supply to the mining industry was curtailed to 90% of historic demand.

The recent changes in the global financial markets have led to a drastic drop in growth, and have effectively alleviated the electricity shortage in the short term.

Regarding the recent electricity supply constraint in South Africa, Anglo Platinum has advised as follows:

Lebowa is particularly well positioned over the six year period to 2014.

The average draw against installed capacity in the Eskom reference period (Sept '06-Oct '07) was approximately 36 MVA. Projects with Eskom approved supply (contract in place and confirmed) included in the original Lebowa long term plan '08 (mining and concentrating) have an additional requirement of 19 MVA to a total of 55MVA by 2014.

Applications for power supply for projects beyond this time period have not been submitted.

Lebowa is an integral part of the Anglo Platinum energy efficiency initiative. As such it is part of a demand side management programme encompassing aspects of compressed air load clipping and lighting energy efficiency projects. Further opportunity exists for improvements in energy efficiency associated with ventilation system optimisation, compressed air utilisation and pumping. Significantly the Brakfontein Merensky section is planned on the basis of electric rock-drilling.

Standby power

LPM does have standby generating capacity (120 kVA) able to support some, but not all, critical operations. Notably shaft winders and hoisting can be supported, however ventilation fans and the concentrators cannot be supported.

Anglo Platinum has decided to increase the generator capacity of Lebowa Platinum Mine, and LPM has budgeted accordingly.

5.5.2 Water

Lebowa Platinum Mine is currently self-sufficient in the supply of industrial and potable water.

Potable water is drawn from a well-field on the Jagdlust property adjacent to the Olifants River which bounds the mining lease area on the north-west. This extraction is limited to 787,000 cubic metres per annum in accordance with the DWAF permit.

Potable water is circulated via booster pumps into dams and then by gravity to all the points of consumption including the shafts, compressors, offices, village and single quarters accommodation.

The well-field has never run dry. However, permanent supply cannot be guaranteed and the existing water use permit is subject to review by the DWAF from time to time.

Industrial water is drawn from numerous used process-water settling dams adjacent to the process plants and is recycled for use underground, in the concentrators and on surface.

There is ingress of groundwater into the underground workings, estimated at nearly 11,000 cubic metres per day (cf. Section 4.6.2). This is pumped to surface and stored in the surface contaminated process-water settling dams mentioned above.

Due to the water cycle described above, there is currently no shortage of available industrial water, even though LPM is situated in an arid region. The available potable water is however limited and there could be a shortage in the future due to the planned expansions at LPM. These potential water shortages have been considered and addressed on a regional scale by Anglo Platinum, together with other affected mining and exploration companies, who have developed a Northern and Eastern Limb water scheme, which includes the raising of the Flag Boshielo Dam wall, the construction of the De Hoop Dam on the Steelpoort River, the construction of the Richmond Dam and a well-field on the Twickenham property. This all forms part of the long-term strategy to ensure adequate water supplies to all of the various Northern and Eastern Limb mines, including LPM.

5.5.3 Compressed air supply

Compressed air is used primarily for pneumatic rock-drilling. Other minor uses include localised pumping in the stopes and in development ends. Compressed air is reticulated through steel columns on surface to the various points of consumption at LPM.

The main compressor station with four units is located adjacent to the existing Vertical Shaft. Another compressor station with one unit is located at the UM2 Inclined Shaft. Total installed capacity is 90,000 cubic feet per minute which is adequate for the current workings. The BRK Merensky Project has been planned to utilise electric drilling and will not require compressed air for drilling.

5.5.4 Concentrators

There are two concentrators on Lebowa Platinum Mine, namely the Merensky Concentrator commissioned in 1991 and the UG2 Concentrator commissioned in 2003. They are located alongside each other, adjacent to the Vertical Shaft (Figure 4.4).

The concentrators are discussed in detail in Section 16 of this technical report.

5.5.5 Tailings dams

There are two tailings dams at LPM, namely the Merensky tailings dam and the UG2 tailings dam. These are located near the Merensky and UG2 Concentrators, respectively.

The Merensky tailings dam has an area of approximately 70 ha and is known as the Consolidated Tailings Dam, being five previous dams, combined into one facility. The UG2 tailings dam is relatively new (commissioned in 2003) and is known as Dam No. 6, with an area covering approximately 63 ha.

Additional tailings dam areas are discussed in Section 16.

5.5.6 Waste rock dumps

Waste rock dumps are located adjacent to the various shafts to accommodate the broken waste rock hoisted from underground. The inert waste rock is used for construction and in future may be used to clad the slopes of the tailings dams.

5.6 Personnel

5.6.1 Organisational structure and complement

LPM's organisational structure is similar to other South African mines, whereby production is divided into the departments of Mining and Engineering and services are provided through Technical Services, Administration/Finance and Safety/Training. In each instance, a Head of Department reports to the General Manager.

As required by South African statute, various persons are legally appointed to their positions, including the Mine Manager and his immediate sub-ordinates as well as the Engineering Manager and his sub-ordinates. Appointed managers are obliged to ensure that the mining activities are carried out according to the Minerals Act regulations and/or codes of practice/standard procedures drafted and adopted by LPM.

LPM employed, at the end of 2008, a total of 3,718 people across all disciplines and in all categories, as shown in Table 5.1. This labour complement includes all personnel necessary for the current operations.

Table 5.1: Lebowa Platinum Mine employee complement as at December 2008.

Patterson Grade	Description	Plan	Actual	Variance
A	Semi-skilled; general worker	236	297	61
B	Skilled; artisan, miner	2 865	2 928	63
C	Supervisor; foreman, shift boss	302	390	88
D	Middle-management	62	93	31
E	Senior management	10	10	0
Total		**3 475**	**3 718**	**243**

The BRK Merensky Project is managed by an Anglo Platinum team and is staffed by Capital Development Corporation, a dedicated Anglo Platinum project development company. When production commences at the BRK Merensky Project, LPM personnel will be deployed. The MPH UG2 Project will be similarly staffed.

5.6.2 Employment policy

LPM's employment policy is to include all core skills from rock-face to manager as permanent 'payroll' employees. Until recently an outsourced labour model was in place on the Middelpunt Hill UG2 adits. This policy was however changed in order to overcome various labour disputes.

LPM does, however, continue to employ contractors in certain non-core activities such as change-house and lamp-room management.

5.6.3 Skills shortage and development

LPM suffers from the industry-wide skills shortage, particularly in the mining, processing and engineering disciplines. Currently LPM subscribes to Anglo Platinum's Retention Policy which includes various initiatives to retain employees and scarce skills, including retention allowances. The current critical skills shortage is summarised in Table 5.2.

Table 5.2: LPM critical skills shortages as at December 2008.

Roles	Complement	Actual	Shortage	Percentage
Shift supervisor	54	53	1	2
Miner	127	99	28	22
Mine overseer	10	7	3	30
Boilermaker	18	23	-5	-28
Rigger	4	3	1	25
Fitter	20	22	-2	-10
Electrician	33	30	3	9
Section Engineer	4	4	0	0
Total	**270**	**241**	**29**	**11**

The skills shortage is also being addressed through training and development and LPM enjoys the benefit of Anglo Platinum's skills capacity pool, its Learnership and Candidateship programmes for miners and mining equipment operators and the bursary programme for knowledge workers.

Deloitte proposes that attention be given to addressing skills shortages into the future.

5.6.4 Skills training

LPM currently enjoys the benefit of various Anglo Platinum training facilities, such as:

- The on-mine training centre including class-room facilities, and occupational skill development facilities including the surface workshops and the underground training stope.
- The Anglo Development Centre in Rustenburg for management, supervisory and leadership training.
- The Klipfontein Development Centre for vocational training of miners.
- The Waterval Training Centre near Rustenburg for training of trackless mechanised mobile equipment operators.
- The Engineering Skills Training Centre in Rustenburg, for artisan and artisan assistant training.

It is understood by Deloitte that these facilities will remain available to LPM after the Lebowa Transaction.

Deloitte also notes that LPM currently has an adult basic education training programme and an HIV/AIDS prevention programme in operation.

6 History

6.1 Prior ownership and ownership changes

Mining operations at Lebowa Platinum Mine began in 1969, when the mine was known as Atok Platinum Mine (Atok). Mining was initiated on the Middelpunt and Zeekoegat properties by Anglovaal, a traditional South African mining house, and OK Bazaars, a South African chain-store.

Expansion of Atok mine was constrained by the ownership of neighbouring properties, primarily Umkoanesstad, which was then owned by RPM. In 1970, Atok was sold to RPM, then a subsidiary of JCI Limited (JCI), another South African mining house in which Anglo American held a significant equity interest. In the mid-1990s, JCI was ‘unbundled’ and its platinum interests were listed separately as Lebowa Platinum Mines Ltd, which was later merged with Anglo Platinum’s other mines to become Anglo Platinum Limited.

Until recently Lebowa was owned by Lebowa Platinum Mines LTD, a wholly-owned subsidiary of Anglo Platinum. In order to effect the Lebowa Transaction, Anglo Platinum has implemented an internal corporate restructuring pursuant to which LPM’s operations were transferred to LPM HoldCo.

6.2 Previous exploration and development work

Over the last 25 to 30 years, JCI/RPM expanded their platinum mining activities primarily across the western limb of the Bushveld Complex, favouring the higher-grade and more platinum-rich Merensky horizon.

On the eastern limb, LPM remained the only modest producer (from the Merensky) of PGMs, together with the Potgietersrust Platinum opencast operations, which mines from the northern limb of the Bushveld Complex.

JCI/RPM had, however, secured significant mineral rights over properties held in trust by the State for various indigenous tribes on the Eastern Limb and exploration was progressed in order to determine the geological extent and characteristics of the UG2 and Merensky mineralisation.

Significant interest in the Eastern Limb was initiated in 1999 and was driven by the increasing demand (and robust forecast of demand) for PGMs.

Whereas previously LPM operations simply replaced worked out areas by expanding on strike, after Anglo Platinum’s strategic 2000/2001 decision to increase total production to 3.5 Moz, the BRK Merensky Project and MPH UG2 Project were initiated and subsequently approved.

6.3 Historical mineral resource and mineral reserve estimates

The Mineral Resources and Mineral Reserves discussed in this document are the latest available estimates for LPM. There is no reliance or reference to historical Mineral Resource and Mineral Reserve estimates in this TR.

6.4 Production history

The production history, as provided by Anglo Platinum, is summarised in Table 6.1

Table 6.1: Production summary for 2005 to 2008 (Source Anglo Platinum).

	F2005	F2006	F2007	F2008
Stoping units (000 sq. m)	474	435	369	266
Stoping width (m)	0.92	0.95	0.96	0.88
4E in-situ grade	5.41	5.44	5.44	5.42
Total Ore tonnes to concentrator (000's)	1 606	1 693	1 342	1 111
UG2 as a percentage of total Ore to concentrator	42.60%	42.30%	38.39%	41.56%
Total tonnes milled	1 609	1 653	1 333	1 098
UG2 as a percentage of total tonnes milled	43%	43%	38%	40%
Recovery	90%	90%	91%	
Pt oz's in metal and concentrate	114 396	114 329	105 353	75 324
4E oz's in metal and concentrate	216 757	216 291	178 083	142 039

7 Geological setting

7.1 Regional geology

The Bushveld Complex (BC) is situated in the northern half of South Africa and exists as an ellipse-shaped body consisting of five lobes (Figure 7.1). The BC is the world's largest known ultramafic igneous intrusion, and extends approximately 450 km east to west and approximately 250 km north to south. It occupies parts of Limpopo Province, North-West Province, Gauteng Province and the Mpumalanga Province. It is estimated to have been formed approximately 2,000 million years ago (Ma). The BC is host to PGM mineralisation in addition to chrome, vanadium, nickel and copper.

The five lobes are referred to as the Western, Eastern, Northern (includes both the Potgietersrus and Villa Nora compartments), South-Eastern, and Far-Western areas. The South-Eastern BC is completely covered by sedimentary successions of the Karoo Supergroup (Figure 7.1), while the remaining four lobes are variably exposed with some areas under extensive soil cover.

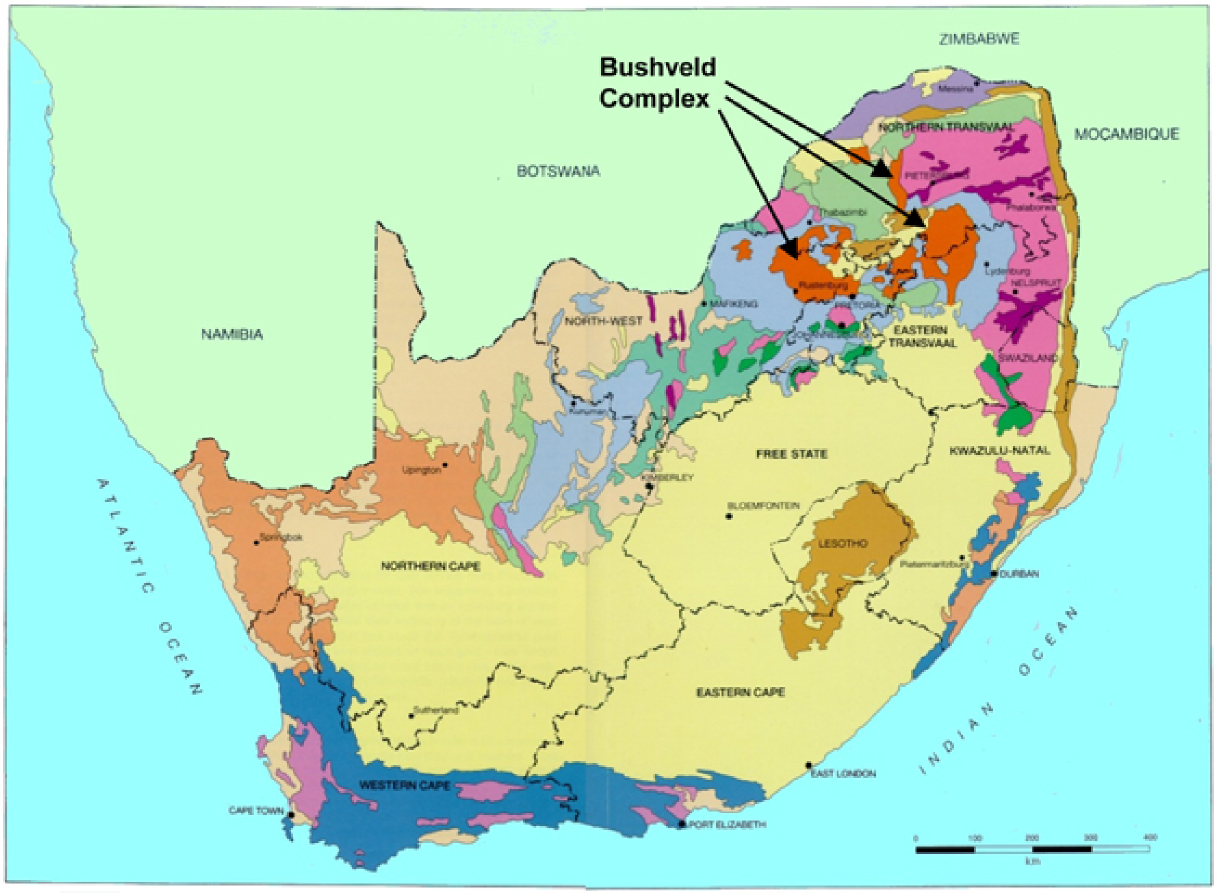


Figure 7.1: Geological map of South Africa, also indicating the location of the Bushveld Complex.

The Bushveld Complex contains the largest known example of A-type granite plutonism, the Lebowa Granite Suite (Kleeman, 1985; Kleeman and Twist, 1989), the largest known accumulation of siliceous volcanism, the Rooiberg Group (Figure 7.2; Twist and French, 1983), and the largest intrusions of layered mafic rocks, the Rustenburg Layered Suite (von Gruenewaldt and Harmer, 1992).

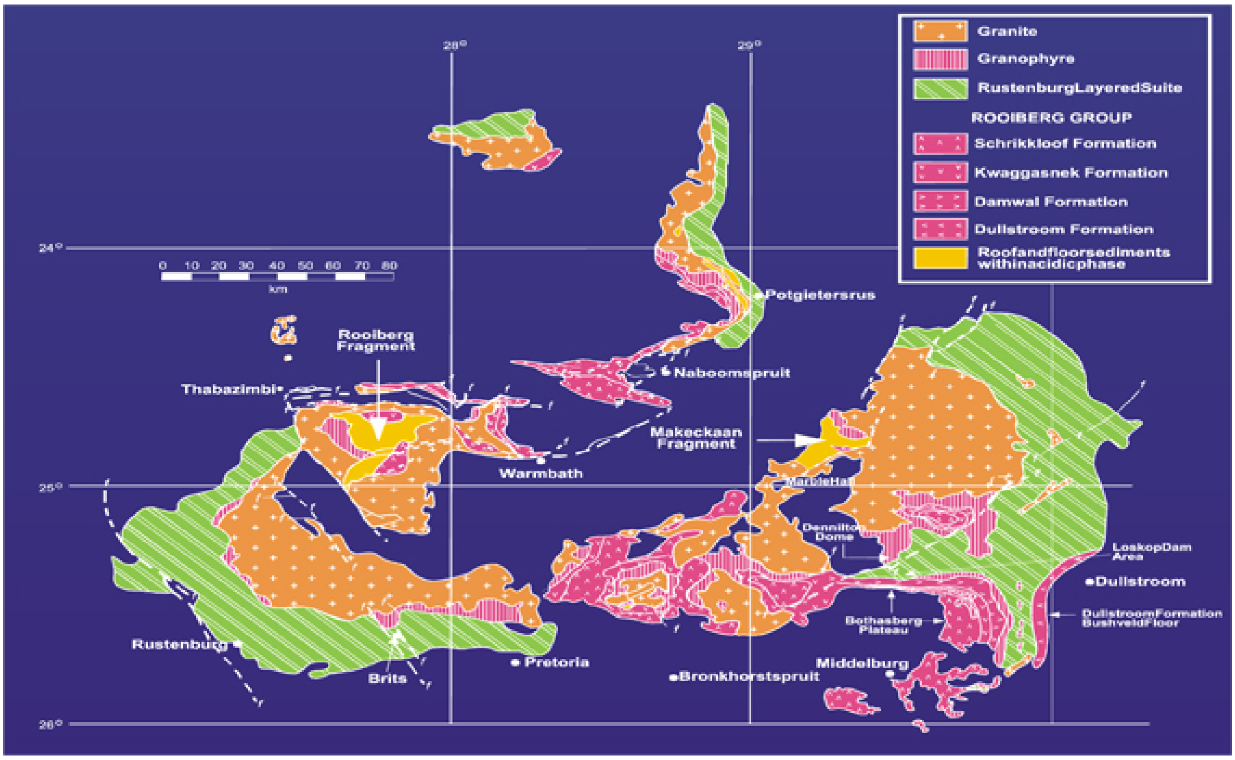


Figure 7.2: Geographic relationship in between the granites, volcanic rocks, granophyres and the mafic rocks of the Rustenburg Layered Suite.

The Rustenburg Layered Suite (RLS) consists of mafic rocks (relatively low SiO_2 contents). It is preserved in the Western-, Eastern-, and Northern Limbs (also termed the Potgietersrus Limb) of the Bushveld Complex. Rocks comprising the RLS are subdivided into several zones, termed, from base to top, Marginal, Critical, Main, and Upper (SACS, 1980; Figure 7.3). It is generally accepted that the strata of the RLS is a result of the injection of several magma pulses at relatively shallow (< 8km) crustal levels (e.g. Harmer and Sharpe, 1985; Hatton and Sharpe, 1989; Hatton, 1989; Kruger, 1994).

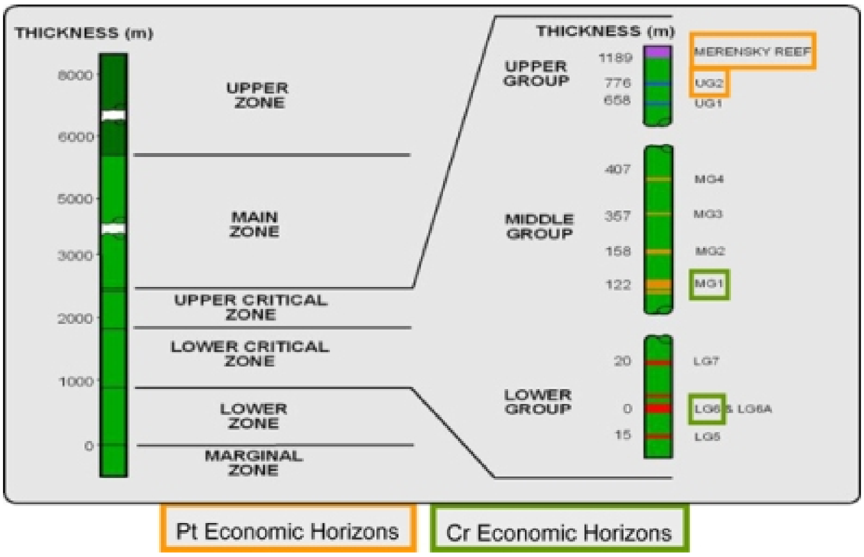
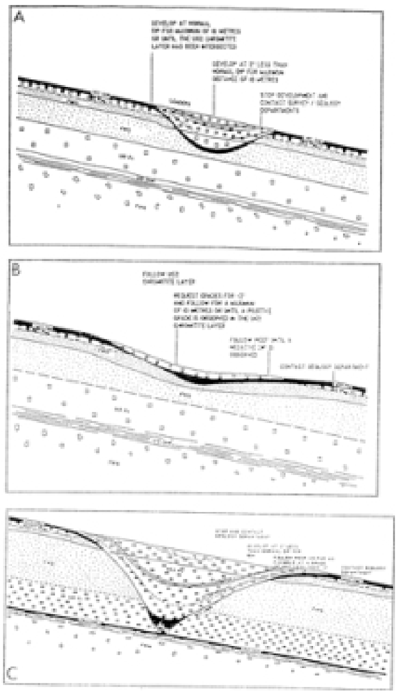


Figure 7.3: The various zones of the RLS. Shown in more detail are the stratigraphic positions of the Lower, Middle and Upper Group chromitite layers of the Critical Zone.



The Lower and Middle Critical Zones contain major chromitite deposits, which are predominantly the LG6, MG1 and MG2 chromitites (Figure 7.3). Platinum Group Elements (PGE's) are derived from the UG2 and the Merensky Reef, chromitite and pyroxenite, respectively. These deposits are located in the Upper Critical Zone (Figure 7.3). Several magnetite layers are located in the Upper Zone, and these are also mined for their vanadium content.

Geological disruptions encountered during mining of the various chromitite layers and the Merensky Reef include dykes, faults, joints, pegmatoids, domes, and potholes. The latter, particularly, can cause major disruptions to mining (Figures 7.4 and 7.5).

Figure 7.4: Type 1, 2 and 3 chromitite potholes as associated with the UG2 chromitite layer (after Hahn and Ovendale, 1994). Potholes straddle different footwall lithologies.

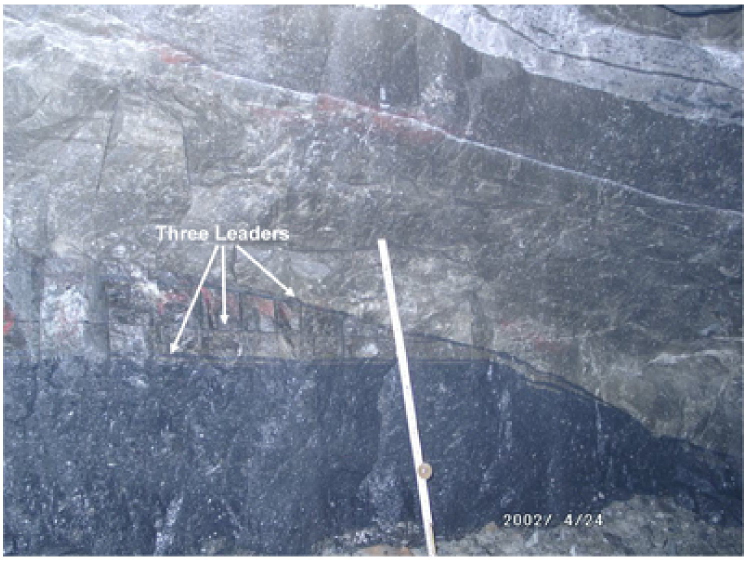


Figure 7.5: Lithological variations as observed at a pothole edge as associated with the UG2. Note the deteriorating middling in between the UG2 and the Three Leaders (or Triplets; white arrows) chromitite bands (photo courtesy of M. Roberts and B. Watson: taken at Middelpunt Hill).

7.2 Geology of the Lebowa area

Lebowa Platinum Mine is located on the northern extremity of the Eastern Limb of the BC (Figures 4.1 and 7.6).

PGM mineralisation is specifically located within the UG2 and Merensky horizons, which form part of the Upper Critical Zone of the Rustenburg Layered Suite (Figures 7.3, 7.6 and 7.7). Both horizons subcrop and in some instances outcrop in the project area along a 16.5 km strike length. The BC layering dips from northeast to southwest at approximately 25º in the northwestern areas (Zeekoegat), and gradually decreases to approximately 18º in the southeastern area (Brakfontein). The general structural geology is characterised by northeast and east trending dykes and faults with associated conjugated joint sets (Figure 7.6).

The mining area is located within the farms Zeekoegat, Middelpunt, Umkoanesstad and Brakfontein (Figure 7.8). The northeastern portion of the mining area is located below a range of pyroxenite hills and the southwestern portion is below the valley floor, overlain by black turf.

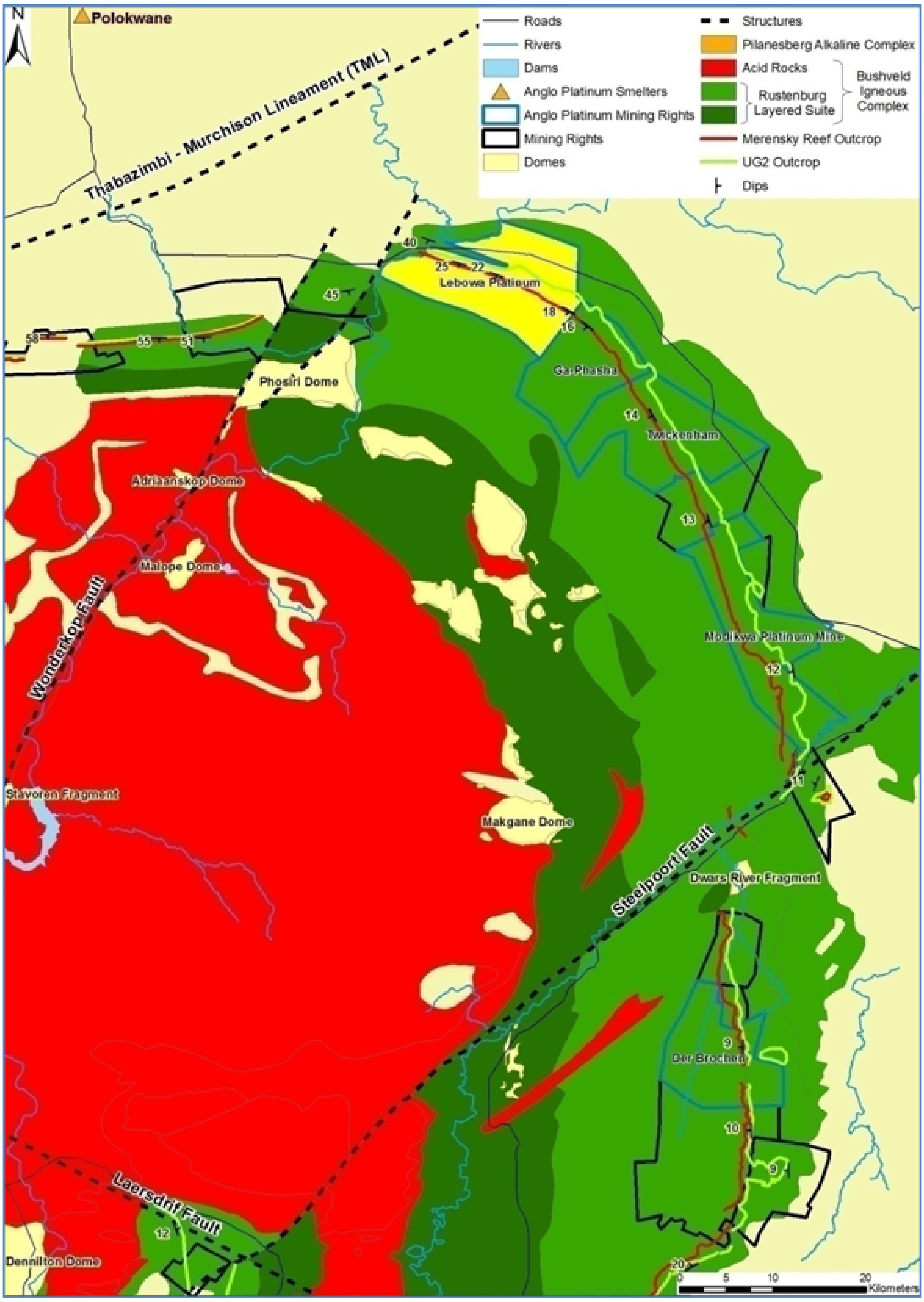


Figure 7.6: Regional Geological Setting - Eastern Limb of the Bushveld Complex.

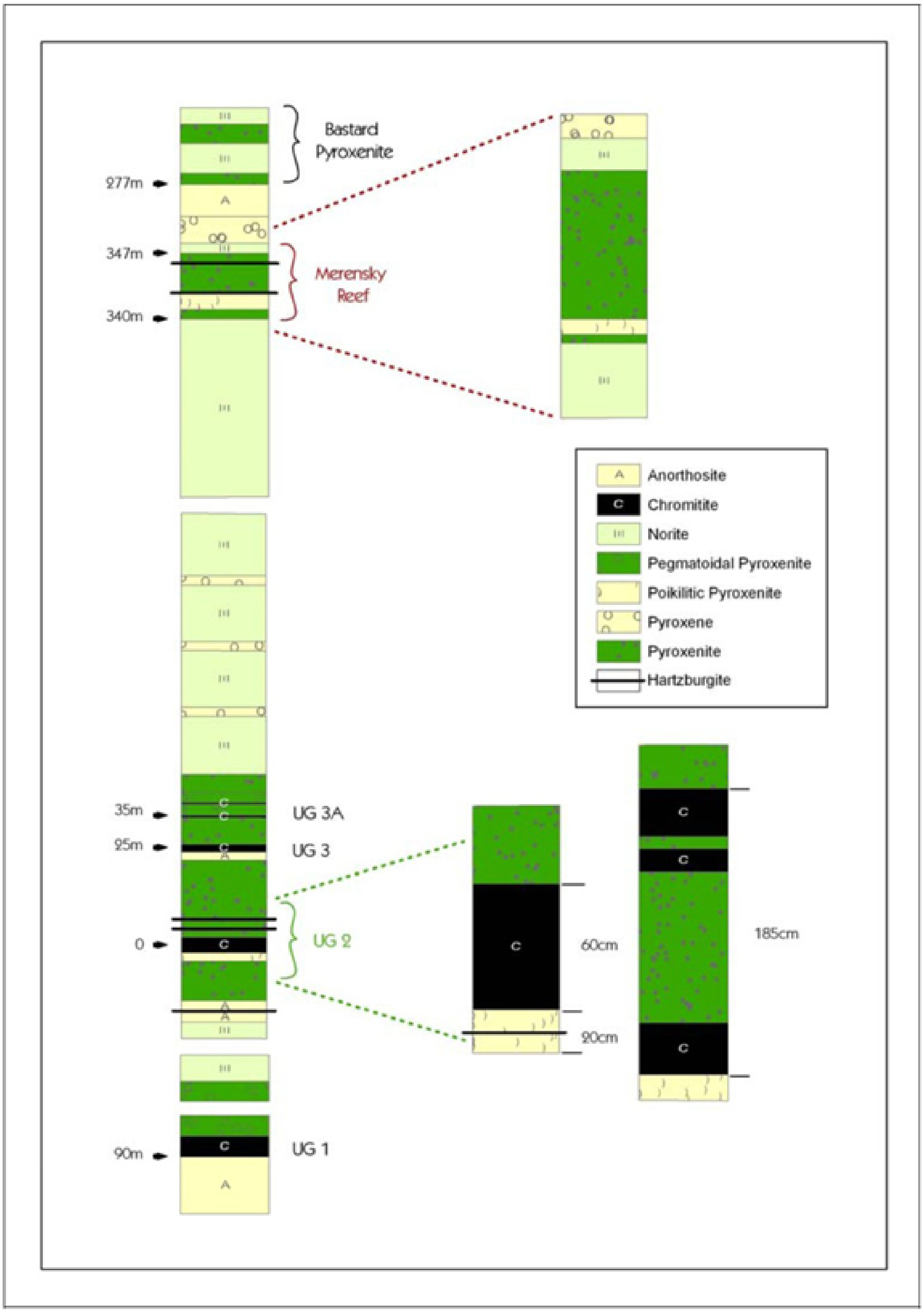


Figure 7.7: Typical stratigraphic column for Lebowa Platinum Mine.

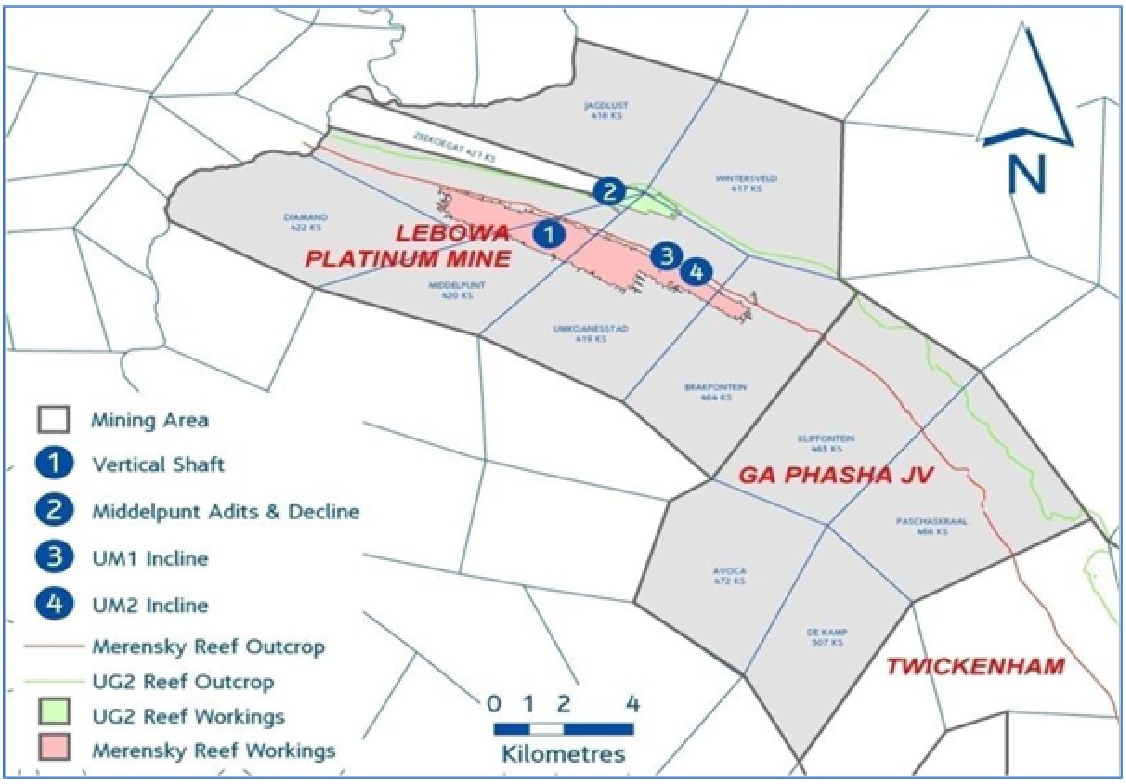


Figure 7.8: Lebowa Platinum Mine mining area and shafts (Anglo Platinum, 2009).

7.3 UG2 and Merensky characteristics

The **UG2** is stratigraphically approximately 350 m below the Merensky and is separated from it by a series of well layered sequences (Figures 7.3 and 7.7). The UG2 is comprised mainly of a well defined chromitite layer together with minor hangingwall and or footwall constituents. The average width of the UG2 is 70 cm. It is overlain by a medium-grained poikilitic feldspathic pyroxenite that averages 9.85 m in width, and hosts a variable number (generally up to 4) of very thin chromitite layers. The position of these stringers is important to the mining of the UG2. The UG2 is underlain by a pegmatoidal feldspathic pyroxenite layer of approximately 0.75 m in width which is commonly host to disseminated chromite and some base metal sulphide occurrences within close proximity to the UG2.

The UG2 elevation isopachs indicate a relatively undisturbed tabular and gently dipping layer. UG2 widths generally increase to the northwest from an average of 67 cm on Umkoanesstad to 74 cm on Zeekoegat. There is no evidence of severe undulations to this layer that would adversely affect the planned mining method. Severe undulations of the UG2 are known to hamper mining by increasing dilution and off-reef mining.

The **Merensky** is a feldspathic pyroxenite reef horizon and is stratigraphically approximately 350 m above the UG2, near the top of the Upper Critical Zone (Figures 7.3 and 7.7).

The Merensky is located below the three to six metre thick Merensky Pyroxenite layer and above the Merensky Norite layers. Two thin chromitite stringers are discontinuously developed with the upper stringer positioned 20 cm to 25 cm from the Merensky Pyroxenite hangingwall contact, and the lower stringer located on or just above the Merensky Pyroxenite's basal contact. In the absence of a consistently developed chromitite stringer, the upper contact of the Merensky Pyroxenite layer assists to define the top position of the Merensky horizon and is a guide for sampling purposes and on-reef mining.

The Merensky footwall stratigraphic sequence has a sharp footwall contact, usually marked by the lower chromitite stringer. While the top contact tends to be planar, the basal contact is undulating as a result of thermo-chemical erosion of the more mafic Merensky lithologies with their underlying lithologies. This contact is often associated with a thin anorthosite layer (approximately 3 cm thick) that probably formed as a secondary reaction product of thermal erosion.

The Merensky hangingwall stratigraphic sequence is typified by medium to coarse grained feldspathic lithologies, ranging in composition from mela-norites to anorthosites.

7.3.1 Potholed UG2

Potholes are magmatic disturbances of the reef plane and are generally deep eroded depressions that have serious structural implications in respect of reef continuity.

The UG2 is known to be affected by potholes (Figure 7.9). UG2 potholes typically have a "soup-bowl" profile. The characteristics of undisturbed UG2 are not preserved in the LPM potholes and the succession often occurs as a variably thickened feldspathic pyroxenite package, containing disrupted and discontinuous chromitite layers. ***As a result, grades within potholes are highly erratic and, invariably, sub-economic.*** UG2 potholes at LPM are commonly destructive and are not economically mineable. Based on current and historic drilling and mining data, geological pothole losses for the UG2 are estimated at 9% of the estimated total geological loss of 15% for the Lebowa UG2.

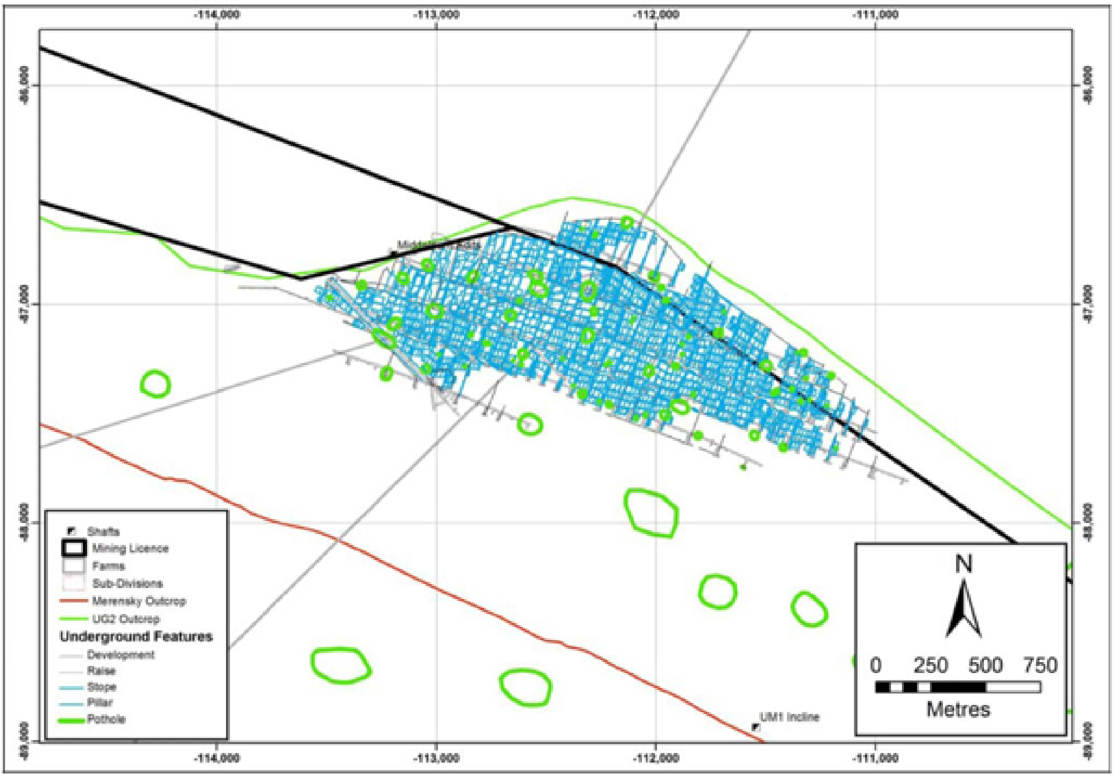


Figure 7.9: Middelpunt Hill UG2 showing pothole distribution (green outlines).

7.3.2 Bifurcated UG2

The UG2 occasionally exceeds a thickness of 95 cm as a direct result of internal xenoliths comprising anorthosite, feldspathic pyroxenite and norite. Although the total chromitite thickness is not affected, *the width between the upper and lower UG2 contacts can exceed 2 m in places.* This causes considerable mining difficulties, due to the unpredictable changes in the UG2 elevation, deteriorating ground conditions and dilution. The current borehole spacing of between 300 m to 500 m is inadequate to accurately delineate these features. Current information suggests that approximately 6.5% of the UG2 resource will contain bifurcated reef, the mining of which will only recover a portion of the reef, and thus a reduced portion of the total metal content (Figure 7.10).

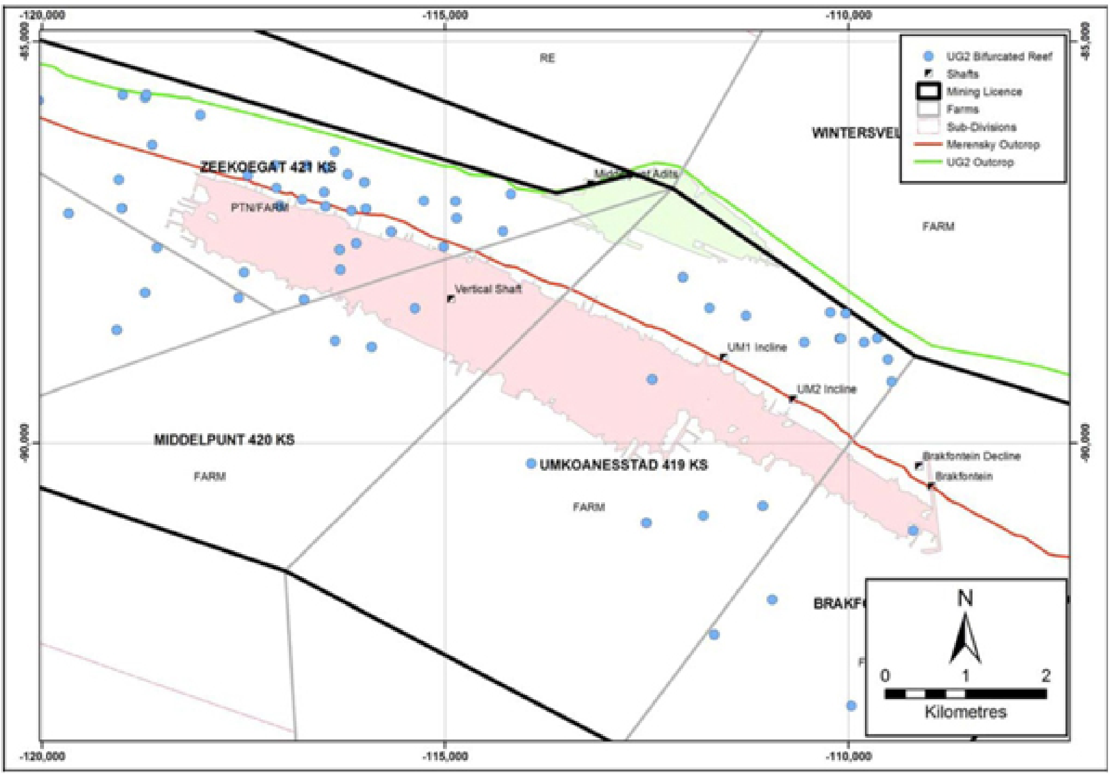


Figure 7.10: Distribution of boreholes that have intersected bifurcated UG2 (shaded blue).

7.3.3 Potholed Merensky

Merensky potholes, including those at LPM, have been well documented (Figure 7.11). Current indications are that potholes account for approximately 16% of the estimated total geological loss of 20%. An additional 5% geological loss has been allowed for the BRK Merensky Project due to the regional pothole area. ***These potholes constitute unmined areas for the Merensky at LPM.***

A regional pothole was discovered in the Brakfontein area which has been divided into two components, a ‘destructive pothole area’ where no recognisable Merensky is preserved and a ‘non-destructive pothole area’ where the Merensky is partially preserved. At this stage the regional pothole areas are excluded from the Mineral Resource due to risks associated with their stability to ensure successful mining.

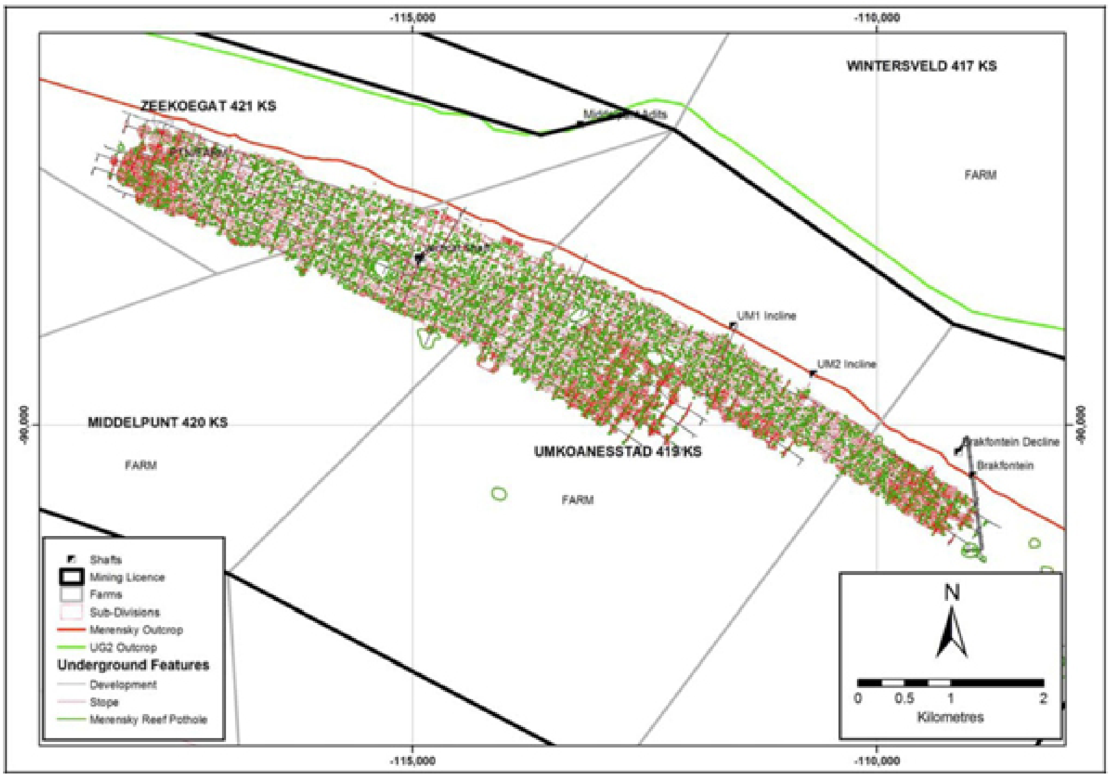


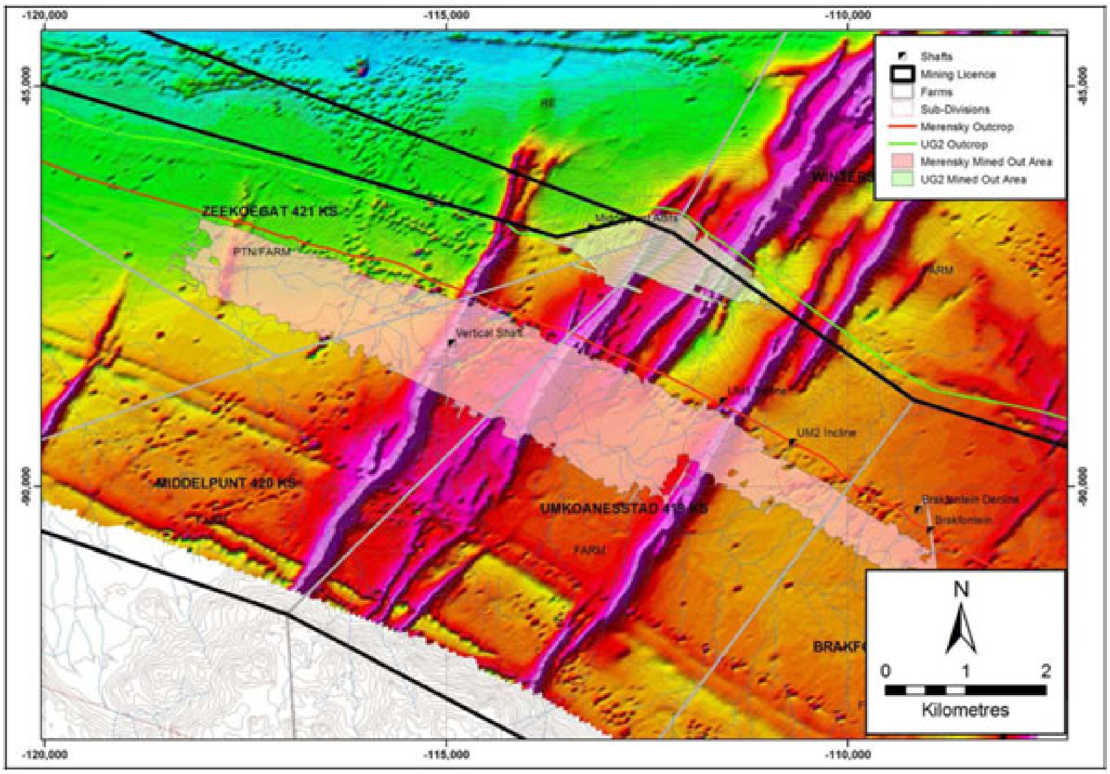
Figure 7.11: Merensky Reef pothole distribution (shaded green).

7.4 Weathering and oxidation

The weathered overburden (soil and calcrete) depth across LPM is highly variable ranging from no overburden in the rocky outcrops and hill areas, to in excess of 50 m in the valley areas. The average overburden depths below surface are Zeekoegat 10 m, Middelpunt 22 m, Umkoanesstad (valley) 30 m, Umkoanesstad (mountain) 2 m and Brakfontein 40 m. The depth of oxidation may be reasonably estimated by adding 25 m to the overburden depth.

7.5 Joints and faults

The geological structure at Lebowa Platinum Mine is not complicated by faulting. According to existing workings, minor faulting is expected to occur, and would consist of dextral and sinistral strike-slip faults, normal and reverse dip-slip faults and faults with more complex combinations of these components. Displacements are expected to be small, at generally ≤ 1 m. Major conjugate joint set orientations were measured from strong macro-lineament features evident from an aeromagnetic survey image (Figure 7.12) and land satellite imagery which provided orientations in the order of 99° and 159°. Joint sets may result in poor ground conditions for mining but are not considered a geological loss.



Purple lineaments indicate the location of intrusives

Figure 7.12: Graphic representation of the aeromagnetic survey.

7.6 Dykes

An airborne magnetic survey has successfully identified three to four swarms of northeast striking dolerite dykes (Figure 7.12). Post-mineralisation dyke occurrences are noted on the Zeekoegat, Middelpunt and Umkoanesstad farms. Current underground workings at Umkoanesstad have intersected dykes up to 10 m wide. No serious problems were encountered during mining through these features, and no significant displacements were noted to be associated with them. The estimated geological loss associated with dykes across the property is 4%. The aeromagnetic response to these features exaggerates the actual width dimension as depicted in Figure 7.12. Not all dykes have magnetic responses and a few (very minor proportion) east-west orientated dykes are known to have no magnetic response.

7.7 Replacement pegmatoids

The BC stratigraphy is sometimes affected by randomly occurring, late-stage replacement pegmatite bodies. These pegmatite bodies have a range of compositions from highly ultramafic to felsic. LPM is no exception to the occurrence of these geological features, but is noted to have minimal evidence for the more mafic replacement pegmatoids. Geological losses are estimated at less than 3% for replacement pegmatoids.

7.8 Geological losses

Geological losses are incurred as a direct result of geological features which remove or prevent the presence of the mineralisation, and therefore the ability to mine.

Geological features which incur geological losses are:

- Potholes
- Pegmatoids and dunite replacement of the reef
- Diabase, lamprophyre, and syenite dykes
- Structural and alteration phenomena, such as faults, shears, joints, veins, alteration zones

LPM has few major faults, a number of dykes and numerous potholes. The geological loss at LPM has been estimated as 20% for Merensky over all of the properties, except Brakfontein where the geological loss is estimated as 25% due to higher pothole occurrence and the regional pothole area. For the UG2 geological losses over all the properties are estimated at 15% primarily due to the lower frequency of potholes.

Deloitte is satisfied that the geological loss factors currently being applied are appropriate.

Deloitte recommends that future studies are conducted at LPM aiming at the quantification of the geological loss associated with bifurcated UG2 reef, as well as geological losses due to potholing below a depth of 650 m (cf. Section 18.9).

8 Deposit types

The UG2 and Merensky are magmatic layers formed by segregation within the mafic BC. These segregated layers have been the favoured lithologies for the accumulation of economic quantities of PGMs and base metals.

Specifically, the Merensky Reef is that portion of the Merensky Pyroxenite that is bounded by a top and bottom chromitite stringer and that has a 4E grade above 1.0 g/t. Higher 4E grades are commonly associated with the chromitite stringer and or a slightly coarser textured pyroxenite.

The UG2 almost exclusively attracts the PGMs, with trivial proportions of mineralisation occurring in the immediate hangingwall and footwall lithologies.

The hangingwall chromitite stringers and chromitite layers are known for their minor PGM occurrences. The immediate UG2 footwall pegmatoidal feldspathic pyroxenite is known to host some mineralisation.

The LPM geology and deposit is described in Section 7 of this TR and is well documented in published literature such as *The Geology of South Africa* (2006). Also references by Eales *et al.* (1993), Eales and Cawthorn (1996), Cawthorn and Boerst (2002 and 2006), and Cawthorn *et al.* (2002), are recommended for regional deposit type description.

9 Mineralisation

9.1 UG2 mineralisation

The UG2 mineralisation is comprised mainly of PGM accumulations that are hosted within the chromitite layers and have variable occurrences in the immediate footwall rocks, but very little in the hanging wall rocks. A 95 cm resource cut in most instances allows for the complete extraction of the mineral content. In the case of the presence of internal lenses (bifurcation) of pyroxenite, anorthosite or norite, the resource cut width may have to be increased to ensure that the UG2 is completely extracted. Ensuing sections of this TR document will refer to the UG2 and its diluting footwall and hangingwall constituents making up the 95 cm resource cut as the UG2.

9.1.1 UG2 geochemistry

The PGM mineralisation occurs in solid solution with sulphides, sulpharsenides, arsenides, bismuthides, tellurides, bismuthotellurides and alloys. PGM-sulphides, tellurides, and alloys are the main constituents of mineralisation in the UG2. The relative proportions of PGM content for the UG2 are colloquially known as the ‘prill split’. Prill splits were determined as part of the Mineral Resource estimation process described in this technical report. The PGM prill split for the UG2 is broadly Pt 43%, Pd 48%, Rh 7%, and Au 2%.

9.2 Merensky mineralisation

At LPM the mineralisation within the Merensky occurs at both the lower and upper chromitite stringers. Most of the PGMs are associated with the upper chromitite stringer and often extend over wider intervals to below the chromitite stringer. Mineralisation associated with the lower chromitite stringer at the base of the Merensky is generally over a very narrow interval and is sometimes absent. High PGM grades are often associated with the lower chromitite stringer, but due to its greater separation from the upper stringer, it was not included in the Mineral Resource estimates in this TR. The Merensky has visible base metal sulphides (commonly pyrite and pyrhhotite) and, as a result, may have viable concentrations of copper and nickel.

9.2.1 Merensky geochemistry

PGMs are commonly associated with base metal sulphides and are associated with the silicate and chromite minerals. At LPM, the Merensky PGM prill split is Pt 61%, Pd 29%, Rh 4% and Au 6%.

9.3 Mineralogy and metallurgy

Several comprehensive reports have been prepared on the mineralogy and metallurgy of the Merensky and UG2 at Lebowa, including those by Malysiak. (2001); Shamaila (2004a and b) and Roberts and Shamaila. (2005).

From a study of 17 UG2 samples taken on Umkoanesstad, Roberts and Shamaila (2005) conclude the following:

The 4E PGE grade of the mining cuts is very variable and in the range 5.8 to 9.3 g/t with a Pt:Pd ratio of 0.6 to 1.0. Altered silicates account for less than 5% of the material and are principally composed of chlorite and serpentine.

Overall, 75 to 96% of the BMS are liberated with pentlandite and chalcopyrite being the most abundant and coarsest of the free sulphides and pyrrhotite the finest and most locked. Of the PGMs, 54 to 94% are free or hosted by liberated BMS and the remainder principally hosted by silicates. PGE-sulphides and PGE-alloys are the most common free PGM-types and PGE(Bi)-tellurides the most locked. Pentlandite and chalcopyrite are the predominant PGM-bearing liberated BMS and pyroxene the major PGM-bearing silicate.

Sixteen mining cuts are readily amenable to flotation and final recoveries of ~90% Pt and 92% Pd or better are obtained due to their high proportion of potentially recoverable PGMs. Lower recoveries of 84% Pt and Pd are achieved for US74 due to the higher proportion of PGMs associated with slow-floating millerite and silicates. Base metal recovery is extremely variable but proportional to head grade.

Differences in the mineralogical and metallurgical characteristics of the chromitite appear to be related to the immediate footwall type. Ore supported by feldspathic pyroxenite contains more BMS, PGE-tellurides and potentially recoverable PGMs and yields the best recovery and concentrate grade. By comparison, chromitite underlain by pegmatoidal feldspathic pyroxenite contains more PGE-alloys but less potentially recoverable PGMs. Consequently, this ore type produces the lowest recovery and concentrate grade.

Smith and Shamaila (2004) conclude the following from their study of Merensky Reef intersections from Brakfontein:

The base metal sulphides represent less than 3% of the sample. Pentlandite is the major BMS with lesser chalcopyrite, pyrrhotite and pyrite. BMS liberation is >88%

The predominant PGM phases are PGE-sulphides and -tellurides. The remaining value minerals are PGE-arsenides, -alloys, -sulpharsenides and gold bearing phases. The PGMs are well liberated.

Metallurgical data indicates that Pt and Pd recoveries are above 95% for all boreholes. Head grades are relatively high (3.2 to 7.2 g/t 4E) for all boreholes except BF151 at 1.9 g/t 4E. The high recoveries are attributed to the high head grade, high liberated PGM component and the low levels of alteration of the reef intersections.

10 Exploration

The geological exploration and evaluation process involves reconnaissance, planning, diamond drilling, core logging and sampling, trenching and sampling, soil sampling, aeromagnetics, ground magnetics, mapping, processing, interpreting and modelling.

Lebowa Platinum Mine has been the focus of various exploration activities since 1964, with six phases of exploration having been carried out, all involving diamond drilling. Activities have centred on the Merensky, and only since 1999 has considerable focus been directed at the UG2.

10.1 Mapping

The UG2 has limited exposure along the hills located along the northern boundary of LPM. Where the outcrop exists on the Umkoanesstad and Wintersveld farms, it has been mapped. A number of dolerite dykes outcrop in these hills and have also been mapped. During 2002, a trenching program was conducted along the western UG2 outcrop areas on the Zeekoegat farm. Twenty-six trenches were excavated across this property, resulting in an accurately mapped UG2 outcrop position.

11 Drilling

11.1 Diamond drilling

At LPM, after reconnaissance and planning, borehole drilling sites are identified using GPS technology and then drilled by a reputable South African contract drilling company. All diamond drilling of recent years has ensured intersections for both the Merensky and UG2 (Figure 11.1). Only boreholes drilled down-dip of the Merensky outcrop have intersected both horizons. The Merensky and UG2 are separated by some 350 m of intermediate stratigraphy. Additional information regarding drilling is provided in Section 17 of this TR.

Deloitte notes that for each borehole with a Merensky and UG2 intersection, three deflections are drilled per parent hole. This is an Anglo Platinum standard and is considered best practice in PGM exploration.

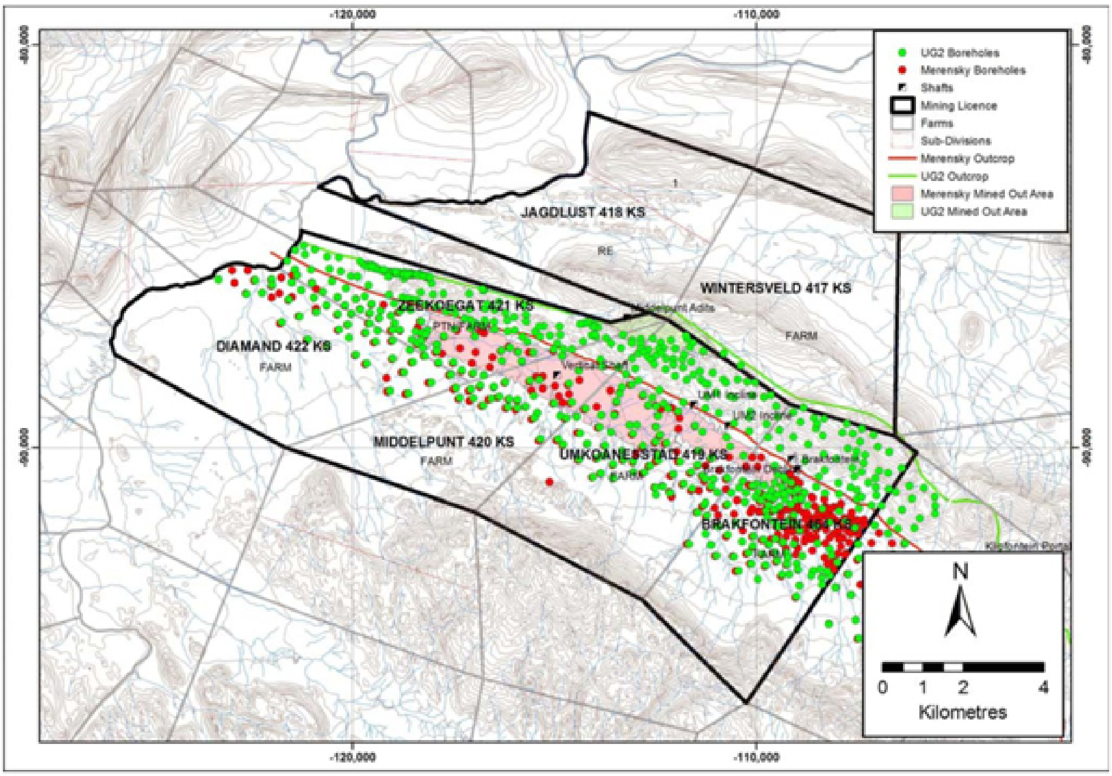


Figure 11.1: Merensky (red) and UG2 (green) borehole distribution (Gray et al., 2008).

12 Sampling method and approach

12.1 Core logging and sampling

Core logging is undertaken by qualified geologists at the Driekop Exploration Base (located approximately 50 km southeast of LPM) where all boreholes and their deflections are accurately logged in terms of lithology, mineralisation, alteration and structure. Logging details are entered directly into laptop computers, making use of the Sable software package designed for this purpose. Geotechnical and structural logging is also carried out by geotechnical staff and structural geologists.

During the logging process, the sampling interval through the mineralised succession is determined and individual samples measured, marked-off and numbered according to standards. Sampling is done continuously throughout the sample section.

Measurements and marking of sample lengths/widths are carried out according to Anglo Platinum standards. Within the UG2 and Merensky horizons, a maximum of 20 cm sample lengths are taken, but general practise is to keep sampling widths to no less than 15 cm. Sample lengths of 15 cm are also applied to the hangingwall and footwall lithologies. The top and the bottom contact samples are taken in such a manner that 2 cm of hangingwall and 2 cm of footwall are included. These samples ensure that high grade material at the top and the bottom contacts of the Merensky and UG2 are included. This facilitates representative modelling of the mineralised horizon and mining widths.

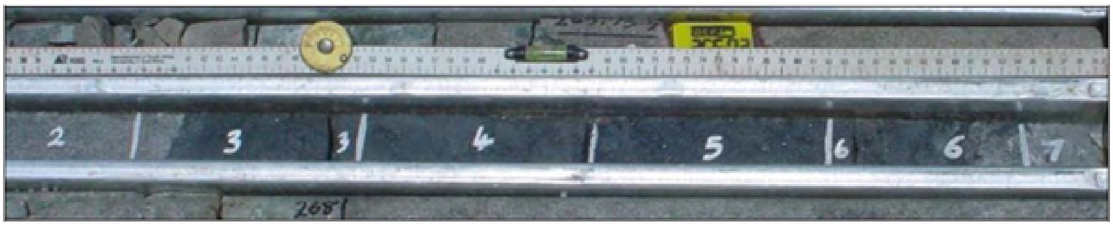


Figure 12.1: The top and bottom UG2 contact samples showing the inclusion of 2 cm of hangingwall and footwall respectively.

The marked core is then submitted to technical staff who proceed to cut the core in half longitudinally. One half of this core is then cleanly broken with a sharp chisel (to avoid sample losses) at individual sample boundary markings and immediately labelled, bagged and sealed.

Precautions are taken to avoid any cross contamination between samples. Individual samples are separated, and in some cases, an additional deflection or intersection is sent for mineralogical/metallurgical investigation.

Once the sampling and logging of the boreholes is completed, the remaining core is stored on core racks. The sample intervals and numbers are replicated onto the remaining core surface for reference, and future re-sampling if necessary.

The sampling data is fully documented and recorded on site, with records of all sampling maintained.

The sampled borehole core (intersections of UG2 and Merensky) is then assayed for individual PGM content, as well as density and Cu and Ni contents. Due to the analytical process's sample mass requirements and detection limits, Rh is only analysed when the Pt+Pd+Au grade exceeds 1.5 g/t.

12.2 Underground sampling

12.2.1 Planning

Typically, the sampler plans sampling at least three days in advance to ensure that working air and water fittings, as well as the required equipment and staff are available. Prior to going underground, the sampler checks the plan for any identified geological disturbances in the area.

12.2.2 Sample spacing

All on-reef development is sampled. The interval between sample sections is a minimum of 10 m and a maximum of 20 m. Advanced strike gully (ASG) sampling is done at 20 m intervals. ASG samples are approximately 30 m apart in the true dip direction. This creates a pseudo grid of 20 m by 30 m. The sampler may vary the sample position by up to 2 m to ensure that:

- Samples are only taken at safe sampling sites.
- Only fully exposed sections are sampled (i.e. exposure of both hangingwall and footwall).
- The maximum amount of exposure of hangingwall is taken.

The sampler is responsible for accurately recording the true distance of the sampled sections from underground survey pegs.

12.2.3 Sampling method

Underground sampling is typically done by means of cutting channels using a rotary diamond saw machine powered by compressed air. Clearance must be obtained from the responsible section miner before using the diamond saw in the working place.

The sampler records all geological features such as reef characteristics, prominent alterations, hangingwall or footwall, faults or dykes, potholes or major rolls and occurrences of reef left in the hangingwall or footwall. Deviations and anomalies are reported to the responsible geologist. The sampler marks the intended sampling positions and the assistant bars, brushes and washes the face with water.

Two parallel lines four centimetres apart are drawn from the hangingwall to the footwall at right angles to the dip.

Two cross-lines, at least 20 cm long, are drawn parallel to the waste contact, from two centimetres below the bottom contact to two centimetres above the top contact. Samples are then marked off parallel to these lines at approximately 15 cm widths. The exposed hangingwall is sampled at 10 cm widths and at least one sample must be taken. The total footwall is sampled at 15 cm widths and at least seven footwall samples must be taken. All grooves are approximately five centimetres deep and the samples are chipped in sequence from the bottom upwards. The sampler records the actual sample lengths and all samples are numbered sequentially as per the sampling tickets and recorded into a PDA. Each sample is carefully placed in a clean plastic bag and a bar coded sampling ticket is pasted on the bag and closed.

12.2.4 Dispatch supervision

Samples are captured in the Mineral Resources Management (MRM) database by the sampler on the same day by down-loading the data from the PDA. The sampler is responsible for ensuring that his sections are captured correctly. The sample bags are then sealed with an impulse sealer and scanned for the 2-D bar code. Historically, the samples are then packed into containers and sent to the Eastern Bushveld Robotic Laboratory (EBRL) via courier.

13 Sample preparation, analyses, and security

13.1 Assay procedure

A variety of analytical techniques have previously been used in assaying samples for Lebowa Platinum Mine:

- Lead-collection fire assay; total 4E PGM was reported with low confidence due to a high temperature cupellation step in the process that caused the loss of the majority of Rh in the samples. This technique has not been employed since 2000.
- Lead-collection fire assay gravimetric prill; individual PGM's determined with similar levels of confidence as above.
- Nickel-Sulphur dissolution; individual PGM's determined with high confidence and accuracy.

From 2000 onwards, diamond core samples have been sent to Anglo American Research Laboratory now (Anglo Research (AR)) in Crown Mines where they are analysed for PGM's, Ni and Cu. The laboratory is operated by a subsidiary of Anglo American and is International Standards Organisation 17025 accredited.

13.1.1 Core samples

Samples are cut, split, bagged and checked against accompanying sample requisition sheets and sample descriptions by the geology department after which they are dispatched to AR for analysis.

Samples are analysed for:

- Pt, Pd and Au using fire assay (lead-collector and gold as co-collector) with inductively coupled plasma (ICP) finish. 3E is Pt+Pd+Au.
- Rh (where 3E is greater than 1.5 g/t) using fire assay (lead- collector and palladium as co-collector) with ICP finish.
- Cu and Ni using X-ray fluorescent (XRF) analysis.
- Density using Grabner pycnometer.

13.1.2 Sample preparation

Samples are crushed in a jaw crusher to two millimetres. The sample (provided the sample is smaller than 3.5 kg) is then milled to 85% ± 5% to minus 75 microns or finer. An eight minute milling time is required.

13.1.3 XRF-AR method for the analysis of Cu Ni

Pulped samples (27 g) are mixed with a styrene/wax binder (3 g) and milled to mix in the binder and further reduce particle sizes. The samples are pressed into briquettes. The briquettes are then read on the ARL PW 1404 XRF for Cu and Ni. Corrections are made for mass absorption coefficient, background and tube spectral interferences (copper only). Mineralogical effects are evident in the briquettes - hence separate ‘type’ calibrations are critical for Merensky and UG2 type samples. Approximately five % of the samples are replicated. Two reference materials are analysed with every batch of up to a maximum of 100 samples.

13.1.4 Fire Assay ICP method for the analysis of Pt, Pd, Au

Samples are weighed out and mixed with an appropriate flux for the material type (i.e. Merensky or UG2). Silver is used as a co-collector.

The samples are fire assayed and the prills dissolved in aqua-regia and read on the ICP for 3E. One blank and two reference materials are analysed with every worksheet (maximum of 35).

Mass taken: 50.0 g for Merensky.

Mass taken: 30.0 g for UG2.

Detection Limit: 0.02 g/t.

All assays are done in duplicate and the average of acceptable replicate pairs is reported. Both sets of results are made available to AR and to Anglo Platinum’s Geology Department.

13.1.5 Fire Assay ICP method for the analysis of Rh

Rh is only analysed if the total of the 3E analysis is greater than 1.5 g/t.

Samples are weighed out and mixed with an appropriate flux for the material type. Palladium is used as a co-collector. The samples are fire assayed and the prills dissolved in aqua regia and read on the ICP for Rh. One blank and two reference materials are analysed with every worksheet (maximum of 35).

Mass taken: 50.0 g for Merensky.

Mass taken: 30.0 g for UG2.

Detection Limit: 0.02 g/t.

Density:

Pulped samples are analysed on a Grabner instrument. Four % replication is performed. Acid washed quartz is analysed with every batch of samples.

Mass taken: 5.00 g.

Reporting of Results

All assays are completed in duplicate and the average of acceptable replicate pairs is reported. Similarly, both sets of results are made available to AR and to Anglo Platinum's Geology Department.

Results are transmitted electronically to AR in Excel format.

13.1.6 Quality Assurance/Quality Control (QA/QC)

AR has a comprehensive assay quality control system that includes blanks, certified reference materials, in-house reference materials, and twin streaming / replicate analyses. It is noted that the Anglo Platinum QA/QC databases, including parameters such as blanks and certified reference material, were not inspected during the course of this investigation.

Care is taken during the handling of samples to avoid potential cross-contamination or misplacement of samples. High and low grade materials are processed in completely separate areas throughout the laboratory, using dedicated and clearly labelled equipment. Samples are weighed and checked upon receipt. Quarry quartz is crushed and milled between individual batches to avoid any possible carry-over. This quartz is analysed with the batch and results are reported on during progress meetings.

For each tray (worksheet) of 3E and Rh analysis, reagent blanks, standard reference material and duplicate samples are included for control purposes. Internationally certified standards as well as internal standards of matched matrices are used. Fire assay pots are only used once to avoid the possibility of cross-contamination of samples. A full calibration of the ICP and atomic absorption spectrophotometry (AAS) is performed prior to sample analysis, and a synthetic check solution is included after every 15 samples. Where the check solution data falls outside the acceptable control limits, the instrument is re-calibrated.

Worksheets are accepted or rejected based on the quality control data of the standards, replicates and blanks. A complete audit trail is maintained in the laboratory to ensure traceability, transparency and ISO compliance. Quartz blanks are designed to monitor the entire process from sample preparation to instrumentation. They are treated as normal samples, i.e. they are prepared with the normal samples. Quartz blanks consist of quarry quartz. As this represents a natural geological material, small amounts of trace elements, e.g. Cu and Ni, are expected to be present. Any contamination introduced during sample preparation and subsequent processes will be reflected in the quartz blank (i.e. it becomes a known amount).

Reagent blanks are introduced during secondary preparation. These are essentially reagents without the sample introduced. For example, in fire assay the reagents would be assayed, and a button made, the prill dissolved, and the solution read. Reagent blanks reflect contamination introduced during the analysis phase but not the primary preparation phase. Spectral density (SPECDENS) and XRF data do not include reagent blanks, but do analyse quartz blanks.

With each method, certified reference material (CRM) and in-house reference materials (IHRM) are run. These are usually type-specified to the method. For example, in

Merensky Reef fire assay for 3E and Rh SARM7 (CRM) and MER001 (IHRM) are run. For UG2 samples, SARM65 (CRM) and UG2001 (IHRM) are run. For XRF analyses, IHRM are run, and for SPECDENS calibration blocks, acid washed quartz, and quartz are run.

For each analysis and standard the precision is calculated and spread about the certified value. When standard values exceed the stated precision from the certified values, the data is rejected and the worksheet analysed. This may involve a total re-analysis or just instrument reading, depending on the problem.

For each method, samples are replicated, if not twin streamed. 3E and Rh are twin streamed, i.e. a 100% replication. For XRF and SPECDENS analyses, 10% replicates are run. Concentration precision curves are then calculated according to the Thompson Howarth algorithm using replicate pairs. This is the stated precision to which AR works and monitors for each method and for each analyte. Replicates are essentially used for measuring the precision of the analyses, and appropriate action is taken if such results are not considered satisfactory.

13.1.7 Rock Density

AR measures the density of each sample as a matter of routine, the results of which are reported with the assay results (see previous section). From this data, average densities for the Merensky and UG2 across the entire Lebowa have been determined (Tables 13.1 and 13.2).

Table 13.1: Average densities for Merensky.

Rock Type	Density (t/m^3)	Samples
Merensky hangingwall	3.43	416
Merensky	3.44	417
Merensky footwall	3.35	421

Table 13.2: Average densities for UG2.

Rock Type	Density (t/m^3)	Samples
Hangingwall pyroxenite	3.44	961
UG2	4.17	1 093
Footwall pyroxenite	3.41	1 191

13.2 Laboratory facilities at Lebowa

Apart from basic sample preparation, there is currently no analytical laboratory at Lebowa. LPM utilises the facilities at the Polokwane Smelter Complex (for assays of the mill feed, tailings and underground samples) and AR (assays of concentrate samples).

The turnaround time from the Polokwane Smelter Complex is about four days and from AR it is up to 16 days.

14 Data verification

During the site visits of 17 to 19 December 2008 and 20 to 21 January 2009, Messrs. Schweitzer, Güler, Lambert and Naidoo met with various employees at LPM, including the Chief Geologist, Chief Surveyor, Acting Manager: Technical Services, the Mine Planner, the Metallurgist and a representative of the Engineering Department. In addition, Messrs. Lambert and Naidoo met with the Exploration Manager – Ore Evaluation and the Senior Mining Engineer at Anglo Platinum's corporate headquarters in Johannesburg, and Mr Kramers met with the Plant Superintendent at LPM from 22 to 23 January 2009.

The mine plans, computer models and databases were viewed together with the responsible persons.

It was not practical to conduct a 10% check of geological logs to assay database to computer database due to the extensive size of the databases which would be required for an exploration target.

However, the Anglo Platinum/Lebowa procedures as provided in the virtual data room maintained in connection with the Lebowa Transaction were reviewed and Messrs Schweitzer and Naidoo are satisfied that the databases are well maintained and validated according to the set procedures.

In addition, Anglo Platinum has a policy of stringent ongoing review processes by their Centralised Services departments which include detailed validation of the geological information and databases. This is also described in Section 17 of this report.

Subsequent to the site visit, Mr Güler also met with the Anglo Platinum Project Manager responsible for the immediate Lebowa projects (BRK Merensky Project and the MPH UG2 Project) for an explanation of the projects' history and the implementation strategy.

The Deloitte representatives have also had access to a due diligence report prepared by RSG Global entitled **LEBOWA PLATINUM MINES, INDEPENDENT TECHNICAL REVIEW OF THE OPERATIONS AND PROJECTS**, Revised March 2008 (RSG Global Report), as well as a previous technical report prepared by Snowden entitled **TECHNICAL REPORT: LEBOWA PLATINUM MINE, LIMPOPO PROVINCE, SOUTH AFRICA**, dated April 2008.

15 Adjacent properties

To the northwest, immediately beyond the Lebowa boundary, the strike of the Merensky and UG2 is truncated by large scale faulting (Figure 7.6). These faults on adjacent northwest properties do not affect the geology of Lebowa.

The Merensky and UG2 extend in a southeasterly direction onto the neighbouring Klipfontein property (Figure 7.8). This property, and its' southeastern neighbour, the Paschaskraal property, are included in the stand-alone Ga-Phasha joint venture project which, after implementation of the Lebowa Transaction, will be effectively owned by Anooraq and Anglo Platinum as to 51% and 49%, respectively as described in Section 4.4.

16 Mineral processing and metallurgical testing

16.1 Introduction

Lebowa Platinum Mines Limited (LPM) currently has two concentrator plants located adjacent to each other. One plant processes Merensky ore and the other UG2 ore. The concentrators are situated close to the Vertical Shaft. Figure 16.1 shows a general view of the concentrators.

Figure 16.1: General view of the concentrators.

The concentrators are generally well-maintained, providing good operating availability.

The Merensky concentrator (rated capacity 85 ktpm) is currently dedicated to processing ore from the Vertical Shaft (35 to 40 ktpm) and the UM2 Inclined Shaft (13 ktpm to 15 ktpm). The ore is delivered to the plant via a conveyor belt system from the Vertical Shaft.

The UG2 concentrator (rated capacity 70 ktpm) is dedicated to processing ore from the Middelpunt Hill UG2 adits and decline, which currently produce 45 ktpm. The UG2 concentrator also treats ore from the neighbouring Twickenham mine, owned by Anglo Platinum, at a rate of approximately 30 ktpm.

UG2 ore is delivered to the concentrator by road.

Historic operational utilisation efficiencies from 2004 to 2007 were about 90 %, and this is considered sustainable for future operations and modelling purposes. The Merensky Concentrator operated at efficiencies near 90 % throughout this time period, whereas the UG2 Concentrator operated at efficiencies between 61 % and 92 % over the same period.

The concentrate arising from metallurgical processing is shipped by road transport to the Anglo Platinum smelter at Polokwane.

Deloitte conducted an independent technical review of the LPM metallurgical processing operations, including a visit to Lebowa Platinum, 22nd and 23rd January, 2009.

The purpose of the review was to confirm that the existing metallurgical processing operations, processing facilities and associated infra-structure conformed to sound Engineering principles and practices. The review included the following:

- Detailed review of the Merensky ore processing concentrator
- Detailed review of the UG2 ore processing concentrator
- Assessment of the technological competency of the existing Merensky and UG2 processes
- Assessment of the current processing equipment selected
- Current and future plant expansion
- Assessment of the general condition and maintenance of the concentrators
- General management and safety aspects
- Concentrator performance review
- Platinum group metal recovery capabilities of the concentrators
- Metallurgical testing
- Tailings dams

16.2 Merensky (MF3) Concentrator Process Review

16.2.1 Merensky concentrator process flow sheet

Figure 16.2 shows the Merensky plant (MF3) metallurgical process flow sheet.

No major problem or flaw with the basic process flow sheet of the Merensky concentrator is apparent. The plant in general is well conceived and designed.

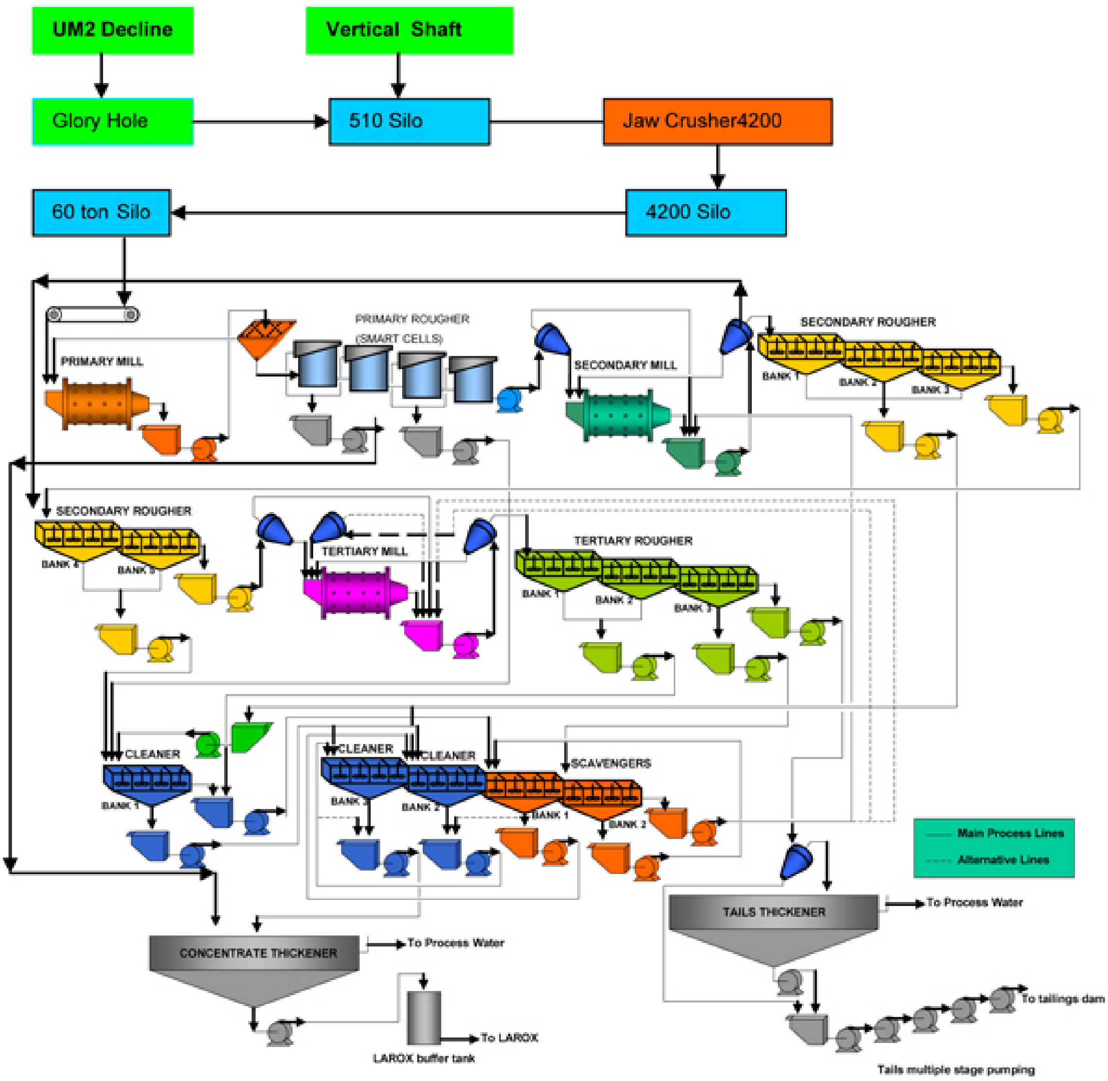


Figure 16.2: Merensky Plant MF3 process circuit (after LPM).

16.2.2 Merensky Ore Stockpiling and Blending

Merensky ores are stored in three bins within the process plant, namely a 510t silo, a 4200t silo and a 60t surge bin ahead of milling. To facilitate a more even run of mine (ROM) particle size distribution (PSD) for the primary fully autogenous grinding (FAG) milling operations, ore blending is conducted in the 4200t silo through distributing the feed to this silo, and withdrawing ROM from the bottom of the silo through 4 control chutes. Figure 16.3 shows the configuration of the silo off-take chutes.

Figure 16.3: Ore removal chutes under 4200t silo.

Merensky ore from the UM2 decline is stored on the ground at the surface tip, prior to being introduced into the primary silo and processed. The first silo is to decouple the shaft system from the crushing operation whilst the larger silo is for blending and storage ahead of milling. The small 60t silo is for mill feed control purposes.

Underground sludge from the Vertical Shaft, which is produced from Merensky mining, is pumped to surface and dried in paddocks. The sludge is blended with the Merensky ore. The quantity of underground mud produced is relatively low.

16.2.3 Merensky Ore Processing

Crushed and blended Merensky ore (nominal 230 mm top-size), is delivered via a conveyor belt to the 60t primary mill silo. On route to the silo, a weightometer is used to provide an integrated reading of the total mass of ROM delivered. Figure 16.4 shows an Accuweigh weightometer on the conveyor belt feeding the mill feed bin.

ROM is then fed at a controlled rate into a Metso primary fully autogenous grinding (FAG) mill measuring 4.27 m in diameter and 4.57 m in length. It is driven by a single 1,150kW installed power motor. The milled slurry is screened at 800 micron and the fine fraction is subjected to primary rougher flotation in four stages of WEMCO 50 smart cells (capacity of 50 cu. m each), whilst the coarse fraction is re-milled. The first concentrate (from smart cells 1 and 2) is transferred to final concentrate, whilst the second concentrate (from smart cells 3 and 4) is pumped to the cleaner flotation circuit. The primary rougher flotation tails undergo hydro cyclone classification. Cyclone underflow is fed to the secondary ball mill, whilst the overflow is fed to the secondary mill discharge sump. Figure 16.5 shows the primary FAG mill with the primary rougher smart cells in the back ground.

Figure 16.4: Merensky ore Accuweigh weightometer.

Figure 16.5: Primary FAG mill with the primary rougher smart cells in the background.

The second stage milling is conducted in a 3.66 m diameter by 5.49 m long overflow ball mill in closed circuit with a large diameter hydro-cyclone. The mill is driven by a single 1,150kW installed power motor. Figure 16.6 shows the secondary ball mill.

Figure 16.6: Secondary ball mill.

The fine fraction of the secondary milled product is subjected to two stages of secondary rougher flotation (first stage is three banks of four WEMCO 120 cells (each cell 8,5 m³) and the second stage is two banks of four WEMC0 144 cells (each cell 14,2 m³), with the concentrate being subjected to cleaner flotation, and the tailings transferred to the tertiary milling circuit, after classification.

The tertiary stage milling is conducted in a 3.66 m diameter by 3.66 m long overflow ball mill driven by a single 1,150 kW installed power motor operating in closed circuit with a hydro-cyclone. The fine fraction is subjected to one stage of tertiary rougher flotation in three banks of WEMC0 144 cells. The concentrate produced is subjected to cleaner flotation, whilst the tailings are classified, thickened in a 40 m diameter thickener, and transferred to the tailings darn by multi-stage pumps.

The cleaner flotation circuit contains three stages of cleaning plus two stages of cleaner scavenging. The first cleaner bank consists of four 0K8 flotation cells (each cell 8 m³), the second stage consists of four OK 3 flotation cells (each cell 3 m³), and the final cleaners also consist of four stages of OK3 flotation cells. Cleaner scavenger flotation is conducted in two banks of four OK5 tank flotation cells. Cleaner tailings are transferred to the tertiary milling circuit for regrinding. Figure 16.7 shows a general view of the flotation cells in the plant.

Figure 16.7: General view of flotation cells in Merensky plant.

Final concentrate is delivered to the Merensky Concentrator thickener as thickened slurry and stored, prior to pressure filtration, in a Larox filter. Merensky concentrate is then transported to a smelter by road. Figure 16.8 shows the Larox filter.

Figure 16.8: Larox filter.

The reagent regime is conventional for Merensky concentrator flotation with sodium isobutyl xanthate (SIBX), Senkol 65, Senfroth 6005 and CMC depressant being employed. The reagent mixing system is located within the Merensky Concentrator building and is not shared with the UG2 Concentrator.

16.2.4 Merensky plant modernization program

The current 85 ktpm Merensky concentrator, built and commissioned between 1989 and 1991, includes the three milling stages with inter-stage flotation circuits. By modern technological standards, the Merensky concentrator employs older technology, but nevertheless maintains high operating efficiencies and availability. The milling plant is in the open, whilst the flotation plant and concentrate filtration, storage, and loading, is under cover.

The Merensky concentrator is currently undergoing an upgrade and modernization, which is planned for implementation in the 2009 financial year. The modernization and technological upgrade will improve the throughput capacity from 85 ktpm to 100 ktpm, and includes:

- Replacement of the current semi programmed-logic-controlled (PLC) process control system with a modern PLC remote-controlled system to increase efficiency. The first phase of this is scheduled for completion in March 2009.
- Increasing the primary mill capacity by converting from fully autogenous grinding (FAG) to semi autogenous grinding (SAG). This will involve changing the mill-shell (the shell currently shows evidence of micro-cracks), and converting it to a SAG mill. Scheduled completion for this is November 2009. This will overcome a problem that is sometimes experienced currently due to inadequate coarse rock in the run of mine feed, to act as grinding medium. The result is poorer than standard grind and throughput rate. Currently, a supply of coarse rock is stockpiled close to the plant feed conveyor system, to enable manual addition of coarse rock if required, using a front end loader.
- The indications are that the current flotation plant equipment capacity will be adequate to meet the planned 100 ktpm expansion. However, appropriate modifications to some pipes and valves will be required.
- Upgrading the motors and drive train systems on all mills is planned.
- Upgrading of appropriate internal pumping systems with variable speed motors etc. is planned.
- Small diameter cluster cyclones to replace the existing large diameter cyclones have been installed between rougher flotation and milling in the Merensky circuit. These are currently being commissioned.

16.3 UG2 (MF2) Metallurgical Processing Plant

The 70 ktpm UG2 Concentrator includes two milling stages with inter-stage flotation circuits (MF2). It is a dedicated concentrator, constructed in 2000 to treat UG2 ore that has subsequently been mined at the Middelpunt Hill adits. The UG2 Concentrator is located adjacent to the Merensky Concentrator and is similarly well-maintained, providing good operating availability.

16.3.1 UG2 Ore Stockpiling and Blending

The UG2 Concentrator (rated capacity 70 ktpm) is dedicated to processing ore from the Middelpunt Hill UG2 adits and decline, which currently produce 45 ktpm. The UG2 Concentrator 'toll-treats' ore, on a temporary basis, from the neighbouring Twickenham Mine, owned by Anglo Platinum, at a rate of approximately 30 ktpm.

UG2 ore from Middelpunt Hill UG2 adits is road delivered and stored on separate stockpiles on the ground at the surface tip near the crushing station. Because the floatability of the ores from the different mining locations is different, e.g. adit 4 material, appropriate blending of Lebowa UG2 and adit 4 in ratios between 1:1 and 1:2 may be required, dependent on the status of the float. Ore blending is achieved by selective delivery of loads of the different ore source materials to the crusher station. In the crushing circuit there are a number of small surge bins as well as a large mill feed bin which assists in providing a homogeneous mix, and provides intermediary stock of the blended ore.

On arrival, oversize rock and underground support timber is removed by means of a back-actor. This can be seen in Figure 16.9. This area may require upgrading in the future.

Figure 16.9: Removed oversize rock and support timber.

Twickenham ore is also delivered by road, to a specially prepared concrete pad area, where the ore is allowed to sun-dry during stock build up, to enable separate processing campaigns of about 1 week in duration. For the rest of the time, the Lebowa UG2 ore is processed. The two ore sources are not mixed. The moisture content of the Twickenham ore is generally significantly higher than that of the Lebowa ore, due to a higher ultra-fines content, thus requiring drying. Figure 16.10 shows the Twickenham ore drying pad.

Figure 16.10: Twickenham ore drying pad.

There is a weather dependency that sometimes leads to throughput and dry crushing problems during the rainy season.

16.3.2 UG2 plant process flow sheet

The UG2 plant process circuit is depicted in Figure 16.11.

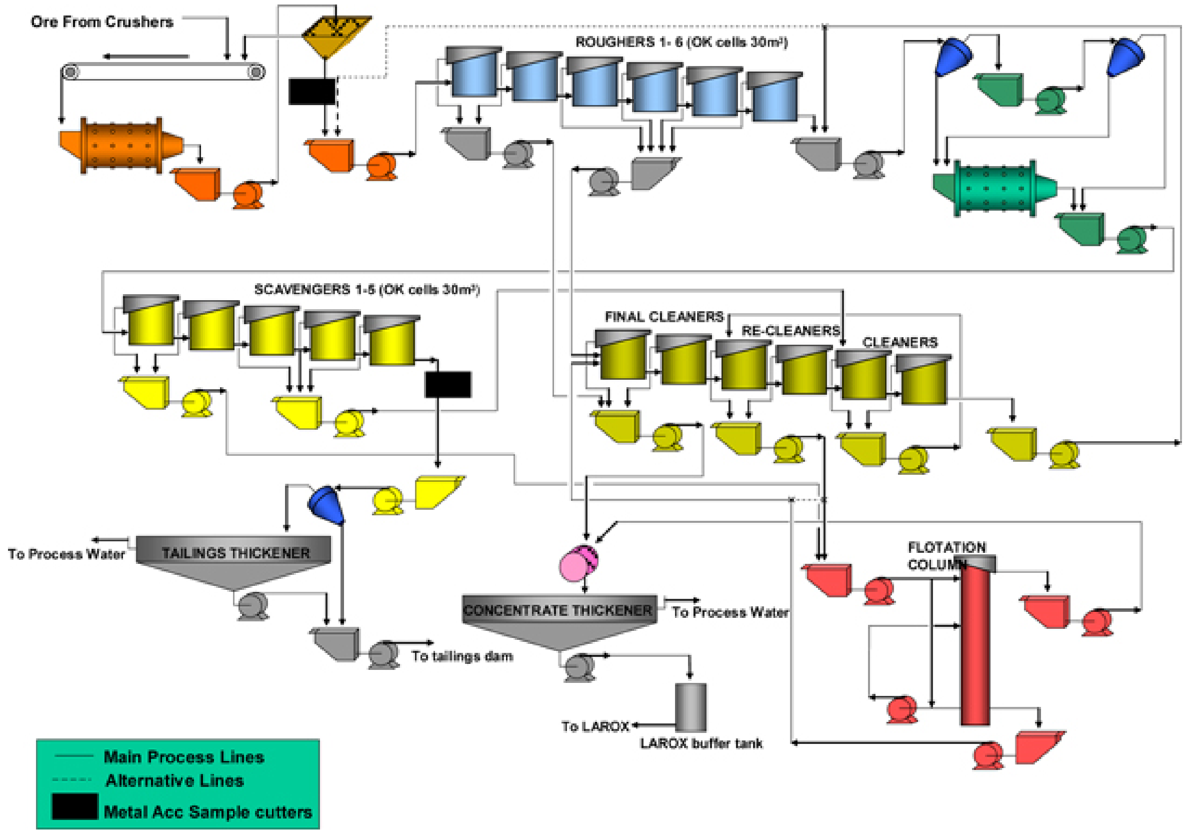


Figure 16.11: UG2 plant process circuit (after LPM).

No major problem or flaw with the basic process flow sheet of the UG2 concentrator is apparent. The plant in general is well conceived and designed.

16.3.3 UG2 Ore Processing

The UG2 concentrator milling and flotation sections are located in the open, but the common Merensky/UG2 concentrate filtration (Larox) is under cover, to avoid material losses due to wind and concentrate moisture increases due to rain.

The reclaimed UG2 ore is dry crushed to about minus 15 mm, using conventional jaw crushing and closed circuit screening and cone crushing technology. Figure 16.12 shows a general view of the dry crushing facility.

Figure 16.12: General view of the UG2 crushing facility.

The crushing station is operated by an outside BEE contractor, and is seldom the cause of delays to the UG2 processing operations. Adequate crushed ore storage capacity between the crusher station and the UG2 primary mill is available to provide flexibility.

The crushed product is then fed into a surge bin ahead of milling. On route to the mill bin, the mass of the UG2 ore fed to the plant is recorded using an Accuweigh belt weightometer. The belt is also equipped with a magnetic tramp iron removal device.

Underground mud mined from the UG2 areas, claimed to contain 25g/t 4E's, is dried in paddocks and shipped directly to the smelter at Polokwane.

The primary mill is a primary grate discharge ball mill measuring 3,66 m in diameter and 4.88 m in length with installed power of 1,250 kW. Figure 16.13 shows the primary UG2 ball mill.

Figure 16.13: Primary UG2 ball mill.

The milled product is screened and the fines fraction subjected to two stages of primary rougher flotation in six OK30 tank cells. Figure 16.14 shows the mill feed belt, the milled product screen, and the primary rougher flotation cells.

Figure 16.14: Mill product screen and the primary rougher flotation cells.

The first stage concentrate is delivered to final concentrate, whilst the second stage is transferred to the cleaner circuit. The tailings are classified in two stages with the coarse fraction being re-milled and the fines subjected to secondary (scavenger) flotation. Figure 16.15 shows the two stage classification.

Figure 16.15: Two stage primary rougher tail classification.

Secondary milling is conducted in an overflow ball mill measuring 3.66 m in diameter and 4.88 m in length, with installed power of 1,250 kW and in open circuit. The milled pulp and the classification overflow are transferred to the secondary flotation circuit, consisting of two stages, and a total of five OK30 scavenger flotation cells. The UG2 concentrate produced is delivered to the cleaner flotation circuit, whilst the tailings are classified and thickened in a 21 m diameter thickener, and transferred to the UG2 tailings dam.

The rougher concentrates are subjected to four stages of cleaning. The first two stages each consist of two OK20 tank cells and the final stage consists of two OK10 tank cells. The intermediate concentrate is cleaned in a column cell to reduce the chrome content and increase the PGM content. Cleaner tailings are transferred to secondary milling for regrinding.

Final UG2 concentrate is then delivered to the UG2 concentrator thickener, with thickened slurry stored prior to pressure filtration in the common Larox filter. Final UG2 concentrate is transported to the Polokwane Smelter Complex by road.

16.4 Concentrate Handling

Merensky and UG2 concentrates are stored separately ahead of the common Larox concentrate filter. Filtration is conducted on a campaign basis and the capacity is adequate for current production.

Merensky concentrate is filtered to a moisture content of about 5% and UG2 concentrate to about 14%. The difference is due to the different particle sizes of the products. The smelter requires a moisture content of less than 15%, so in both cases the concentrate is within the moisture specification. Figure 16.16 shows the Merensky and UG2 filtered concentrate storage facilities, from which concentrate is transported in specially designed road transporters for delivery to the smelter.

Figure 16.16: Merensky and UG2 filtered concentrate storage facilities.

It is stated by the mine that when changing the UG2 processing campaigns from Lebowa to Twickenham, the crushing plant and mill feed bin are emptied of ore and the weightometer readings taken. On completion of a Twickenham ore processing campaign and the start of a Lebowa UG2 processing campaign, the first two truck loads of Lebowa UG2 concentrates are loaded as Twickenham. Likewise, at the completion of a Lebowa UG2 run and the start of a Twickenham run, the first two concentrate loads are loaded as Lebowa concentrate. This ensures non-mixing of the concentrates and accountability at the smelter. It seems that the Twickenham ore is different to the Lebowa ore in that the recovery and concentrate grade is lower, although no data apart from the annual results was readily available.

Normal metallurgical accounting is conducted from the concentrator mill feeds, tailings streams to concentrate.

16.5 Processing plants condition and management

16.5.1 Concentrator safety

During the site visit it was observed that moving equipment in the concentrators is well guarded. Conveyor belts are equipped with appropriate emergency pull wires and safety interlock systems. The plant is well equipped with safety notices, emergency showers etc. Appropriate hand rails, walk ways and access are available in the plants. During the visit, plant operators were observed to be wearing the necessary personal protection equipment and using equipment lockout systems during maintenance. Plant management could detail all the recent injuries that had occurred on the concentrator.

Well organized safety induction and training is in place. The concentrators exhibit good housekeeping and general maintenance.

Provided that the basic safety issues are addressed by the staff and management at the Lebowa concentrator, the author is of the opinion that there is no reason to believe that safety will become a major issue or concern.

16.5.2 Plant condition and maintenance

In general, the plants operate well with few major equipment breakdowns. The most common reasons for current plant stoppages are due to lack of ore from the shafts and planned maintenance.

From a physical plant inspection, it appears that both concentrators are generally in good order with no evidence of excessive corrosion or decay to either structure or installed equipment. Evidence of this can be seen in the photographs in the previous sections of this report. There is some evidence of spillage, however.

A planned maintenance program is in place which provides for extended maintenance stoppages on key equipment. Stockpiling capacity in the form of Merensky silos and UG2 ground stockpiles assist in managing extended stoppages in ore delivery to the concentrators.

The existing programmed-logic-controlled (PLC) and computer based process control system monitor both concentrators, but the system has only a limited capacity to operate the concentrators remotely, via the process control system. The general condition of both the Merensky and UG2 concentrators at Lebowa Platinum is regarded as fair and reasonable. There were no significant major problematic operational issues identified. Where spillage occurred during plant start up or stoppage, this was immediately taken care of. The process equipment, concrete work, structural steel and sheeting are generally in good condition and where there has been evidence of damage, maintenance and repair work has been conducted.

For example, reinforcing cable straps have been installed around the 4200 silo to avoid concrete damage as shown in Figure 16.17.

Figure 16.17: Cable reinforcing around the 4200 silo.

Planned maintenance is managed through the mine SAP system and the planned maintenance system is regarded as adequate.

The concentrator maintenance team consists of boilermakers, electricians, fitters and management. This team completes routine maintenance activities whilst major maintenance such as mill relining, flotation equipment replacement and conditioning monitoring are contracted out to specialist supply companies (Multotec, Metso, etc.), or to the original equipment manufacturer (OEM).

It is reported by plant management that the aim is to have a 48 hour concentrator shutdown on each plant every month (but not at the same time) and an entire plant stoppage every three months. This is regarded as an appropriate maintenance philosophy for this and any concentrator.

16.5.3 Process Control

The plant has a PLC and computer based process control system to monitor both concentrators. Figure 16.18 provides a view of the control room.

Figure 16.18: Process control room.

The process control system is not fully integrated into the operation and only monitors process conditions (flows, densities, levels, etc.). In a few cases it is capable of stopping and starting process equipment remotely. All major and most minor process equipment is operated manually (stopped and started) from the field, and not through the control system.

There are no in-stream or on-line analysers to measure (particle size distribution or contained metal) and control the operating conditions of the plant to desired levels. In addition, there is no feed forward control system to optimize and modify reagent addition based on the tonnage being processed or other factors. This is regarded as a possible weakness in the operation, but as the performance is more than adequate, this is not a serious concern. The entire PLC system is currently being up graded.

The plants are equipped with automatic sample cutters for feed and tails grade analysis. Figure 16.19 shows one of the tailings stream sample cutters.

Figure 16.19: Automatic sample cutter in the Merensky plant tailings stream.

16.5.4 Spares and Stocks

There appears to be adequate stocks of spares with regard to maintenance items such as flotation equipment, conveyor belting, mill liners, mill major items (mill ends, bearings, motors, gearboxes, etc.) and Larox filter equipment. Considering logistics and stores management, the current situation is that the Polokwane store is the central store and receiving facility, and maintains the stock level of spare parts.

16.5.5 Laboratory facilities

Apart from basic sample preparation, there is no analytical laboratory on site. Lebowa Platinum utilizes the facilities at the Polokwane Smelter (assays of the feed and tailings and underground samples) and Anglo Research Centre (assays of concentrate samples). The turnaround time from Polokwane is about four days and from ARC between 12 and 16 days. This is regarded as inadequate control, irrespective of the plant performance being regarded as good. It is important that results be obtained as soon as possible from routine shift and daily samples, in order that corrective action can be taken immediately, if required.

16.6 Concentrator performance

The recent concentrator performance at Lebowa is summarized below:

16.6.1 Merensky Concentrator

The following are the reported values achieved from the Merensky processing plant:-

- Tons milled — 71 ktpm to 80ktpm (business plan 78 ktpm — capacity 85ktpm
- Head grade — 3.85g/t 4E to 4.15g/t 4E (business plan — 4.33g/t 4E)
- Concentrate mass pull — 2.34%
- Concentrate grade — 185g/t 4E (business plan — 175g/t 4E)
- Concentrate contained chrome level — 0.7% Cr_20_3
- Recovery— 92.4% to 93.0% (business plan — 91%)
- Tailings grade — 0.32g/t 4E to 0.40g/t 4E

Real historical annual data which is available is depicted in Table 16.1.

16.6.2 UG2 Concentrator

The following are the nominal values achieved from the UG2 processing plant:-

- Tons milled from Lebowa only — 55ktpm to 58ktpm (business plan — 68ktpm) as a result of under delivery — capacity 70ktpm
- Head grade — 4.50g/t 4E to 4.81 g/t 4E (business plan — 4.68g/t)
- Concentrate mass pull — 2.16%
- Plant Power draw — 39kWhr/ton and is typical for UG2 ores
- Concentrate grade — 149g/t 4E to 178g/t 4E (business plan — 170g/t 4E) — this is variable and is related to plant operations in obtaining improved recovery at the expense of concentrate grade
- Concentrate contained chrome level (typical for UG2 Conc.) — 4% to 5% Cr_20_3
- Recovery — 87.8% to 89.3% (business plan — 87.0%)
- Tailings grade — 0.52g/t 4E to 0.60g/t 4E

Real historical annual data (sourced from audited LPM Metallurgy Department, partially confirmed by Anglo Platinum Annual Report, 2007) is presented in Table 16.1

Table 16.1: Historical concentrator performance data (Data from Anglo Platinum).

Parameter	Year	Merensky	UG2
Tonnage (000's)	2004	1016	773
	2005	917	692
	2006	957	590
	2007	949	511
	2008	662	600
Head Grade (4E's g/t)	2004	4.74	4.21
	2005	4.66	4.67
	2006	4.29	4.77
	2007	4.33	4.77
	2008	4.03	4.82
Tail Grade (4E's g/t)	2004	0.47	0.56
	2005	0.41	0.57
	2006	0.34	0.59
	2007	0.3	0.52
	2008	0.32	0.59
Conc Grade (4E's g/t)	2004	207	282
	2005	174	237
	2006	150	185
	2007	164	178
	2008	187	200
Concentrator Mass pull %	2004	2.1	1.3
	2005	2.4	1.7
	2006	2.7	2.3
	2007	2.5	2.4
	2008	2.1	2.2

Over the years 2004 to 2008, annual tonnages of Merensky and UG2 ore processed through the plants can be seen to be reducing significantly each year. It was stated that the major reason for this was related to ore supply from mining, rather than any plant short coming.

The lower than usual tonnage milled in 2008 was also attributed to the power supply problems experienced during the year.

The reported values stated by plant management during the site visit are comparable with the actual plant achievement during the last five years as shown in Table 16.1.

Acceptable concentrate grades for the smelting operations are achievable at high recoveries of the PGM metals for both the Merensky and UG2 concentrators.

There has been a general decline in the annualized head grade of the ore processed by the Merensky concentrator from 4.74g/t 4E's in 2004 to 4.03g/t 4E's in 2008. However, there has been a general increase in the head grade of the ore processed by the UG2 concentrator from 4,21g/t 4E's in 2004 to 4,82g/t 4E in 2008. Tailings grades have consistently been well controlled.

Historic operational utilisation efficiencies from 2004 to 2007 were about 90%, and this is considered sustainable for future operations and modelling purposes. The Merensky Concentrator operated at efficiencies near 90% throughout this time period, whereas the UG2 Concentrator operated at efficiencies between 61% and 92% over the same period.

16.6.3 Concentrator 4E metal Recoveries

Table 16.2 summarizes the average audited annual 4E metal recoveries actually achieved by the Merensky and UG2 concentrators for the years from 2004 to 2008.

Table 16.2: Average 4E metal recoveries 2004 to 2008 (Data from Anglo Platinum).

Year	Merensky (%)	UG2(%)
2004	90.6	86.9
2005	91.4	87.9
2006	92.2	87.9
2007	93	89.3
2008	92.5	88.2

The Merensky concentrator recoveries of the 4E'S are good, due to the accommodating mineralogy of the Lebowa ore-body. Typically, PGM's are associated with base metal sulphides and clearly, liberation is very good at the current plant grinds.

The average 4E's recovery over the 5 year period is 91.9% over a range 90.6% to 93.0% (annual recoveries). Deloitte is of the opinion that the Merensky concentrator metal recovery performance is good.

Similarly, the UG2 PGM recoveries are good with a five year average of 88.0% over a range 86.9 to 89.3 (annual recoveries). Deloitte is of the opinion that the UG2 concentrator performance also rates as being one of the best in the platinum industry.

In future, if higher throughput rates are treated by the plants, it can be expected that recoveries may reduce marginally due to reduced milled pulp residence time in the flotation circuits. In addition, it is noted that as UG2 production increases, cognisance has to be taken of the increased overall chrome content for smelting. The ratio of Merensky and UG2 concentrates, in terms of the chrome content, will be affected and may necessitate reducing the chrome content in the UG2 concentrate. This may be at the expense of 4E recovery from the UG2 plant. This aspect would have to be considered when the chrome content specifications and penalties are defined with Polokwane Smelter.

16.6.4 Individual 4E element recoveries

Table 16.3 summarizes current recoveries of individual 4E elements as reported by Anglo Platinum. Current recoveries of Nickel and Copper are given in Table 16.4.

Table 16.3: Individual 4E element recoveries (Data from Anglo Platinum).

Element	Merensky (%)	UG2(%)
Platinum	94.19	88.27
Palladium	94.13	90.26
Rhodium	92.14	87.58
Gold	79.03	78.46

Table 16.4: Nickel and Copper recoveries (Data from Anglo Platinum).

Element	Merensky (%)	UG2(%)
Nickel	66.63	53.97
Copper	76.21	62.45

16.7 Concentrator expansions

Concentrator management stated that the upgrade of the Merensky plant from 85 ktpm to 100 ktpm will proceed according to plan. Most of the major equipment for the expansion is already in the procurement system. However, the original expansions planned for the UG2 plants are being reconsidered due to the global financial problems that currently exist.

The original plan was that two new UG2 concentrators would need to be constructed in the future. Space exists, adjacent to the UG2 plant, for this expansion.

It was planned that the first new UG2 concentrator with a capacity of 1,200 ktpa (100 ktpm) would be commissioned in 2012 and a second UG2 concentrator with a capacity of 1,260 ktpa (105 ktpm) would be commissioned in 2019.

The Merensky Concentrator would treat Merensky ore exclusively, while the second UG2 Concentrator would treat an 80:20 blend of ore from Merensky and UG2 respectively.

16.8 Tailings management and expansion

The Merensky consolidated tailings dam and UG2 tailings dam are operated by Fraser Alexander — a specialist tailings dam operating company, and the competent person for annual review purposes is SRK Consulting.

The current tailings dams have a combined capacity of 170 ktpm at a maximum rate of rise of 2.5 m per annum. This is adequate for current production.

It is anticipated that the existing Merensky tailings dam will run out of capacity in 2012, and a new dam will need to be in operation by that date. The new site is planned to have a 30 year capacity at 100 ktpm.

The existing UG2 tailings dam has a capacity for 145 ktpm and a life of 23 years.

There are physical provisions for future tailings dams, known as Site B for Merensky and Site D for the UG2, these tailings storage facilities will be adequate for the LOM.

The tailings dam design includes the provision of adequate surface area to limit the rate of rise and to facilitate sun-drying and the desiccation of tailings. This enables the walls to be constructed with tailings material only - and not with rock, which reduces the potential for liquefaction.

The Merensky consolidated tailings dam has had water sprayers installed around the top edges of the dam to provide for dust-allaying. The walls of the dam have also been partially grassed with limited effect.

According to the management of LPM, as the mines release a significant quantity of water, there is adequate water for dust allaying, by spraying the tailings dam. However, this is not considered a permanent solution and further investigation into vegetation of the Merensky consolidated tailings dam is required.

16.9 Metallurgical testing

The Merensky and UG2 Concentrators have been operating since 1991 and 2000, respectively. From time to time, various metallurgical tests have been carried out to identify how the process may be made more efficient.

In 2006/7, an investigation was undertaken by Anglo Platinum to optimize the grind of the Merensky Concentrator primary milling circuit, and specifically, to determine the particle size range for optimal flotation. Also, an investigation was undertaken into the technical and practical feasibility of the use of rubber liners, rather than steel liners, in the Merensky concentrator primary mill.

The indications were that after mill modification, appropriate selection of the ball size and mill load may improve the mill grind.

The recommendations of this investigation were:

- The ball size should be increased to a nominal diameter of at least 75 mm.
- The mill ball-load should be increased to at least 35%.

The above recommendations are expected to improve the mill grind and facilitate better primary flotation. Appropriate testing will be conducted on completion of the modification.

Other recommended upgrades include:

- Increasing the size of selected conveyor belt motors to increase rock handling capacity, and subsequently the rate of plant throughput.
- Installing steel capped (Hardox 500) plated rubber liners, instead of normal rubber liners. (The normal rubber liners were shown to be uneconomical in other plants).
- Currently, metallurgical test work is being conducted on the optimization of small diameter cluster cyclones, rather than the existing single large diameter cyclones, between the Merensky plant rougher flotation and milling stages. The small diameter cluster cyclones have already been installed and are currently being commissioned. Spigot sizes, vortex finder sizes and cyclone feed pressures are currently being investigated.
- Reagent optimization is ongoing test work.

- The influence of the particle size distribution of the grind on concentrate grade is currently under investigation.

Other metallurgical test work is carried out from time to time either by LPM metallurgical staff, Anglo Platinum metallurgical consultants and/or in conjunction with Anglo Research, and/or private companies specializing in PGM metallurgy.

16.10 Sale of Concentrate Agreement

All of Lebowa's Concentrate is currently supplied to Anglo Platinum's Polokwane Smelter Complex pursuant to an existing sale of concentrate agreement between LPM OpCo and Anglo Platinum (through RPM). This agreement, effective 1 January 2008, requires that LPM OpCo is obliged to sell (and RPM is obliged to purchase) all concentrate produced from the LPM properties, through the Merensky and UG 2 Concentrators.

This concentrate sale agreement will continue for an initial period of five years with an option for LPM HoldCo to renew this agreement for a further five year period after the first four years of the initial five year term.

16.11 Prices and Charges

The refiner (RPM) will pay LPM HoldCo monthly for LPM concentrate.

The price payable will be based on a fixed market-related percentage of the equivalent ZAR price for the various metals based on the preceding month, taking into account the costs of smelting and refining incurred by RPM.

16.12 Penalties

As is common practice in concentrate sale agreements, various penalties, relating to concentrate not meeting agreed specifications, are provided for in the off-take agreement, and are deductible from the price payable for concentrate. These include penalties for excess concentrate moisture or chrome content as well as concentrate grade below specified values of g/ton (4E).

The Sale of Concentrate Agreement makes allowance for various other commercial terms, commonly applicable in a market-related concentrate sale agreement of this nature, including the annual inflationary escalation of charges and penalties according to various South African and United States of America rates.

The agreement also includes provision for a Joint Evaluation Committee for purposes of agreeing on the weighing, sampling and assaying of concentrate and other commercial issues are provided for such as force majeure and dispute resolution, amongst others.

16.13 Conclusions

No major problems or flaws with the basic process flow sheets of either the Merensky or UG2 concentrators at LPM are evident. By modern technological standards, the concentrators employ older technology, but nevertheless maintain high operating efficiencies and availability. In general, the plants operate well with few major equipment breakdowns. The most common reasons for current plant stoppages are due to lack of ore from the shafts and planned maintenance. The Merensky Concentrator historically (2004 to 2007) operated at utilization efficiencies near 90%, whereas the UG2 Concentrator efficiencies varied between 61% and 92% over the same time period.

The Merensky Concentrator recoveries of the 4E'S are good, due to the accommodating mineralogy of the Lebowa ore-body. Typically, PGM's are associated with base metal sulphides, and clearly, liberation is very good at the current plant grinds.

The average 4E's recovery over the past 5 year period (2004 to 2008) is 91.9%, over a range 90.6% to 93.0% (annual recoveries). Deloitte is of the opinion that the Merensky Concentrator metal recovery performance is good.

Similarly, the UG2 PGM recoveries are good with a five year (2004 to 2008) average of 88.1%, over a range 86.9% to 89.3% (annual recoveries). The concentrate grades here too are acceptable for smelting. Deloitte is of the opinion that the UG2 concentrator performance is good.

There has been a general decline in the annualized head grade of the ore processed by the Merensky concentrator from 4.74g/t 4E's in 2004 to 4.17g/t 4E's in 2008. However, there has been a general increase in the head grade of the ore processed by the UG2 concentrator from 4.21 g/t 4E's in 2004 to 4.85 g/t 4E in 2008. Both concentrators' tailings grades have consistently been well controlled.

The Merensky concentrator is currently undergoing upgrading and modernization, planned for implementation in the 2009 financial year. The modernization and technological upgrade will improve the plant throughput capacity from 85 ktpm to 100 ktpm. The fully autogenous primary grinding (FAG) mill will be replaced by a semi autogenous grinding mill (SAG), and the current process control system will be improved. The indications are that the current flotation plant equipment capacity will be adequate to meet the planned 100 ktpm expansion. However, appropriate modifications to some pipes and valves will be required.

The current rated capacity of the existing UG2 concentrator at 70 ktpm will be maintained.

In future, if higher throughput rates are to be treated by the existing plants, it can be expected that PGM recoveries may reduce marginally, due to reduced milled pulp residence time in the flotation circuits. In addition, it is noted that as UG2 production increases, cognisance has to be taken of the increased overall chrome content for smelting. The ratio of Merensky and UG2 concentrates, in terms of the chrome content, will be affected and may necessitate reducing the chrome content in the UG2 concentrate. This may be at the expense of 4E recovery from the UG2 plant. This aspect would have to be considered when the chrome content specifications and penalties are defined with Polokwane Smelter.

No significant technological and/or operational issues or major problems were identified with the concentrators by Deloitte. They are considered to be designed and operated in accordance with sound Engineering principles. The basic process flow design and equipment selection relevant to both the Merensky and UG2 plants are considered to be completely satisfactory, albeit they exhibit older technology and equipment, in relation to modern technology.

The Merensky concentrator and UG2 concentrator have been operating since 1991 and 2000, respectively. Metallurgical research and development and test work has been conducted on a continuous basis to improve the efficiency of the concentrators.

Concentrator management indicated that the upgrade of the Merensky plant from 85 ktpm to 100 ktpm will proceed according to plan, since most of the major equipment for the expansion is already in the procurement system. However, the original expansions planned for the UG2 plants are in question, due to the global financial problems that currently exist.

Over the years 2004 to 2008, annual tonnages of Merensky and UG2 ore processed through the plants have reduced. This has been attributed to ore supply from mining, rather than any plant short coming.

The general condition of both the Merensky and UG2 concentrators at LPM is regarded by Deloitte as fair and reasonable. The process equipment, concrete work, structural steel and sheeting are generally in good condition. Where there has been evidence of damage, maintenance and repair work has accordingly been conducted.

There appears to be adequate stocks of spares with regard to maintenance items such as flotation equipment, conveyor belting, mill liners, mill major items (mill ends, bearings, motors, gearboxes, etc.) and Larox filter equipment.

Provided that basic safety issues are addressed by the staff and management at the LPM concentrators, there is no reason to believe that safety will become a major issue or concern.

It is anticipated that the existing Merensky tailings dam will run out of capacity in 2012, and a new dam will need to be in operation by that date. The new site is planned to have a 30 year capacity at 100 ktpm.

The existing UG2 tailings dam has a capacity for 145 ktpm and a life of 23 years.

A dust problem exists during periods of high winds with the current Merensky dam, albeit that water spraying and vegetation of the upper edges of the dam wall are being implemented.

17 Mineral Resource and Mineral Reserve estimates

The Mineral Resource and Mineral Reserve figures reflect the position as at 31 December 2008 by allowing for depletion, losses and any additions which have occurred following the last update of the Mineral Resource and Mineral Reserve estimates in December 2007 (Tables 17.1 and 17.2). Neither the 2007 nor the 2008 Resource and Reserve estimates were externally audited. The 2007 Resource and Reserve estimates were reviewed by Snowden (2008).

Table 17.1: December 2007 Mineral Resources (Snowden, 2008).

Category	Tonnage (Mt)	4E grade (g/t)	4E contained metal (Oz)	Pt grade (g/t)	Pd grade (g/t)	Rh grade (g/t)	Au grade (g/t)
Merensky							
Measured	**25.0**	**5.68**	**4.57**	**3.65**	**1.51**	**0.21**	**0.30**
Indicated	**27.4**	**5.51**	**4.68**	**3.46**	**1.52**	**0.20**	**0.33**
Measured and Indicated	**52.4**	**5.61**	**9.43**	**3.55**	**1.52**	**0.20**	**0.32**
Inferred	**103.2**	**5.30**	**17.58**	**3.34**	**1.45**	**0.20**	**0.31**
UG2							
Measured	**107.6**	**6.60**	**22.84**	**2.70**	**3.23**	**0.55**	**0.12**
Indicated	**71.3**	**6.56**	**15.32**	**2.70**	**3.20**	**0.53**	**0.13**
Measured and Indicated	**178.9**	**6.58**	**38.16**	**2.70**	**3.22**	**0.54**	**0.12**
Inferred	**145.0**	**6.61**	**30.82**	**2.72**	**3.23**	**0.53**	**0.13**

Notes: Mineral Resources are exclusive of Mineral Reserves.

- Tonnes and ounces have been have been rounded and this may have resulted in minor discrepancies.
- The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh), and gold (Au).
- Oxidised zones and all geological losses have been excluded from the resources.
- Measured and Indicated Mineral Resources are generally located within 650 m below the surface.
- Cut-off grades of 2.4 to 3.5 g/t 4E depending on reef characteristics are applied to Merensky Mineral Resource statements.
- A cut-off grade of 1.8 g/t 4E is applied to UG2 Mineral Resource statements.

Snowden (2008) concludes in their review that the 2007 Mineral Resource and Reserve estimates are in accordance with SAMREC and the CIM definition standards.

Both the 2007 and 2008 Mineral Resource and Reserve estimates were compiled by Anglo Platinum personnel, and have been stated to be in accordance with the Australasian Code for the Reporting of Mineral Resources and Mineral Reserves (The JORC Code, 2004) as a minimum standard and in compliance with The South African

Code for Reporting of Mineral Resources and Mineral Reserves, the SAMREC Code, 2007, in Anglo Platinum's Competent Persons Sign-off Report, dated 30 November 2008.

The 2008 Resource estimate is based on the 2007 geological model, as drilling in 2008 did not result in material changes to the geological model. The 2008 Reserve estimate was derived, as was the 2007 Reserve estimate, by applying modifying factors to the resource.

Table 17.2: December 2007 Mineral Reserve Statement (Snowden, 2008).

Category	Tonnage (Mt)	4E grade (g/t)	4E contained metal(Oz)	Pt grade (g/t)	Pd grade (g/t)	Rh grade (g/t)	Au grade (g/t)
Merensky							
Proven	**23.1**	**4.25**	**3.20**	**2.62**	**1.20**	**0.15**	**0.28**
Probable	**5.4**	**4.06**	**0.70**	**2.50**	**1.12**	**0.16**	**0.28**
Total Reserve	**28.5**	**4.22**	**3.90**	**2.59**	**1.19**	**0.16**	**0.28**
UG2							
Proven	**34.1**	**5.29**	**5.80**	**2.18**	**2.57**	**0.44**	**0.10**
Probable	**9.4**	**5.04**	**1.50**	**2.11**	**2.39**	**0.44**	**0.09**
Total Reserve	**43.5**	**5.23**	**7.30**	**2.17**	**2.53**	**0.44**	**0.10**

Notes: Mineral Reserves are exclusive of Mineral Resources.

- Tonnes and ounces have been rounded and this may have resulted in minor discrepancies.
- The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh), and gold (Au).
- Only Measured and Indicated Resources have been converted to the Mineral Reserves.
- Mineral Reserves have been determined by applying modifying factors to the Mineral Resource.
- Mineral Reserves have been depleted by current mining since December 2007.
- The Merensky pay limit varies between 1.3 and 4.8 g/t 4E across all operations of Anglo Platinum. The UG2 pay limit varies between 1.3 and 4.4 g/t 4E across all operations of Anglo Platinum. The variability is a function of various factorsincluding depth of the orebody, geological complexity and infrastructure.

A new mine plan will form the basis of the 2009 Life of Mine plan and the 2009 Reserve estimate. As the revised mining plan is currently being designed and scheduled, detailed parameters cannot as yet be derived.

17.1 The Mineral Resource and Mineral Reserve statement, December 2008

Mineral Resource and Mineral Reserve estimates are reported for the LPM properties including the existing Lebowa Mine, and planned project expansions at Brakfontein and Middelpunt. Mineral Resources (Remaining Resources) are also reported for the regions outside of the existing mine plans and project expansion plans but within the LPM properties.

17.1.1 Estimation methodology

The Mineral Resources were classified according to the standardised Anglo Platinum methodology, which includes the following considerations:

- Kriging variance and kriging efficiency,
- Range of the semi variogram,
- Borehole and sample density distribution,
- Geological framework,
- Aeromagnetic survey if available,
- Mining history,
- Risk assessment,
- Competent Persons' (SAMREC equivalent to a Qualified Person) assessment, and
- Conditional simulation.

Where conditional simulation risk analysis has been performed, the risk shall be scaled to be consistent with a 12-month production period and determined at 90% confidence limits, using the following guidelines:

- A Mineral Resource is consistent with the Measured Resource category when the scaled risk associated with the accumulated metal estimate is less than 10% (inclusive).
- A Mineral Resource is consistent with the Indicated Resource category when the risk associated with the accumulated metal estimate is between 10% and 20% (inclusive).
- A Mineral Resource with a risk associated with the accumulated metal estimate greater than 20% is consistent with an Inferred Resource.

The Mineral Resources quoted are over an economic and mineable resource cut as appropriate to the specific ore deposits. The Mineral Resources quoted are in addition to those resources that have been converted to produce Mineral Reserves and are based on auditable geological sampling, databases, modelling and estimation methodologies according to the standardised Anglo Platinum methodology. Mineral Resources are reported after all known and expected geological losses have been applied.

According to Anglo Platinum policy, where mine plans exist, the relevant Measured Mineral Resources are transformed in part or wholly to Proved Mineral Reserves and the Indicated Mineral Resources are transformed to Probable Mineral Reserves. Where uncertainties exist with regard to the conversion factor or mine design the Measured Mineral Resources may only be converted to Probable Mineral Reserves.

17.1.2 Definition of pay limits based on economic criteria

Based on the global assumptions as per third quarter of the year, i.e. September 2008, (see Table 17.3), the 4E pay limits for the UG2 and Merensky Reef were calculated and are summarised in Tables 17.4 and 17.5.

Table 17.3: Global assumptions as per third quarter such as commodity prices and exchange rates (after Anglo Platinum, 2008).

FY 2008/2009 Real money terms

Year	R/US$		Metal prices US$ /oz (real)				Metal prices (US$/lb)		
	Nominal	Real	Pt	Pd	Rh	Au	Ni	Cu	Co
2008	**7.30**	**7.30**	**1300**	**270**	**3500**	**850**	**8.00**	**3.00**	**30.00**

Note: The cost (working) figures used were based on September 2008 costs. The revised costs following the global metal prices fall were not available at the time of compilation of this TR. Therefore, Deloitte recommends the paylimit determination study be revised using the most recent acceptable input parameters for the commodity prices.

Table 17.4: Pay limits for UG2 4E.

UG2 4E Pay Limits

Source	Marginal Pay Limit			Shaft Head Pay Limit			Planning Pay Limit		
	Tonnes	g/t	Contents	Tonnes	g/t	Contents	Tonnes	g/t	Contents
Recovered		1.32	42 311 063		1.83	58 620 664		2.40	76 929 205
Final Floatation Tails		0	0		0	—		0	—
Milled	32 062 513	1.00	42 311 063	32 062 613	1.83	58 620 664	32 062 513	2.40	76 929 205
Shortfall/Excess	0			—			—		
Survey called for (generated)	32 062 513	1.39	44 537 961	32 062 513	1.92	6 175 962	32 062 613	2.53	80 978 111
Back area sweepings	0	0	0	—	0	—	—	0	—
Development on Reef	836 456	3.14	2 625 179	836 456	3.14	2 625 179	836 456	3.14	2 625 179
From stoping	31 226 057	1.34	41 912 782	31 226 057	1.89	59 080 784	31 226 057	2.51	78 352 932
Waste from stopes	1 644 524	0.50	817 312	1 644 524	0.50	817 312	1 644 524	0.50	817 312
From stope faces	29 581 533	1.39	41 095 470	29 581 533	1.97	58 263 471	29 581 533	2.62	77 535 620
Pay limits (g/t)		**1.39**			**1.97**			**2.62**	

Table 17.5: Pay limits for Merensky 4E.

Merensky 4E Pay Limits

	Marginal Pay Limit			Shaft Head Pay Limit			Planning Pay Limit		
Source	Tonnes	g/t	Contents	Tonnes	g/t	Contents	Tonnes	g/t	Contents
Recovered		1.46	31 651 688		2.27	49 316 197		2.65	57 548 523
Final Floatation Tails		0	—		0	—		0	—
Milled	21 711 968	1.46	31 651 688	21 711 988	2 27	49 316 197	21 711 988	2.65	57 548 523
Shortfall/Excess	1 150 735			1 150 735-			1 150 735		
Survey called for (generated)	20 561 253	1.62	33 317 566	20 561 253	2 52	51 911 786	20 561 253	2.95	60 577 392
Back area sweepings	—	0	—	—	0	0	0	0	—
Development on Reef	1 364 215	2.16	2 942 537	1 364 215	2.16	2 942 357	1 364 215	2.16	2 942 357
From stoping	19 197 037	1.58	30 375 208	19 197 037	2.55	48 969 429	19 197 037	3.00	57 635 035
Waste from stopes	1 870 199	1.50	2 800 132	1 870 199	1.50	2 800 132	1 870 199	1.50	2 800 132
From stope faces	17 326 838	1.59	27 575 076	17 326 838	2.66	46 169 297	17 326 838	3.16	54 834 903
Paylimits (g/t)		1.59			2.66			3.16	

17.1.3 The Mineral Resource and Mineral Reserve statement, December 2008

Due to the lack of any material effect of the additional drilling during 2008, the Geological Model compiled for the 2007 resource estimation was re-used for the 2008 resource statement. The 2007 model was, however, depleted, i.e. the areas mined out during the course of 2008 were subtracted from the model. This depleted 2007 Geological Model was therefore used to constrain the 2008 Block Model.

In addition, the on-mine Block Model is routinely updated as part of other ongoing processes within the mine. When the Block Model was created for the 2008 Resources and Reserves, it therefore included information that was obtained in 2008, such as the MRM samples (face/channel samples).

The structural model, which is also used in constructing a Block Model, was revised in 2008. The most notable revision was that the size and shape of the Brakfontein Pothole was revised on the basis of drilling conducted in 2008. A revision of the size and shape of the Brakfontein Pothole resulted in some changes to the Resources and Reserves calculated over the block in which the pothole occurs.

Lastly, dip of a reef horizon within a block controls the ratio between the apparent thickness and the true width of the reef within the block. The true width is one of the factors that controls the volume, and hence the tons, represented by the block. The dips for the UG2 and Merensky Reef used in the 2008 Resource and Reserve Statement were assigned to the blocks in the Block Model on the basis of the farm in which this block was located, i.e. a single dip value was assigned to all blocks falling within a particular farm. However, this practice changed very recently with the assigned dip value for each block in the Block Model interpolated individually, i.e. each block was assigned a unique dip value. The results of the study based on this particular change were available at the date of writing of this report.

Mineral Resource and Mineral Reserve estimates are reported (see Tables 17.6 and 17.7, respectively) for the Lebowa properties including the existing Lebowa mine, and planned project expansions at Brakfontein and Middelpunt. Mineral Resources (Remaining Resources) are also reported for the regions outside of the existing mine plans and project expansion plans but within the Lebowa properties.

Table 17.6: December 2008 Mineral Resources.

Category	Tonnage (Mt)	4E grade (g/t)	4E contained metal (Moz)	Pt grade (g/t) (kriged)	Pd grade (g/t) (kriged)	Rh grade (g/t) (kriged)	Au grade (g/t) (kriged)
Merensky Reef							
Measured	**25.92**	**5.64**	**4.71**	**3.63**	**1.50**	**0.21**	**0.30**
Indicated	**27.39**	**5.51**	**4.85**	**3.46**	**1.52**	**0.20**	**0.33**
Measured and Indicated	**53.31**	**5.58**	**9.56**	**3.54**	**1.51**	**0.20**	**0.32**
Inferred	**102.89**	**5.30**	**17.53**	**3.34**	**1.45**	**0.20**	**0.31**
UG2 Reef							
Measured	**108.47**	**6.60**	**23.03**	**2.70**	**3.23**	**0.55**	**0.12**
Indicated	**71.91**	**6.56**	**15.18**	**2.70**	**3.20**	**0.53**	**0.13**
Measured and Indicated	**180.38**	**6.58**	**38.21**	**2.70**	**3.22**	**0.54**	**0.12**
Inferred	**145.00**	**6.61**	**30.82**	**2.72**	**3.23**	**0.53**	**0.13**

Notes: Mineral Resources are exclusive of Mineral Reserves.
Tonnes and ounces have been rounded and this may have resulted in minor discrepancies.
The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au).
A zero cutoff grade was applied.
The UG2 Resources include areas of bifurcated UG2.
The Mineral Resources were calculated by Anglo Platinum (Mafoko and Hanekom, 2008).

Table 17.7: December 2008 Mineral Reserves.

Category	Tonnage (Mt)	4E grade (g/t)	4E contained metal (Moz)
Merensky Reef			
Proven	**21.71**	**4.34**	**3.03**
Probable	**5.43**	**4.16**	**0.73**
Total Reserve	**27.14**	**4.31**	**3.76**
UG2 Reef			
Proven	**32.1**	**5.43**	**5.60**
Probable	**9.1**	**5.17**	**1.50**
Total Reserve	**41.2**	**5.37**	**7.10**

Notes: Mineral Reserves have been determined by applying modifying factors to the Mineral Resources.
Tonnes and ounces have been rounded and this may have resulted in minor discrepancies.
The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au).
Only Measured and Indicated Resources have been converted to Mineral Reserves.
Mineral Reserve grade is based on the hoisted ore grade.
The Mine Call Factors used in the Mineral Reserve calculations for both reefs are 97% and 98% for the Proven and Probable Reserves, respectively.
However, Deloitte notes that the agreed Mine Call Factor for the UG2 Reef was 94%.
Recent changes in the economic commodity market are not considered.
The Mineral Reserves were calculated by Anglo Platinum (Mafokoand Hanekom, 2008).

17.1.3 Summary

For both reefs the 2007 resource model was used in the 2008 estimations since there were insignificant additions or changes. A limited amount of drilling was effected. Most of the drilling was from underground.

Surface drilling has revealed that the Brakfontein Pothole is slightly bigger than previously estimated and this pothole intersected at 4.0 Raise and 6 East 18 Raise contributed to the slight decrease in tonnage. Underground development at 1.20 East Raise has shown that the UG2 pothole in this area was slightly bigger than estimated which also contributed to the slight decrease in the UG2 Mineral Resource.

UG2 Reef: The Measured Mineral Resource decreased by 1.04% (1.13 Mt) whilst there has been a minimal decrase of Indicated Mineral Resource 1.07% (0.77Mt). There is a slight tonnage decrease and the grade remains the same at 6.59 g/t for the remaining Mineral Resources from 327 Mt at 6.59 g/t to 325.38 Mt at 6.60 g/t. Insignificant changes in reserve tonnage resulted in a slight increase in grade from 5.23 g/t to 5.37 g/t. The differences are mainly due to the application of a depletion factor.

Merensky Reef: The Measured Mineral Resource decreased by 1.93% (-0.50Mt) whilst there has been a minimal decrease of Indicated Mineral Resource 0.15% (-0.04Mt). The conversion of the additional resources to ore reserves resulted in a slight decrease in resource cut and the grade remains the same at 5.30 g/t for the remaining Mineral Resources from 157 Mt to 156 Mt.

The Measured Mineral Resource decreased by 1.93% (0.05 Mt) whilst there has been a minimal decrease of Indicated Mineral Resource 0.15% (0.04 Mt). Insignificant changes of a slight increase of 0.1 g/t in Reserve grade with a straight forward depletion decrease in tonnages from 28.5 Mt to 27.1 Mt.

In summary, the 2007 and 2008 Resource and Reserve statements have been prepared by in-house Anglo Platinum personnel. Neither statement has been externally audited.

In Deloitte's opinion, the terms, Measured, Indicated and Inferred Mineral Resources, as used in this report are consistent with the CIM Definition Standards and the SAMREC Code

The 2008 Reserve estimates, as stated in the Anglo Platinum Annual Report, are based on applying modifying factors to the mineral resources, and are not derived from a scheduled mining plan. Although mining plans for both reefs do exist, these were not used during reserve estimation.

17.2 Disclosure

Mineral Resources reported in Section 17 of this TR were prepared by Anglo Platinum's personnel. Mineral Resources that are not Mineral Reserves have a reasonable prospect of economic extraction, but have not yet been demonstrated as viable by a mining plan within an approved project.

17.2.1 Lebowa Competent Persons procedure

It is the responsibility of the Mine Technical Services (MTS) Manager at Lebowa to ensure that the processes defined in the Anglo Platinum procedures are appropriately carried out. The Mine Geological Services, the MTS and the Exploration and Mineral Strategy departments of Anglo Platinum provided the necessary technical support services and assistance for the 2008 Mineral Resource and Mineral Reserve estimates.

The Anglo Platinum companies responsible for the Lebowa mines, projects and other independent services have ensured that technical teams responsible for the preparation of the Mineral Resource and Mineral Reserve statements in this TR were managed by suitably qualified Competent Persons.

17.2.2 Minimum standards

The following minimum standards are used as the basis for reporting of Mineral Resources and Mineral Reserves on a mine or project basis.

A statement to demonstrate competence and compliance with the policy/relevant reporting codes must accompany the estimated Mineral Resources and Mineral Reserves.

The company responsible for the mine or project must officially sign-off on the statements of the estimated Mineral Resources and Mineral Reserves to Anglo Platinum Exploration and Mineral Strategy Department by the end of November, each year.

17.2.3 Lebowa Competent Person(s) for Mineral Resource estimates

S Malenga – Chief Geologist, Lebowa Platinum Mines; B.Sc. (Hons.) in Geology, SACNASP No. 400181/06, plus five years relevant experience. He was responsible for the Mineral Resource modelling, for the resource model acceptance, for the mine geological databases (Sable, MRM and structural components) and management of the resource data. Due to his transfer to another mine, P Stevenson is the responsible Competent person and Signing-off on behalf of Lebowa Platinum Mines during 2008.

P Stevenson – Manager Evaluation, B.Sc. in Geology, SACNASP No. 400433/04 plus 14 years platinum mine experience.

The following technical specialists were involved in the preparation of the Mineral Resources and have appropriate experience in their field of expertise to the activity that they are taking and consent to the inclusion in the report of the matters based on their technical information in the form and context in which it appears.

D Siboza and C Radzivhoni – Shaft Geologists, Lebowa Platinum Mines. They were responsible for underground geology, logging, collecting and interpretation of the data.

D Nowak – Mineral Resource Manager. He was responsible for the resource modelling and verification.

D Jarman – Project Geologist. He was responsible for logging and validation of borehole information, and

A Mokhasipe – Chief Geologist, Lebowa Platinum Mines. He was responsible for underground geology, logging, collecting and interpretation of data including validation of borehole information.

The above people have sufficient experience relevant to the style and type of mineral deposit under consideration and to the activity which is being undertaken to qualify as Competent Persons as defined in the SAMREC Code.

17.2.4 Lebowa Competent Person(s) for Mineral Reserve estimates

D M Mafoko – Chief Surveyor, Lebowa Platinum Mines; NHD Mineral Resource and MSCC, PLATO MST 0075, plus eleven years gold and three years platinum mine experience.

K Hanekom – Senior Mining Engineer, NDT Metal Mining, MMC, SAIMM 703110, plus more than thirty years gold and one year platinum mine experience. He was responsible for the conversion of Mineral Resources to Mineral Reserves.

The above people have sufficient experience relevant to the style and type of mineral deposit under consideration and to the activity which is being undertaken, to qualify as a Competent Person as defined in the SAMREC Code.

17.2.5 Technical specialists for Mineral Reserve estimates

The following technical specialists were involved in the preparation of the Mineral Reserve estimates and have appropriate experience in their field of expertise to the activity that they are undertaking.

J A Swanepoel – Shaft Valuator, Lebowa Platinum Mines. He was responsible for the historical data collation and for the definition of modifying factors.

D M Mafoko – Chief Surveyor, Lebowa Platinum Mines. He was responsible for the conversion of Resources to Reserves.

K Hanekom – Senior Mining Engineer. He was responsible for the mineral extraction studies.

17.2.6 Review process

Internal peer reviews and audits of the Mineral Resources and Mineral Reserves presented in this TR were conducted by Mr. P Stevenson, B.Sc. (Geology) Witwatersrand University, Comp Dipl., Pr.Sci.Nat. No 400433/04, of Anglo Platinum, with the following conclusions:

As more information becomes available, the methodology of modelling the UG2 with bifurcation may need to be re-visited. The erratic Merensky Reef footwall grade needs to be investigated for its economic potential.

The Mineral Resource and Mineral Reserve estimation as presented in this report have been undertaken by subordinates and their work has been reviewed and has been accepted as being a true reflection of the Mineral Reserves and Mineral Resources of the Lebowa area which has been undertaken in accordance with the AAplc Policy for Reporting of Mineral Resources and Mineral Reserves.

Divisional Technical Reviews were conducted by Anglo Platinum, Alan Field, Acting Head of Mining Technical Services, Theo Pegram, Head of Mine Geological Services and Ron Hieber, Head of Exploration and Mineral Strategy with the following conclusions:

The Ore Reserves and Mineral Resources presented in this report have been reviewed and have been accepted as being a true reflection of the Ore Reserves and Mineral Resources on the Lebowa section in accordance with the AAplc Policy for Reporting of Mineral Resources and Mineral Reserves.

17.3 Merensky Mineral Resource estimation

The Merensky Mineral Resource estimates for precious and base metal grades, thickness and density take a practical mining width cut into account. The total 4E PGM grade is the summation of the individual prill split grades for Pt, Pd, Rh and Au.

17.3.1 Modelling overview and estimation criteria

The modelling procedure adopted was as follows:

- The overall dip per farm was used to calculate the true reef thickness composites.
- Modelling was completed using Datamine mining software in two dimensions.
- The validated boreholes and underground samples were combined, compared and investigated geostatistically in order to characterise and optimise the estimation process.
- Only boreholes were utilised during grade determination.
- The following cuts were investigated (see Figure 17.1):
  - The Merensky Reef channel,
  - A fixed 10 cm. hangingwall cut, and
  - Fixed 25 cm/35 cm/40 cm/50 cm footwall cuts.
- The entire area of interest was kriged.
- Cell dimensions of 250 m by 250 m were used within and immediate adjacent to workings. Sample points had to be either contained within the cell or up to the boundary of the cell for a 250 m cell definition. All other cells were defined as 500 m by 500 m. Drill grids in down-dip areas approximate 500 m intervals.
- Reef grade and thickness were modelled over the Merensky Reef.
- The Merensky Reef channel thickness shows a gradual increase in thickness from the NW to the SE on Lebowa Mine. In order to remove this trend, a boundary along the easting co-ordinate line -112600 was applied (see Figure 17.5).
- The resource tonnage is calculated using:
  - Kriged density,
  - Dip correction factor, and
  - Geological loss factor.
- For the mining cut, the individually kriged hangingwall and footwall cuts were combined and weighted according to the kriged density and the thickness to produce the kriged resource cut/grade/thickness/density/prill grade/Cu-Ni grade.
- The underground samples were only used for the modelling of the reef thickness since the samples do not have any prill data, resulting in their exclusion in the modelling of the PGE grade. The prills were individually modelled and combined at the end of the PGE resource cut grade.
- A minimum of 7 and a maximum of 30 samples were required within the search ellipse for interpolation. This was kept the same for all variables.

Figure 17.1 shows the amount of footwall mining necessary to achieve a minimum mining cut of 85 cm in the west and to achieve the optimum economic cut in the east.

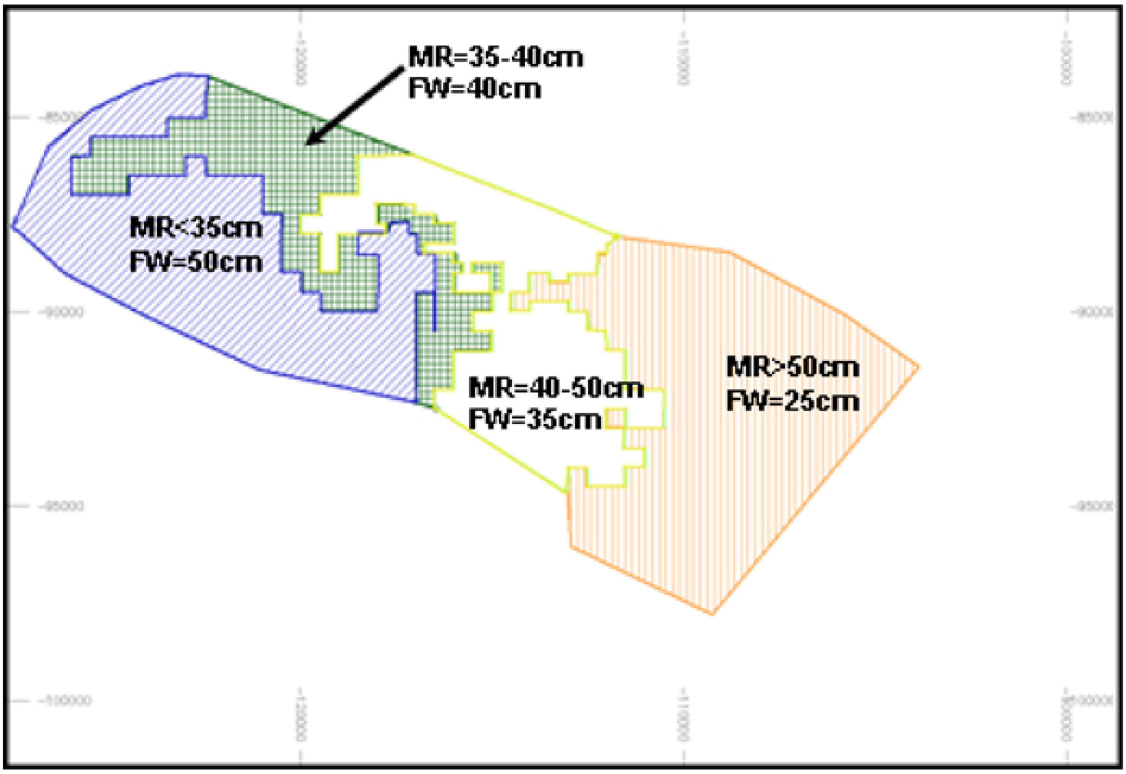


Figure 17.1: Footwall thickness distribution used for modelling (25 cm, 35 cm, 40 cm and 50 cm fixed footwall cut).

Geotechnical considerations

The geological structure of the mine is relatively simple, being unaffected by high frequency of faults, folds and dykes. The hangingwall of the Merensky Reef can be affected by a calcite break situated about 1.2 m above the top reef contact for the majority of the lease area.

Therefore, no geotechnical considerations were necessary for consideration during resource estimation due to the absence of chromitite stringers or other sharp lithological contacts located in the direct hangingwall of the Merensky Reef.

Data sources

The Mineral Resource Management (MRM) underground sampling database and the Sable borehole database were used for Resource estimation. The Sable database has provision for storing prill, density and base metal analyses whereas the underground sample sections (MRM) can only store 4E grades. Since the previous estimate, no new underground sampling data was available due to a delayed start-up of the new Eastern Bushveld Robotic Laboratory (EBRL).

Historically, boreholes contain differences in stored information, with the newer boreholes having prill/density data compared to the holes drilled prior to 2000. In addition, older boreholes did not always have their deflections (wedges) sampled. Borehole data has been subject to several comprehensive validation stages in Sable, Excel, and Datamine. Erroneous data is recorded and noted for correction and is discarded from the database prior to Resource estimation.

Coordinate conversion, PGM correction factor and Density

All data used was converted to WGS84 (LO31) format in 2003.

No PGM correction factor was applied to the borehole or underground sample values.

Where samples have missing density values the mean density values per rock type were assigned and then used during compositing and estimation. Assigned values were not used for statistical analysis or variogram modelling.

17.3.2 Merensky – Borehole distribution plan

Surface drillholes (see Figure 17.2) are distributed across the Lebowa mining licence area, with a closer drill grid spacing across the Brakfontein property. This is due to the targeting of the Brakfontein Merensky project and its associated study level requirements for obtaining higher confidence levels. The deeper areas have appropriately increased the drill grid spacing and are confirming the presence of the Merensky horizon.

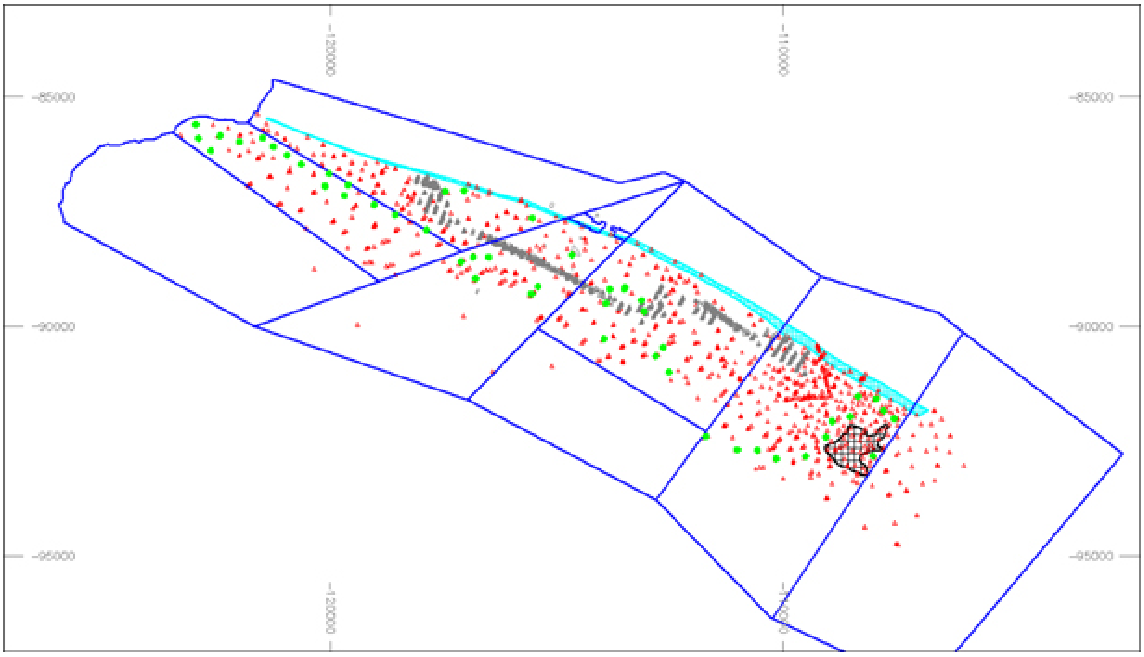


Figure 17.2: Additional boreholes drilled since 2006 are shown in green (Anglo Platinum, 2008).

The 2007 Merensky Reef Resource estimate was based on 738 borehole intersections and 1 025 MRM channel intersections. Table 17.8 details the changes to the drillhole and MRM database numbers since the 2006 estimate. The number of boreholes drilled during 2008 was not available at the time of writing of this TR.

Table 17.8: Merensky Borehole and MRM data.

	2007	New	2006	Change (%)
Parent boreholes	366	64	302	21.2
Borehole intersections	738	148	590	25.1
MRM channels	1 025	0	1 025	0

Boreholes without assays

In 2007, 44 borehole intersections did not have assay data due to pending assay results, intersections to be used for metallurgical analysis, geotechnical holes and historical intersections which were never assayed.

Pothole intersections

Of the available borehole data in 2007, a total of 47 borehole intersections were identified to have intersected potholes and 127 were identified to have intersected pothole edges. These intersections were excluded from the resource estimate.

17.3.3 Merensky Reef – Statistical analysis

The Merensky sample data has been considered for a number of scenarios or ‘Resource cuts’.

The mean statistical parameters of the data for the various layers and mining cuts are presented in Table 17.9. Statistical analysis demonstrates continuity of the sample data per domain and supports the use of the ordinary kriging estimation technique, with single, normal data distributions. For example, Figure 17.3 shows hangingwall PGE cut grade distribution within the 10 cm cut. The hangingwall PGE cut grade decreases from the NW to the SE. A local high-grade anomaly in the Zeekoegat area exists. However, strong grade zonations are present.

PGM accumulation estimation

In order to determine whether it was necessary to krige the PGM accumulation value (4E grades multiplied by width), Anglo Platinum investigated the relationship between PGM grades and channel thickness. Scatter plots demonstrate no definitive relationship between the reef grade and thickness (see Figure 17.4) for both, the borehole and sample sections. As a result, the PGM grade and thickness were estimated individually and no accumulation estimate was deemed necessary.

Table 17.9: Statistical parameters Merensky Reef, hangingwall and footwall cuts. Footwall cuts are measured from the base of the Merensky Reef (Malenga, 2007).

Merensky Reef	Width (cm)	Number of samples	Mean 4E grade (g/t)
Hangingwall cut	10	1576	3.25
Merensky Reef West domain (cut at 20 g/t)	44	960	7.75
Merensky Reef East domain (cut at 20 g/t)	68	741	5.94
Footwall cut	25	1700	3.34
Footwall cut	35	1672	3.09
Footwall cut	40	1656	3.00
Footwall cut	50	1604	2.87

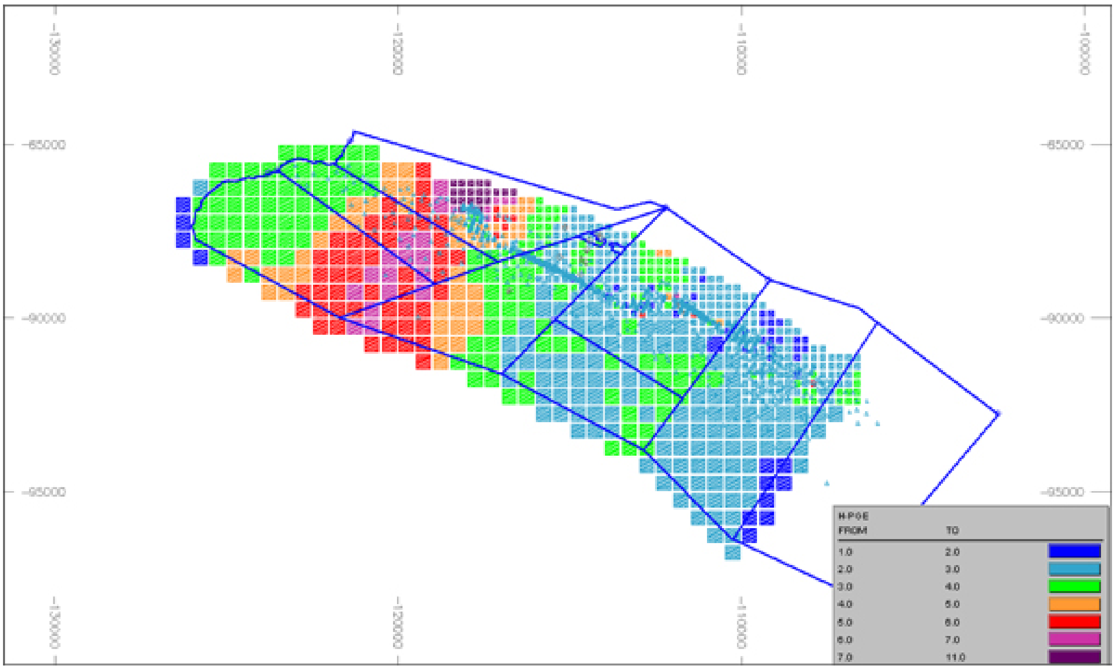


Figure 17.3: Hangingwall PGE cut grade distribution within the 10 cm cut (Anglo Platinum, 2008).

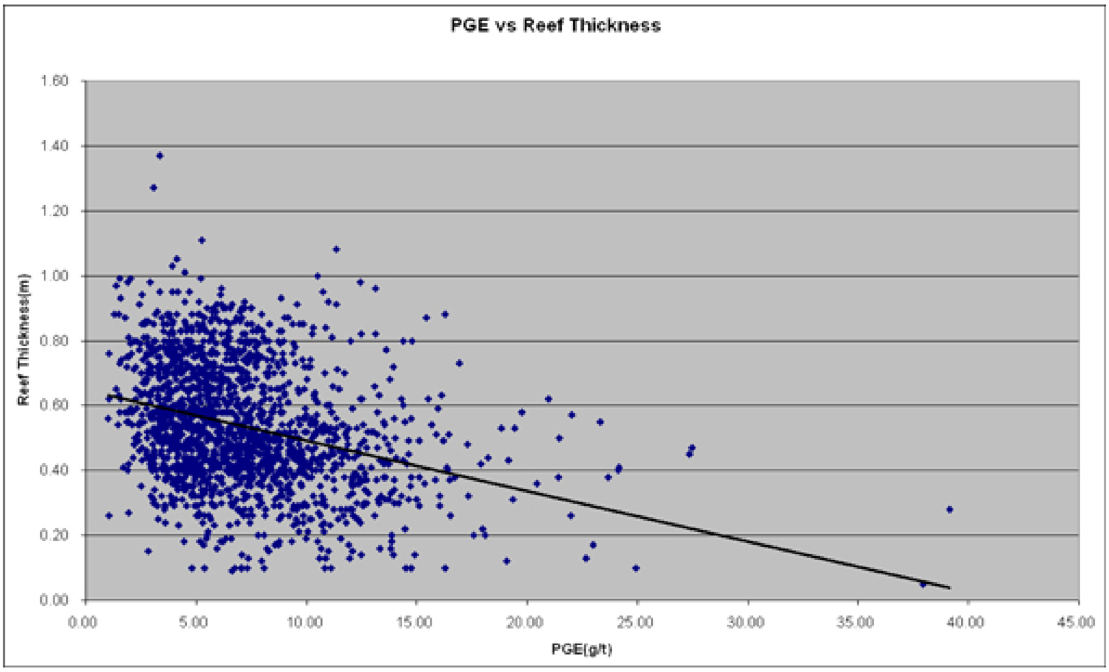


Figure 17.4: The relationship between the Merensky Reef PGE grade and thickness (Anglo Platinum, 2008).

17.3.4 Merensky Reef - Brakfontein Regional Pothole area

During the last few years, several additional boreholes were drilled to determine the down dip extent and limit of the 'Brakfontein Regional Pothole Area'. The Brakfontein Regional Pothole Area was divided into two components in 2005:

- Destructive pothole area – Merensky reef not preserved.
- Non-destructive pothole area – Merensky reef partially preserved.

No resource has been estimated for the non-destructive pothole area as more information is required to determine the geometry of the regional pothole.

The destructive pothole area remains the same as in 2006 at 349,000 m^2 while the non destructive pothole area has decreased from 665,000 m^2 (2006) to 403,700 m^2 (2006).

The purple, hatched area in Figure 17.5 defines the extent of the destructive pothole. The area outlined in green illustrates the extent of the non-destructive pothole as interpreted by Anglo Platinum in 2005 and following additional drilling, in 2007.

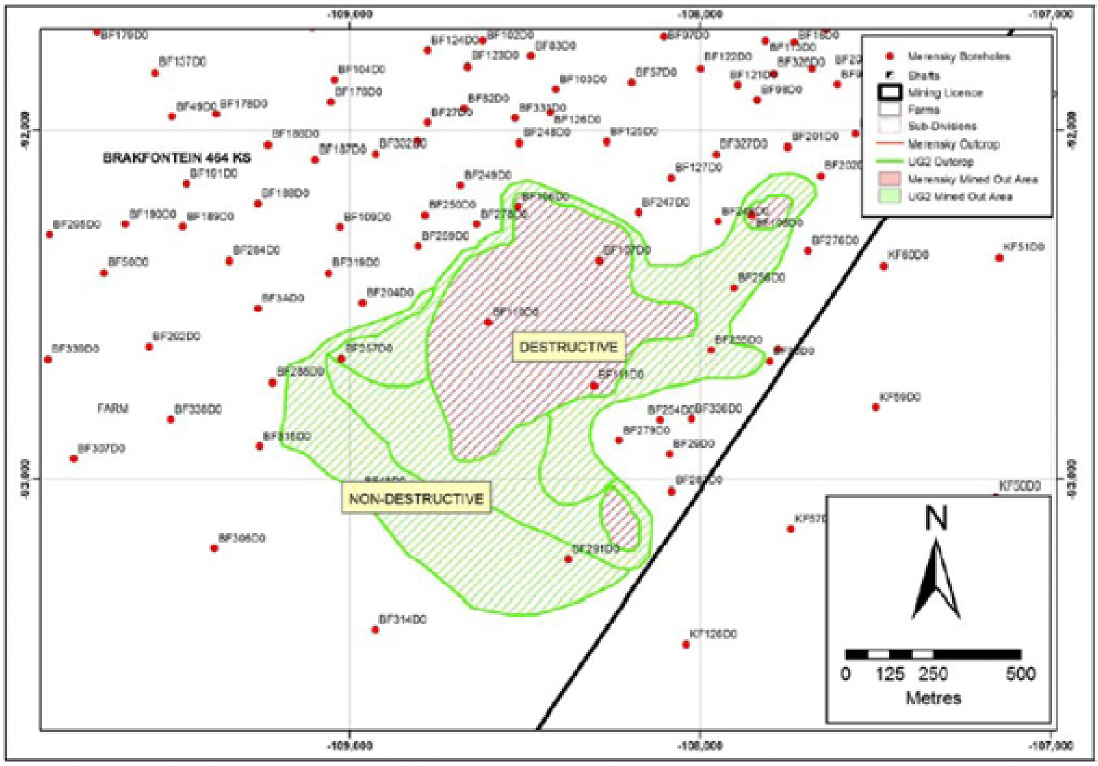


Figure 17.5: Merensky Reef – Brakfontein regional pothole area (Anglo Platinum, 2008).

17.3.5 Merensky – Geozone definitions

The Merensky Reef channel thickness shows a gradual increase from the northwest to the southeast across Lebowa. This trend was removed by delineating a boundary along the easting co-ordinate line X-112 600, separating the Merensky Reef horizon into two geozones (see Figure 17.6).

Variograms for the thickness were constructed for data segregated about this boundary line and resulted in significantly improved variography. The thickness estimation was performed using soft boundary conditions either side of this co-ordinate line.

It was found that applying the same facies boundary limit to the grade information also resulted in improved variograms for the grade data.

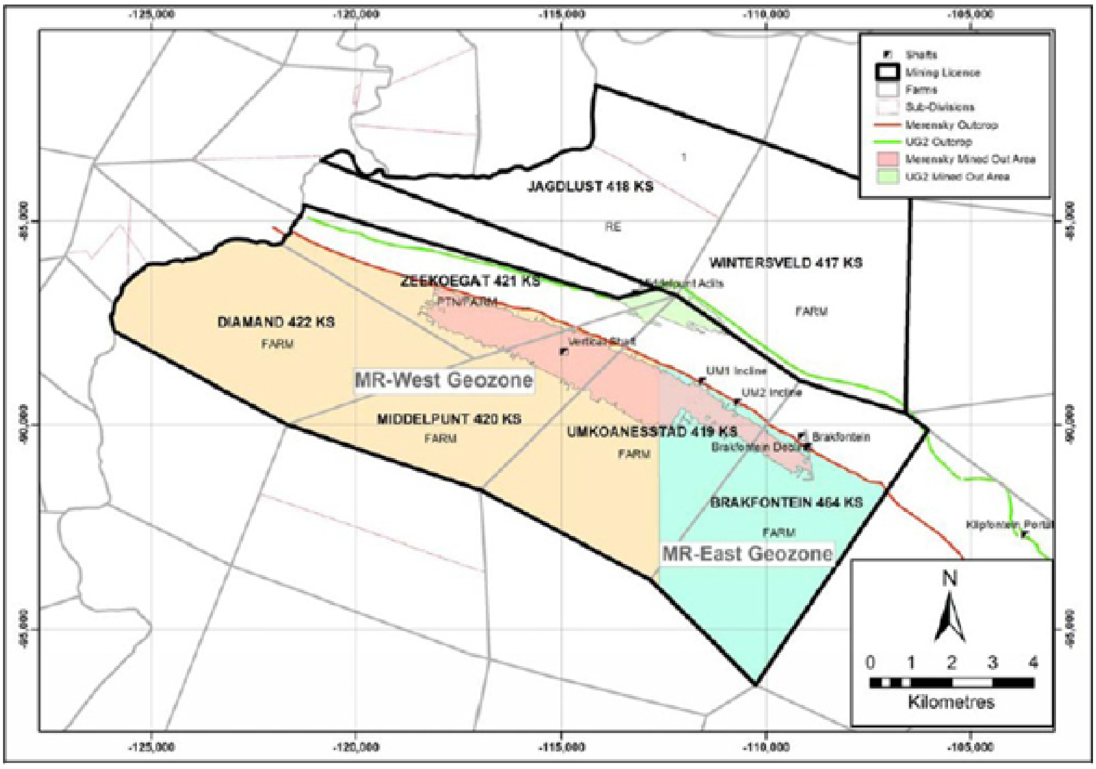


Figure 17.6: Merensky Reef East and West Geozones (Anglo Platinum, 2008).

17.3.6 Merensky Reef – Variogram analysis

Reef thickness and grade continuity variogram models were developed for each Resource cut scenario and geozone. Variogram models were developed for reef thickness, Pt, Pd, Rh, Au, Ni, Cu and density. The model parameters were then used in Datamine mining software for grade estimation.

Capping strategy

For the variogram analysis capping was applied to remove the 'noise' in the experimental variograms. No capping of the borehole or sample data was, however, applied during the estimation process.

17.3.7 Merensky – Estimated Mineral Resource model plans

Individual estimates of the spatial distribution are modelled for the 4E elements, widths and densities. These estimates are then combined into a single model for reporting of the overall Mineral Resource/Reserve estimates.

Merensky Reef thickness distribution

Reef thickness increases from NW to SE. The 2007 model shows that the Merensky Reef thickness varies between 0.27 m and 0.86 m with a mean thickness of 0.50 m (see Figure 17.7).

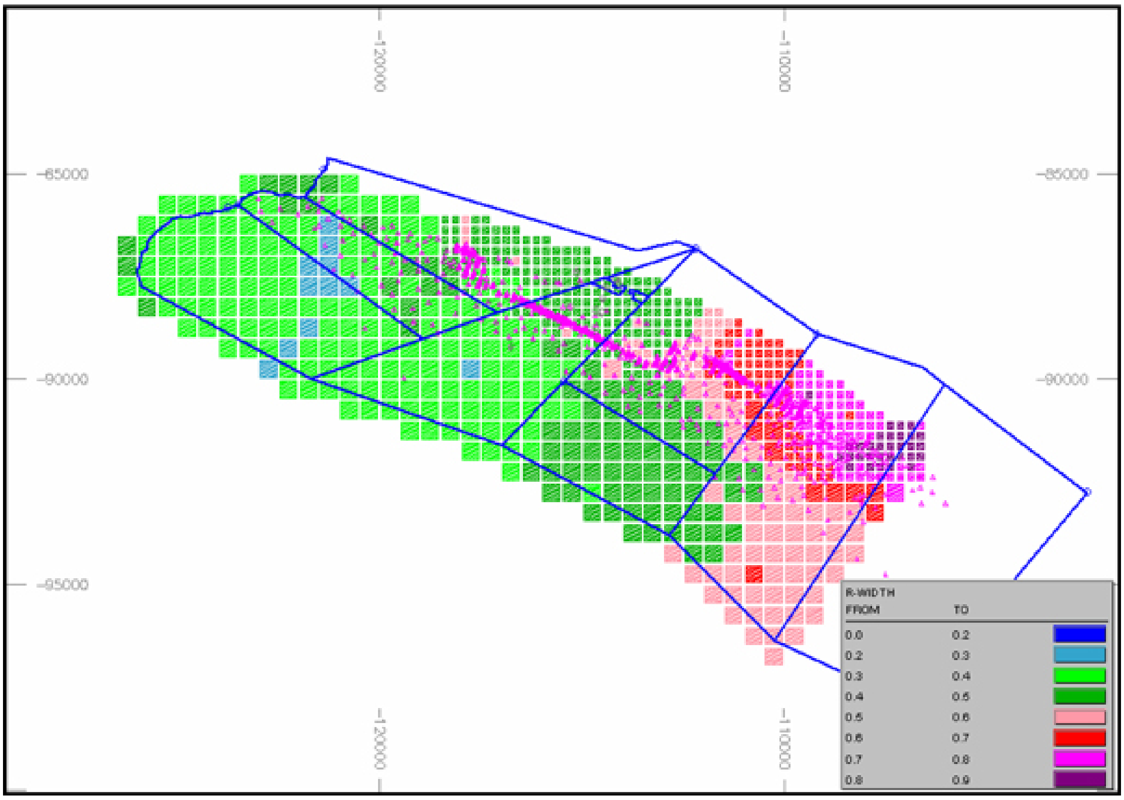


Figure 17.7: Merensky Reef thickness distribution (Anglo Platinum, 2008).

Merensky Reef PGE grade (g/t) distribution

The low grades along the eastern boundary of the Brakfontein property are substantiated by the boreholes in the Brakfontein and Klipfontein properties. The reef PGE grade decreases from the northwest to the southeast (see Figure 17.8). The reef grade varies between 3.92 g/t and 12.81 g/t with a mean grade of 7.57 g/t.

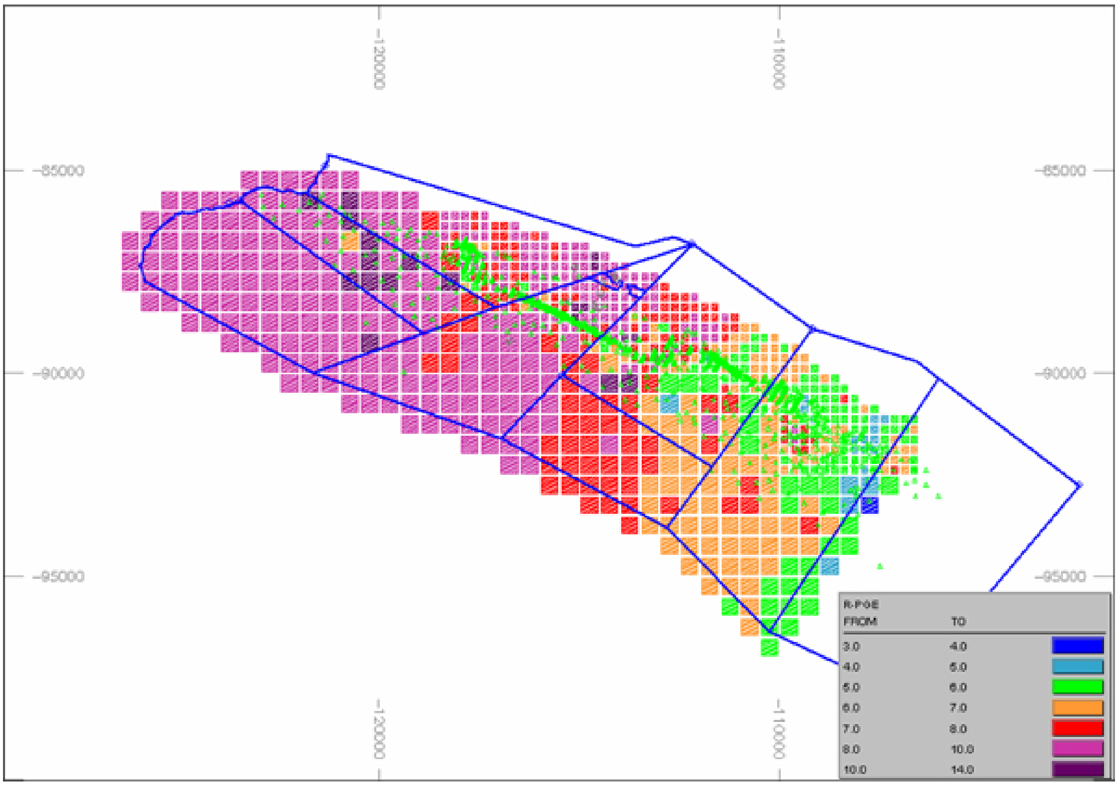


Figure 17.8: Merensky Reef PGE grade (g/t) distribution (Anglo Platinum, 2008).

Merensky Reef density distribution

The density is unchanged in the informed areas. The reef density decreases slightly from the northwest to the southeast (see Figure 17.9). The reef density varies between 3.4 and 3.6 g/cm³. Mean density is 3.45 g/cm³.

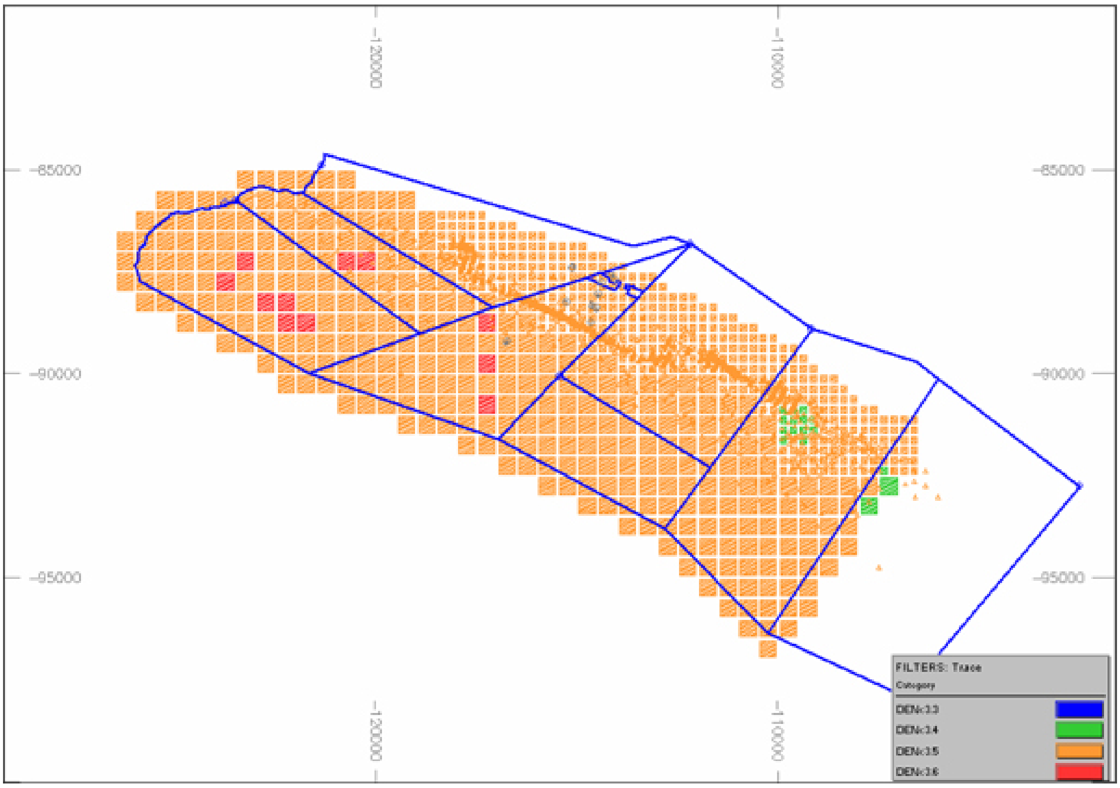


Figure 17.9: Merensky Reef density distribution (t/m³) (Anglo Platinum, 2008)

Merensky Reef Resource cut width distribution

The resource cut increases from 0.8 m in the northwest to more than 1.2 m in the southeast (see Figure 17.10).

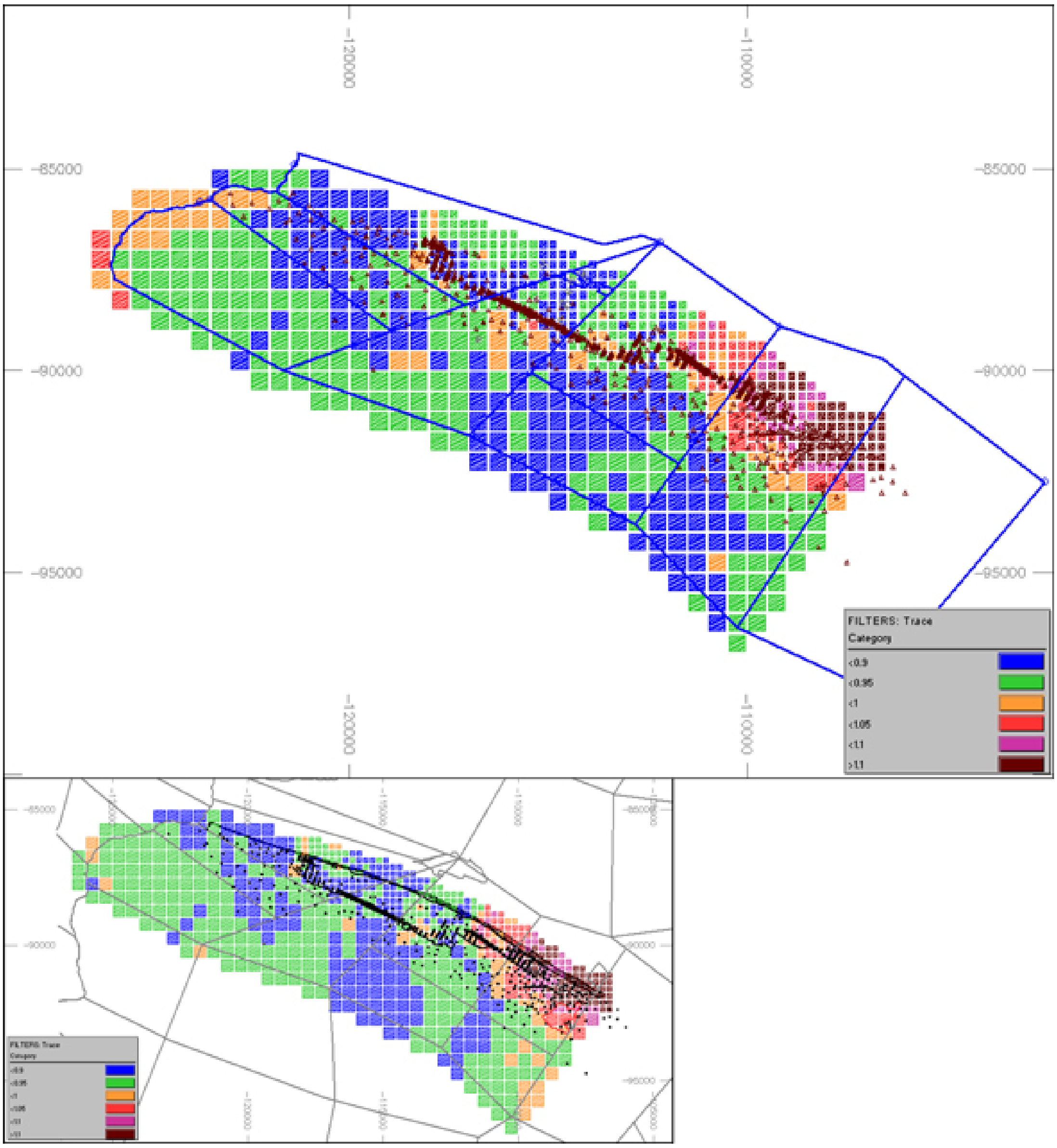


Figure 17.10: Merensky Reef Resource cut width distribution (m). Note: Insert is for 2006 (Anglo Platinum, 2008).

Merensky Reef - Resource cut PGE grade distribution (g/t)

The low grade along the eastern boundary of Brakfontein is substantiated by the boreholes in Brakfontein and Klipfontein. The grade distribution varies between 3.5 g/t and 8.6 g/t (see Figure 17.11).

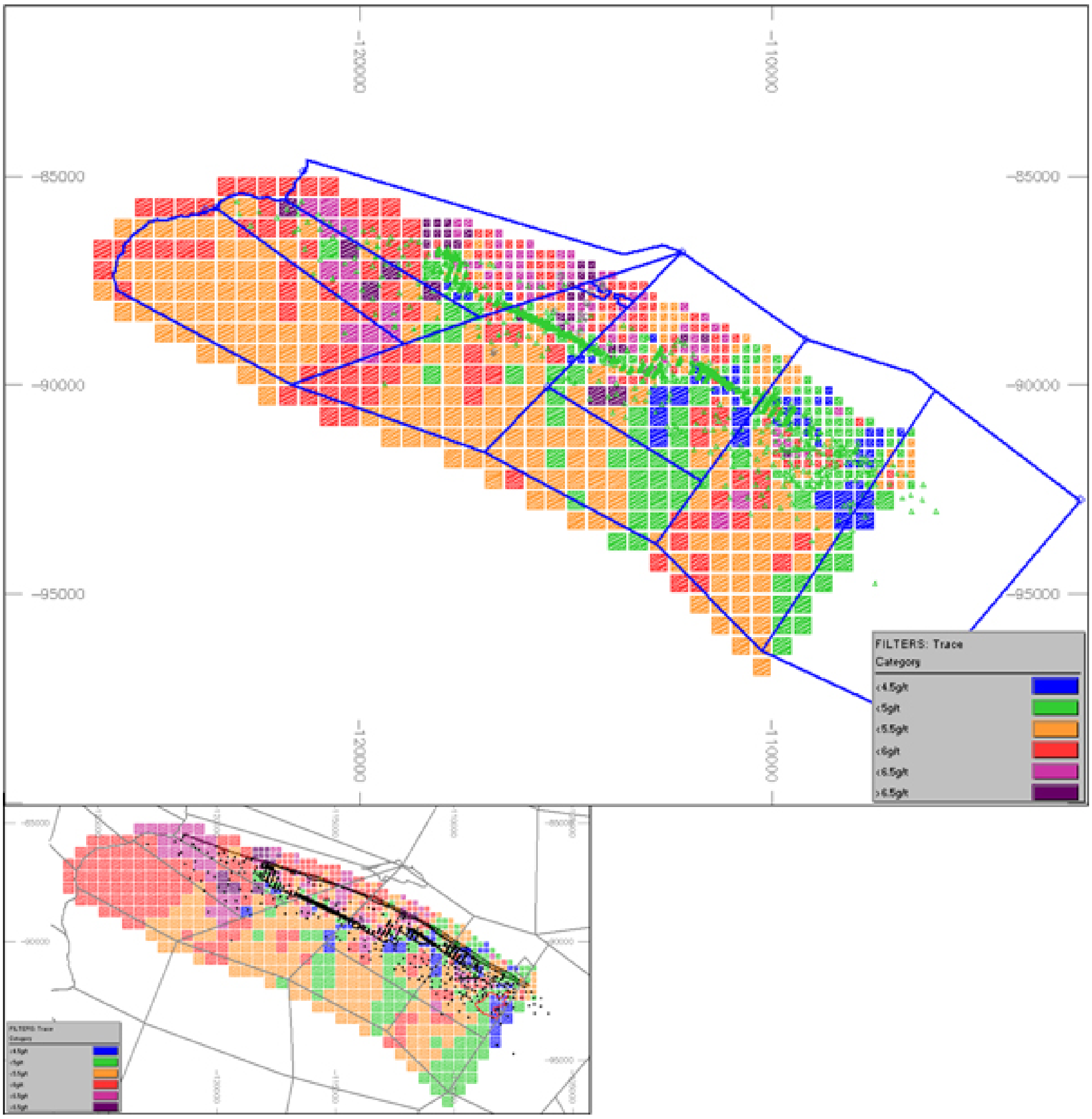


Figure 17.11: Merensky Reef - Resource cut PGE grade distribution (g/t). Insert is for 2006 (Anglo Platinum, 2008).

Merensky Reef 4E PGM kriging variance

The kriging variance is generally less than 0.2. The split on the geozone boundary of X 112 600 is clearly visible (see Figure 17.12).

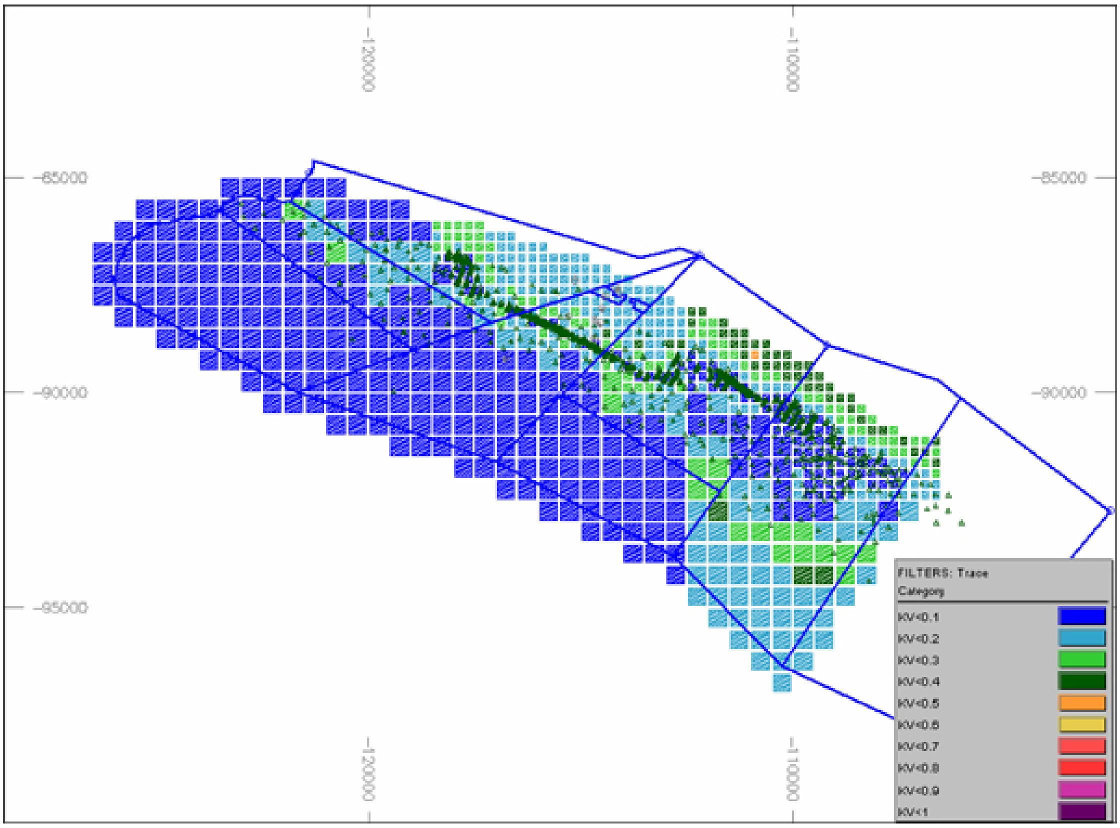


Figure 17.12: Merensky Reef - 4E PGM kriging variance (Anglo Platinum, 2008).

Merensky Reef 4E PGM kriging efficiency

The kriging efficiency is more than 0.6. The split on the geozone boundary of X-112 600 is clearly visible (see Figure 17.13).

Deloitte observes that the kriging efficiency appears very high in the areas of limited borehole data. This is not seen as a serious problem as these areas fall into the Inferred Resource category and warrants further investigation.

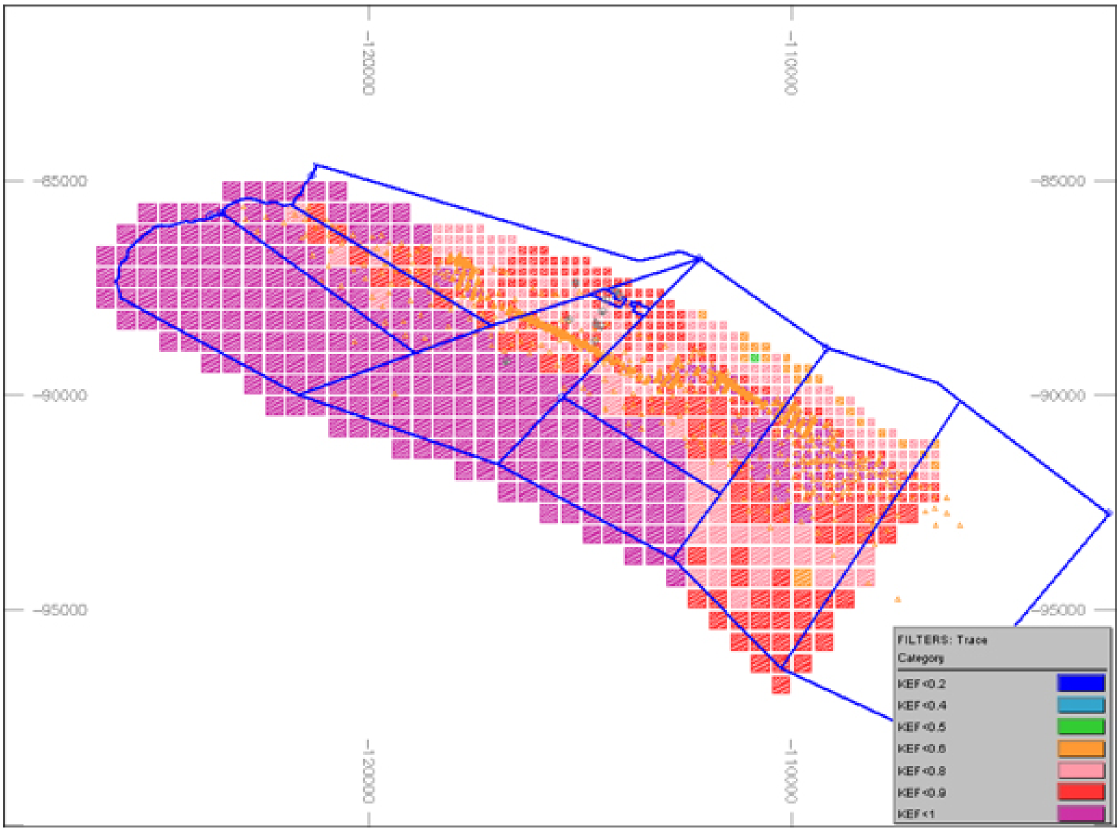


Figure 17.13: Merensky Reef - 4E PGM kriging variance (Anglo Platinum, 2008).

17.3.8 Merensky Mineral Resource classification

Kriging variance and efficiency are the main criteria for the definition of the various Mineral Resource categories. The Mineral Resource classification was reviewed and agreed to by Anglo Platinum Competent Person, Paul Stevenson in January 2008 (see Figure 17.14)

The classification in terms of Measured, Indicated and Inferred Mineral Resource is based on the borehole distribution, MRM sample distribution, variography (grade continuity), geological continuity, aeromagnetic survey, mining history, conditional simulation, risk assessment and the Competent Person's (SAMREC equivalent to a Qualified Person) overall assessment.

Deloitte is satisfied that the Mineral Resource classification complies with industry best practice.

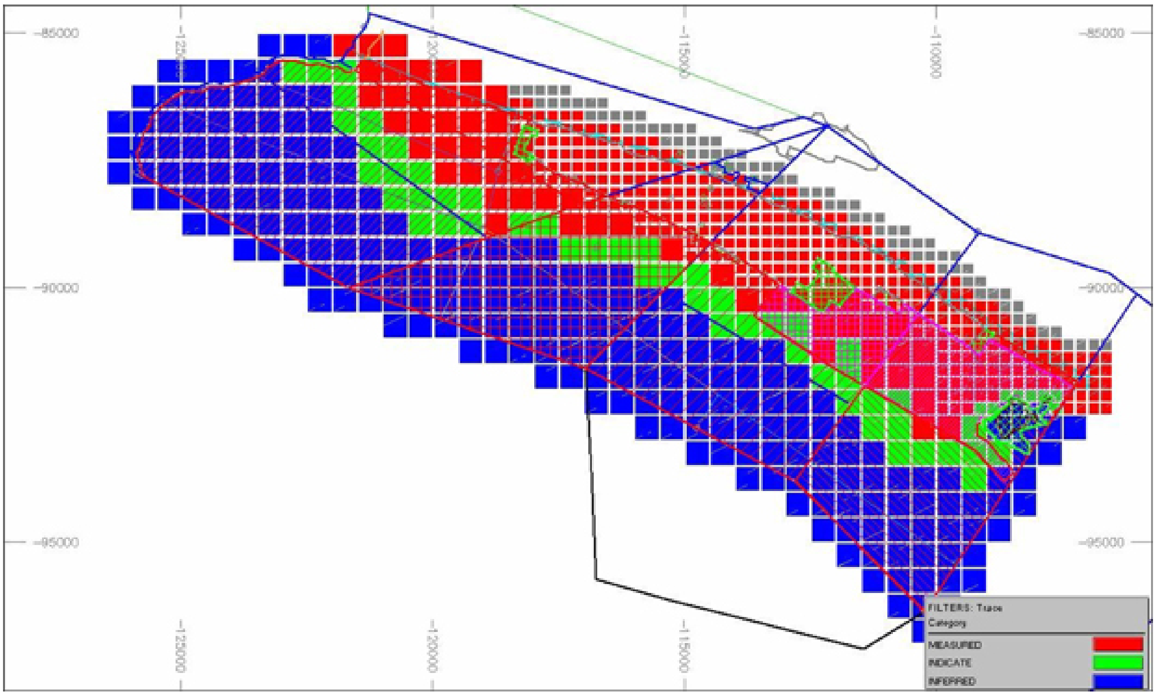


Figure 17.14: Merensky Reef - Mineral Resource classification (Anglo Platinum, 2008).

17.3.9 Merensky Reef - Geological losses

Geological losses are expressed as percentages of the reef area and are estimates of reef losses associated with geological structures. These estimated geological losses have been benchmarked according to the existing underground mapping information.

The geological losses applied to the Umkoanesstad, Zeekoegat, Middelpunt, and Diamand properties was 20% and for the Brakfontein property 25%. An additional 5% geological loss has been allowed for the Brakfontein property to cater for the regional pothole area.

17.3.10 Merensky Reef – Mineral Resource cut evaluation and resource reporting

During September 2006, the Brakfontein Merensky Project's mine plan was adjusted by Anglo Platinum to coincide with the LPM plan area together with the latest results of the updated Brakfontein regional pothole.

Tables 17.10 to Table 17.12 summarise the Mineral Resource estimates for each mining and project area after taking the optimum stope width cut into account and the respective classification categories. A minimum of 25 cm footwall was included for each mining cut for LPM and the Remaining Resource areas.

The areas defined for Resource and Reserve reporting are presented in Figure 17.15.

Table 17.10: Merensky Reef - LPM Mineral Resources and classifications.

		Dip (o)	Density (g/cm3)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade (g/t) kriged	Rh grade (g/t) kriged	Au grade (g/t) kriged
Block 1.1 VERTICAL WEST	**MEASURED**	**25.0**	**3.42**	**0.56**	**5.22**	**0.09**	**495.0**	**3.34**	**1.44**	**0.17**	**0.27**
Block 1.2 MP-WEST 10 L WINZE	**MEASURED**	**22.0**	**3.40**	**0.05**	**6.55**	**0.01**	**604.8**	**4.23**	**1.70**	**0.24**	**0.38**
Block 1.3 MP-EAST 10 L WINZE A	**MEASURED**	**22.0**	**3.41**	**0.02**	**4.68**	**0.00**	**408.3**	**3.13**	**1.17**	**0.20**	**0.18**
Block 1.4 US 10L WINZE B	**MEASURED**	**18.0**	**3.41**	**0.04**	**4.19**	**0.01**	**383.4**	**2.65**	**1.14**	**0.17**	**0.22**
Block 1.5 US 10L WINZE C	**MEASURED**	**18.0**	**3.40**	**0.04**	**5.85**	**0.01**	**548.3**	**3.50**	**1.84**	**0.20**	**0.31**
Block 1.6 US CENTRAL EAST	**MEASURED**	**18.0**	**3.44**	**0.64**	**4.30**	**0.09**	**390.7**	**2.75**	**1.13**	**0.16**	**0.27**
Block 1.7 US UM2 5 LEVEL	**MEASURED**	**18.0**	**3.43**	**0.12**	**4.45**	**0.02**	**447.5**	**2.90**	**1.11**	**0.18**	**0.26**
Block 1.8 BF UM2 2 LEVEL	**MEASURED**	**16.0**	**3.37**	**0.15**	**4.09**	**0.02**	**457.5**	**2.56**	**1.13**	**0.16**	**0.25**
Block 22 US VERTICAL EAST	**MEASURED**	**18.0**	**3.42**	**0.33**	**4.38**	**0.05**	**392.4**	**2.79**	**1.16**	**0.17**	**0.26**
SUMMARY	**MEASURED**		**3.42**	**1.95**	**4.66**	**0.29**	**437.7**	**2.97**	**1.25**	**0.17**	**0.27**

Merensky Reef - BRK Project Mineral Resources

Table 17.11: Merensky Reef - Brakfontein Project Mineral Resources and classifications.

	Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3)	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade (g/t) kriged	Rh grade (g/t) kriged	Au grade (g/t) kriged
Umkoanesstad (Block 1.9)	MEASURED	18.0	0.91	3.42	20.0	6.19	4.92	0.98	450	3.04	1.37	0.18	0.34
	INDICATED	18.0	0.89	3.40	20.0	2.17	4.82	0.34	429	2.98	1.27	0.19	0.38
	MEASURED AND INDICATED	18.0	0.91	3.41	20.0	8.36	4.89	1.32	444	3.02	1.34	0.18	0.35
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Brakfontein (Block 1.11)	MEASURED	16.0	1.02	3.41	25.0	11.43	5.21	1.92	534	3.16	1.53	0.19	0.34
	INDICATED	16.0	1.00	3.40	25.0	2.67	4.94	0.42	492	3.01	1.42	0.19	0.32
	MEASURED AND INDICATED	16.0	1.02	3.41	25.0	14.09	5.16	2.34	526	3.13	1.51	0.19	0.34
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Total Brakfontein Project	MEASURED		0.98	3.41	23.2	17.62	5.11	2.90	503	3.12	1.47	0.18	0.34
	INDICATED		0.95	3.40	22.7	4.84	4.89	0.76	463	3.00	1.35	0.19	0.34
	MEASURED AND INDICATED		0.98	3.41	23.1	22.46	5.06	3.66	494	3.09	1.45	0.19	0.34
	INFERRED		0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00

17.3.11 Merensky Reef - Remaining Mineral Resources evaluation

The Remaining Mineral Resources within Lebowa and not described above are as follows (Table 17.12):

Table 17.12: Merensky Reef – Remaining Mineral Resources and classification.

	Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade kriged	Rh grade (g/t) kriged	Au grade kriged
Block 0.1 (Oxidised)	MEASURED	25.0	0.86	3.43	20.0	0.51	5.95	0.10	509	3.83	1.59	0.20	0.32
	INDICATED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
	MEASURED AND INDICATED	25.0	0.86	3.43	20.0	0.51	5.95	0.10	509	3.83	1.59	0.20	0.32
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 0.2 (Oxidized)	MEASURED	16.0	1.13	3.44	25.0	0.86	4.62	0.13	524	2.90	1.24	0.18	0.31
	INDICATED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
	MEASURED AND INDICATED	16.0	1.13	3.44	25.0	0.86	4.62	0.13	524	2.90	1.24	0.18	0.31
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 1.0 ZK	MEASURED	25.0	0.88	2.42	20.0	6.18	5.73	1.14	503	3.70	1.53	0.20	0.30
	INDICATED	25.0	0.90	3.46	20.0	0.01	6.68	0.00	601	4.30	1.83	0.19	0.36
	MEASURED AND INDICATED	25.0	0.88	2.42	20.0	6.19	5.73	1.14	503	3.70	1.53	0.20	0.30
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 2.0 DT	MEASURED	25.0	0.88	3.42	20.0	10.06	5.83	1.88	515	3.81	1.50	0.23	0.29
	INDICATED	25.0	0.89	3.43	20.0	12.04	5.77	2.23	511	3.71	1.53	0.21	0.31
	MEASURED AND INDICATED	25.0	0.89	3.43	20.0	22.10	5.79	4.12	513	3.76	1.52	0.22	0.30
	INFERRED	25.0	0.92	3.44	20.0	38.61	5.45	6.76	503	3.48	1.46	0.20	0.31
Block 3.0 MP	MEASURED	22.0	0.93	3.42	20.0	4.93	5.47	0.87	507	3.48	1.49	0.21	0.29
	INDICATED	22.0	0.92	3.43	20.0	4.73	5.13	0.78	471	3.21	1.44	0.19	0.28
	MEASURED AND INDICATED	22.0	0.92	3.42	20.0	9.66	5.30	1.65	489	3.35	1.47	0.20	0.29
	INFERRED	22.0	0.91	3.40	20.0	22.81	5.35	3.93	488	3.45	1.42	0.21	0.28
Block 4.0 US	MEASURED	18.0	0.92	3.41	20.0	2.75	5.49	0.49	504	3.46	1.52	0.21	0.31
	INDICATED	18.0	0.89	3.42	20.0	5.42	5.28	0.92	469	3.23	1.48	0.19	0.39
	MEASURED AND INDICATED	18.0	0.90	3.42	20.0	8.17	5.35	1.41	481	3.30	1.49	0.20	0.36
	INFERRED	18.0	0.89	3.42	20.0	23.15	5.00	3.72	443	3.09	1.39	0.18	0.34
Block 5.0 BF	MEASURED	16.0	0.92	3.34	25.0	0.63	5.54	0.11	510	3.17	1.84	0.20	0.33
	INDICATED	16.0	0.89	3.36	25.0	5.19	5.51	0.92	489	3.35	1.62	0.20	0.34
	MEASURED AND INDICATED	16.0	0.89	3.36	25.0	5.82	5.52	1.03	492	3.33	1.64	0.20	0.34
	INFERRED	16.0	0.89	3.37	25.0	17.89	5.30	3.05	470	3.21	1.55	0.20	0.34
Block 1.11	MEASURED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
	INDICATED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
	MEASURED AND INDICATED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
	INFERRED	16.0	0.99	3.39	25.0	0.43	4.76	0.07	470	3.03	1.27	0.20	0.27
TOTAL RESOURCES INCLUDING OXIDIZED BLOCKS	MEASURED		0.90	3.42	20.1	25.92	5.69	4.71	509	3.66	1.51	0.21	0.30
	INDICATED		0.89	3.41	21.0	27.39	5.51	4.85	492	3.46	1.52	0.20	0.33
	MEASURED AND INDICATED		0.89	3.42	20.6	53.31	5.60	9.57	500	3.56	1.52	0.21	0.31
	INFERRED		0.91	3.41	21.0	102.89	5.30	17.53	480	3.34	1.45	0.20	0.31
TOTAL RESOURCES EXCLUDING OXIDIZED BLOCKS	MEASURED		0.90	3.42	20.1	24.55	5.69	4.49	509	3.66	1.51	0.21	0.30
	INDICATED		0.89	3.41	21.0	27.39	5.51	4.85	492	3.46	1.52	0.20	0.33
	MEASURED AND INDICATED		0.89	3.42	20.6	51.94	5.59	9.34	500	3.55	1.52	0.21	0.31
	INFERRED		0.91	3.41	21.0	102.89	5.30	17.53	480	3.34	1.45	0.20	0.31

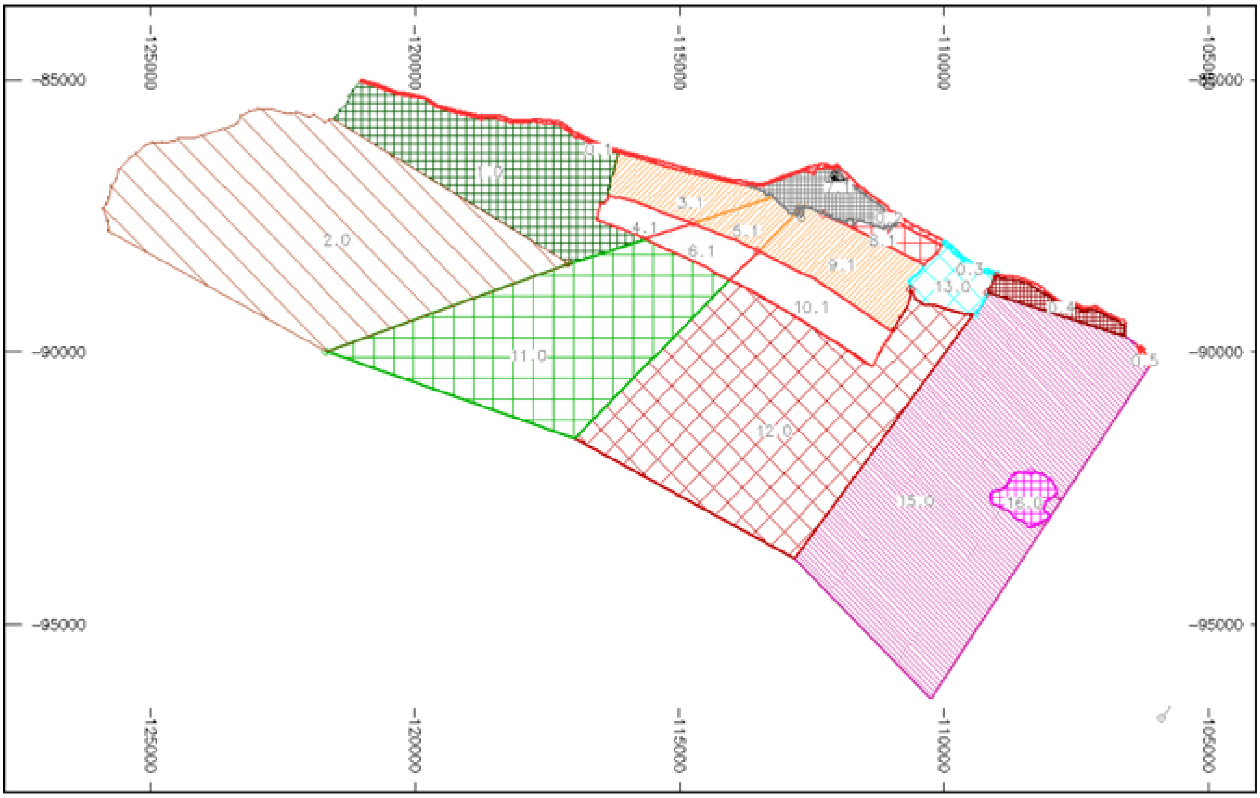


Figure 17.15: Merensky Reef - Lebowa mining lease area polygons (Anglo Platinum, 2008).

17.3.12 Merensky Reef - Mineral Reserve Statement

The Mineral Resource to Mineral Reserve conversion for the Merensky Reef was based on the block model grades. The resource tonnage and grade was converted to a plant head grade by applying the relevant modifying factors.

After applying the geological losses and modifying factors the entire Merensky Reef horizon is mined, therefore no cut-off grade needed to be applied.

The following modifying factors were applied to the Lebowa Mineral Resource estimates (see Table 17.13). These factors are supported in the documented Mineral Resource/Reserve conversion process for the Brakfontein Merensky Project (Mafoko and Hanekom, 2008).

A minimum stope width of 85 cm has been used. The stope width is the sum of the channel thickness, the hangingwall cut and the appropriate footwall cut.

Table 17.13: Modifying factors employed for the Merensky Reef Mineral Reserve calculations (Anglo Platinum, 2008).

Mineral Reserves Estimation Process Elements

		Proved Reserve		Probable Reserve	
3.1 Extraction	Comment	percent	grade (4E)	percent	grade (4E)
- Pillar Loss	- Regional and instope pillars				
	- Assuming that some Regional pillars can be superimposed on geological losses. This can either be calculated or determined	5.5%		5.5%	
	- % of Resource available for conversion.[2]				
- Post calculation depletion	- Area mined between calculation date and 31 December (reserves statement date)				
Available to mine after pillar loss	Actual mineable area after all losses.		5.07		4.89
% Extraction (% of dip corrected area)		94.5%		94.5%	
3.2 Stoping					
3.2.1. Stoping (Dimensions)					
Stoping reef cut	Percent of m2 from stoping	97.9%	5.07	97.9%	4.89
+ ASGs	additional % tonnage of ASGs	10.0%	1.50	10.0%	1.41

+ Winch beds	additional % tonnage from winch beds	0.3%	1.50	0.3%	1.41
+ Sliping	additional % tonnage from sliping	0.0%	0.00	0.0%	0.00
+ Other (to be clearly defined)		0.0%	0.00	0.0%	0.00
Sub Total b/f			4.74		4.56
3.2.2. Stoping (Modifying factors)					
- Off reef mining loss	loss in content (grade x tons)	1.6%	5.07	1.6%	4.89
+ Off reef mining (add back)	add back tons at no grade				
- RIF/RIH loss	loss in content (grade x tons)	0.5%	2.00	0.5%	2.00
+ RIF/RIH (add back)	add back tons at reduced grade				
+ Off reef development (re-dev)	add tons at no grade	1.9%		1.9%	
+ Overbreak/ dilution	additional tonnage at reduced/no grade	0.5%	1.82	0.5%	2.02
+ Scaling	additional tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00
+ FoG	additional tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00
+ Other (to be clearly defined)	(comment here)	0.0%	0.00	0.0%	0.00
Total - Stoping (net)			4.56		4.39
3.3 Development					
3.3.1. Development (Dimensions)					
Dev reef cut	at reef grade	2.1%	5.07	2.1%	4.89
+ Dev Waste cut	additional tonnage at reduced/no grade	250%	1.50	250.0%	1.41
+ Cubbies - Reef cut	at reef grade	1.0%	4.60	1.0%	4.89
+ Cubbies - Waste cut	additional tonnage at reduced/no grade	2.0%	0.00	2.0%	0.00
+ Other (to be clearly defined)	(comment here)	0.0%	0.00	0.0%	0.00
Sub Total b/f			2.51		2.40

3.3.2. Development (Modifying factors)

- Off reef mining loss	loss in content (grade x tons)	20.0%	5.07	20.0%	4.89
+ Off reef mining (add back)	add back tons at no grade				
- RIF/RIH loss	loss in content (grade x tons)	4.5%	5.11	5.0%	4.89
+ RIF/RIH (add back)	add back tons at reduced grade				
+ Off reef development (re-dev)	add % tonnes at low/no grade	0.0%	0.00	0.0%	0.00
+ Overbreak/ dilution	add % tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00
+ Scaling	add % tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00
+ FoG	add % tonnage at reduced/no grade	0.1%	0.00	0.0%	0.00
+ Other (to be clearly defined)	(comment here)	0.0%	0.00	0.0%	0.00

Table 17.14 summarizes the Proven and Probable Mineral Reserves for the Merensky Reef.

Table 17.14: Merensky Reef - Mineral Reserve Statement – December 2008.

Category	Tonnage (Mt)	4E grade (g/t)	Contained metal (Moz)
Proven	**21.71**	**4.34**	**3.03**
Probable	**5.43**	**4.16**	**0.73**
Total	**27.14**	**4.31**	**3.76**

Notes: Mineral Reserves have been determined by applying modifying factors to the Mineral Resources Tonnes and ounces have been rounded and this may have resulted in minor discrepancies. The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au). Only Measured and Indicated Resources have been converted to Mineral Reserves. The Mine Call Factors used in the Mineral Reserve calculations are 97% and 98% for the Proven and Probable Reserves respectively. The Mineral Reserves were calculated by Anglo Platinum (Mofoko and Hanekom, 2008).

17.4 UG2 Mineral Resource estimation

The UG2 Mineral Resource estimates for precious and base metal grades, thickness and density take a practical stope width cut into account. The total 4E PGM grade is the summation of the individual prill split grades for Pt, Pd, Rh and Au.

17.4.1 Modelling overview and estimation criteria

The modelling procedure adopted was as follows:

- Datamine modelling was done in two dimensions
- The regressed borehole composite database was used
- Several geostatistical investigations were performed on the validated boreholes and underground samples. The following cuts were investigated:
    - The geotechnical hangingwall cut (boreholes)
    - The UG2 reef cut (boreholes and samples)
    - The pegmatoidal footwall cut (boreholes)
- The statistical analysis was completed using Snowden Supervisor software.
    - For the histograms and semivariogram analysis, see Appendix: “Lebowa UG2 Geological - Resource Modelling 2007 – Validation – Statistics”
- Various Q-Q plots were completed in Datamine
- There were no changes to the overall dips per farm from 2006 and the same dips were used to calculate the composites true reef width. The following dips were used:

Zeekoegat and Diamand:	25 degrees
Middelpunt:	22 degrees
Umkoanesstad:	18 degrees
Brakfontein:	16 degrees

- Various variance analysis plots were completed in Datamine
- Various scatter plots were completed in Excel.
- A new kriging neighbourhood study, as conducted in 2007, was applied.
- The entire area of interest was kriged
    - Ordinary kriging was used for the UG2 reef, geotechnical hangingwall cut, and pegmatoidal footwall cut.
    - Cell dimensions of 250 m by 250 m were used within and immediately adjacent to workings. All other cells were defined as 500 m by 500 m.
- The reef prill, base metal grades and the thickness/density were modelled for the UG2 reef, geotechnical hangingwall, and pegmatoidal footwall.

- The reef Cu grade was divided into two geozones. One area is to the west of X-112 600 and one area to the east of this coordinate.
- The resource tonnage is calculated using:
  - Combined kriged density for the geotechnical hangingwall, reef, and pegmatoidal footwall.
  - Dip correction factor.
  - Geological loss factor.
- If the combined geotechnical hangingwall cut, the UG2 reef and the pegmatoidal footwall cut is less than the minimum resource cut of 0.95m, additional footwall material (norite) will be added with a grade of 0.0g/t.
- The default density in the underground sampling has not been used for the density modelling.
- Mean densities were used for the older boreholes which do not contain density values. These mean density values were used for compositing and estimation and were not used for variogram modelling.
- The UG2 reef thickness from the sample database was used for the evaluation. The overall PGE grade was obtained by combining the individually modelled prill grades. The underground samples do not contain prill grades and were not used for the estimation of the PGE grades.
- The model is then validated against the composited data and reconciled against the previous year's model.
- The risks associated with the resource estimation are assessed.
- The risks, geostatistical confidence indicators and the geological framework is assessed and with due consideration for the previous year's resource classification, the new classification is assigned.
- The resources are evaluated by resource category.
- Resources are compared against those generated from the previous model.

Data sources

The Mineral Resource Management (MRM) underground sampling database and the Sable borehole database data were used for Resource estimation. The Sable database has provision for storing prill, density and base metal analysis whereas the underground sample sections (MRM) can only store 4E grades. Since the previous estimate, no new underground sampling data was available due to a delayed start-up of the new Eastern Bushveld Robotic Laboratory (EBRL).

Historically, boreholes contain differences in stored information, with the newer boreholes having prill/density data compared to the holes drilled prior to 2000. In addition, older boreholes did not always have their wedges sampled. Borehole data has been subject to several comprehensive validation stages in Sable, Excel, and Datamine. Erroneous data is recorded and noted for correction and is discarded from the database prior to Resource estimation.

Coordinate conversion, All the data has been converted to WGS84 (LO31) format.

No PGM correction factor was applied to the borehole data. Underground samples are assayed at the lab using the fire assay technique. A 1.002 correction factor has been applied to the 4E assay results in order to reduce the effects of data bias (Nowak, 2005).

UG2 bifurcation

The UG2 bifurcation has been the subject of a study done by R Brown and D. Nowak (2003, 2005). As a result, the UG2 shows two different reef facies types:

- A facies with a solid chromitite layer with minor inclusions of non-chromitite material such as feldspathic pyroxenite.
- A facies of bifurcated UG2. In these areas UG2 chromitite commonly bifurcates with pyroxenites, resulting in extra dilution and increasing widths (Figure 17.16).

The studies characterise bifurcated UG2 as having widths greater than 0.9 m and dilution greater than 20%.

In addition, it is noted that at least 50% of all the UG2 intersections have some form of waste inclusions/inter-layers and that the average dilution within the UG2 is 9.3%. Figures 17.17 and 17.18 show the areas of bifurcated UG2.

Bifurcated UG2 - Mining constraints

The bifurcated UG2 impacts on the mine plan because it affects the optimum stope width cut. As more borehole intersections have encountered bifurcated UG2 it is necessary to classify the bifurcated UG2 as mineable or un-mineable depending on the degree of dilution.

The minimum stope width cut applied at the current Middelpunt Hill UG2 area is 95 cm. The maximum stope width is a function of the UG2 reef thickness, the geotechnical hangingwall cut and a minimum of 0.1 m footwall cut. Where the UG2 bifurcation exceeds the maximum stoping width of 1.20 m, the stoping width reverts back to 0.95 m and only the top UG2 layer is mined.

17.4.2 Flood-line and River Pillars

A 300 m wide “flood line-pillar” of 300 m for Resources lying above 500 m depth and a 100 m wide pillar for resources below 500 m has been maintained for the resources along the Olifants River. The guideline was obtained from M Button, the Senior Mining Engineer, in 2003. These volumes are excluded from the Resource statements.

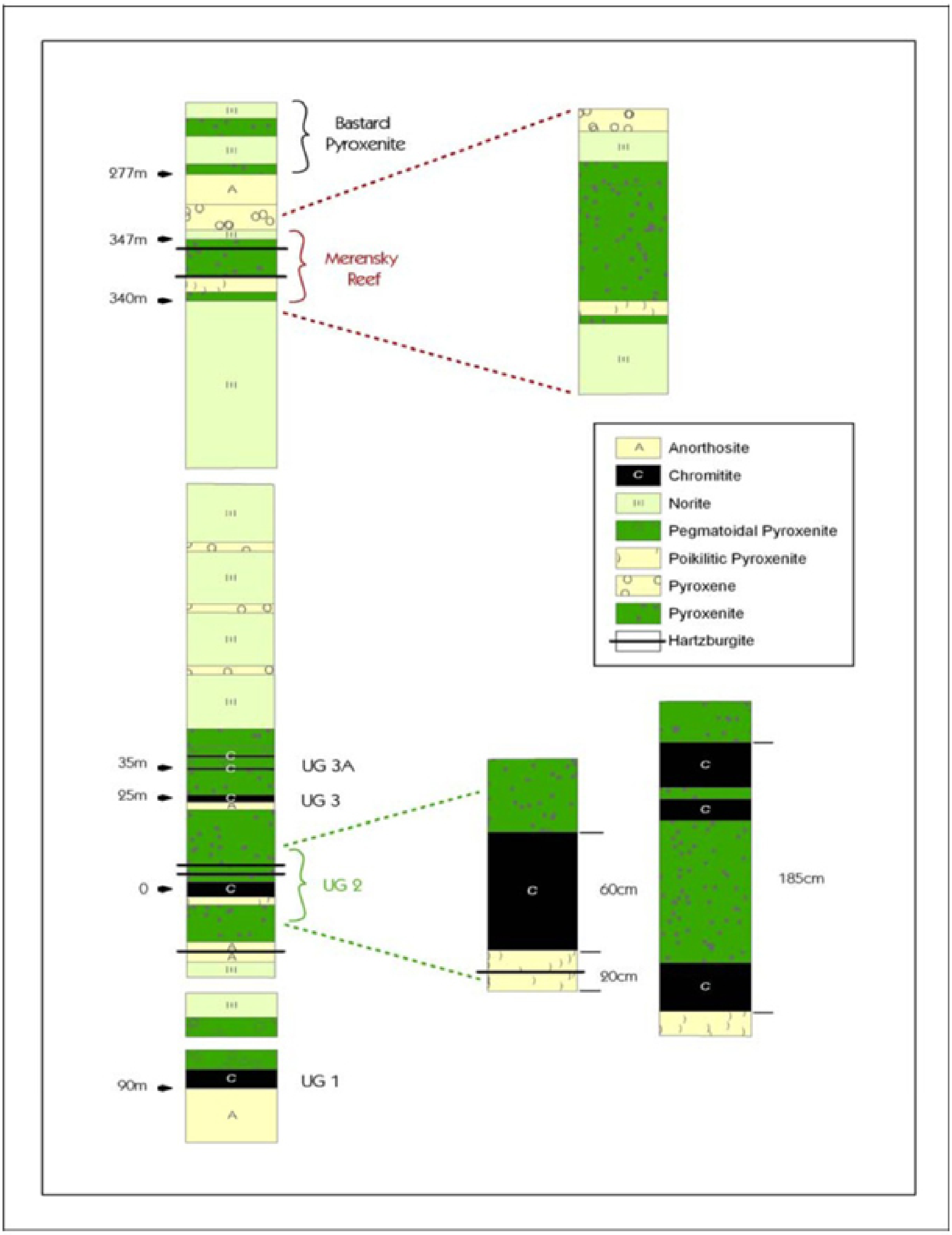


Figure 17.16: Stratigraphic description of bifurcated UG2 (Brown, 2003).

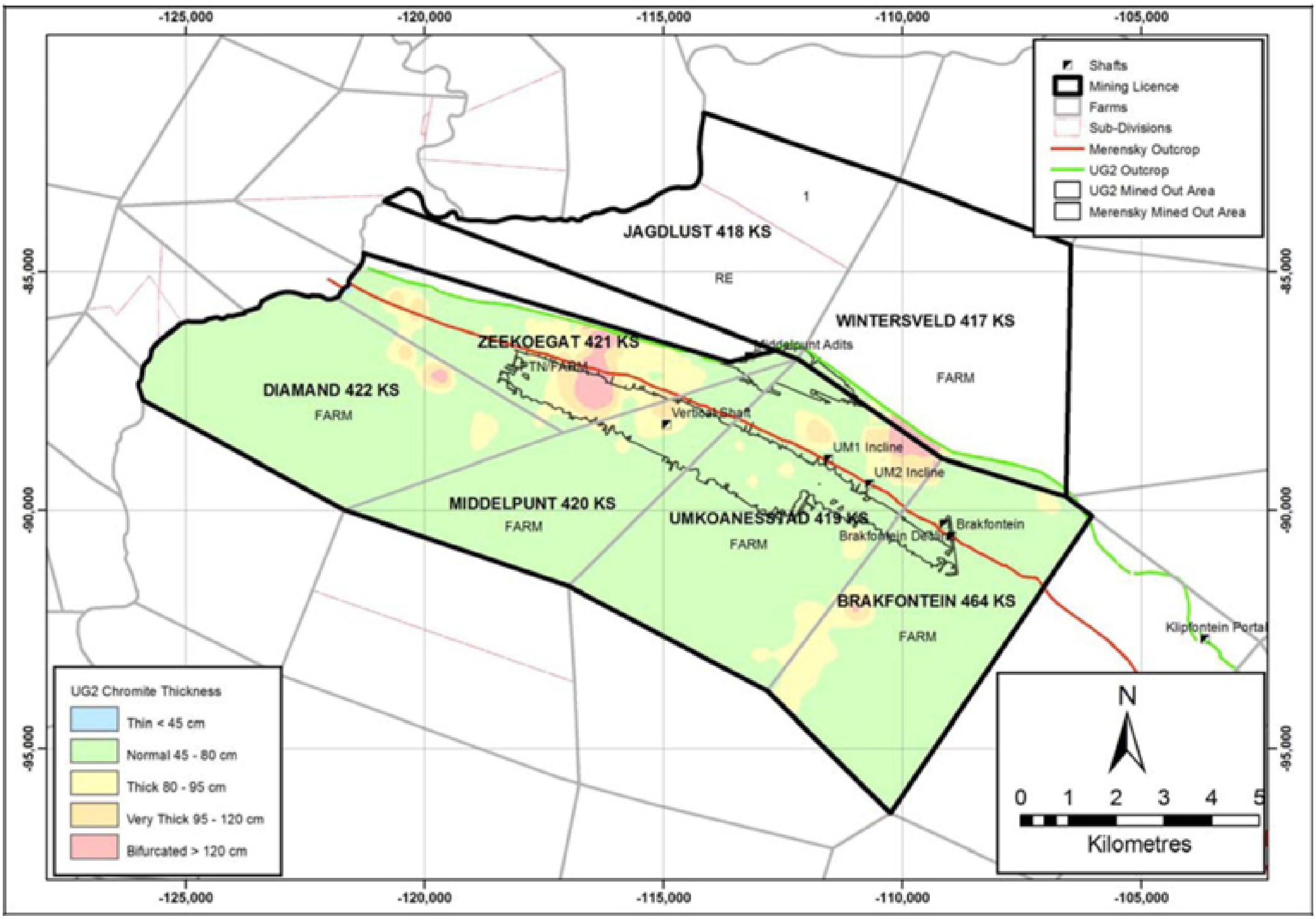


Figure 17.17: Areas with abnormal UG2 thickness (chromitite and waste interlayers) (Anglo Platinum, 2008).

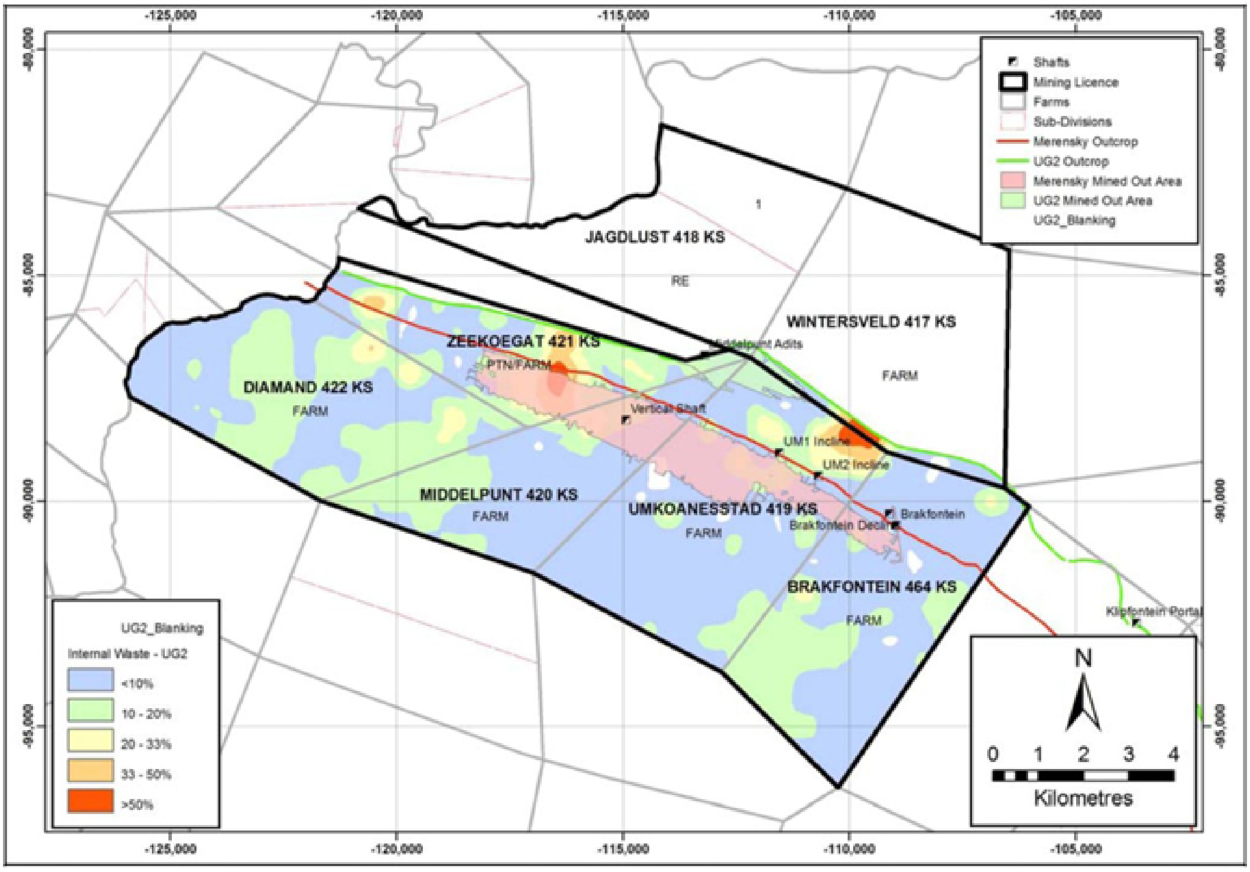


Figure 17.18: Areas with high dilution (internal waste) within the UG2 (Anglo Platinum, 2008).

17.4.3 UG2 - Data sources

The borehole data is stored in a Sable database and contains density, 4E prill split values and base metal values for the assayed samples. The underground sampling data is stored in the MRM database and can only store 4E grades. It is noted that no new underground sampling data was available for the 2006 or 2007 modelling due to problems at EBRL. Deloitte recommends that these problems be addressed as a matter of priority.

Co-ordinate conversion, PGM correction factor

All data has been converted to WGS84 (LO31) format in 2003. No PGM correction factor was applied to the borehole data. Underground samples are assayed at the lab using the fire assay technique. A 1.002 correction factor has been applied to the 4E assay results in order to reduce the effects of data bias (Nowak, 2005).

17.4.4 UG2 Mineral Resource block model

The UG2 Mineral Resource Block Model contains the following kriged estimates:

- UG2 thickness including dilution.
- 4E (Pt, Pd, Rh, Au), Cu, Ni grades.
- density

Mineral Resources are stated for the Lebowa Mine plan, the Middelpunt Hill Project areas and the remaining areas outside of planned mining.

17.4.5 UG2 - Borehole distribution plan

Figure 17.19 illustrates the distribution of UG2 boreholes across Lebowa. There is a closer grid spacing for the Brakfontein Project area located in the southeast. A breakdown of the available data sources is shown in Table 17.15.

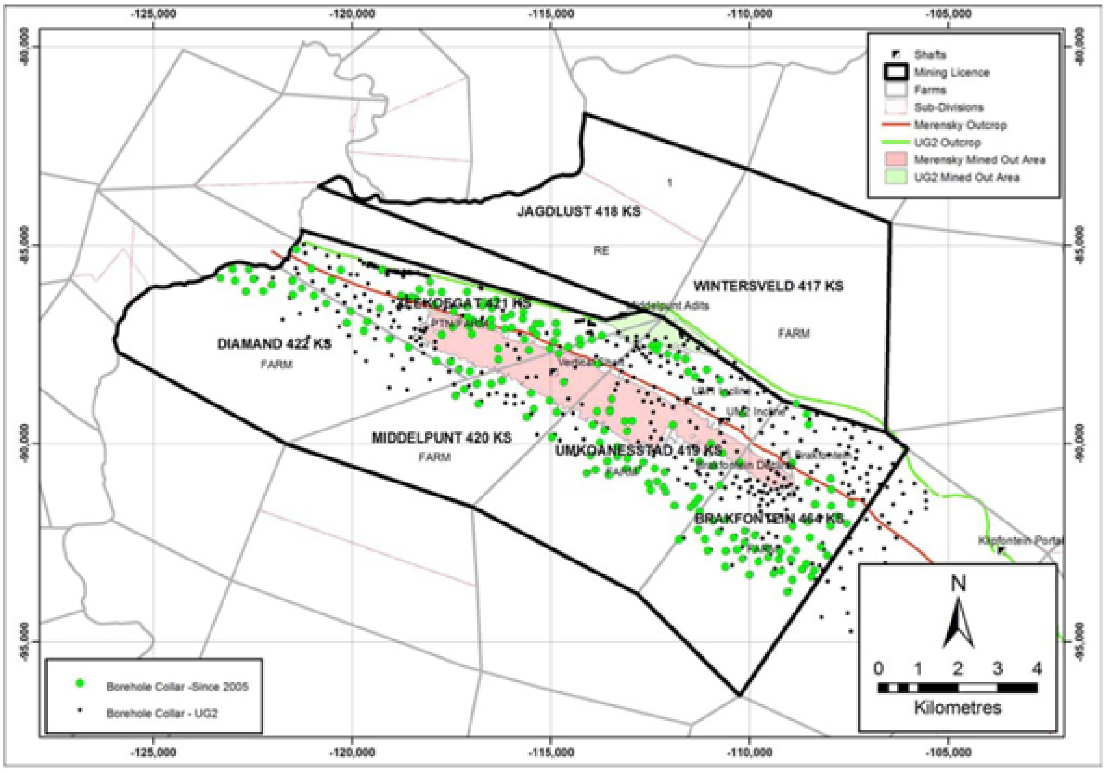


Figure 17.19: UG2 borehole data with new (2007) boreholes in green (Anglo Platinum, 2008).

Table 17.15: UG2 borehole and MRM data (Anglo Platinum, 2008).

	2007	New	2006	Change (%)
Parent boreholes	**553**	**70**	**483**	**14.5**
Borehole intersections	**1497**	**232**	**1 265**	**18.3**
MRM channels	**829**	**0**	**829**	**0**

Pothole Intersections

33 intersections were identified by Anglo Platinum to have intersected potholes or pothole edges on the basis of the chromitite thickness and footwall/hangingwall lithology and stratigraphic sequences. These intersections were excluded from the Mineral Resource estimation. Boreholes with a UG2 thickness of less than 30 cm were treated as pothole intersections.

Validated borehole data were composited for statistical analysis, variography and estimation. The compositing process used density and length weightings, and was corrected to a true thickness.

17.4.6 UG2 - Statistical analysis

Statistical analysis (Table 17.16) was performed for composites of the:

- Geotechnical hangingwall.
- UG2 grades, thickness, and density.
- Pegmatoidal pyroxenite footwall grades, thickness and density.

Table 17.16: UG2 statistical parameters for hangingwall and footwall cuts (Malenga, 2007).

Merensky Reef	Width (cm)	Number of samples	Mean 4E grade (g/t)
Geotechnical Hangingwall	**16**	**582**	**3.76**
UG2 Horizon (borehole only cut below 2 g/t and 14 g/t)	**71**	**1144**	**7.98**
Pegmatoidal Footwall	**51**	**1016**	**2.18**

Grade and width relationships

Scatter plots show no discernable relationship between the composited UG2 grade and the UG2 thickness for borehole and sample sections respectively. Grade and width were therefore estimated individually. The accumulation (grade x width) variable was consequently not estimated.

17.4.7 UG2 - Capping strategy

The grade within the first 3 cm above the top reef contact of the UG2 was capped. Due to the standardised sampling procedure, the first 2 cm of the top reef contact of the UG2 is sampled together with UG2. In order to avoid an over-estimation of the geotechnical thickness grade, the grade over the 2 cm was excluded.

For the variogram analysis the following capping was applied:

- Geotechnical hangingwall cut - Pd capped at 10.0 g/t, Rh at 1.0 g/t, Pt at 2.0 g/t, Cu at 0.1%, Thickness at 0.02 m and 0.50 m, Density at 4.0 g/cm3.
- UG2 cut - Pd capped at 9.0 g/t, Pt at 2.0 g/t, Thickness at 1.4 m.
- Pegmatoidal footwall cap - Pd capped at 3.0 g/t, Rh at 1.0 g/t, Au at 0.02 and 0.05 g/t, Cu at 0.1%, Ni at 0.04 and 0.20%, Thickness capped at 2.0 m.

For the estimation the following capping was applied:

- Geotechnical hangingwall cap - Pd at 10.0 g/t, Cu at 0.2%, Ni at 0.35%. No capping was performed for the Pt and Rh grade.
- UG2 cap - Pd at 9 g/t, Au at 0.3 g/t, Ni at 0.35%.
- Pegmatoidal footwall cap - Pt at 2 g/t, Pd at 3 g/t, Rh at 0.6 g/t, Au at 0.1 g/t.

Sporadic high grades were capped in order to minimise overestimation of the local kriged grades.

17.4.8 UG2 - Geozone definitions

The UG2 Cu grade shows a gradual decrease in grade from the northwest to the southeast, across the Lebowa properties and the histograms indicate that there are two populations of data. These two populations of data were separated along a Cu geochemical soft boundary corresponding to the X co-ordinate line X-112 600, resulting in two geochemical zones (see Figure 17.20).

There was no discernable difference in the spatial distribution of the 4E PGM grade across this boundary and as a result geological domains were not applied during the 4E grade estimation.

Geotechnical considerations

Geotechnical considerations were necessary for modelling due to the presence of chrome leaders in the direct hangingwall of the UG2. These chromitite leaders form natural planes of weakness, resulting in hangingwall instability when located too close to the mining stope’s hangingwall. The Rock Engineering Department at Anglo Platinum recommended that the geotechnical hangingwall beam be 0.30 metres (see Figure 17.21). This will ensure the ability of miners to perform the in-stope roof bolting and mine planning.

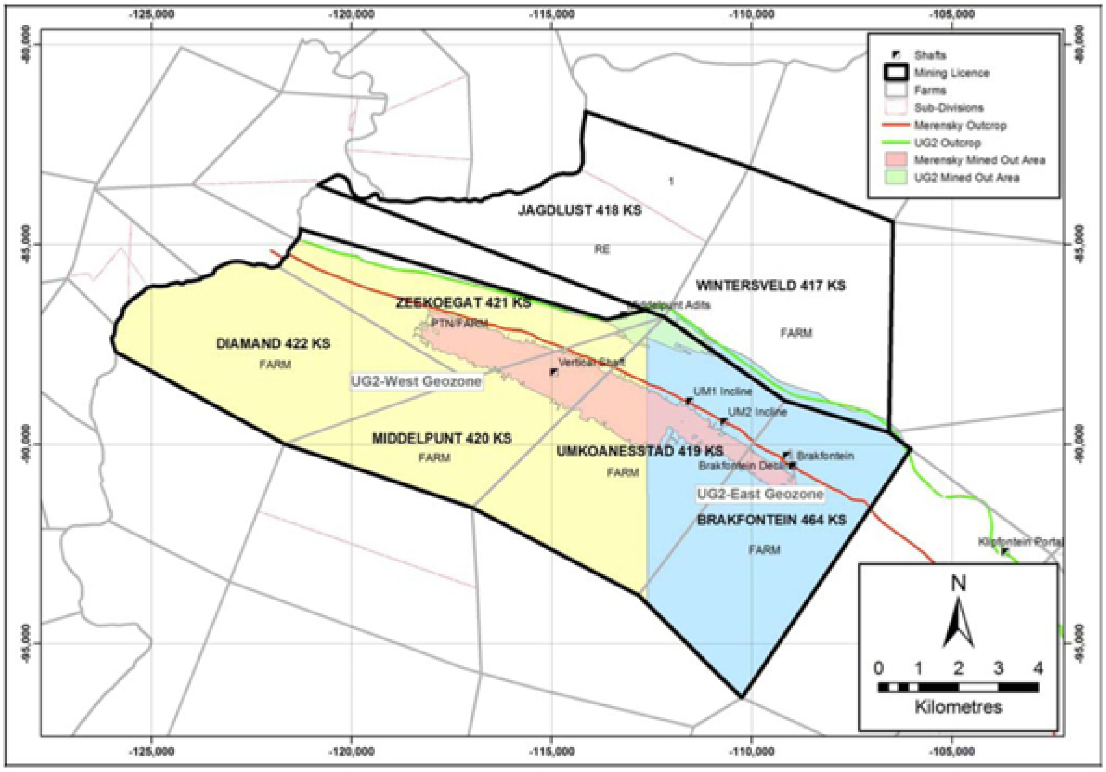


Figure 17.20: Two Cu geochemical geozones were separated into a West and East area either side of the X -112 600 coordinate (Anglo Platinum, 2008).

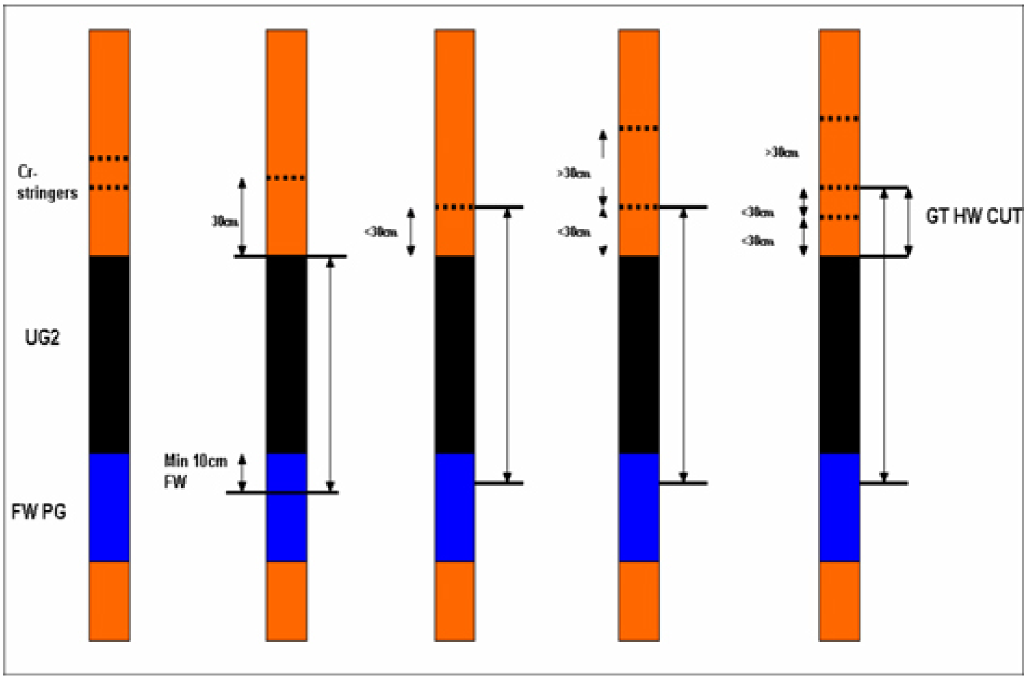


Figure 17.21: Chromitite leaders relative to the mining cut (Gray et al., 2008).

Pegmatoidal footwall modelling

In 85% of the boreholes, pegmatoidal footwall is identified. For the remaining 15% various other rock types have been logged. Only the logged and flagged pegmatoidal pyroxenite footwall was used for the evaluation. In cases where the pegmatoidal footwall has not been logged/identified, the grade and the thickness information from the surrounding boreholes was used.

The grade and the thickness of the pegmatoidal footwall vary from borehole to borehole and even between the mother-hole and its deflections. Due to the variability of the pegmatoidal grade distribution the grade of the total width of the pegmatoidal footwall was used for the resource cut even though only a portion of the pegmatoid may be required to make up the resource cut.

Kriging neighbourhood study

The findings of the 2007 kriging neighbourhood study have been applied:

- 250 m by 250 m cell dimensions were used in the well-informed areas and 500 m by 500 m cells in the poorly informed area.
- A minimum of seven and a maximum of thirty samples were required within the 1st and 2nd search ellipses for kriging. This was kept the same for all elements.

17.4.9 UG2 – Estimated Mineral Resource Model Results

Block model plans were produced for the individual 4E PGM elements, reef widths, densities, kriging variance and kriging efficiency. These estimates were combined into a single block model for the overall Mineral Resource/Reserve estimates. The block models are illustrated in the following figures.

UG2 Reef thickness

The UG2 thickness varies between 0.43 m and 1.90 m with a mean of 0.74 m. There is a general increase in thickness due to the inclusion of bifurcated boreholes in the Zeekoegat and the Umkoanesstad properties (see Figure 17.22).

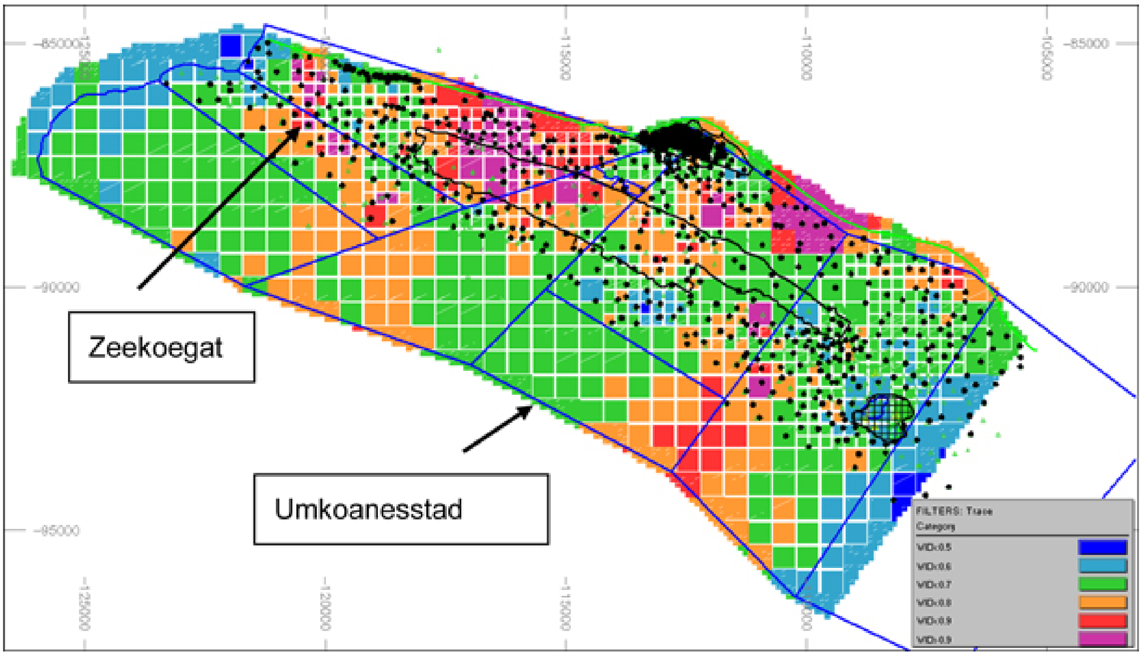


Figure 17.22: UG2 thickness distribution (Anglo Platinum, 2008).

UG2 4E PGE grade

The UG2 4E PGE grade varies between 4.27 g/t and 9.65 g/t with a mean of 7.87 g/t. There is a general decrease in grades due to the inclusion of bifurcated boreholes (see Figure 17.23).

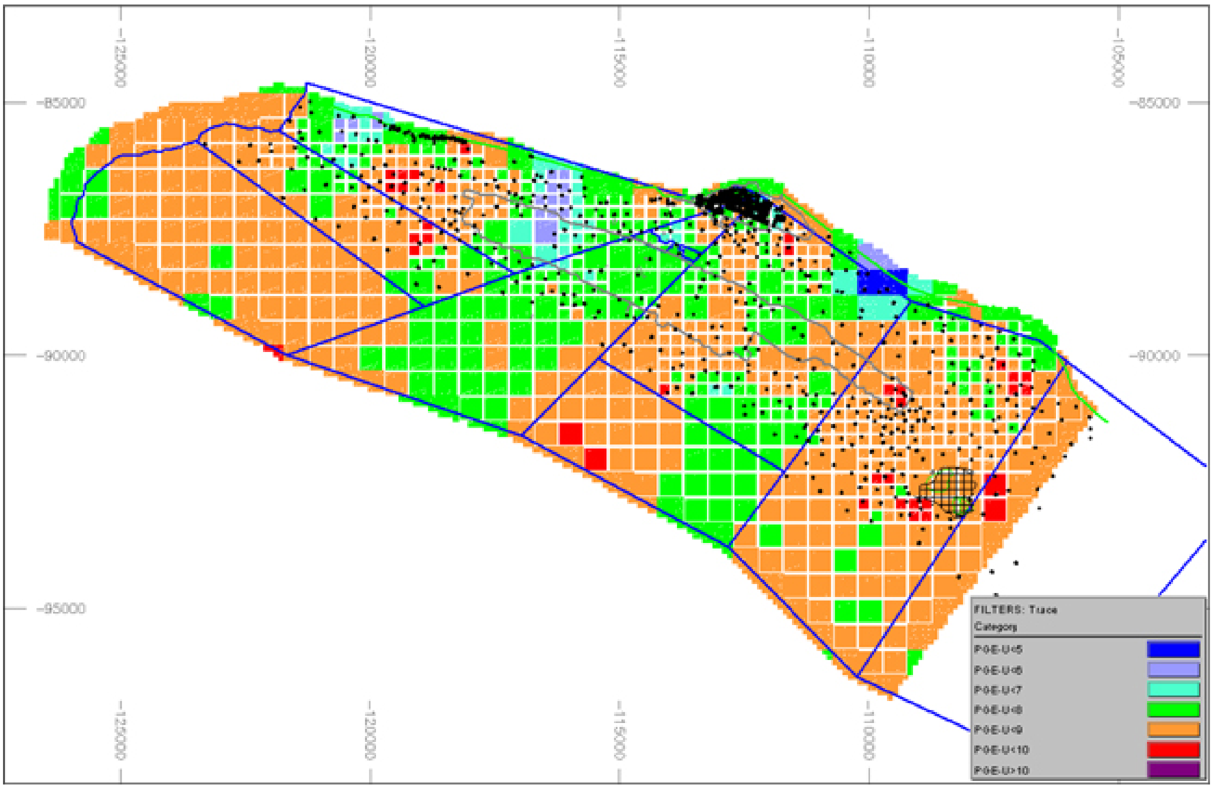


Figure 17.23: UG2 4E PGE grade distribution (Anglo Platinum, 2008).

UG2 Reef density distribution

The UG2 Reef density varies between 3.77 g/cm³ and 4.49 g/cm³ with a mean of 4.20 g/cm³. The density is lower in areas with a cluster of boreholes which intersected bifurcated UG2, on Zeekoegatand Umkoanesstad (see Figure 17.24).

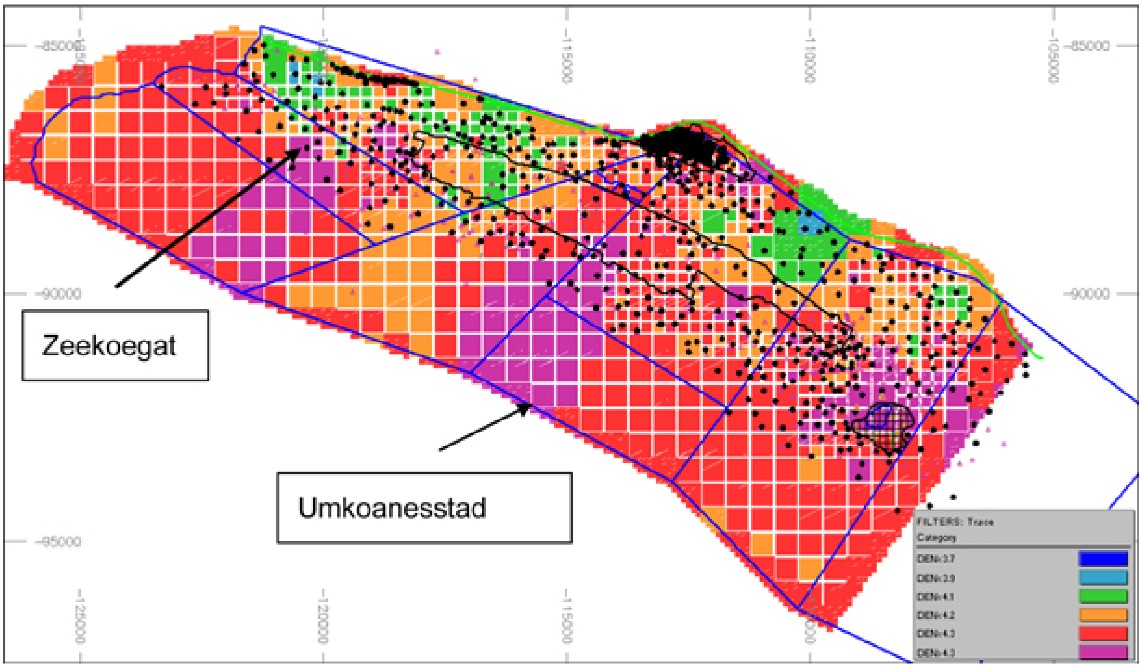


Figure 17.24: UG2 Reef density distribution (Anglo Platinum, 2008).

UG2 Reef internal dilution distribution

The UG2 Reef internal dilution shows a mean of 10.1% (8.5% - 2006). The UG2 internal dilution is higher in areas with a cluster of bifurcated reef intersections, Zeekoegat, Umkoanesstad and the deeper areas of Brakfontein (see Figure 17.25).

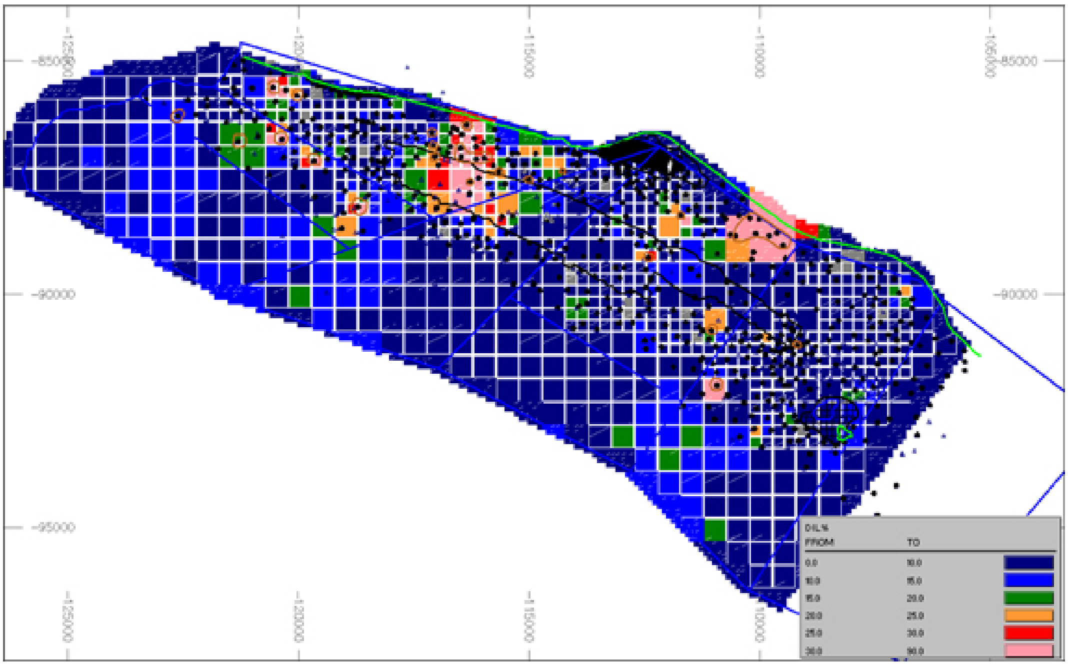


Figure 17.25: UG2 internal dilution distribution (Anglo Platinum, 2008).

UG2 Reef Resource mining cut thickness distribution.

The resource cut thickness distribution varies between 0.95 m and 2.04 m with a mean of 1.03 m (see Figure 17.26). In 2007, there was an increased cluster of bifurcated reef intersections within the Zeekoegat Farm. The change in the geotechnical hangingwall beam from 0.20 m to 0.30 m in 2007 increased the overall resource cut thickness.

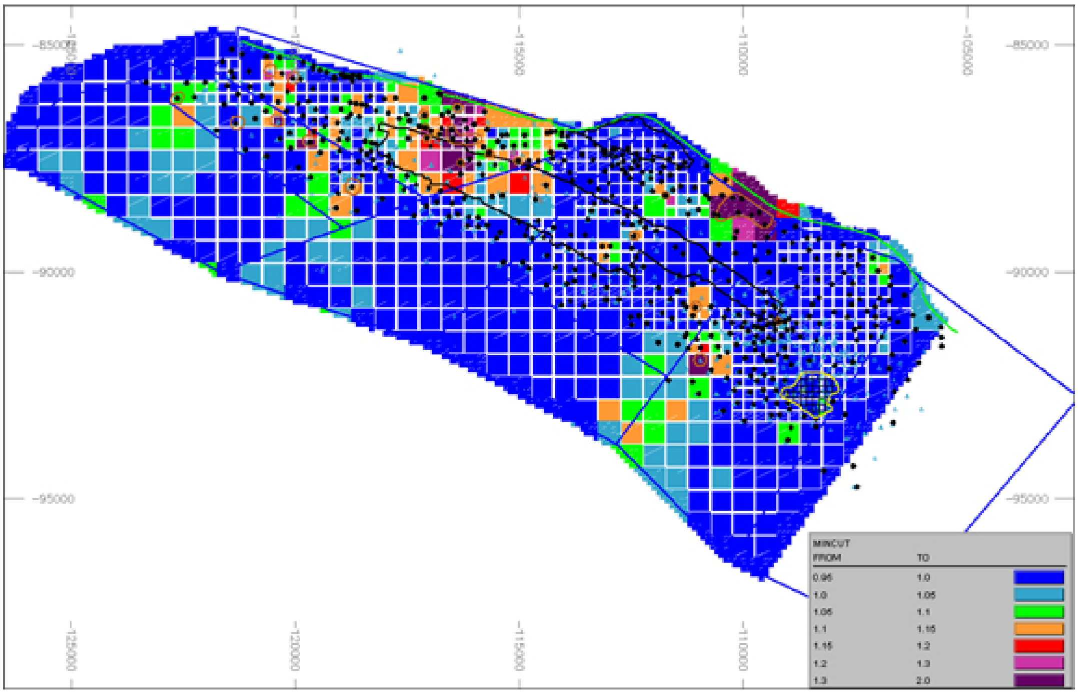


Figure 17.26: UG2 Reef Resource mining cut thickness distribution (Anglo Platinum, 2008).

UG2 Reef Resource mining cut 4E PGM grade distribution.

The resource cut grade distribution varies between 4.36 g/t and 7.61 g/t with a mean of 6.45 g/t (see Figure 17.27). Resource cut grades of less than 5 g/t exist in the northwest area of Zeekoegat and in the bifurcated area at Umkoanesstad.

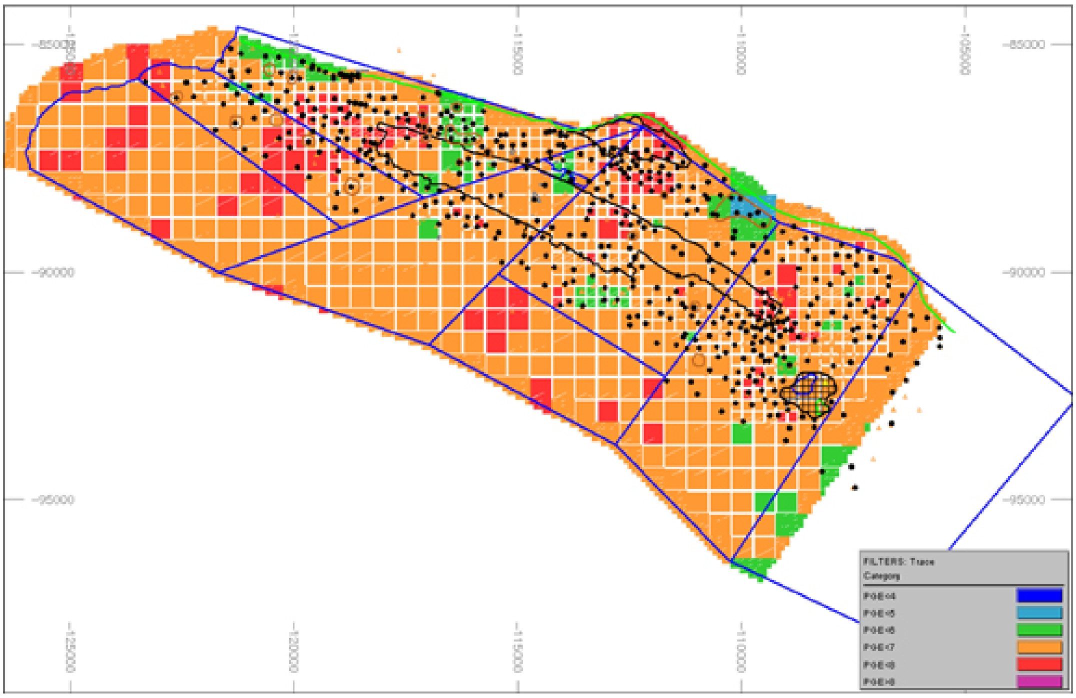


Figure 17.27: UG2 Reef Resource mining cut 4E PGM grade distribution (Anglo Platinum, 2008).

UG2 Reef Pt kriging variance distribution

The kriging variance is generally less than 0.2 in the area with dense drilling. In the shallow area the kriging variance is generally less than 0.3. An overall reduction in the kriging variance in the well informed area is noted (see Figure 17.28). A new kriging neighbourhood study in 2007 increased the block size in well informed areas from 100 m by 100 m in 2006 to 250 m by 250 m block size in 2007.

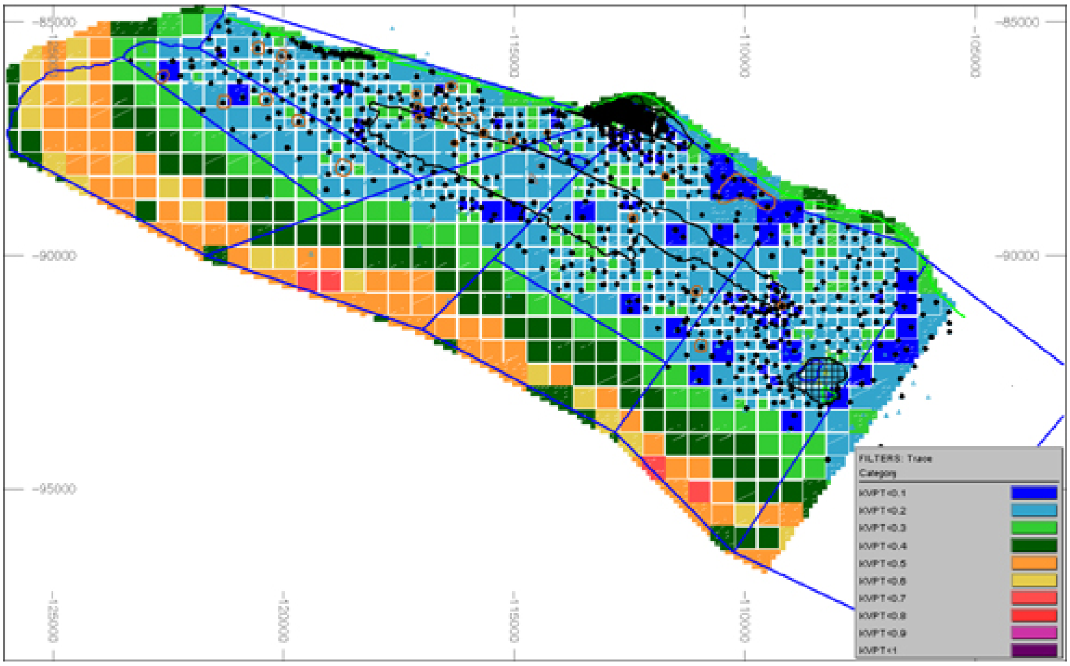


Figure 17.28: UG2 Reef Pt kriging variance distribution (Anglo Platinum, 2008).

UG2 Reef kriging efficiency distribution

The kriging efficiency generally exceeds 0.6 in the area with dense drilling (see Figure 17.29).

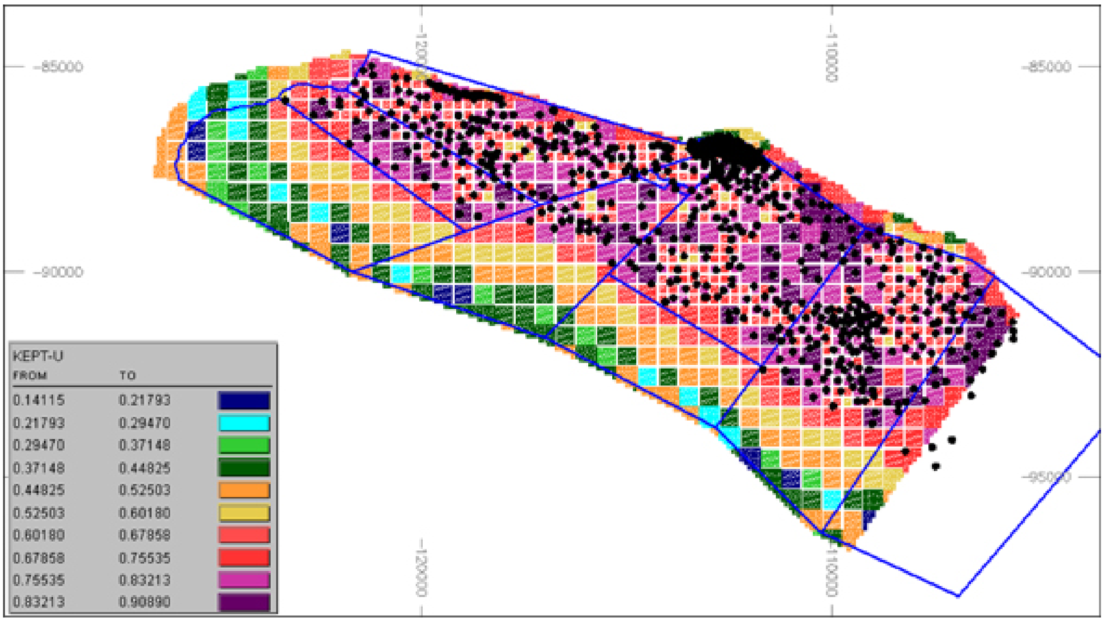


Figure 17.29: UG2 Reef kriging efficiency distribution (Anglo Platinum, 2008).

UG2 Reef Resource Mining cut thickness distribution

The UG2 Resource mining cut thickness distribution varies between 0.95 m and 1.80 m with a mean of 1.02 m (see Figure 17.30).

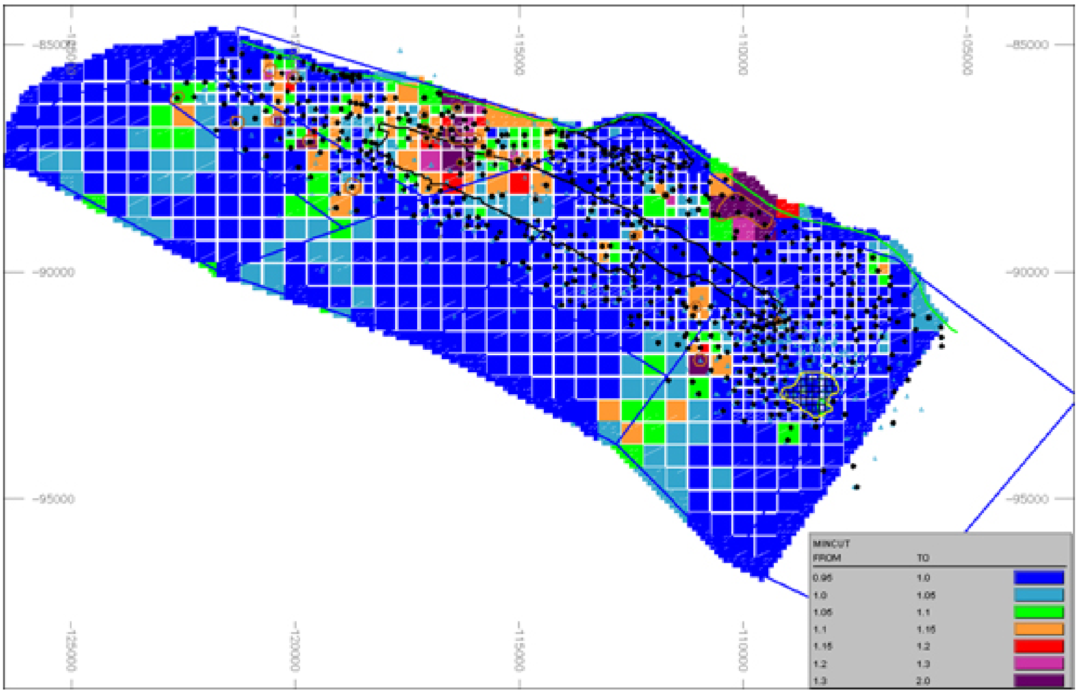


Figure 17.30: UG2 Reef Resource mining cut thickness distribution (Anglo Platinum, 2008).

UG2 Resource mining cut 4E PGE grade distribution

The resource cut grade distribution varies between 4.36 g/t and 7.61 g/t with a mean of 6.45 g/t (see Figure 17.31).

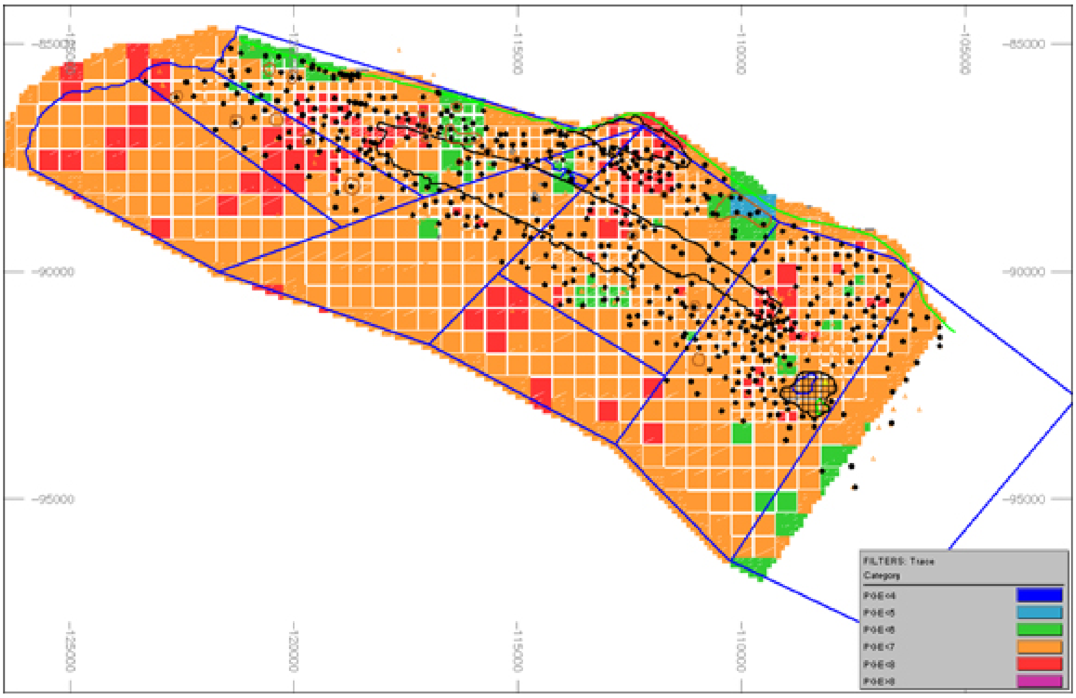


Figure 17.31: UG2 Resource mining cut 4E PGE grade distribution (Anglo Platinum, 2008).

17.4.10 UG2 Mineral Resource classification

The resource classification is based on the following parameters:

- kriging variance,
- kriging efficiency,
- borehole distribution,
- MRM sample distribution,
- variography (grade continuity),
- geological continuity,
- aeromagnetic survey,
- mining history,
- risk assessment, and
- the Competent Persons assessment.

The Mineral Resource at LPM was classified by S Malenga and reviewed by P Stevenson and M Patterson. The following criteria were used for the resource classification.

Measured Mineral Resource classification

A kriging variance of less than 0.15 has been used as one of the criteria to guide the determination and extent of the Measured Resource.

In addition, a kriging efficiency of greater than 0.8 was used to identify the Measured Mineral Resource area.

Indicated Mineral Resource classification

A kriging variance of less than 0.3 and greater than 0.15 has been used as the criteria to guide the determination of the extent of Indicated Mineral Resources.

In addition a kriging efficiency of greater than 0.6 was used to determine the Indicated Mineral Resource.

Figure 17.32 presents the UG2 Mineral Resource classification.

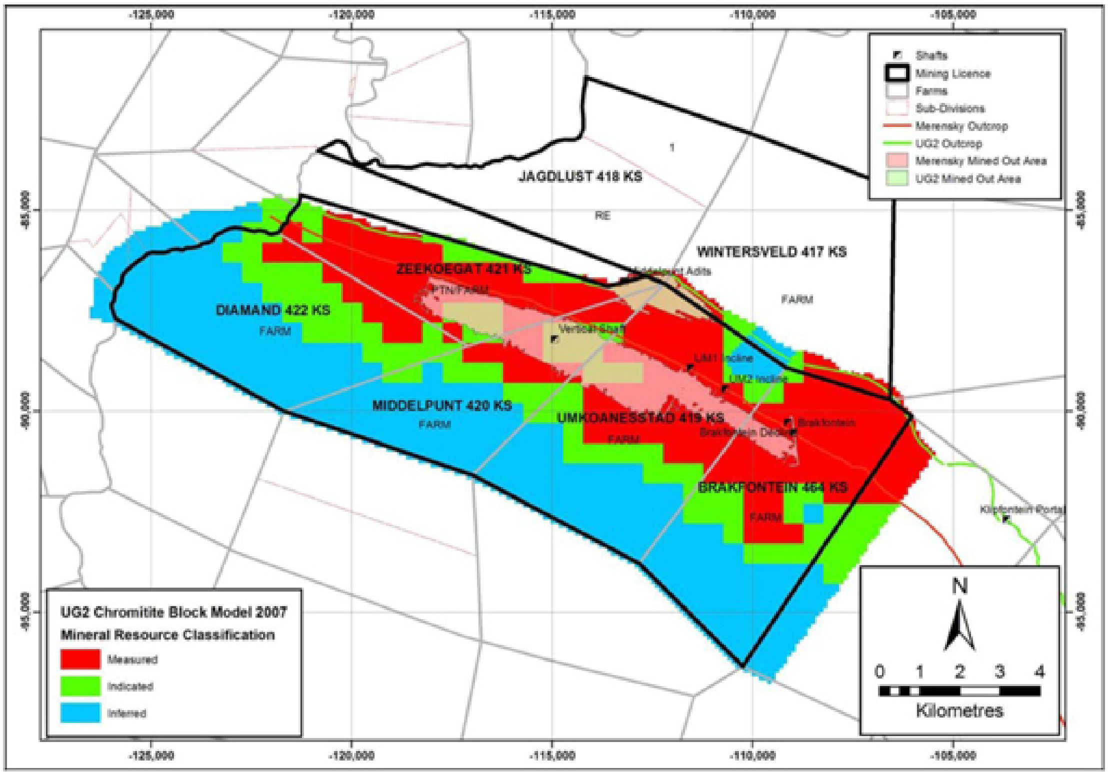


Figure 17.32: UG2 Mineral Resource classification as defined in 2007 (Anglo Platinum, 2008).

17.4.11 UG2 - Resource mining cut evaluation and reporting

Figure 17.33 shows the current Lebowa mining area and the UG2 Project areas used for resource reporting.

Tables 17.17 to 17.21 summarise the *in situ* mining width cuts of the Mineral Resources and their respective classifications.

Geological losses

The UG2 geological loss comprises 9% potholes, 4% dykes and 2% faulting. The current operation at the MPH UG2 Project has a geological loss of less than 5% potholes, 2% dykes and less than 1% faulting. However, due to the fact that the mined-out area is only a small proportion of the UG2 Mineral Resource, the 15% geological loss as utilised in the 2007 resource model has been applied.

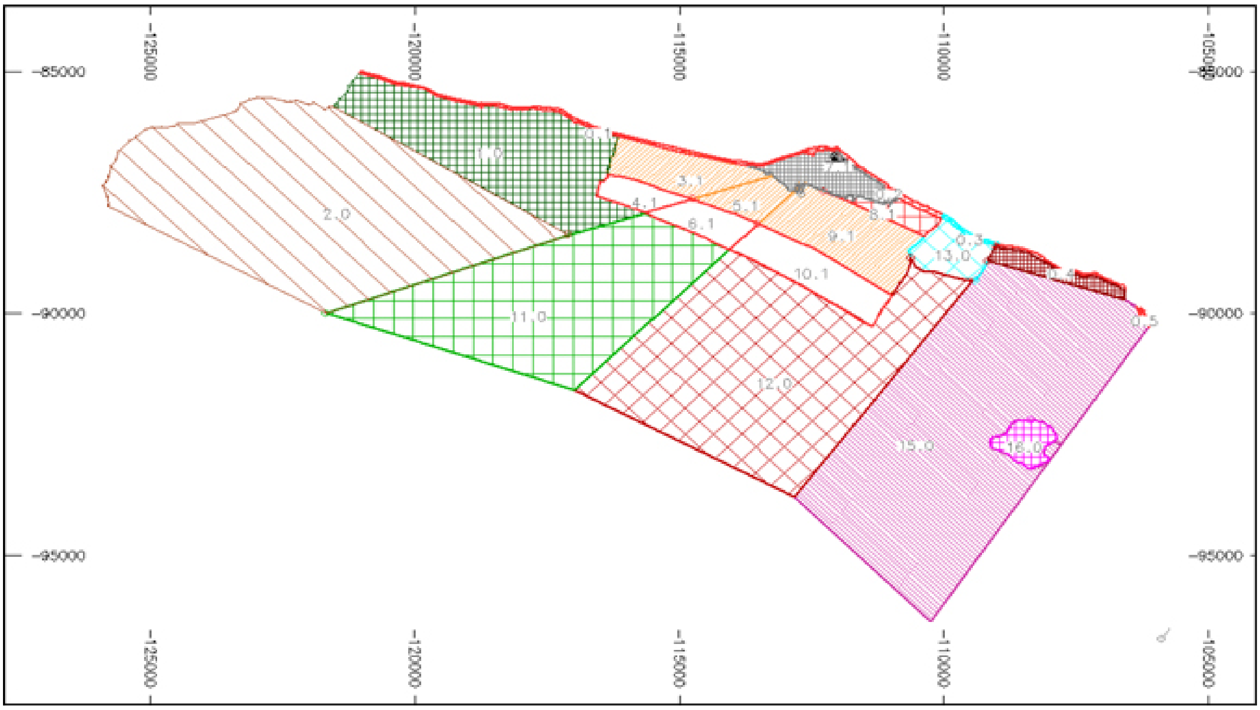


Figure 17.33: UG2 - Mineral Resource evaluation polygons (Anglo Platinum, 2008).

Table 17.17: UG2 Reef - Lebowa Mine Mineral Resource cut evaluation.

Resource Classification	Dip (o)	Width cut (m)	Density (g/cm3)	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)
MEASURED	18.1	0.95	4.04	15.0	1.16	6.71	0.25
INDICATED	18.0	1.19	4.03	15.0	1.16	6.01	0.22
MEASURED AND INDICATED	18.1	1.06	4.04	15.0	2.32	6.36	0.47
INFERRED		0.00	0.00	0.0	0.00	6.00	0.00

Table 17.18: UG2 Reef - Lebowa Mine Mineral Resource cut evaluation (exclusive of oxidised zones).

Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3)	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade (g/t) kriged	Rh grade (g/t) kriged	Au grade (g/t) kriged
MEASURED	18.1	0.95	4.04	15.0	1.16	6.71	0.25	637	2.89	3.13	0.57	0.12
INDICATED	18.0	1.19	4.03	15.0	1.16	6.01	0.22	718	2.59	2.77	0.54	0.10
MEASURED AND INDICATED	18.1	1.06	4.04	15.0	2.32	6.36	0.47	673	2.74	2.95	0.55	0.11
INFERRED		0.00	0.00	0.0	0.00	6.00	0.00	0	0.00	0.00	0.00	0.00

Table 17.19: UG2 Reef – Middelpunt Hill Project Mineral Resource cut (0-8 Level, excluding oxidised zones).

Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3)	Geo loss (%)	Tonnage after geo loss (Mt)	4E grade (g/t)	Content (4E Moz)
MEASURED	**20.5**	**1.04**	**4.03**	**15.0**	**31.53**	**6.50**	**6.59**
INDICATED	**21.3**	**1.09**	**4.04**	**15.0**	**8.11**	**6.27**	**1.63**
MEASURED AND INDICATED	**20.7**	**1.05**	**4.04**	**15.0**	**39.64**	**6.45**	**8.22**
INFERRED		**0.00**	**0.00**	**0.0**	**0.00**	**6.00**	**0.00**

Table 17.20: Middelpunt Hill Project – UG2 Resource Classification.

	Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3)	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade (g/t) kriged	Rh grade (g/t) kriged	Au grade (g/t) kriged
Block 3.1 ZK	MEASURED	25.0	1.12	4.01	15.0	7.66	6.21	1.53	696	2.56	3.02	0.51	0.12
	INDICATED	25.0	1.14	3.99	15.0	0.94	6.18	0.19	702	2.58	2.96	0.52	0.12
	MEASURED AND INDICATED	25.0	1.12	4.00	15.0	8.60	6.21	1.72	696	2.56	3.01	0.52	0.12
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block ZK	MEASURED	25.0	1.22	4.02	15.0	2.95	6.11	0.58	745	2.53	2.96	0.51	0.12
	INDICATED	25.0	1.85	3.91	15.0	0.72	5.54	0.13	1026	2.17	2.79	0.47	0.11
	MEASURED AND INDICATED	25.0	1.31	4.00	15.0	3.67	6.00	0.71	786	2.46	2.93	0.50	0.12
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 5.1 MP	MEASURED	22.0	0.97	4.04	15.0	2.58	6.26	0.52	605	2.63	2.98	0.54	0.11
	INDICATED	22.0	0.95	4.09	15.0	0.71	6.30	0.14	599	2.68	2.94	0.56	0.11
	MEASURED AND INDICATED	22.0	0.96	4.05	15.0	3.28	6.27	0.66	604	2.64	2.97	0.55	0.11
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 6.1 MP	MEASURED	22.0	1.21	4.04	15.0	0.67	6.32	0.14	767	2.64	3.02	0.53	0.13
	INDICATED	22.0	1.06	4.07	15.0	3.48	6.21	0.70	658	2.64	2.90	0.55	0.12
	MEASURED AND INDICATED	22.0	1.08	4.07	15.0	4.16	6.23	0.83	674	2.64	2.92	0.55	0.12
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 9.1 US	MEASURED	18.0	0.99	4.04	15.0	11.18	6.73	2.42	668	2.76	3.30	0.55	0.12
	INDICATED	18.0	1.15	3.96	15.0	0.78	6.00	0.15	693	2.45	2.92	0.52	0.11
	MEASURED AND INDICATED	18.0	1.00	4.03	15.0	11.97	6.68	2.57	669	2.74	3.28	0.55	0.12
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Block 10.1 US	MEASURED	18.0	0.97	4.07	15.0	6.49	6.75	1.41	652	2.77	3.30	0.56	0.12
	INDICATED	18.0	0.98	4.10	15.0	1.47	6.93	0.33	676	2.86	3.37	0.57	0.12
	MEASURED AND INDICATED	18.0	0.97	4.07	15.0	7.96	6.78	1.74	657	2.79	3.31	0.56	0.12
	INFERRED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
Total	MEASURED		1.04	4.03	15.0	31.53	6.50	6.59	673	2.68	3.17	0.54	0.12
	INDICATED		1.09	4.04	15.0	8.11	6.27	1.63	684	2.62	2.99	0.54	0.12
	MEASURED AND INDICATED		1.05	4.04	15.0	39.64	6.45	8.22	675	2.66	3.13	0.54	0.12
	INFERRED		0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00

Table 17.21: UG2 Reef Mineral Resources.

Resource Classification	Dip (o)	Resource cut (m)	Density (g/cm3)	Geo Loss (%)	Tonnage after Geo Loss (Mtons)	4E grade (4E g/t) (sum of prills)	Content (4E Moz)	Content cmg/t	Pt grade (g/t) kriged	Pd grade (g/t) kriged	Rh grade (g/t) kriged	Au grade (g/t) kriged
MEASURED	25.0	1.08	3.96	15.0	26.54	6.51	5.56	701	2.64	3.23	0.52	0.13
INDICATED	25.0	1.16	3.97	15.0	8.17	6.17	1.62	716	2.53	3.01	0.51	0.12
MEASURED AND INDICATED	25.0	1.10	3.96	15.0	34.71	6.43	7.18	704	2.61	3.17	0.52	0.13
INFERRED		0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
MEASURED	25.0	1.06	4.05	15.0	10.32	6.78	2.25	717	2.74	3.38	0.52	0.14
INDICATED	25.0	1.00	4.05	15.0	23.17	6.79	5.05	679	2.78	3.35	0.52	0.14
MEASURED AND INDICATED	25.0	1.02	4.05	15.0	33.49	6.78	7.31	691	2.76	3.36	0.52	0.14
INFERRED	25.0	0.99	4.02	15.0	49.84	6.82	10.93	673	2.82	3.31	0.54	0.14
MEASURED	22.0	0.99	4.07	15.0	8.00	6.55	1.68	649	2.78	3.07	0.56	0.14
INDICATED	22.0	0.98	4.05	15.0	9.67	6.50	2.02	638	2.75	3.08	0.54	0.13
MEASURED AND INDICATED	22.0	0.99	4.06	15.0	17.67	6.52	3.70	643	2.76	3.08	0.55	0.13
INFERRED	22.0	0.97	4.05	15.0	30.49	6.60	6.47	642	2.76	3.20	0.51	0.13
MEASURED	18.0	0.97	4.02	15.0	20.54	6.53	4.31	633	2.69	3.16	0.56	0.12
INDICATED	18.0	0.97	4.06	15.0	14.78	6.63	3.15	641	2.73	3.22	0.55	0.12
MEASURED AND INDICATED	18.0	0.97	4.04	15.0	35.32	6.57	7.46	636	2.71	3.19	0.56	0.12
INFERRED	18.0	0.98	4.09	15.0	30.76	6.74	6.66	658	2.74	3.31	0.56	0.12
MEASURED	0.0	0.00	0.00	0.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
INDICATED	18.0	1.22	3.92	15.0	0.13	5.95	0.02	728	2.41	2.93	0.50	0.11
MEASURED AND INDICATED	18.0	1.22	3.92	15.0	0.13	5.95	0.02	728	2.41	2.93	0.50	0.11
INFERRED	18.0	1.52	3.93	15.0	6.43	5.28	1.09	804	2.08	2.64	0.46	0.10
MEASURED	16.0	0.98	4.03	15.0	2.17	6.72	0.47	656	2.79	3.22	0.59	0.12
INDICATED	16.0	1.04	4.02	15.0	0.44	6.41	0.09	668	2.60	3.13	0.56	0.13
MEASURED AND INDICATED	16.0	0.99	4.03	15.0	2.61	6.67	0.56	658	2.76	3.21	0.59	0.12
INFERRED	16.0	1.14	4.03	15.0	0.04	6.13	0.01	701	2.45	3.01	0.54	0.12
MEASURED	16.0	0.96	4.02	15.0	38.95	6.66	8.35	642	2.73	3.25	0.57	0.12
INDICATED	16.0	1.03	4.04	15.0	14.23	6.44	2.95	667	2.64	3.16	0.54	0.11
MEASURED AND INDICATED	16.0	0.98	4.03	15.0	53.18	6.61	11.29	648	2.70	3.23	0.56	0.11
INFERRED	16.0	1.00	4.02	15.0	26.91	6.43	5.56	644	2.61	3.18	0.53	0.11
MEASURED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
INDICATED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
MEASURED AND INDICATED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
INFERRED	18.0	1.62	3.99	15.0	0.04	5.34	0.01	863	2.18	2.57	0.49	0.09
MEASURED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
INDICATED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
MEASURED AND INDICATED	0.0	0.00	0.00	15.0	0.00	0.00	0.00	0	0.00	0.00	0.00	0.00
INFERRED	18.0	1.31	3.92	15.0	0.03	5.74	0.01	752	2.28	2.85	0.51	0.10
MEASURED		0.99	4.02	15.0	1.95	6.57	0.41	651	2.73	3.16	0.55	0.12
INDICATED		1.16	4.00	15.0	1.32	6.17	0.26	716	2.52	3.01	0.52	0.12
MEASURED AND INDICATED		1.06	4.02	15.0	3.27	6.41	0.67	677	2.65	3.10	0.54	0.12
INFERRED		1.66	3.96	15.0	0.45	5.19	0.07	863	2.06	2.56	0.48	0.09
MEASURED		1.00	4.01	15.0	106.52	6.60	22.62	662	2.70	3.23	0.55	0.12
INDICATED		1.01	4.04	15.0	70.59	6.57	14.91	667	2.71	3.20	0.53	0.13
MEASURED AND INDICATED		1.01	4.02	15.00	177.11	6.59	37.53	664	2.70	3.22	0.54	0.12
INFERRED		1.00	4.04	15.0	144.55	6.61	30.74	662	2.72	3.23	0.53	0.13
MEASURED		1.00	4.01	15.0	108.47	6.60	23.03	663	2.70	3.23	0.55	0.12
INDICATED		1.02	4.04	15.0	71.91	6.56	15.17	669	2.70	3.20	0.53	0.13
MEASURED AND INDICATED		1.01	4.02	15.0	180.38	6.59	38.20	666	2.70	3.22	0.54	0.12
INFERRED		1.00	4.04	15.0	145.00	6.61	30.82	664	2.72	3.23	0.53	0.13

17.4.13 UG2 - Mineral Reserve Statement

The modifying factors applied for the Lebowa Approved Projects supporting the documented Mineral Resource to Reserve conversion process are detailed in Table 17.22. A minimum stope width of 0.95 m has been used. The mining cut has been approved by C Ackhurst (MTS Manager). The resource cut is based on the sum of the geotechnical hangingwall cut, the UG2 and a minimum of 0.1 m footwall component.

Table 17.22: UG2 modifying factors (Anglo Platinum, 2008).

		Proved Reserve		Probable Reserve		
3.1 Extraction	**Comment**	**percent**	**grade (4E)**	**percent**	**grade (4E)**	**content (g)**
- Pillar Loss						
	- Regional and in stope pillars					
	- Assuming that some Regional pillars can be superimposed on geological losses. This can either be calculated or determined	15.0%		9.0%		
	- % of Resource available for conversion.[2]					
- Post calculation depletion	- Area mined between calculation date and 31 December (reserves statement date)					
Available to mine after pillar loss	Actual mineable area after all losses.		6.51		6.23	49,027,244
% Extraction (% of dip corrected area)		85.0%		85.0%		
3.2 Stoping						
3.2.1. Stoping (Dimensions)						
Stoping reef cut	Percent of m2 from stoping	98.8%	6.51	98.5%	6.24	48,241,091
+ ASGs	additional % tonnage of ASGs	5.0%	0.50	5.1%	0.51	201,367
+ Winch beds	additional % tonnage from w/b	1.0%	0.49	0.9%	0.50	36,488
+ Sliping	additional % tonnage from sliping	0.0%	0.00	0.0%	0.00	0
+ Other (to be clearly defined)	(comment here)	0.0%	0.00	0.0%	0.00	0
Sub Total b/f			6.17		5.91	48,478,946
3.2.2. Stoping (Modifying factors)						
- Off reef mining loss	loss in content (grade x tons)	1.6%	6.51	2.3%	6.32	-977,741
+ Off reef mining (add back)	add back tons at no grade					
- RIF/RIH loss	loss in content (grade x tons)	0.2%	5.50	0.6%	5.50	-254,025
+ RIF/RIH (add back)	add back tons at reduced grade					
+ Off reef development (re-dev)	add tons at no grade	2.0%		0.0%		
+ Overbreak/ dilution	additional tonnage at reduced/no grade	5.0%	1.00	5.1%	1.00	391,443
+ Scaling	additional tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00	0

+ FoG	additional tonnage at reduced/no grade	0.8%	0.00	0.0%	0.00	0
+ Other (to be clearly defined)	X-tram of Waste from Dev Ends	0.0%	0.00	0.0%	0.00	0
Total - Stoping (net)			**5.69**		**5.53**	**47,638,623**
3.3 Development						
3.3.1. Development (Dimensions)						
Dev reef cut	at reef grade	1.2%	6.54	1.5%	6.15	786,153
+ Dev Waste cut	additional tonnage at reduced/no grade	156%	0.96	196%	0.94	236,807
+ Cubbies - Reef cut	at reef grade	0%	0.00	0%	0.00	0
+ Cubbies - Waste cut	additional tonnage at reduced/no grade	0%	0.00	0%	0.00	0
+ Other (to be clearly defined)	(comment here)	0%	0.00	0%	0.00	0
Sub Total b/f			3.14		2.70	1,022,960
3.3.2. Development (Modifying factors)						
- Off reef mining loss	loss in content (grade x tons)	0.0%	6.54	1.1%	6.15	-8,625
+ Off reef mining (add back)	add back tons at no grade					
- RIF/RIH loss	loss in content (grade x tons)	0.0%	6.54	0.0%	6.15	0
+ RIF/RIH (add back)	add back tons at reduced grade					
+ Off reef development (re-dev)	add % tonnes at low/no grade	1.6%	0.00	1.1%	0.00	0
+ Overbreak/ dilution	add % tonnage at reduced/no grade	5.7%	0.52	3.8%	0.52	2,505
+ Scaling	add % tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00	0
+ FoG	add % tonnage at reduced/no grade	0.0%	0.00	0.0%	0.00	0
+ Other (to be clearly defined)	X-tram of Waste from Dev Ends	36.1%	0.00	91.9%	0.00	0
Total - Development b/f			2.66		2.02	1,016,840

The Mineral Resource to Mineral Reserve conversion for the UG2 was based on the kriged block model grades. The resource tonnage and grade was converted to a plant head grade by applying the relevant modifying factors.

After applying the geological losses and modifying factors, the entire UG2 horizon is mined therefore no cut-off grade needs to be applied.

Table 17.23 summarizes the Proven and Probable Mineral Reserves for the UG2 horizon.

Table 17.23: UG2 - Mineral Reserve Statement - December 2008.

Category	Tonnage (Mt)	4E grade (g/t)	Contained metal (Moz)
Proven	**32.10**	**5.43**	**5.60**
Probable	**9.10**	**5.17**	**1.50**
Total	**41.20**	**5.37**	**7.10**

Notes: Mineral Reserves have been determined by applying modifying factors to the Mineral Resources. Tonnes and ounces have been rounded and this may have resulted in minor discrepancies. The 4E elements are platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au). Only Measured and Indicated Resources have been converted to Mineral Reserves. The Mine Call Factors used in the Mineral Reserve calculations are 97% for the Proven and Probable Reserves. The Mineral Reserves were calculated by Anglo Platinum (Mafoko and Hanekom, 2008).

17.4.14 Risk assessment results

The risk assessment is the first step, after which further action is taken, depending on whether the controls to be applied require elimination, mitigation or tolerance of the risk.

Areas of concern

- The difference between pre 2000 and post 2000 grade needs to be investigated to determine whether a fire assay correction factor should be applied.
- Special mining considerations may need to be considered where bifurcated reef exceeds 1.20 m i.e. mining at stoping widths greater than 1.20 m will require specific support patterns including more areal support (packs rather than sticks).
- The MRM system should be upgraded in order to cater for storage of prill split data for the underground sampling.
- The lateral continuity of the bifurcated UG2 intersections needs to be investigated by pitting and trenching where the bifurcated reef outcrops.

17.5 Known issues that materially affect mineral resources and mineral reserves

LPM has successfully mined and processed Merensky Reef and UG2 and has economically produced 4E concentrate for the last 25 years. However, Deloitte notes that the SAMREC Code clearly states that for all Mineral Reserves, parameters should be detailed with engineering studies completed to at least a pre-feasibility level, and a LOM plan for an operation. This is not the case for the 2007 estimate, as it includes areas such as Zeekoegat and the Brakfontein UG2 (Snowden, 2008).

The mine call factor (MCF) applied to the 2008 UG2 Reserve estimations was 97%, whereas a

MCF of 94% had been agreed to, based on historical information. The use of a 97% MCF instead of 94% resulted in a 3% increase in the grade and contained metal content of the reported reserves, but does not affect the tonnage.

The UG2 and Merensky Reef 2007 and 2008 Reserve estimates were determined by applying modifying factors based on historical insights and experience, independent from the mining plan and scheduling process. This approach, according to Deloitte’s interpretation of the SAMREC Code, may invalidate code compliancy, as the SAMREC Code requires Reserves for operating mines to be based on a Life of Mine plan. National Instrument 43-101, which follows the CIM Definition Standards, requires a pre-feasibility study as a minimum basis on which to base reserve estimation.

The total 2008 declared UG2 Reserve is 41.2 Mt, however the mining schedule generates a 44.9 Mt reserve. The total 2008 declared Merensky Reserve is 27.1 Mt, as compared to a scheduled output of 25.8 Mt. These discrepancies can be attributed to reserve estimation being conducted independently from mine planning and scheduling.

Areas for continuous improvement

Anglo Platinum completes a risk assessment for each of their Mineral Resource estimates. The assessment is based upon the SAMREC codes Appendix 1 checklist and provides their competent persons with a comprehensive understanding of the risks that may affect their estimates. The risk assessment is an initial step identifying risk items that require elimination, mitigation or tolerance of the risk, after which action is taken, and controls are implemented for monitoring progress. The risk assessment follows a standard method of a semi-quantitative measurement based on the concept of risk as a product of probability and consequence. The following areas of concern are noted from Anglo Platinum’s risk assessment:

Merensky

- Borehole logs do not always identify potholes or the edge of potholes.
- Further work is necessary to quantify the geological loss more accurately.
- Further work is necessary to understand the sporadic footwall mineralisation so that the mining cut resource extraction can be optimised.

UG2

- The difference between pre 2000 and post 2000 grade needs to be investigated to determine whether a fire assay correction factor should be applied.
- Special mining considerations may need to be considered where bifurcated reef exceeds 1.20 m i.e. mining at stoping widths greater than 1.20 m will require specific support patterns including more areal support (packs rather than sticks).
- The assaying problems at EBRL need to be resolved as they are affecting the updating of the underground sampling database and the resource estimates.

- The MRM system should be upgraded in order to cater for storage of prill split data for the underground sampling.
- The lateral continuity of the bifurcated UG2 intersections needs to be investigated by pitting and trenching where the bifurcated reef outcrops.

Deloitte concurs that these areas of concern need to be addressed. It is the QP's opinion that these risks will not have a material effect on future Mineral Resource estimates and may be viewed as continuous improvement items.

18 Other relevant data and information

Coffey Mining (South Africa) Pty. Ltd. trading as RSG Global was engaged by Anooraq Resources Corporation to prepare an independent Technical Review of the operations and projects owned by LPM. The purpose of this study was to confirm that the existing operations and the plans of the proposed mining operations, processing facilities and all other associated existing infrastructure and services were based on appropriate and adequate data and were developed in terms of sound engineering principles and generally accepted environmental guidelines and practices. The report submitted on 9 April 2008 presents, in detail, all outcomes of the study based on the agreed mining operations and the experienced commodity prices then.

However, due to the very recent world wide economic crisis including a very significant decrease in the commodity prices and financial markets, a need for a modification on the agreed previous mining activities, especially for the short term, i.e. 4-5 years, was established.

Deloitte has reviewed the previous study and the “new plan” with the other associated relevant information and corroborated the information found therein with their own findings on the site visit and through information found in the virtual data room.

18.1 Grade-tonnage curves

In this section the characteristics of the deposit, such as grade, potential cutoff grades and reef thickness, are considered, for both the UG2 and the Merensky Reef. In addition, the *in-situ* value of the material is analysed and viewed in the light of the currently prevailing economic climate.

Figure 18.1 considers the entire UG2 deposit at Lebowa Platinum based on kriged block data (files MRW-21-1-2008-cad250/500.grid and UGW-13-1-2008-cad50/100/250/500.grid). The curves can be applied to either the Resource or Reserve estimates provided that the weathering depth, geological losses and mined-out ore are non-selective in terms of grade or indicated stoping width. The crude negative relationship between stoping width and grade is depicted in Figure 18.2, whereas the kriged stoping width distribution is shown in Figure 18.3. Since the bulk of the deposit has stoping widths between 95 and 130 cm (Figure 18.4), the probability of possible selective depletion due to the relevant factors appears negligible.

Figure 18.5 shows the grade-tonnage curve for the Merensky Reef. The arguments for the applicability of this curve to the resource and reserve estimates are the same as for the UG2. The bulk of the Merensky Reef shows narrower stoping widths combined with higher grades (Figures 18.6 and 18.7). The stoping width increases towards the northwest of the LPM property and stoping widths above 105 cm represent approximately 15% of the total deposit.

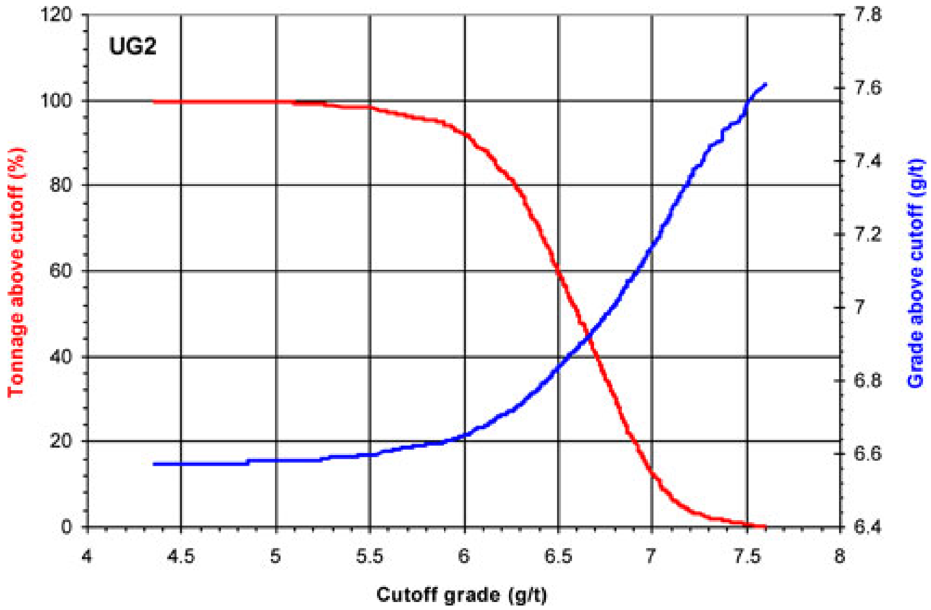


Figure 18.1: Grade-tonnage curves for UG2. The critical part of the tonnage curve lies at a cutoff grade of between approximately 6 and 7 g/t.

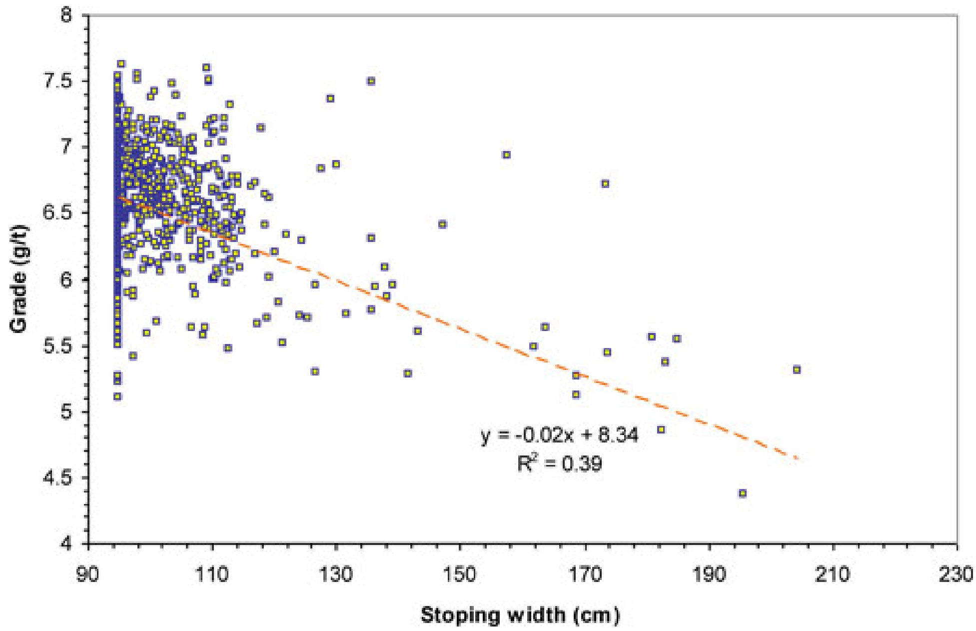


Figure 18.2: Grade versus stoping width of UG2 kriged data. Crude negative correlation is suggested.

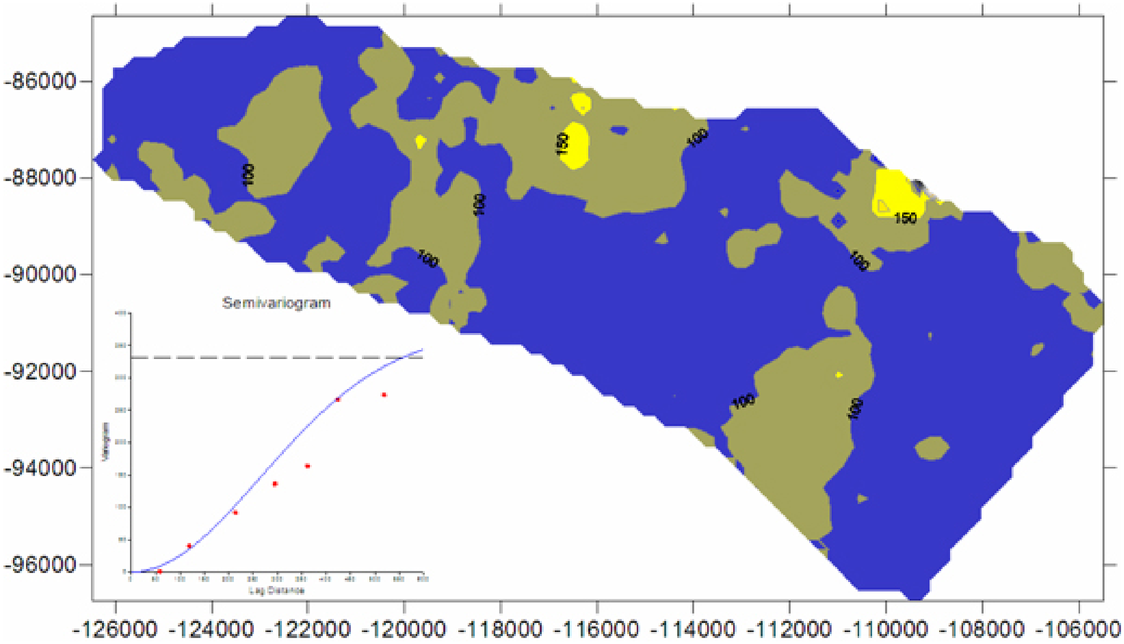


Figure 18.3: Stoping width of UG2 across Lebowa Platinum (kriged diagram of kriged widths). Semivariogram (inset) is omnidirectional and suggests stoping width isotropy.

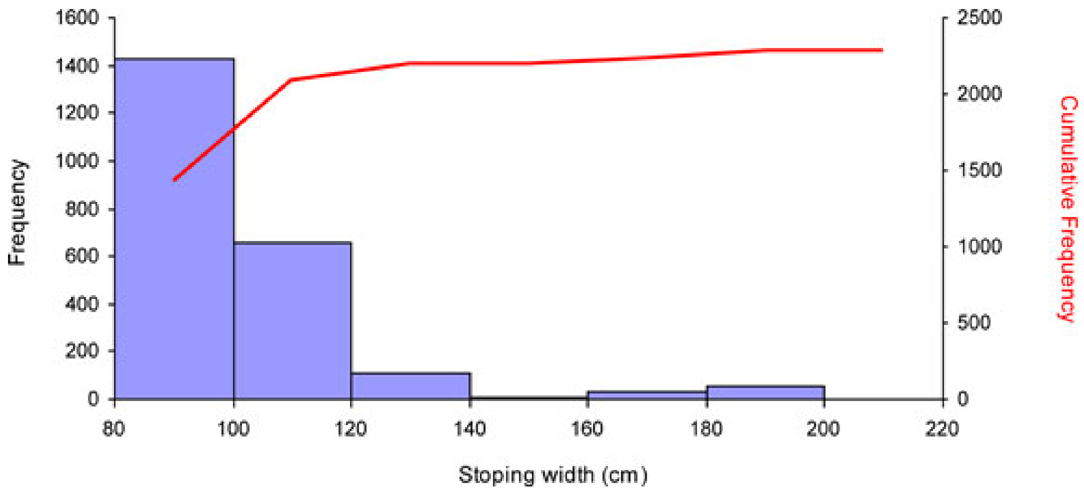


Figure 18.4: Stoping width distribution of UG2.

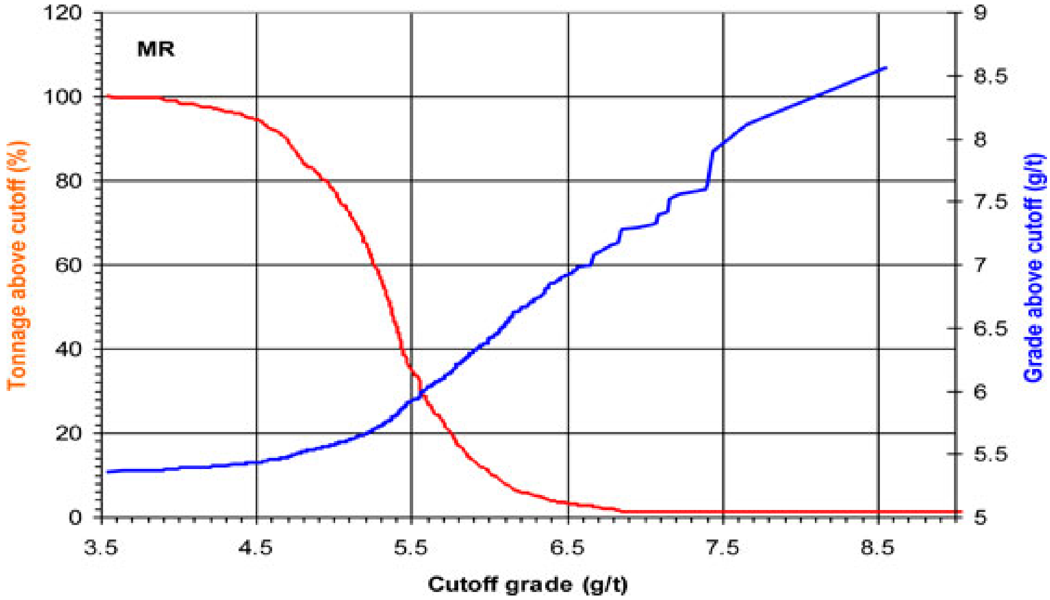


Figure 18.5: Grade-tonnage curves for Merensky Reef. The critical part of the tonnage curve lies at a cutoff grade between approximately 4.5 and 6.5 g/t.

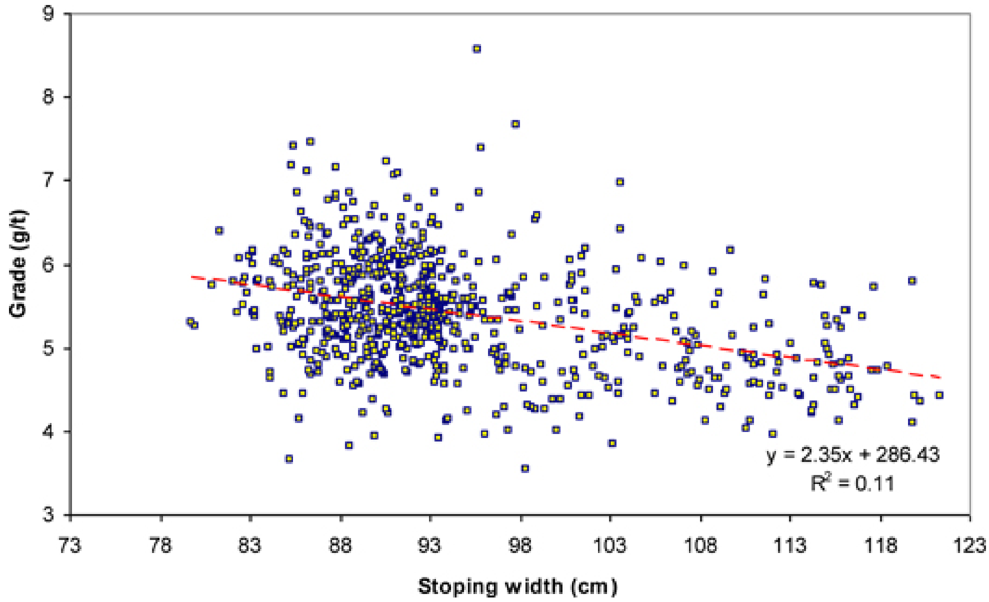


Figure 18.6: Grade vs. stoping width of Merensky Reef kriged data. Crude negative correlation is suggested.

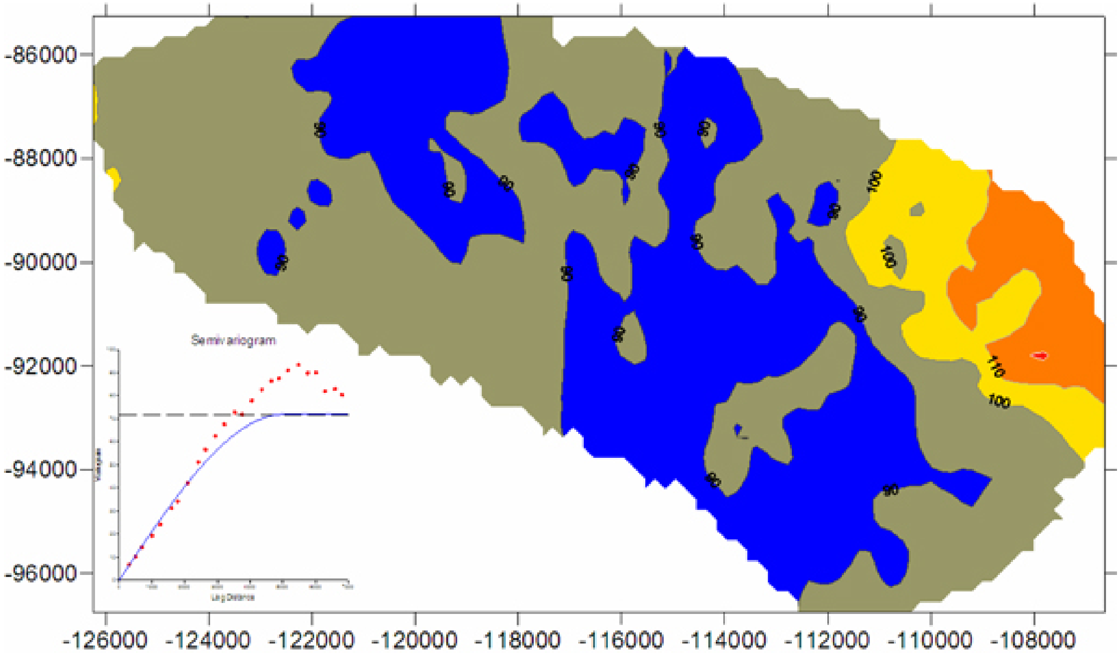


Figure 18.7: Stoping width of Merensky Reef across Lebowa Platinum (kriged diagram of kriged widths). Semivariogram (inset) is directional and suggests north-south anisotropy of stoping width.

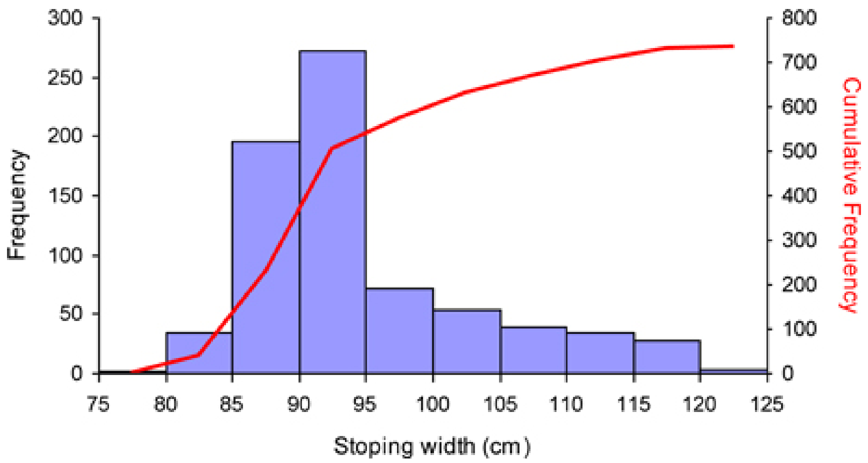


Figure 18.8: Stoping width distribution of Merensky Reef.

Variability of grade-tonnage in the UG2

The final kriged block model data for the UG2 (file FINMOD07-UG2.csv) contains references to Measured, Indicated and Inferred Resources. Since these resource groups out of necessity refer to different areas across the LPM property, they offer an opportunity to evaluate the lateral variability of the UG2 ore as well as the variance due to a relative increase in information. It is also known that the Inferred Resources mostly refer to areas deeper than 650 m.

Grade-tonnage curves provided in Figure 18.9 suggest that the Measured and Inferred resource estimates are for practical purposes indistinguishable. However, the Indicated estimate shows a substantial difference from the other two. The most likely source of this variance could result from a difference in sample size used to draw the grade tonnage curves (Table 18.1).

Table 18.1: Sample sizes considered for grade tonnage curves.

	n	Volume (km^3)
Measured	957	0.043
Indicated	407	0.026
Inferred	1035	0.049

The observed consistency in the grade–tonnage curves suggests an inherent lateral chemical uniformity of the UG2, which is generally true for Bushveld chromitite layers.

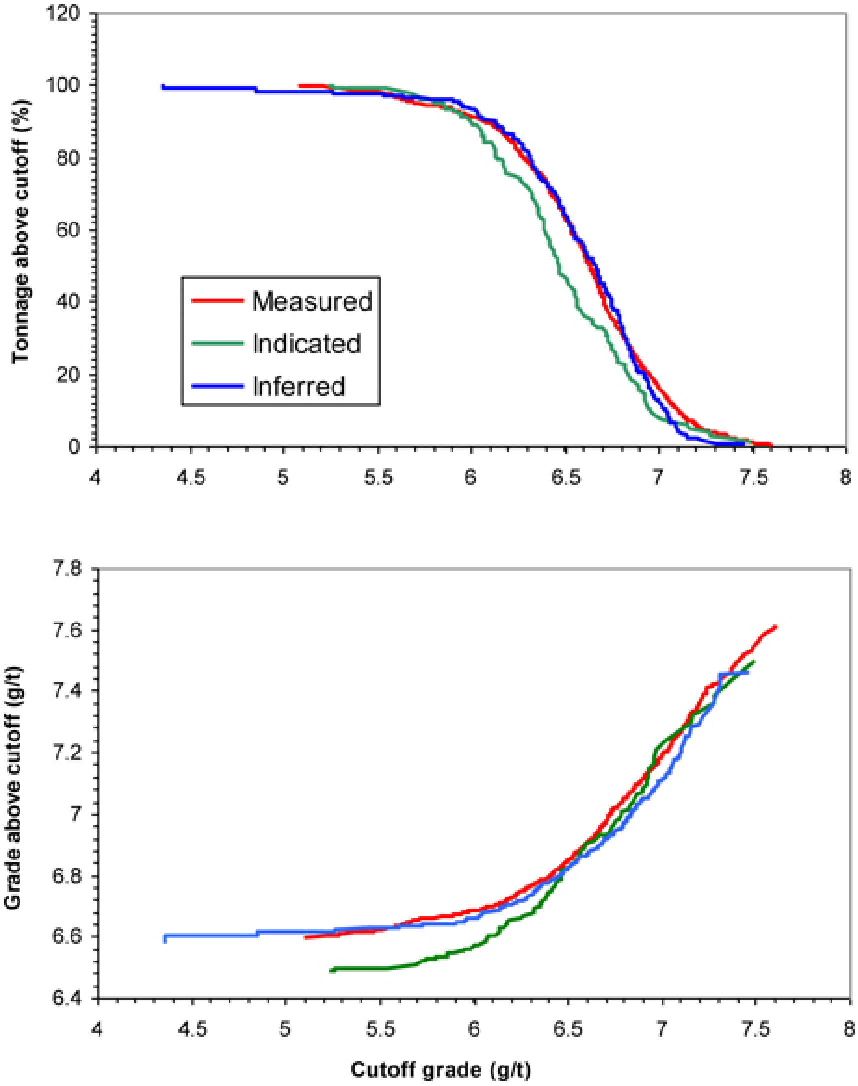
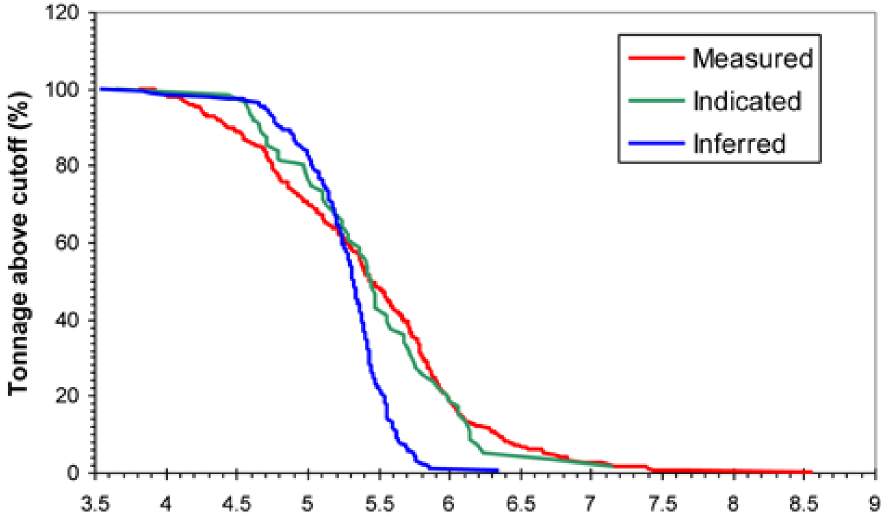


Figure 18.9: Grade–tonnage curves for Measured, Indicated and Inferred Resources in UG2. Legend in upper diagram (grade-tonnage) also applies to lower diagram (grade above cutoff grade).

A similar exercise applied to the Merensky Reef (file FINMOD-07.csv) produced more inconsistent results (Figure 18.10) for the different resource categories. This is to be expected in terms of the higher lateral chemical and width variability in this unit relative to the UG2.



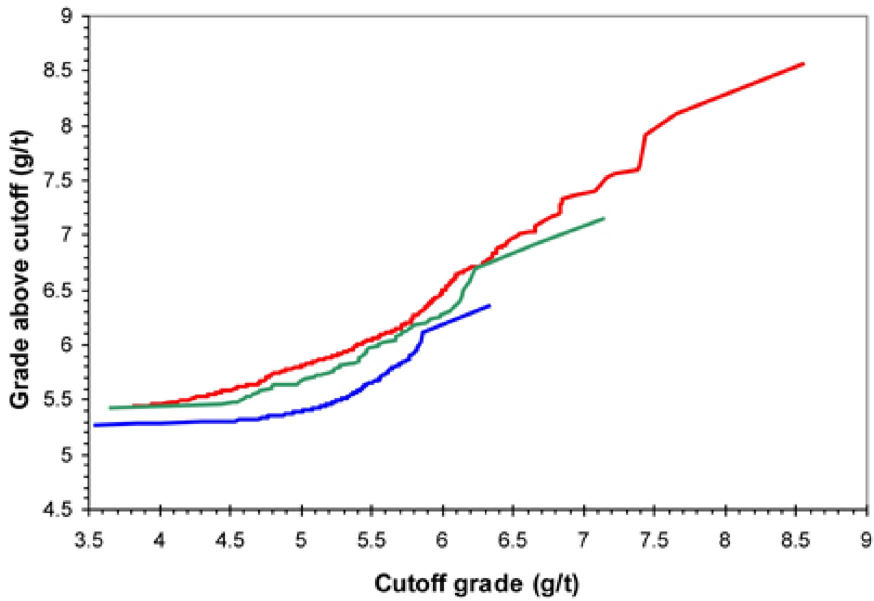


Figure 18.10: Grade–tonnage curves for Measured, Indicated and Inferred Resources in Merensky Reef. Legend in upper diagram (grade-tonnage) also applies to lower diagram (grade above cutoff grade).

Financial erosion of in situ value of the Merensky Reef and UG2

The metal prices and the ZAR/USD exchange rate deteriorated significantly since October 2007, as shown in Table 18.2.

Table 18.2: Metal prices and exchange rate for period under consideration.

Metal price	Pt (US$/oz)	Pd (US$/oz)	Rh (US$/oz)	Au (US$/oz)	Ni (US$/lb)	Cu (US$/lb)	Co (US$/lb)
Avg Oct 07	1410.96	365.61	6172.17	754.60	13.80	3.75	30.00
Avg Sept 08	1223.18	247.93	4515.68	829.93	7.90	3.40	33.00
Avg Jan 09	947.39	187.85	1006.22	851.42	5.00	1.40	18.50

Exchange rate	ZAR/USD
Avg Oct 07	6.78
Avg Sept 08	8.05
Avg Jan 09	9.86

The observed drop in metal prices combined with the deterioration of the ZAR/USD exchange rate had a measurable influence on the in situ value of the Merensky Reef and UG2 ore. Table 18.3 summarizes this effect for the 4PGE's. The observed differences between the Merensky Reef and the UG2 are attributable to different metal ratios.

It is noted that this analysis specifically focuses on spot metal prices, which may differ significantly from long term forecasts.

Table 18.3: Relative percentage loss of value of in situ ore (4PGE only) due to deterioration in metal prices and ZAR/USD exchange rates.

	Dollar (USD)			Rand (ZAR)		
	% loss between Oct 2007 and Sep 2008	% loss between Sep 2008 and Jan 2009	% loss since Oct 2007	% loss between Oct 2007 and Sep 2008	% loss between Sep 2008 and Jan 2009	% loss since Oct 2007
Merensky Reef	-16	-30	-42	-1	-15	-15
UG2	-21	-43	-55	-6	-30	-34

Conclusions

In summary, it is concluded that grades are fairly evenly distributed, but scope for domaining exists.

A significant margin exists between cutoff grades and paylimits, pointing to the inherent economic viability of the orebodies under consideration.

Considering the financial erosion of the *in-situ* value of the two orebodies, and the depreciation of the rand against the US dollar, the UG2 shows significantly more deterioration when compared to the Merensky Reef (Table 18.3). This is attributed to the differing prill splits of the orebodies, with the UG2 being characterised with higher rhodium contents.

18.2 LPM Depletion Strategy

18.2.1 Mining blocks

Lebowa is extensive, covering some 15,500 ha including a strike length of almost 16.5 km and a dip length of almost 10 km.

The Merensky and UG2 mineralised horizons are in the order of 300 m apart and either outcrop (in the hills) or subcrop (in the valleys) extend to depths beyond 2000 m below surface, towards the south-western boundary.

Figure 18.11 illustrates the LPM properties, the property boundaries, the outcrop/shallow subcrop of the Merensky and UG2, the current workings and the various access shafts.

The extent of Lebowa Platinum Mine requires a tailored depletion strategy, including various phases. Therefore the mining rights area has been divided into a series of mining blocks according to:

- strike length, of approximately six kilometres – the realistic strike that can be serviced by one shaft system, and
- depth, no more than nine production levels per shaft system.

The various blocks which relate to the Lebowa properties are shown in Figures 18.12 and 18.13 for Merensky and UG2, respectively.

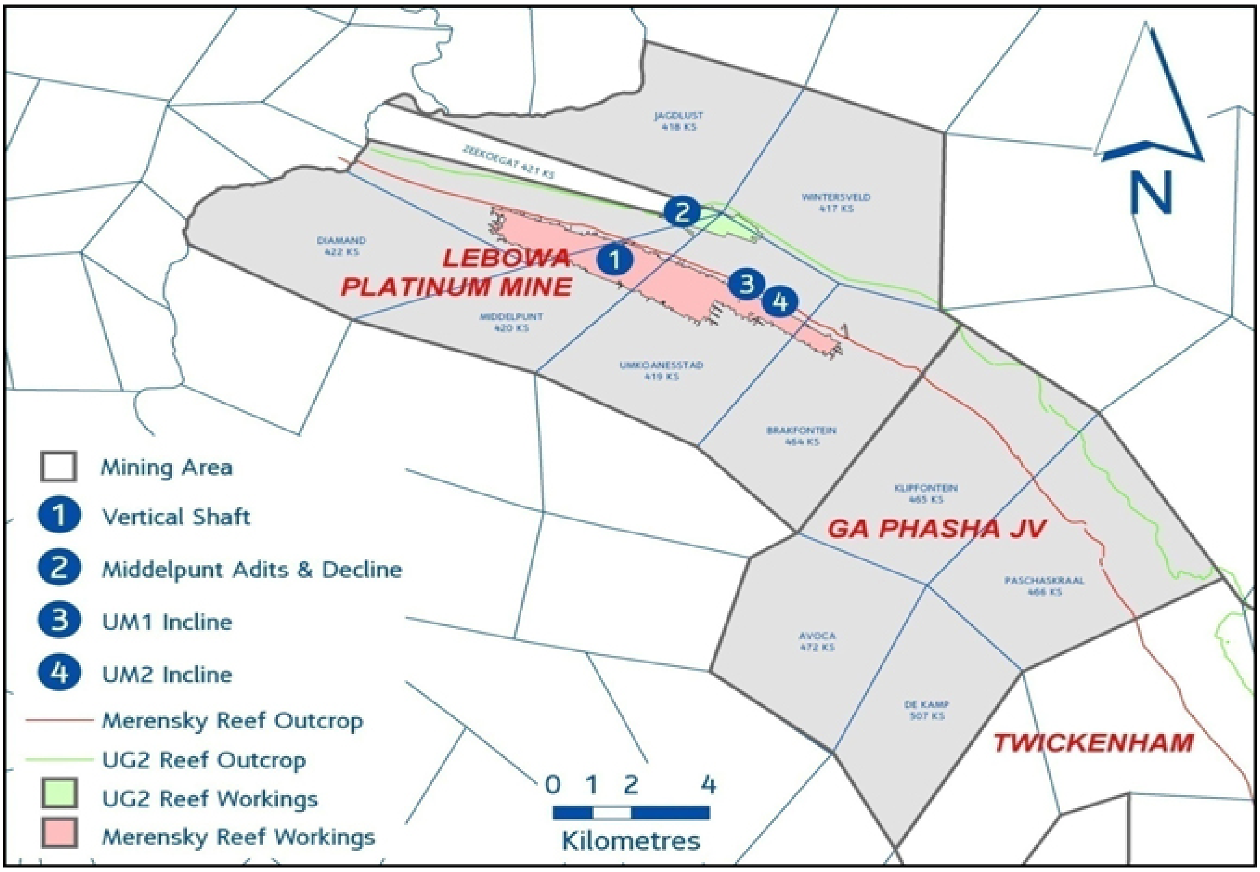


Figure 18.11: LPM mining rights area (Anglo Platinum, 2008).

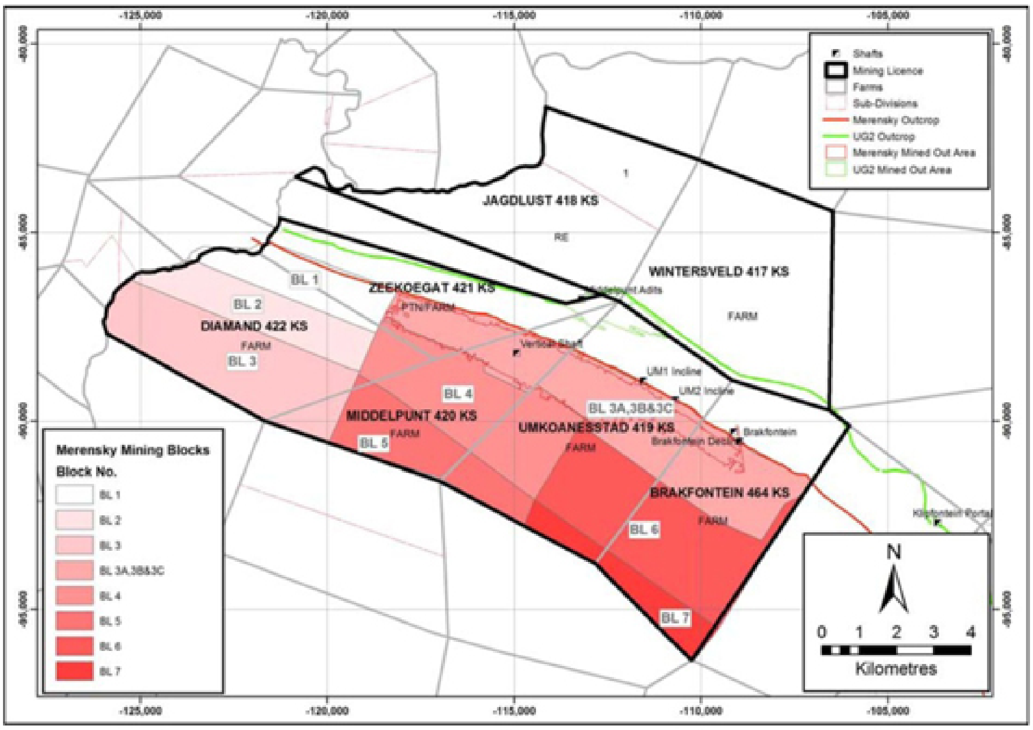


Figure 18.12: Merensky mining blocks (Anglo Platinum, 2008).

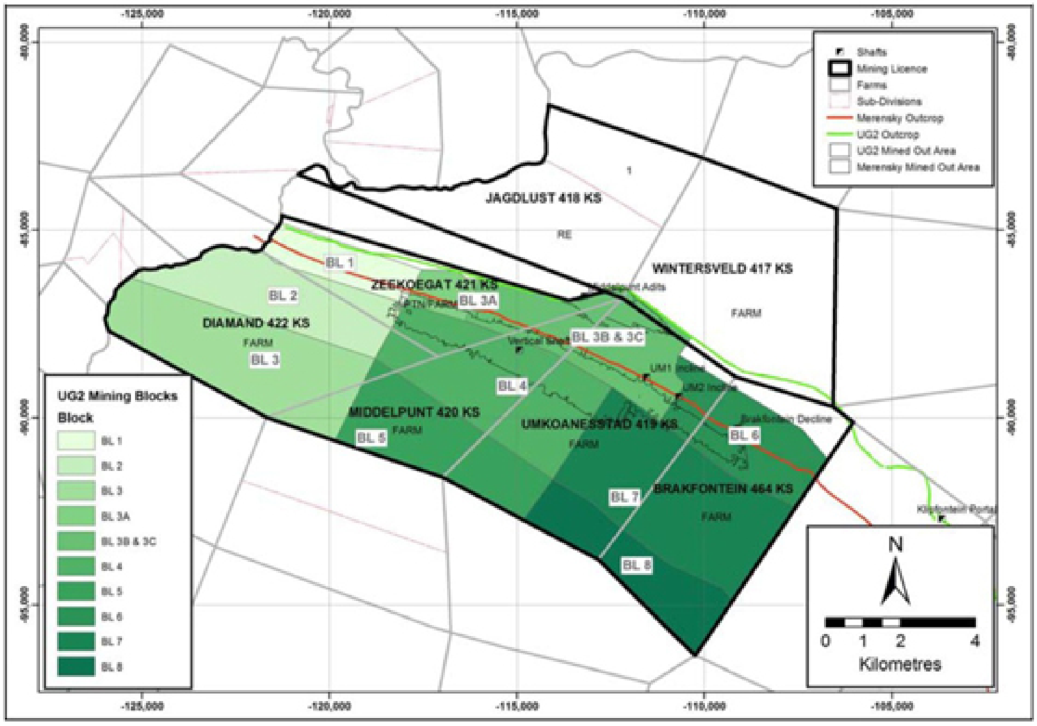


Figure 18.13: UG2 mining blocks (Anglo Platinum, 2008).

18.2.2 Access strategy

LPM's previous long-term depletion plan includes a generic access strategy which proposes a system of declines and vertical shafts to exploit the mining blocks (see Figures 18.12 and 18.13).

The previous access strategy (initial 2008 mining plan) is comprised of various phases (see Figure 18.14). These phases are:

Phase 1

Initial access to the shallow Merensky Reef and UG2 horizons is by means of separate decline shaft systems. These decline shaft systems will facilitate mining for a maximum of nine levels, to a depth of approximately 650 m below surface. From the lowest level, a single, 650 m deep, vertical shaft will be raise-bored to surface and equipped to provide men and material access plus ventilation so that UG2 mining can continue for a further three levels.

Phase 2

A main vertical shaft will be sunk from surface to a depth of approximately 900 m. The first station will be at about 630 m and will service another sub-decline shaft system to allow Merensky mining beyond No. 9 Level (i.e. Phase 2 mining). A second station will be established at approximately 810 m (12 Level) from which a twin haulage (approximately 1100 m long) will be developed to meet with the extended UG2 Phase 1 decline shaft system. The original Phase 1 vertical shaft will then become an up-cast ventilation shaft.

Phase 3

The Phase 2 main vertical shaft will be deepened from 12 Level to 18 Level (about 360 m). From a station on 18 Level a twin-haulage will crosscut to the Merensky and UG2 reefs. Two separate decline shaft systems will then be established for the mining of each reef horizon between 18 Level to the mining boundary.

The relationship between phases and blocks is detailed in Table 18.4.

The 2008 Mineral Resource and Mineral Reserve estimates were based on the 2008 LOM schedule, which includes:

- Implementing Delta 80 at MPH in 2009,
- Aggressive ramp up at Brakfontein to 120 ktpm by 2011, and
- Suspending operations at Vertical (1st quarter 2010) and UM2 (end 2009).

Although the long term plan remains unchanged, due to the current commodity prices and financial markets, the short term plan has been revised. This revised short term strategy includes:

- Delta 80 at MPH postponed to ±2017,
- Ramp up period extended at Brakfontein to 2013/2014,
- Mining at UM2 and Vertical Shafts extended to 2012/2013, and

- Plant upgrade is postponed.

At the time of writing of this report, detailed scheduling of the above, long term scheduling was in progress and all changes will only be reflected in the 2009 Life of Mine and 2009 Mineral Resource and Mineral Reserve statement.

Figure 18.14 depicts a schematic diagram of Phases 1, 2 and 3 development and associated shafts.

Table 18.4: LPM mining blocks.

	Merensky Reef	UG2
Phase 1 (by decline shaft)	**Block A, B and C**	**Block A, B and C**
	(Current Vertical Shaft and UM2 Shafts) plus Brakfontein declines	**(Middelpunt Hill adits and declines)**
		Block 6 Brakfontein
	Block 1 Zeekoegat	**Block 1 Zeekoegat**
Phase 2 (by decline shaft)	**Block 2 Zeekoegat**	**Block 2 Zeekoegat**
	Block 4 Middelpunt	**Block 4 Middelpunt**
	Block 6 Brakfontein	**Block 7 Brakfontein**
Phase 3 (by vertical shaft and sub vertical)	**Block 3 Zeekoegat**	**Block 3 Zeekoegat**
	Block 5 Middelpunt	**Block 5 Middelpunt**
	Block 7 Brakfontein	**Block 8 Brakfontein**

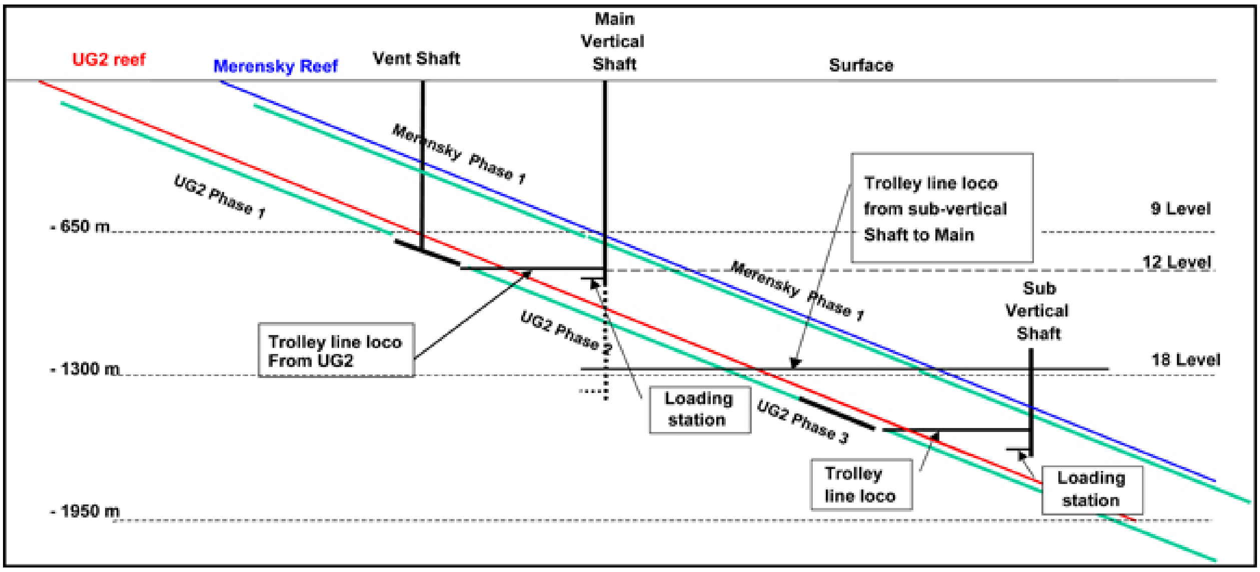


Figure 18.14: Schematic diagram of Phases 1, 2 and 3 development and associated shafts (Anglo Platinum, 2008).

18.3 Mining methods

18.3.1 Decline shaft system (Phase 1)

The decline shaft systems were designed to produce between 120 ktpm and 130 ktpm of run-of-mine ore from four operating production levels at any one time. The three declines will be developed at a dip of 10° in an apparent dip direction by mobile mechanised trackless equipment. The declines will be located 40 m below the reef elevation and about 20 m below the footwall haulages.

The 5.5 m wide by 3.15 m high conveyor decline and 4.6 m wide by 3.15 m high material decline will be separated by a lateral distance of 25 m, centre to centre and will be connected every 105 m via 20 m long horizontal laterals. Additional laterals will be developed between the material and conveyor declines to provide for substations and dams with pump stations. A 3.2 m wide by 2.8 m high chairlift decline will transport personnel to the operating levels and will commence from the decline access portals to comprise a dedicated chairlift decline on apparent dip to 1 Level. Below 1 Level an old raise-line will be modified to accommodate the chairlift, which will then continue on true dip.

Inclined crosscuts of various lengths will connect the material decline to transfer stations on each footwall haulage level. The haulages will first be developed to the tip positions above the belt decline. The main tips on each level will comprise two ore passes approximately 20 m long.

18.3.2 Main vertical shaft system (Phases 2 and 3)

All vertical production shafts from surface will be equipped with hoisting facilities for rock, personnel and material to support a production rate of approximately 250 ktpm of ore. These will support clusters of sub-declines below the respective reef horizons.

The main vertical shaft will be equipped with a single cage and counterweight in the personnel/material compartments, designed for 1600 persons per hour. The winder will be a 6 m diameter double drum winder designed for a capacity of 250 ktpm of ore. Broken rock will be hoisted from a conventional loading station comprising ore and waste storage silos discharging onto a loading conveyor belt which in turn feeds a measuring flask and finally into a 20 tonne skip.

The initial raise-bored vertical shaft will have only one shaft station and only one production level, i.e. 9 Level.

Merensky ore will be transported up the decline conveyor, transferred into silos and then trammed by wagon trains to the main vertical shaft ore-passes and loading arrangements.

UG2 ore will similarly be transported up the sub-decline conveyor before being moved to the vertical shaft via a horizontal conveyor system on 12 Level. The conveyor will discharge into separate ore and waste storage silos.

The sub-vertical shaft will support mining operations and will be sunk from 17 Level to just below 30 Level. Ore and waste will be hoisted up to 17 Level and discharged into an ore pass to feed 18 Level. On 18 Level, the rock will be transported by means of a high speed train over a distance of about 4 km to the main vertical shaft where it will be transferred into the main shaft rock handling system. A dedicated personnel and material haulage is proposed between the main vertical shaft and the sub-vertical shaft.

18.3.3 Primary development

Primary development (i.e. haulages, cross-cuts, return-air-ways) is conducted by electrohydraulic mobile drill rigs (at Brakfontein) with Load Haul Dump (LHD) mucking.

The return-air-way (4.0 m wide by 4.0 m high) is positioned just below the oxidised zone which, in the case of UG2, extends some 50 m vertically below surface. This interval increases to 100 m below surface in the case of the Merensky Reef, which sub-outcrops in the valley.

All levels below the return-air-way will be 5.6 m wide by 3.7 m high single end haulages equipped with conveyor belts. These will be developed between 15 m to 30 m vertically below the reef horizon and at a gradient of 1:200. Conventional 3.0 m wide by 3.2 m high cross-cuts, spaced 200 m apart, are developed to serve each raise line. The crosscuts are stopped short of the reef horizon and a travelling way is developed to access the reef horizon. A 15 m long material bay equipped with a mono-winch will precede each travelling way.

One short ore-pass serving four panels and a longer twin (Y-leg) ore-pass serving the remaining 10 panels will be established between the crosscut and the raise. Other development will include dams, substation cubbies and explosives cubbies on each level. More than one development raise will be in progress on each half level at any stage.

18.3.4 Secondary development

Secondary development (i.e. raises, advanced strike gullies and winch chambers) are drilled and blasted conventionally with pneumatic hand held drills and cleaned with scraper winches.

Raises and advanced strike gullies will be 2.1 m high by 1.5 m wide and developed 20° above strike. Strike gully winch positions will be developed concurrent with the advancing raises at 35 m centres to be available for ledging purposes.

Stope panel spans will be limited to 35 m (pillar centre to pillar centre). Raise lines will be positioned 200 m apart and a stability pillar left at the mining limit between adjacent connections. Rock engineering considerations require that stability pillars be introduced at a depth of 200 m below surface to augment the support provided by panel pillars.

Haulages will be positioned 15 m in the footwall at shallow depth (< 600 m), and crosscuts with short inclined travelling ways will access the reef horizon. As the depth of mining increases the middling to reef will be increased, and haulages are positioned 20 m in the footwall at a depth of 700 m, 25 m in the footwall at a depth of 800 m, and 30 m in the footwall at a depth of 1000 m.

18.3.5 Conventional stoping

Conventional narrow reef stoping is currently applied at the Merensky Vertical Shaft and UM2 Inclined Shaft. The depth of the Vertical Shaft reaches to 9 level with a hoisting capacity of 80 ktpm. The mining operations are rail bound and compressed air is the only source for drilling operations. UM2 Shaft is an inclined shaft deepening up to 5 level with a connection to the Brakfontein at 2, 3 and 4 levels.

The conventional mining method will be practiced at both shafts during the next five years i.e. 2012/2013, after which they will be phased out and replaced by the hybrid mining method at the Brakfontein Shaft. It is noted that the in-stope process described under conventional stoping is applicable to the hybrid mining method except for ore handling in the crosscut and footwall drives (i.e. mechanised mobile equipment will be used in place of locomotives and hoppers in the hybrid mining method).

The ledging crew will establish the Advanced Strike Gullies (approximately 3 m in advance) at 35 m centres and at 20º above strike so that the gullies remain clear of water. The travelling way will be over-stoped during ledging. An average stoping panel will have a working face length parallel to the raise line of 29.5 m with ASGs 1.5 m wide and 4.0 m wide strike pillars. Additional critical parameters used in the mining design and scheduling are given in Table 18.5.

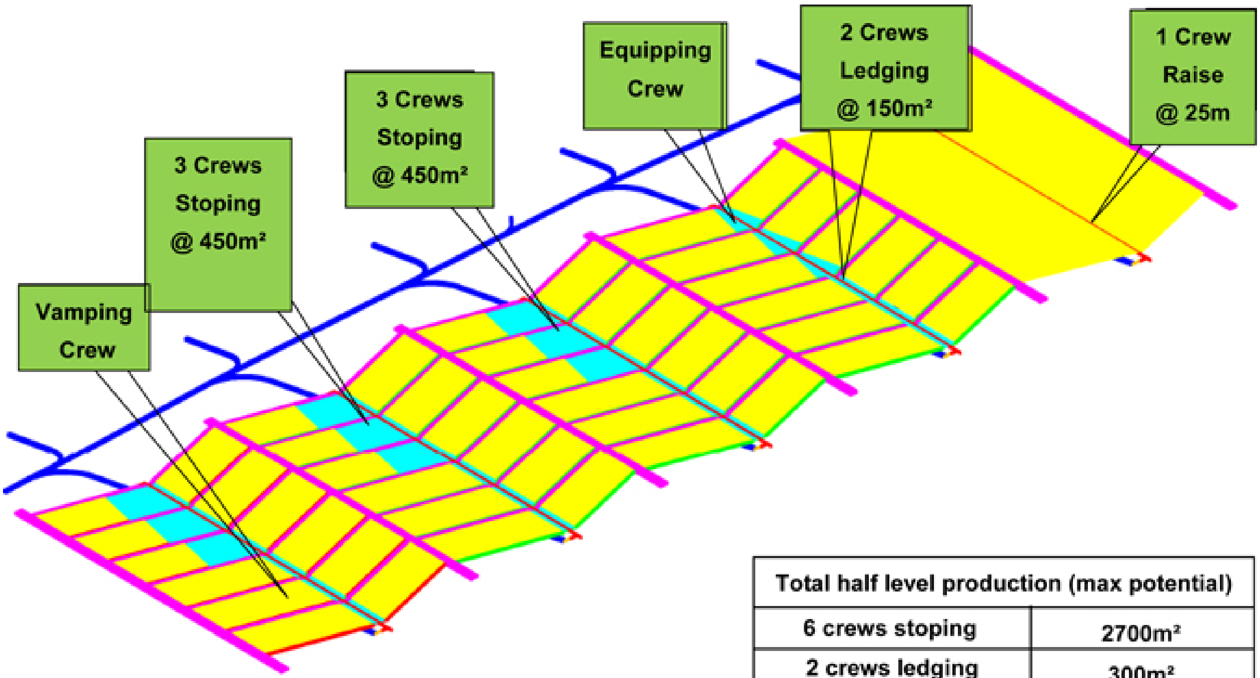
Table 18.5: Parameters used in the mining design and scheduling for Vertical and UM2 Shafts.

	Actual 2008	Plan 2009
Face Length (m)	**31.5**	**31.5**
Face Advance (m, per month)	**7.3**	**10.1**
m² (per stoping crew per month)	**231**	**320**
Haulage Advance (m, per month)	**22.0**	**30**
Cross cut Advance (m, per month)	**16.8**	**30**
Boxhole Advance (m, per month)	**11.0**	**15**
Raise Advance (m, per month)	**20.4**	**25**
Winze Advance (m, per month)	**20.9**	**20**

The revised mining scheduling in the Vertical and UM2 Shafts highlights that tunnel development on the east and west sides of levels 5 to 9 will be recommenced and some development work will be commenced on available winzes, especially with the winzes to 10 level, as a contingency plan.

The planned tonnage from the Vertical Shaft starts at 37 ktpm and ramps up 40.9 ktpm by the end of 2013. However, the planned tonnage at the UM2 Shaft starts at 14 ktpm at 2009 and reduced to 8.1 ktpm by the end of 2013.

5 half levels (4 half levels from 5 to 8 levels with a 10 ktpm production rate and 1 half level at 9 level with a 5 ktpm production rate) have been designed along the west side of the Vertical Shaft. On the other hand, there will be 3 half levels with a 10 ktpm production rate and one half level with 5 ktpm production rate at the east side of the Vertical Shaft. A typical half level design to be employed at these two shafts is given in Figure 18.15.



Total half level production (max potential)	
6 crews stoping	2700m²
2 crews ledging	300m²
Total	3000m²

Figure 18.15: Half level design parameters for Middelpunt Shaft (schematic view).

It is envisaged that the whole raise will be ledged for 6 m by one crew before stoping teams take over. An average stoping panel will have a working face length parallel to the raise line of 29.5 m with ASGs 1.5 m wide and 4.0 m wide strike pillars.

Six stoping crews, two ledging and one raise crew will be responsible for all drilling and blasting operations in a half level. Two rows of 32 mm holes on a 40 cm burden are drilled utilising pneumatic handheld rock drills with 1.1 m long drill steel, giving a planned advance per blast using Anfo and capped fuse and igniter cord. (Note the mine is currently converting to a shock tube initiation system). Night-shift crews will muck all the broken ore from the mining faces with 37 kW and 55 kW winches into hoppers via ore-passes for transport to the decline tips. Sweepings will be carried out with the advancing faces.

In addition, vamping operations will be carried out to ensure that all re-usable material and equipment is recovered before a worked out area is sealed off. A Mine Overseer will be appointed to compile and implement a vamping programme with redeploying non production labour for vamping and reclamation. It should be noted that the current plan includes some additional Pt recovery from this operation, i.e. 75 ounces and 160 ounces Pt per month from UM2 and Vertical Shafts respectively.

Broken ore and waste will be transported by rail-bound hoppers to the main ore-passes. Ten tonne battery locomotives pulling ten 6 tonne hoppers (about 50 tonnes per trip) equates to 10 trips for every four panels cleaned. Two locomotives are allocated per half level to transport stope ore, as well as stope store material i.e. timber support, scraper winch ropes, cement and roof bolts to the working faces. Store material will be transported in one shift utilising rail-bound material cars, flat cars, pipe bogies or rail bogies. An additional locomotive, and hoppers, will be used to clear the development ends.

Personnel will either travel down the chairlift decline until 1 Level before further transportation by chairlift (via a slyped out raise), or be lowered to the appropriate level in the main compartment of the shaft before being transported further in personnel carriages. The chairlift will have offloading landings on each level, from where people will either walk or be transported again via carriages to their respective working places. At Zeekoegat, personnel will leave the cage at their designated level.

18.3.6 Hybrid stoping

Hybrid stoping is currently applied at Middelpunt Hill and will be the preferred mining method for future expansion and replacement projects at LPM.

The proposed mine layout comprises the use of mechanised development with conventional mining on strike, using pneumatic rock drills and scraper winches. This is a well established mining method in South Africa. Cleaning of the stope panels is by means of a 37 kW face winch per panel, with a 55 kW strike gully winch feeding the centre gully winch. From the centre gully ore will be scraped into the stope ore-pass and loaded to the conveyor system.

Material will be transported down the material decline in trackless vehicles and will continue to the stope crosscut where the material will be off loaded.

Personnel will access the workings by means of the chairlift system described above.

Stoping operations will be conducted via conventional stoping methods with ore removed from the stope on a 1 050 mm strike conveyor belt to discharge into decline passes that will in turn feed the main decline conveyor belt.

The strategy for the short term plan will be as follows:

- to accelerate development on 1 Level and winzes on zero level
- recommence decline development to 2 level
- implement four half level models with 14 ktpm rate on 1 and 2 levels
- Implement a reclamation and vamping programme, and
- Complete 1 level main ore passes and conveyor system to eliminate the current bottle neck.

Table 18.6 details the parameters used in the mining plan and scheduling for Middelpunt Shaft.

Table 18.6: Parameters used in the mining plan and scheduling for the Middelpunt Shaft.

	Actual 2008	Plan 2009
Face Length (m)	**31.5**	**31.5**
Face Advance (m, per month)	**13.7**	**14.3**
m² (per stoping crew per month)	**431**	**450**
Haulage Advance (m, per month)	**31.0**	**30**
Cross cut Advance (m, per month)	**23.4**	**30**
Boxhole Advance (m, per month)	**17.7**	**15**
Raise Advance (m, per month)	**26.0**	**25**
Winze Advance (m, per month)	**nil**	**20**

The short term strategy for the Brakfontein Shaft includes the completion of the infrastructure to 3 and 4 levels with an implementation of a half level model on 3 and 4 levels and a production rate of 5 ktpm once infrastructure is completed. Table 18.7 details the parameters used in the mining plan and scheduling for the Brakfontein Shaft.

Table 18.7: Parameters used in the mining plan and scheduling for the Brakfontein Shaft.

	Actual 2008	Plan 2009
Face Length (m)	**31.5**	**31.5**
Face Advance (m, per month)	**7.8**	**15**
m^2 (per stoping crew per month)	**245**	**473**
Haulage Advance (m, per month)	**4.4**	**30**
Cross cut Advance (m, per month)	**5.4**	**30**
Boxhole Advance (m, per month)	**0**	**15**
Raise Advance (m, per month)	**28.5**	**25**
Winze Advance (m, per month)	**20**	**20**

In addition, productivity rate of a stoping crew will be 400 m^2 for the first 6 months and 473 m^2 thereafter.

18.4 Mining Production and Projects

18.4.1 Current production

Anglo Platinum indicates that current production of Merensky ore is 35 ktpm to 40 ktpm from the Vertical Shaft workings and 13 ktpm to 15 ktpm from the UM2 Inclined Shaft. UG2 production from the Middelpunt Hill Adits is 45 ktpm. This conforms to the production of 1100 kt, reported in Anglo Platinum's Annual Report (2008), which represents an average total monthly production of approximately 92 ktpm.

Merensky

The Vertical Shaft services the working places on the north-west extremity on the Zeekoegat property and on the lowest levels of the Vertical Shaft on the Middelpunt and Umkoanesstad properties while the UM2 Inclined Shaft services the workings on the southeast extremity within the Umkoanesstad and Brakfontein properties.

The current workings on Zeekoegat and Brakfontein and on portions of Umkoanesstad are remote from the shafts and as such productivity is less than optimal. Similarly, some working places on the Middelpunt property and on the nearer portions of Umkoanesstad, are accessed by winzes i.e. are below the lowest working level, and are particularly difficult to access. However, the current mine plan involves all Merensky workings until the end of 2013. Then, the decline in Merensky ore production will be replaced with the build-up of Merensky ore production from the Brakfontein Merensky Project which is discussed below.

UG2

Until 2006, UG2 ore production from the Middelpunt Hill Adits was steady at 55 ktpm. Since 2006 UG2 ore production from these adits has declined and has to some extent been supplemented by UG2 ore production from Level 0 of the Middelpunt Hill Decline. UG2 ore production from the Middelpunt Hill Adits is planned to ramp up from 35 ktpm to 60 ktpm until 2014.

18.4.2 Phase 1a Expansion - Approved Projects

There were two approved projects in Phase 1a expansion, the Brakfontein Merensky Project at 120 ktpm (BRK Merensky Project) and the Middelpunt Hill UG2 Expansion Project at 125 ktpm (MPH UG2 Project).

Prior to the Anglo Platinum's 'Investment Proposal' (IP) process, Anglo Platinum project teams – with support from selected mining engineering consulting companies, prepared Feasibility Studies including detailed capital budget estimates and mine plans.

Both of these projects have been through the IP process and funding is therefore approved. However, due to the currently subdued financial market, these expansion projects have been slowed down. The ramp up on the Brakfontein project has been reconsidered to more realistic levels and the Middelpunt Delta 80 project has been delayed to approximately 2017.

Merensky

The BRK Merensky Project, approved by Anglo Platinum in 2004, is planned to produce 120 ktpm of Merensky ore from the area below the oxidised zone and above 650 m below surface, via a Decline Shaft System located on the Brakfontein property.

In the new mining plan it is indicated that Brakfontein Merensky ore production will replace Vertical/UM2 Shafts production in 2014 at 1113ktpa and steady-state production of 1440 ktpa is expected from 2016 through till 2027.

UG2

The MPH UG2 Project (Delta 80), approved by Anglo Platinum in 2007 was planned to produce 125 ktpm of UG2 ore from below the oxidised zone to 650 m below surface, within the Middelpunt property. The project included the continued sinking of the Middelpunt Hill Decline and the sinking of a new Decline Shaft System, from near the site of the current UM1 Shaft, through the hangingwall and the UG2, to intersect the Middelpunt Hill Decline on No.2 Level. Below No.2 Level the Decline Shaft System will continue as a conventional footwall decline shaft system as described in Section 18.2.2.

This project has been postponed to ± 2018. Therefore, UG2 production is expected to commence in 2013 and maintained until 1525 ktpa is achieved in 2021, which will be maintained until 2043.

18.4.3 Phase 1b Expansion - Proposed Projects

There were three proposed projects in the Phase 1b expansion:

- The Brakfontein UG2 Project (BRK UG2),
- The Zeekoegat Merensky Project (ZKG Merensky), and
- The Zeekoegat UG2 Project (ZKG UG2)

The Brakfontein UG2 Project was at a Pre-Feasibility Study level and the Zeekoegat Projects were at Conceptual Study level. The former included a Preliminary Cost Estimate and the latter an Order of Magnitude Cost Estimate. All these projects were deferred, and do not currently form part of the mine plan.

18.4.4 Life-of-Mine production profile

As a result of the recent decision to revise all current mining plans in order to minimize the negative impact of the prevailing economic climate, the 2008 Life of Mine scheduling for LPM has changed.

It became evident during the discussions with Head Office personnel, as well as mine representatives, that mine personnel are still busy, at the time of preparation of this report, changing the Life of Mine scheduling, i.e. short term and long term. The critical parameters for short-term scheduling have been agreed on by all parties.

The Deloitte team was provided with a 5 year short-term, i.e. 2009 to 2013, mining scheduling based on certain parameters. However, actual mining scheduling using CADSMINE and other, relevant software was still in progress at the time of compilation of this report. The short-term plan, once completed, will form the foundation for long term planning.

In addition, Deloitte was also provided with a financial model by Anooraq. This information includes details such as tonnage profile, grade, total metal recovery, OPEX, CAPEX, etc. However, this information is spreadsheet-based and still needs to be converted and/or confirmed with the actual mining plan and scheduling process. It must also be noted that the production rates of both reefs are in agreement when comparing, for the five years, the data from the mine and Anooraq.

Merensky

The mining scheduling for the short term, as provided by Lebowa Mine, indicates that Merensky ore production will ramp up from 60 ktpm in 2009, to 90 ktpm in October 2010, and remain at that level between 90 ktpm and 105 ktpm during November 2010 to December 2013. Both Vertical and UM2 Shafts will reach the end of production by December 2013. Production will be replaced by Brakfontein Shaft.

The fluctuation of the head grade of the Brakfontein Shaft is less than those of the UM2 and Brakfontein Shafts.

The current Merensky Concentrator capacity is 85 ktpm. However, Merensky Reef production will reach 85 ktpm during November 2009 and will then ramp up to 106 ktpm in October 2011. Therefore, the expansion of the Merensky Concentrator capacity in October 2009 will be sufficient to handle the ramp up.

UG2

Overall, UG2 production will ramp up from 39 ktpm in 2009 to 66.5 ktpa in 2013. During this period, changes in head grades are relatively small, i.e. 5 g/t to 6 g/t.

The current UG2 Concentrator capacity is 60 ktpm. It is planned that the production rate of the UG2 will achieve a level of 60 ktpm in the middle of April 2012. However the overall concentrator capacity, i.e. Merensky and UG2 Concentrators, will be sufficient to handle the ore that is anticipated to be mined.

18.5 Capital Expenditure

The current LOM plan of Lebowa Mine indicates that the mining activities will take place until the end of 2043 at various phases. However, during the execution of this study, Deloitte was informed that only the short-term LOM plan, i.e. 2009 to 2013, was in preparation and included an appropriate mining scheduling plan taking into account certain design and scheduling parameters agreed on with the mine personnel. Therefore, Deloitte could only discuss the short-term LOM plan in Section 17.

Nevertheless, Deloitte was also presented a detailed financial study taking into account the full LOM plan, i.e. 2009 to 2043, which is not accompanied by a detailed mining plan The following section details the financial model developed by Anooraq with the assumption that the presented production targets will be accompanied with a detailed mining plan in the near future.

18.5.1 Capital expenditure budgeting and categories

Anglo Platinum distinguishes three categories of capital expenditure:

- Expansion capital expenditure for material increases in production.
- Replacement capital expenditure for assets that replace existing assets.
- Stay-in-Business (SIB) capital expenditure to maintain the life of an existing asset e.g. plant, machinery and ore reserve development.

The current mining plan, as well as Anooraq's financial model, considers two approved projects (BRK Merensky Reef and MPH Delta 80), which have gone through the Feasibility Study and Investment Proposal phases. These have approved Control Budget Estimates (CBEs) to a +90% level of confidence.

For the previously proposed projects, the BRK UG2 was at a Pre-Feasibility Study level and the Zeekoegat (ZKG) Merensky and ZKG UG2 Projects were at a Conceptual Study level. The ZKG Merensky and ZKG UG2 Projects are not included in the financial model, whereas the BRK UG2 Project (at Pre-Feasibility level) is included in the financial model from 2026 onwards.

For the BRK UG2 Project (pre-feasibility study) there was a Preliminary Cost Estimate (PCE) with a confidence of 75% to 90% while for the ZKG Merensky and ZKG UG2 Projects the budget estimates were at an Order of Magnitude Estimate (OME) level and has a confidence level in the order of 75%.

18.5.2 Previously approved Project capital expenditure

Brakfontein Merensky Project (120 ktpm)

The BRK Merensky Project was approved in 2004 and is currently in the development stage, with first production achieved in 2008. As of December 2008 approximately ZAR960 M had been spent and a further ZAR186 M is expected to be spent to complete the project.

The BRK Merensky project has, in the short term plan, been limited to deepening to 6 level, considering SIB capital. The construction of 6 level ventilation shaft will be delayed. Infrastructure to 3 and 4 levels will be completed.

Table 18.8 represents the 2004 terms approved CBE for the BRK Merensky Project.

Table 18.8: BRK Merensky Project CBE in 2004 terms.

Description	ZAR M 2C04 Terms
Mining	291.13
Civils	117.33
Structural	2.35
Painting	—
Mechanicals	271.27
Platework	—
Piping	73.39
Electricals	75.35
Instrumentation	34.94
Consumables	14.97
Administration	27.28
Service overheads	6.32
Non Process Equipment	2.43
Reimbursables	32.63
Reimbursables – professional services	109.06
Job totals	1,058.44
Contingency at 10%	98.69
Total (2004 terms)	**1,157.14**

Middelpunt Hill UG2 Expansion Project (125 ktpm)

Table 18.9 details the originally approved, 2006 terms CBE for the MPH UG2 Project. As of December 2008, approximately ZAR143 M had been spent on the project, with a further ZAR23 M expected to be spent before completion of the project. The current short term mining plan considers the increase of tonnages through vamping and accelerated development. Four half levels are anticipated to produce 14 ktpm per half level. Simultaneously, logistical bottle necks and constraints are planned to be reduced.

Table 18.9: MPH UG2 Project CBE in 2006 terms.

Description	ZAR M 2006 Terms
Mining	264.91
Civils	130.22
Structure als	48.66
Painting	—
Mechanicals	206.74
Platework	—
Piping	31.33
Electricals	82.47
Instrumentation	27.34
Consumables	16.37
Administration	—
Service Overheads	39.62
Non Process Equipment	—
Reimbursables	33.56
Reimbursables Prof. Services	106.29
Reimbursables to CBE	—
Reimbursables non-fee Items	4.30
Job Totals	992.31
Contingencies @ 10%	99.23
Total (2006 terms)	**1,091.54**

18.5.3 Previously proposed projects capital expenditure estimates

The capital cost estimates for the originally proposed projects, are based on conceptual design layouts, information from existing operations and budget prices from suppliers. These three projects (BRK UG2, ZKG MR and ZKG UG2) have been postponed, and do not form a part of Anooraq's financial model as discussed in this report.

Anglo American Technical Division (ATD) assisted in determining the capital required at the mining unit level (i.e. ZAR919 M for a decline cluster).

The generic CAPEX cash-flow for a decline shaft system and a vertical shaft system is illustrated in Figure 18.16.

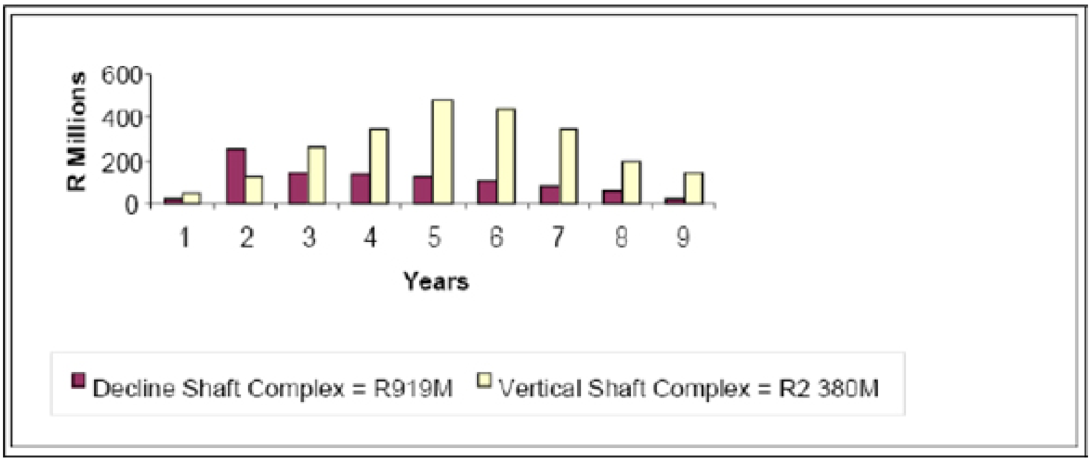


Figure 18.16: Cost and timing of the capital estimates (2005 terms), of postponed projects.

18.5.4 Mining 'Stay-in-Business' capital expenditure estimates

For ongoing capital mining development – 'Stay-in-Business (SIB) capital expenditure', Anglo Platinum includes an amount equal to eight % of shaft and operating expenditure.

For the period up to 2021, during which average monthly production is 178 ktpm, this amounts to ZAR 18.1 M per month on average. The annual amount thereafter is ZAR 161.5 M for an average production rate of 246 ktpm. This is equivalent to a reduction of ZAR 47 per tonne from ZAR 102 per ton to ZAR 55 per ton produced.

The reduced capital expenditure per annual ton of production is a reflection of the assumed reductions in operating cost.

18.5.5 Summary of capital expenditure estimates

Table 18.10 summarises the CAPEX estimates used in the DCF. The capital expenditure estimate for the LOM plan features in Figure 18.17.

Table 18.10: LOM capital expenditure estimates.

Category	Reef Type UG2 ZAR (M)	Reef Type Merensky ZAR (M)	Total ZAR (M)
Project Expansion	1 971.7	0	1 971.7
Project Replacement	3 846.2	266.3	4 112.5
Sub Total	**5 817.9**	**266.3**	**6 084.2**
Stay in Business	3 578.1	2 805.7	6 383.9
Total	**9 396.0**	**3 072.0**	**12 468.1**

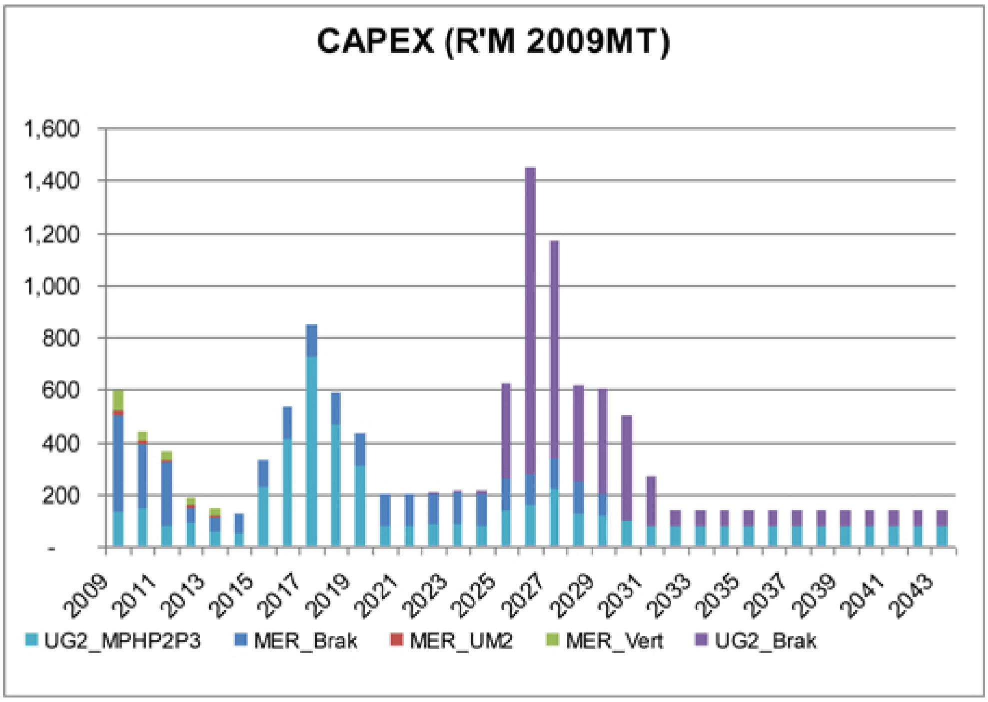


Figure 18.17: Capital expenditure plan for the LOM.

18.5.6 Concentrator capital expenditure estimate

The concentrator expansion strategy was discussed in Section 16.6.

The DCF model provides concentrator expansion capital expenditure as recommended in the RSG Global Report (2008) and is corroborated by Snowden (2008). Deloitte considers this to be reasonable.

18.5.7 Concentrator 'Stay-in-Business' capital expenditure estimate

The annual provision of SIB capital expenditure for processing is equal to eight percent of processing operating expenditure. The original model considered four percent, which was regarded as reasonable by Snowden (2008). Deloitte considers the revision reasonable in terms of the changes in financial conditions.

18.5.8 Working capital expenditure

The original DCF model has provided for adequate working capital expenditure throughout the LOM, including the various production build-up phases.

Primarily the working capital represents the physical 'work in progress' i.e. concentrate progressing through the smelting and refining stages according to the terms of the Offtake Agreement.

Working capital also allows for average creditors' terms of 46 days.

18.6 Operating Costs

Anglo Platinum distinguishes between on-mine and off-mine costs.

On-mine costs include the following categories:

- Shaft-head costs, including labour, stores and contractors directly attributable to mining plus mining overhead charges.
- Concentrator costs, including labour, stores and contractors directly attributable to processing plus its share of overhead processing cost.
- Indirect costs, which cannot be directly attributed to a specific area or activity (e.g. geological unit) which offers services across shafts.
- Off-mine costs include smelting, refining and other costs, such as marketing, research, exploration, corporate office charges, etc.

Anglo Platinum have used various methods to forecast operating costs for purposes of the feasibility studies including benchmarking with other (relevant) Anglo Platinum operations and zero-based estimation by the various project teams.

Operating unit costs during the production build up and at the various stages throughout the LOM are given in the Tables 18.11 and 18.12 respectively. The current and anticipated operating costs at LPM are amongst the highest when compared to the other Anglo Platinum mines.

Table 18.11: Operating unit costs during the production build-up (first 13 years).

	Years Tonnage (kt)	2009 1339	2010 1577	2011 1884	2012 1920	2013 1920	2014 1920	2015 1920	2016 1977	2017 2132	2018 2461	2019 2814	2020 2877	2021 2973
Unit Operating Costs (Real)														
Shaft Head R/T	R/t	576	589	552	500	502	451	446	436	433	427	422	420	418
Concentrator	R/t	106	105	105	95	97	97	97	96	93	88	84	83	82
Indirect Cost	R/t	226	216	214	194	196	165	166	165	155	138	123	121	118
Off Mine cost	R/t	16	17	18	19	19	20	20	20	20	20	19	19	19
Additional Closure provision	R/t	0	0	0	0	25	0	0	0	0	0	0	0	0
Total Cost	R/t	924	927	889	808	840	733	728	716	700	672	648	643	637

Table 18.12: Operating unit costs at various production stages throughout the LOM.

	Years Tonnage (kt)	2023 3037	2027 2954	2030 2946.9	2033 2946.9	2036 2947	2039 2947	2041 2946.9	2043 2946.9
Unit Operating Costs (Real)									
Shaft Head R/T	R/t	415	422	408	404	401	397	395	393
Concentrator	R/t	81	81	80	79	79	78	77	77
Indirect Cost	R/t	115	116	115	114	113	112	112	111
Off Mine cost	R/t	19	19	18	18	18	18	18	18
Additional Closure provision	R/t	0	0	0	20	0	0	0	0
Total Cost	R/t	630	638	621	636	611	606	602	599

Steady state real term operating costs for the various projects are detailed in Table 18.13.

Table 18.13: Steady state operating costs of the projects.

Project Year		Brakfontein Merensky 2016	Middelpunt Hill UG2 2021	Brakfontein UG2 2030
Tonnage	kt	1437	1526	1441
Shaft Head Cost	R/t	394.9	444.9	380.3
Concentrator Cost	R/t	95.5	81.8	79.9
Indirect Cost	R/t	165.1	117.9	115.3
Smelt	R/t	20.2	18.7	18.2
Total Unit Cost	R/t	675.7	663.2	593.7

18.7 Markets

A generic view of the PGM market is provided here and is based upon publicly available information.

The demand for PGM’s over the last ten years is reflected in Table 18.14.

Table 18.14: Annual PGM demand.

Metric tonnes	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006
Platinum	159.6	167.0	173.9	176.7	193.8	201.2	203.1	203.4	208.1	210.7
Palladium	235.8	267.5	291.4	278.7	210.5	150.2	168.9	204.0	228.7	206.3
Rhodium	14.5	15.8	IS.4	25.3	18.0	18.4	19.3	22.7	25.7	26.0

PGM demand remained robust, under-pinned by various sound fundamentals particularly:

- Demand in the automotive sector for use in catalytic converters as emission regulations in developed countries become more stringent and the demand for diesel-powered passenger vehicles increases.
- Industrial demand where PGMs are used in chemical applications, electrical applications including computer hard-discs and LCD glass manufacturing.

The monthly spot prices of the primary PGMs are illustrated in Figures 18.18 to 18.21. However, it should be noted that confidential contractual purchase and sales agreements may or may not reference the spot market prices.

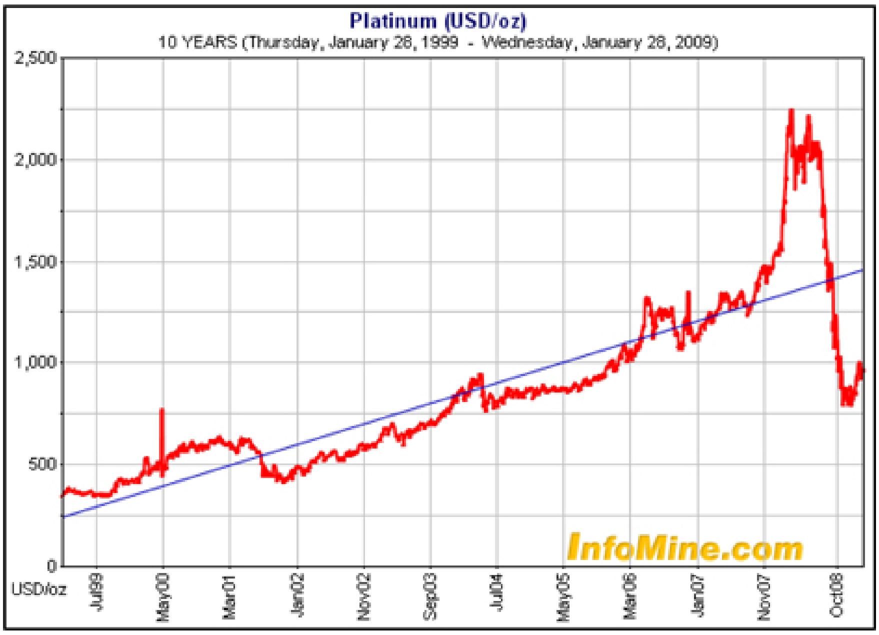


Figure 18.18: Average Platinum spot prices until January 28, 2009.

Figure 18.19: Average Palladium spot prices until January 28, 2009.

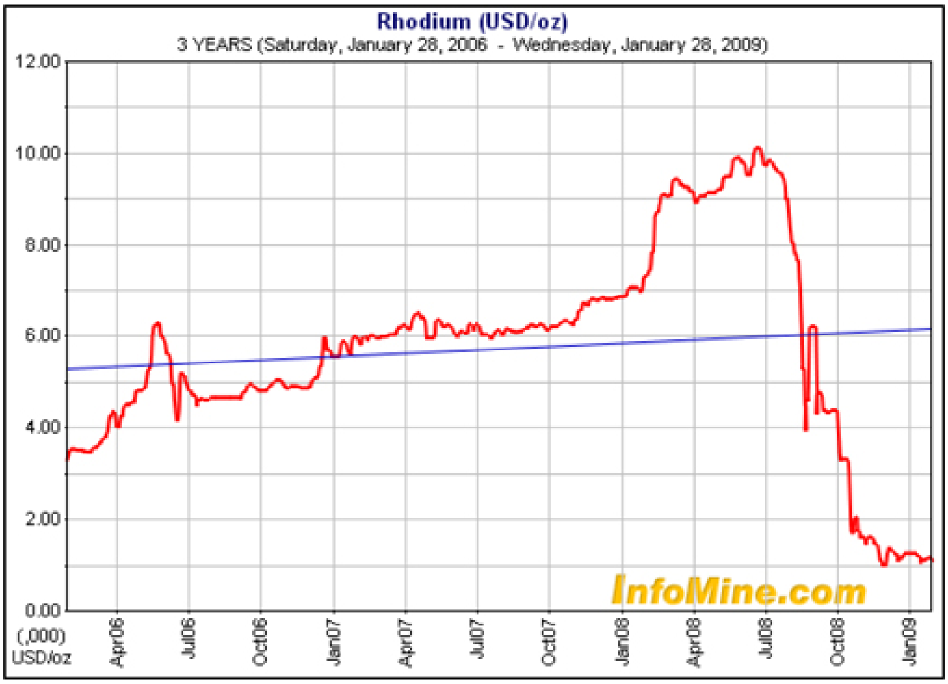


Figure 18.20: Average Rhodium spot prices until January 28, 2009.

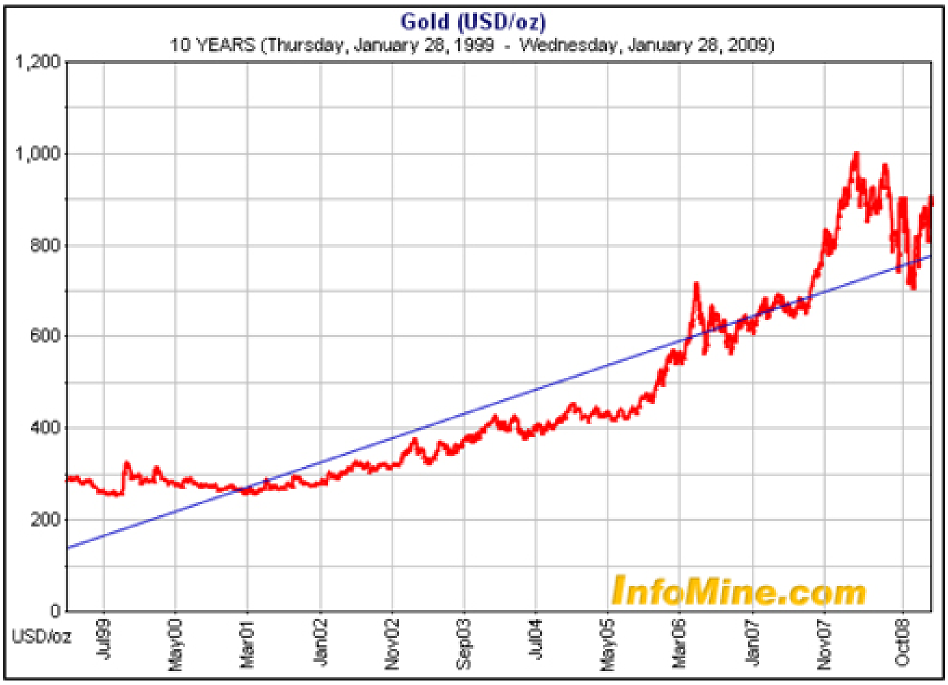


Figure 18.21: Average Gold spot prices until January 28, 2009.

Platinum

The single biggest consumer of platinum is the automotive industry (49 % net after ‘scrap’ recovery in 2006) followed by the jewellery industry (24% in 2006) while the chemical, electrical and glass industries make up a further 18% of consumption.

Platinum demand is increased in the automotive industry (10.5%) driven by more stringent emission standards in developed countries and the on-going growth in vehicle sales. There was however a decrease in demand from the jewellery sector (18.3%) which was nevertheless off-set by the increases in demand in the automotive and industrial sectors.

The shortfall between supply and demand was the key driver of increasing prices.

Notably platinum is more dominant than palladium in the Merensky Reef (2:1) than it is in the UG2 where ratios are about equal.

Palladium

The demand for palladium fell some 10% between 2005 and 2006.

The primary reason was the significant decrease in demand from jewellery manufacturing industry (30% in 2006) which reflected the consumer’s resistance to the metals’ higher prices since late 2005. This was not off-set by the increase in demand from the automotive industry (4% gross in 2006) and the electronics industry (9.6%).

The palladium market was in over-supply with significant material (21%) reporting to stock even though Russian supply fell. Nevertheless the price in US$ terms increased year on year between 2005 and 2006.

Rhodium

Notably the rhodium market is an order of magnitude less than platinum or palladium market. Nevertheless rhodium demand has grown for five consecutive years to 2006 (Table 18.14).

The single biggest consumption of rhodium is in catalytic converters making up 87% of the market in 2006 and 83.7% in 2005. The single biggest supplier of Rhodium is South Africa (87%) and notably rhodium is found in greater quantities in the BC Eastern Limb mines such as Lebowa.

18.8 Economic analysis

18.8.1 Commodity prices

Anooraq appointed external consultants to determine current consensus PGE and BM prices. Tables 18.15 and 18.16 summarise the current consensus, mean prices used in the DCF model for the first four years.

Table 18.15: Metal prices used in the cashflow model (nominal terms).

	2009	2010	2011	2012	Trend
Platinum (US$/Oz)	1052	1237	1369	1398	1339
Palladium (US $/Oz)	235	293	349	363	378
Rhodium (US$/Oz)	2831	3421	4049	4436	3700
Nickel (US$ /lb)	5.6	6.7	7.5	7.9	7.2
Copper (US$ /lb)	1.9	2.3	2.7	2.6	1.9

Following is the weighted unit revenue for the 4E basket of metals for the first four years of production. The basket price includes the weighting according to prill-split and source of production i.e. Merensky or UG2.

Table 18.16: Weighted 4E metal prices for the first four years of the original cash flow model (real (2009MT) and nominal terms).

			2009	2010	2011	2012	LOM
4E basket	US$/Oz	nominal	753	888	999	1035	
		real	753	868	950	956	967
Exchange rate	ZAR/US$	nominal	9.67	9.42	9.43	9.81	
		real	9.67	9.21	9.02	9.17	9.60
4E basket	ZAR/kg	nominal	234 238	268 718	302 764	326 336	
		real	234 238	256 901	275 404	281 904	297 371
SA CPI			0.0%	4.6%	5.1%	5.3%	
US CPI			0.0%	2.3%	2.8%	2.9%	

18.8.2 Company tax

The current South African Income Tax regime for companies applies to LPM and the DCF model therefore includes the following tax regime:

- Company income tax rate of 28% on taxable income.
- Secondary tax on companies, a tax on dividends declared, of 10%.
- A withholding tax of 10% for dividends payable to non-residents.

Tax shield

The South African mining sector enjoys immediate tax relief on capital expenditure i.e. capital expenditure can be off-set against gross profit in the year it is incurred (or can be carried forward to create a tax shield) i.e. capital expenditure is not depreciated or amortised for tax purposes.

Royalty

The delayed South Africa Royalty Act is discussed in Section 4.3.3.

The DCF model uses the third and final draft average rate to calculate royalties payable to the State, which is based on gross sales less allowable beneficiation related expenses and transport expenses between the seller and buyer of the final product. The effective royalty rate over the Lebowa LOM is 5.6%.

18.8.3 Financial indicators

Following in Table 18.17 are the real term financial indicators of the LOM plan.

Table 18.17: Financial indicators (in real terms -2009MT).

	Units	Total
Production		
Tonnes treated	000's	92 740
Grade (4E headgrade)	4E g/t	5.06
PGMs produced	4E oz	13 684 167
Revenue	ZARm	126 749
Gross revenue	ZARm	134 226
Royalties	ZARm	-7 477
Operating cost	ZARm	64 067
Unit operating cost	ZAR/t	703.34
Gross profit	ZARm	62 682
CAPEX	ZARm	12 468
Real term tax	ZARm	14 937
Effective tax rate	%	22.0
Working CAPEX	ZARm	1 303
Net profit (after working CAPEX)	ZARm	33 974
Margin	%	24.7

Cashflow

A composite discounted cash-flow (DCF) model has been used to evaluate each of the projects making up the life of mine.

The LOM real term (2009MT) cash-flow model (abridged) is detailed in Table 18.18.

Table 18.18: LOM real term cash-flow (abridged).

Real terms cashflow - abridged

	Units	Total	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
Production													
Tonnes treated	000's	92,740	1,339	1,577	1,884	1,920	1,920	1,920	1,920	1,977	2,132	2,461	2,814
Grade (4E headgrade)	4E g/t	5.06	4.54	4.59	4.86	5.10	5.15	4.80	4.96	4.71	4.38	4.59	4.86
PGMs produced	4E oz	13,684,167	176,774	210,822	266,560	284,199	286,816	267,866	277,098	271,498	272,903	329,918	399,166
Revenue	**ZARm**	**126,749**	**1,442**	**1,882**	**2,476**	**2,630**	**2,821**	**2,636**	**2,744**	**2,750**	**2,818**	**3,304**	**3,904**
Gross revenue	ZARm	134,226	1,442	1,889	2,545	2,757	2,971	2,790	2,888	2,866	2,894	3,456	4,133
Royalties	ZARm	(7,477)	—	(8)	(69)	(127)	(150)	(154)	(144)	(117)	(76)	(152)	(230)
Operating costs	ZARm	64,067	1,238	1,462	1,676	1,552	1,612	1,407	1,398	1,416	1,493	1,655	1,824
	ZAR/t	703.34	924.13	926.80	889.36	808.48	839.61	732.76	727.95	716.04	700.28	672.31	648.16
Gross Profit	ZARm	62,682	204	420	800	1,078	1,209	1,229	1,346	1,334	1,325	1,650	2,080
Capex	ZARm	12,468	597	441	365	185	146	125	328	531	847	589	428
Real terms tax	ZARm	14,937	—	—	—	227	298	309	285	225	134	297	463
Effective tax rate	*%*	*22.0%*	*0.0%*	*0.0%*	*0.0%*	*21.1%*	*24.6%*	*25.2%*	*21.2%*	*16.9%*	*10.1%*	*18.0%*	*22.2%*
Working capex	ZARm	1,303	198	52	86	56	40	9	34	13	15	83	105
Net profit (after working capex)	ZARm	33,974	(590)	(73)	349	609	726	787	699	565	329	681	1,085
Margin	*%*	***24.7%***	***–41.0%***	***–3.9%***	***14.1%***	***23.2%***	***25.7%***	***29.8%***	***25.5%***	***20.6%***	***11.7%***	***20.6%***	***27.8%***

Table 18.18 (cont.): LOM real term cash-flow (abridged).

Real terms cashflow - abridged

	Units	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Production														
Tonnes treated	000's	92,740	2.877	2,973	3,018	3,037	2,972	2,951	2,941	2,954	2,948	2,948	2,947	2,947
Grade (4E headgrade)	4E g/t	5.06	4.96	5.00	4.92	4.84	4.81	4.76	4.79	4.81	4.98	5.25	5.39	5.39
PGMs produced	4E oz	13,684,167	415,791	433,284	433,210	428,445	417,042	409,300	410,961	413,795	425,862	446,428	457,602	457,602
Revenue	**ZARm**	**126,749**	**4,024**	**4,182**	**4,187**	**4,148**	**4,046**	**4,022**	**4,126**	**4,055**	**4,011**	**4,027**	**4,077**	**4,050**
Gross revenue	ZARm	134,226	4,295	4,469	4,471	4,426	4,315	4,239	4,252	4,204	4,222	4,237	4,319	4,319
Royalties	ZARm	(7.477)	(271)	(286)	(284)	(278)	(268)	(216)	(126)	(149)	(211)	(211)	(242)	(269)
Operating costs	ZARm	64,067	1,850	1,895	1,912	1,912	1,878	1,859	1,861	1,946	2,000	2,131	2,036	2,031
	ZAR/t	703.34	643.07	637.33	633.38	629.56	632.13	630.02	632.76	658.69	678.34	722.85	690.90	689.13
Gross Profit	ZARm	62,682	2.174	2,288	2,275	2,236	2,168	2,163	2,265	2,110	2,011	1,896	2,040	2,019
Capex	ZARm	12.468	195	199	204	210	214	622	1,452	1,167	618	599	501	264
Real terms tax	ZARm	14,937	554	585	580	567	547	432	228	281	420	418	489	549
Effective fax rate	*%*	*22.0%*	*25.5%*	*25.6%*	*25.5%*	*25.4%*	*25.2%*	*20.0%*	*10.1%*	*13.3%*	*20.9%*	*22.1%*	*23.9%*	*27.2%*
Working capex	ZARm	1,303	42	46	24	18	11	20	37	11	18	21	46	22
Net profit (after working capex)	ZARm	33,974	1,382	1,457	1,467	1,441	1,396	1,090	548	650	956	857	1,005	1,185
Margin	*%*	***24.7%***	***34.3%***	***34.8%***	***35.0%***	***34.7%***	***34.5%***	***27.1%***	***13.3%***	***16.0%***	***23.8%***	***21.3%***	***24.7%***	***29.3%***

Table 18.18 (cont.): LOM real term cash-flow (abridged).

Real terms cashflow - abridged

	Units	Total	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	Total
Production															
Tonnes treated	000's	92,740	2,947	2,947	2,947	2,947	2,947	2,947	2,947	2,947	2,947	2,947	2,947	2,947	92,740
Grade (4E headgrade)	4E g/t	5.06	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.39	5.06
PGMs produced	4E oz	13,684,167	457,602	457,602	457,602	457,602	457,602	457,602	457,602	457,602	457,602	457,602	457,602	457,602	######
Revenue	**ZARm**	**126,749**	**4,035**	**4,041**	**4,034**	**4,033**	**4,033**	**4,032**	**4,031**	**4,031**	**4,030**	**4,030**	**4,029**	**4,028**	**126,749**
Gross revenue	ZARm	134226	4,319	4,319	4,319	4,319	4,319	4,319	4,319	4,319	4,319	4,319	4,319	4,319	134226
Royalties	ZARm	(7,477)	(284)	(278)	(285)	(286)	(286)	(287)	(287)	(288)	(289)	(289)	(290)	(290)	(7,477)
Operating costs	ZARm	64,067	2,026	2,078	2,015	2,010	2,005	2,000	1,995	1,990	1,985	1,979	1,974	1,969	64,067
	ZAR/t	703.34	687.37	705.16	683.85	682.10	680.36	678.62	676.88	675.15	673.43	671.71	669.99	668.28	703.34
Gross Profit	ZARm	62,682	2,010	1,963	2,019	2,023	2,028	2,032	2,037	2,041	2,046	2,050	2,055	2,059	62,682
Capex	ZARm	12,468	139	139	138	138	137	137	137	136	136	135	135	135	12,468
Real terms tax	ZARm	14,937	581	568	584	585	587	588	589	591	592	593	595	596	14,937
Effective tax rate	*%*	*22.0%*	*28.9%*	*28.9%*	*28.9%*	*289%*	*28.9%*	*28.9%*	*28.9%*	*28.9%*	*28.9%*	*28.9%*	*29.0%*	*29.0%*	*22.0%*
Working capex	ZARm	1,303	23	18	31	25	25	25	25	25	25	25	25	25	1,303
Net profit (after working capex)	ZARm	33,974	1,266	1,238	1,266	1,275	1279	1282	1,286	1,289	1293	1296	1,300	1,303	33,974
Margin	*%*	***24.7%***	***31.4%***	***30.6%***	***31.4%***	***31.6%***	***31.7%***	***31.8%***	***31.9%***	***32.0%***	***32.1%***	***32.2%***	***32.3%***	***32.3%***	***24.7%***

The real term cash-flow is illustrated in Figure 18.22 below. Current PGM prices, combined with Lebowa's grades, indicate a margin of approximately 40% once operations reach steady state.

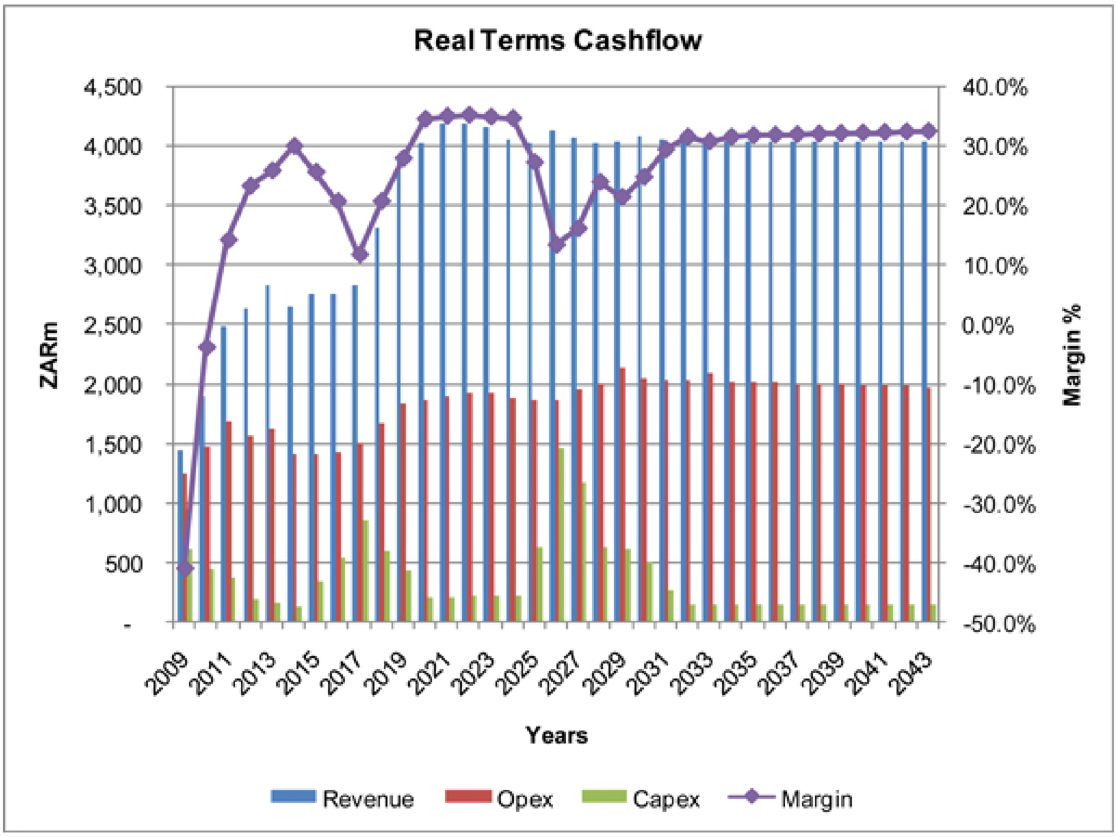


Figure 18.22: LOM real term cash-flow (2009MT).

Production profiles

The LOM production profile used in the financial model (see Figure 18.23) indicates that during the 35 year LOM some 92.7 Mt of ore (66.9 Mt UG2, 25.8 Mt Merensky) will be processed at a head grade of 5.1 4E g/t. It is noted that the currently defined UG2 and Merensky Reef Reserve estimates are 41.2 Mt and 27.1 Mt, respectively. The defined UG2 Reserve therefore has a shortfall of about 25.7 Mt. However, scope exists to upgrade the significant Measured and Indicated UG2 Resources (180.38 Mt) to Reserve status.

The production and grade profiles are shown in Figure 18.23.

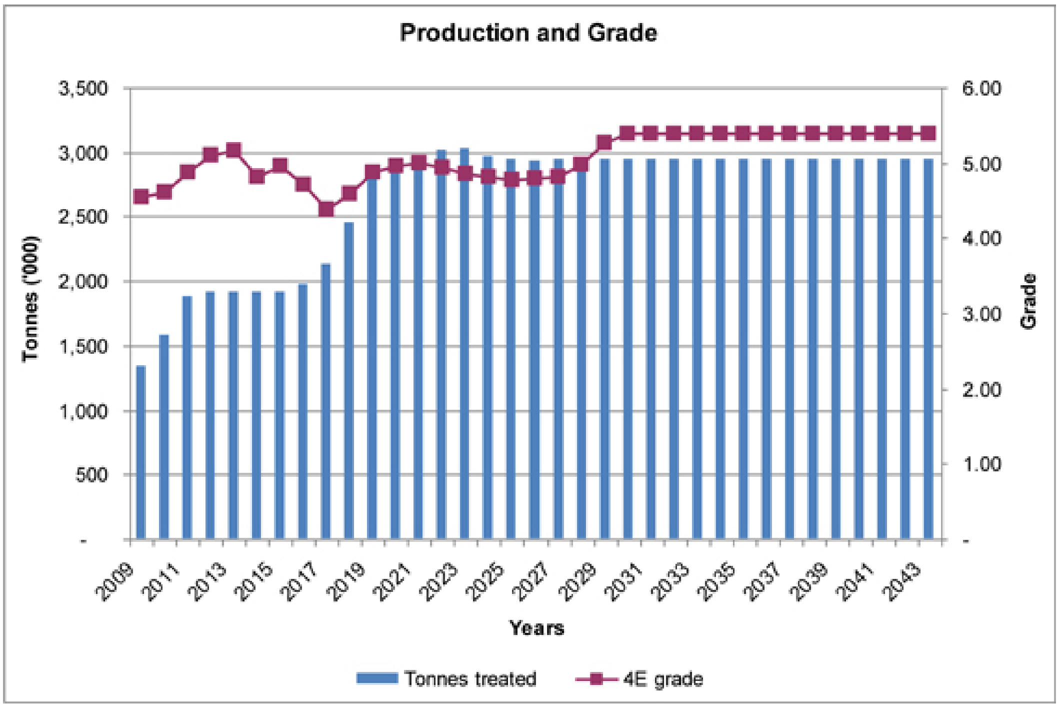


Figure 18.23: LOM production profile - tonnes.

The head-grade is 5.01 4E g/t and is made up of 2.44 g/t platinum, 2.10 g/t palladium, 0.37 g/t rhodium and 0.14 g/t gold (Figure 18.24).

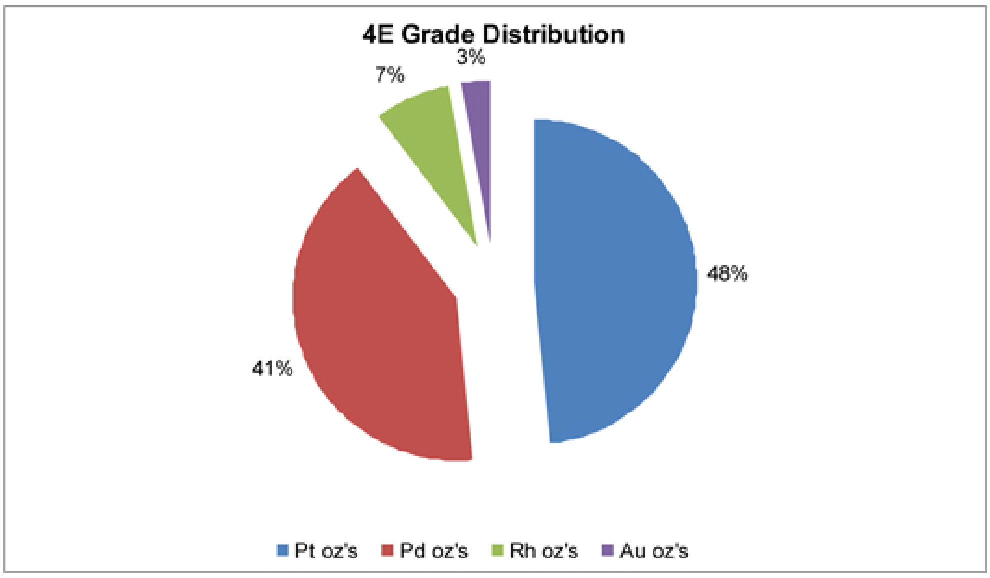


Figure 18.24: 4E grade distribution.

Revenue profiles, PGM and BM

Base metals (nickel, copper and cobalt) contribute 7.5 % of the revenue over the LOM. Nickel contributes 84 % of the value of the base metal contribution to revenue (Figures 18.25 and 18.26).

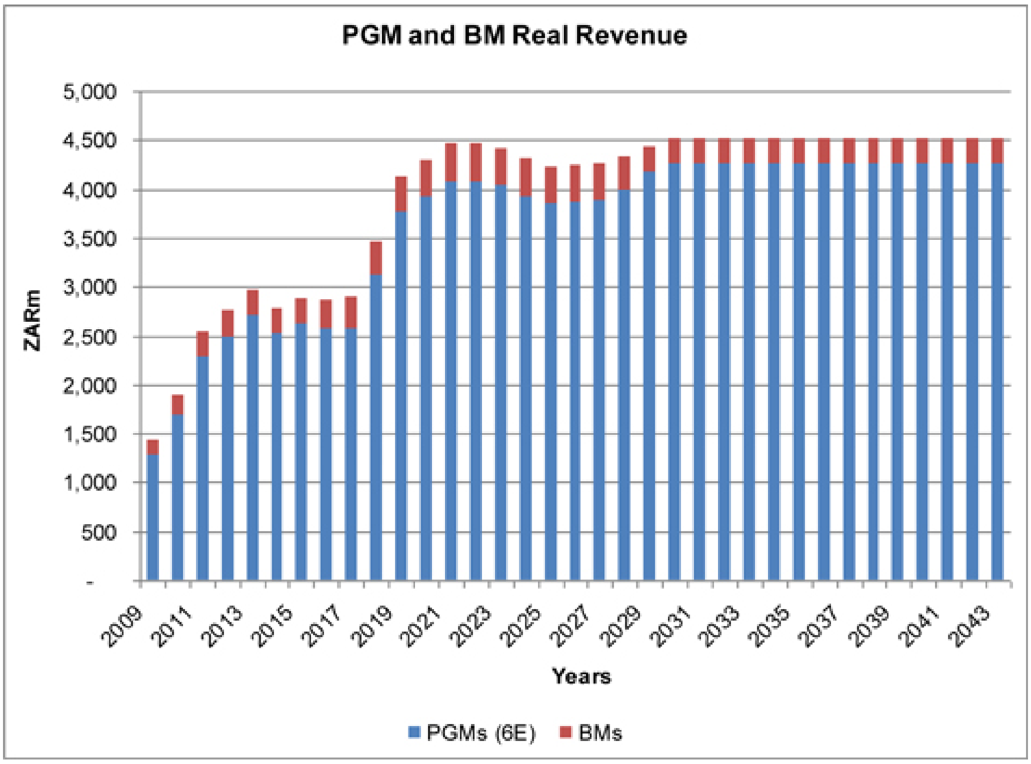


Figure 18.25: LOM real revenue by PGM and BM (2009MT).

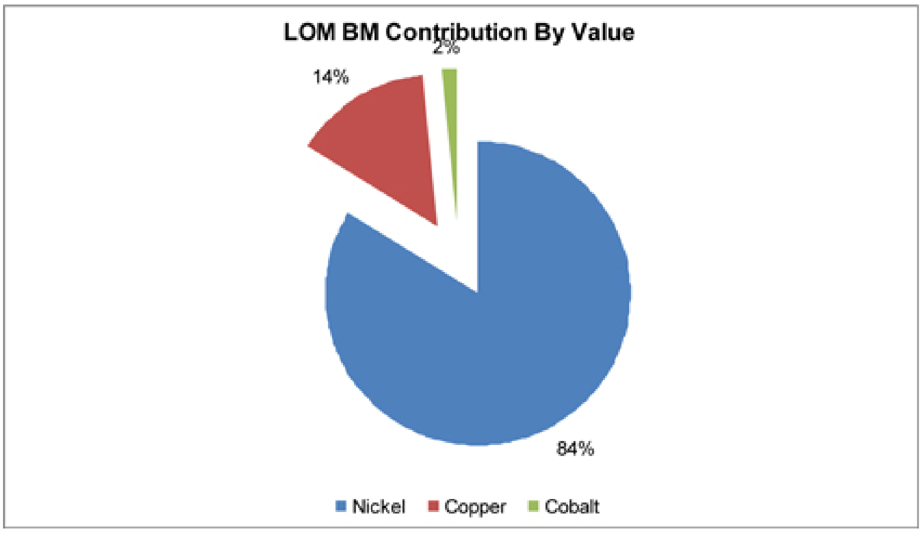


Figure 18.26: LOM BM contribution by value.

Revenue profiles by 4E

Figure 18.27 illustrates real revenue by 4E metal while Figure 18.28 details revenue contribution of each 4E metal of the LOM plan.

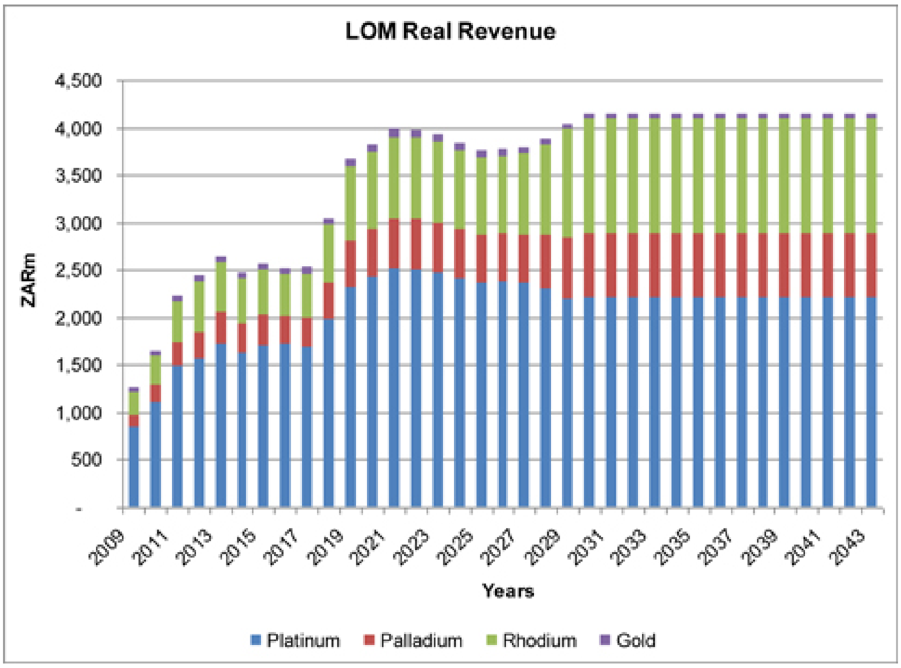


Figure 18.27: LOM real revenue by 4E (2009MT).

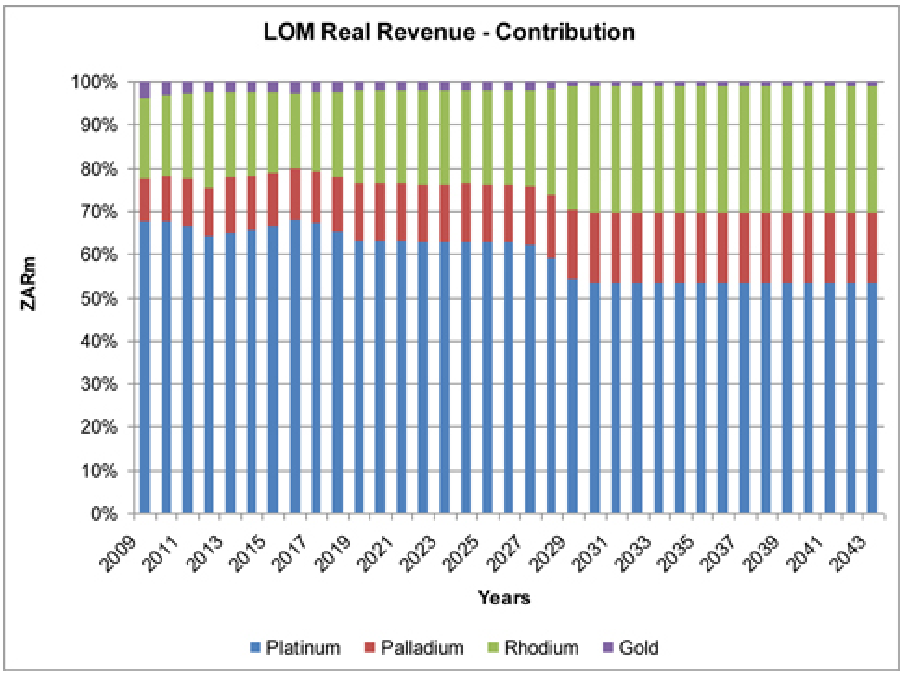


Figure 18.28: LOM revenue contribution by 4E (2009MT).

Operating cost profile

The originally planned real terms operating costs over the LOM are illustrated in Figures 18.29 and 18.30.

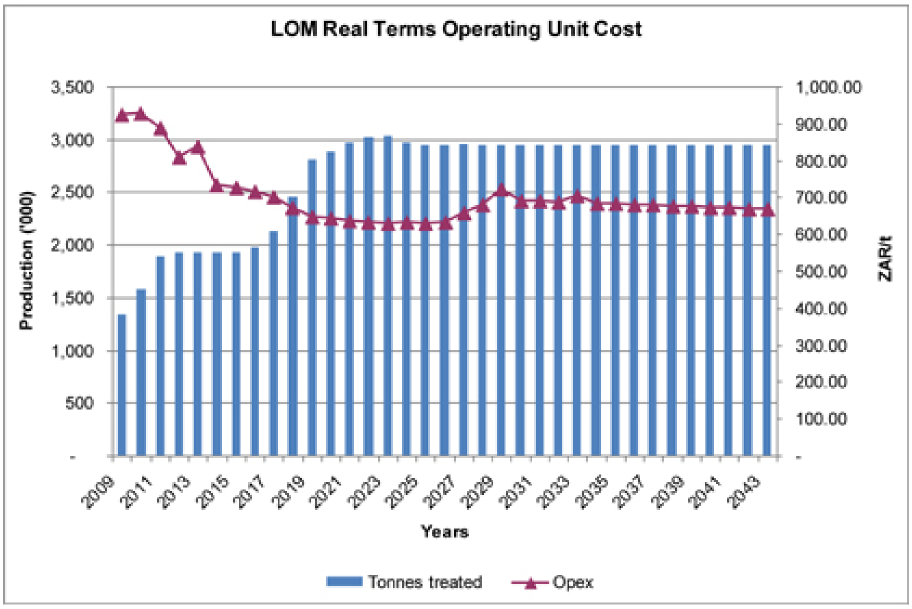


Figure 18.29: LOM real term operating unit cost (2009MT).

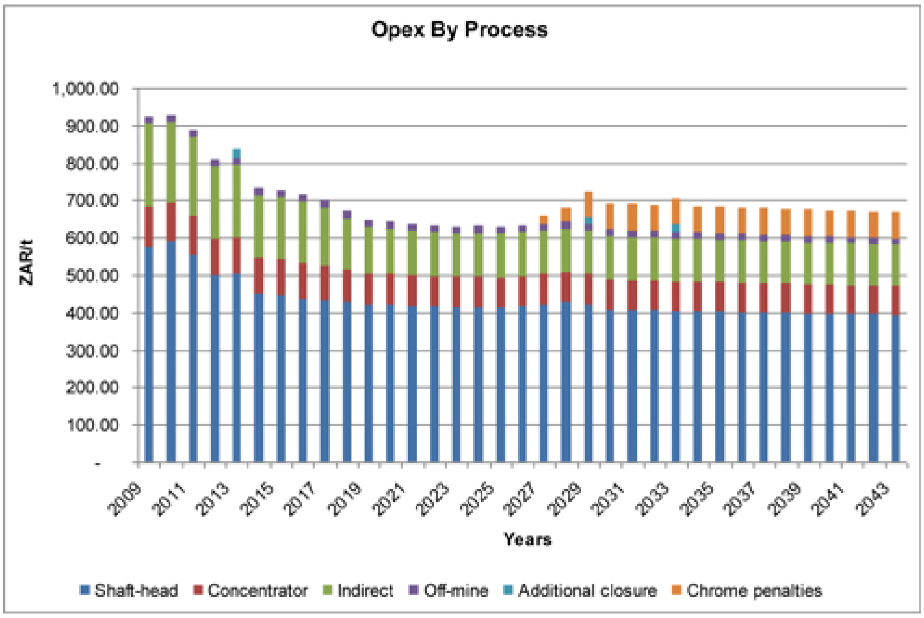


Figure 18.30: LOM real term operating unit cost (2009MT).

18.8.4 Payback and Internal rate of return

Although LPM is, according to the LOM mining plan, in an expansion phase, the advanced stage of the BRK Merensky Project in particular means there is always an overall net positive cash-flow and as such there is no overall payback period (see abridged cash-flow above).

18.8.5 DCF sensitivity analysis

Sensitivities have been calculated in the DCF model for revenue, operating costs and working costs.

This valuation is most sensitive to a change in revenue. A 10.0 % decrease in revenue results in a 28 % decrease in value in the case of NPV at a discount rate of 7.5% as shown in Figure 18.31.

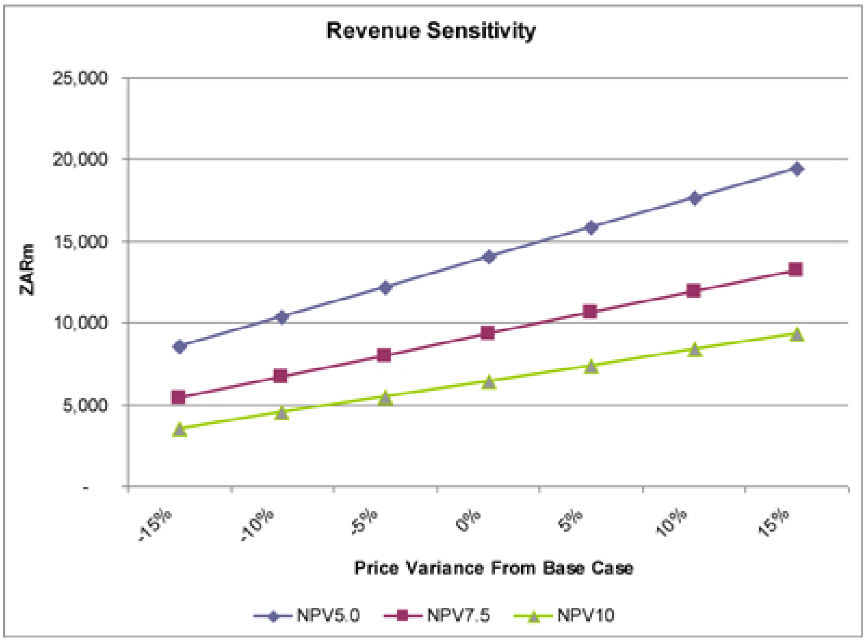


Figure 18.31: DCF valuation sensitivity to revenue.

The valuation is not particularly sensitive to capital expenditure. An increase in capital of 10 % decreases the value by just 4.0 % in the case of NPV at a discount rate of 7.5%, as is shown in Figure 18.32.

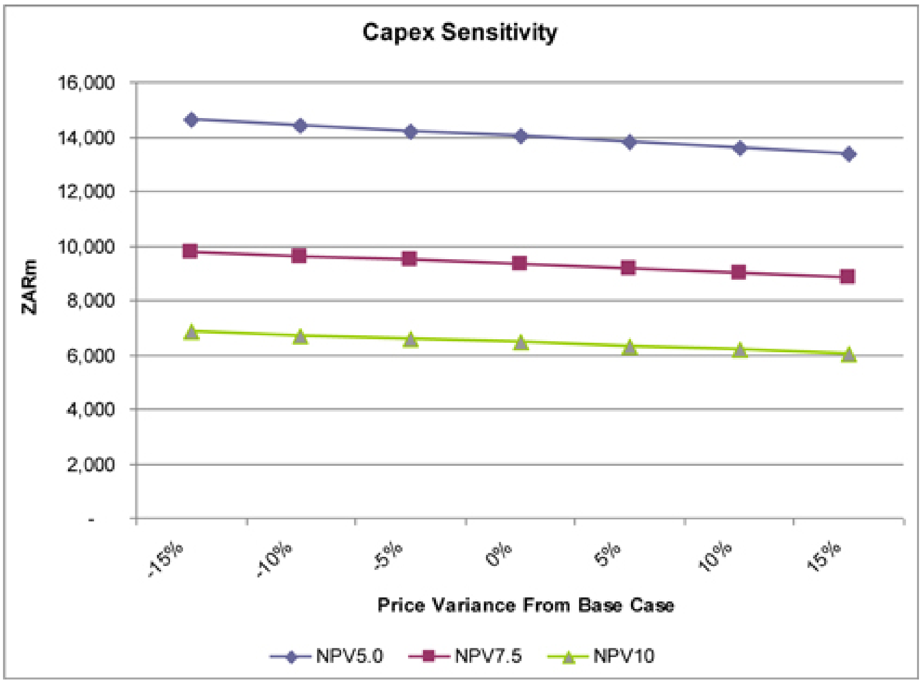


Figure 18.32: DCF valuation sensitivity to capital costs.

The valuation is sensitive to a variance in operating costs. An increase of 10.0 % decreases the NPV by 12.6 % in the case of NPV at a discount rate of 7.5% as shown in Figure 18.33.

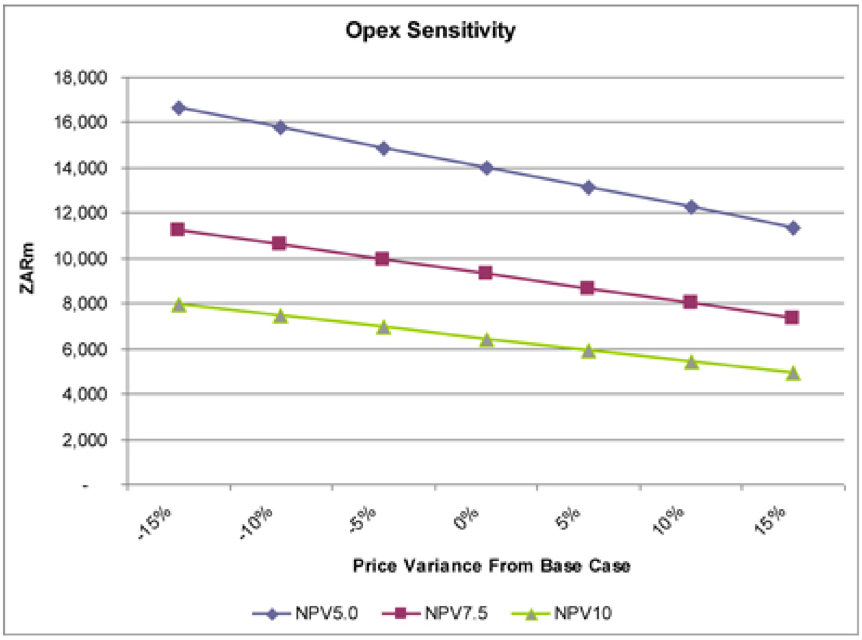


Figure 18.33: DCF valuation sensitivity to operating costs.

A series of sensitivities for various changes in combined operating costs and revenue, have been calculated for a NPV discount rate of 7.5% and are included in Table 18.19.

Table 18.19: Sensitivity matrix, OPEX and revenue.

		NPV7.5 (ZARm) LOM Real Average Basket price ZAR 297,371 kg								
	9,290	**–20%**	**–15%**	**–10%**	**–5%**	**0%**	**5%**	**10%**	**15%**	**20%**
	– 20%	6,710	8,008	9,299	10,598	11,908	13,215	14,523	15,837	17,154
	– 15%	6,056	7,355	8,652	9,942	11,240	12,550	13,857	15,164	16,478
LOM Real Average	– 10%	5,399	6,702	8,000	9,294	10,584	11,882	13,192	14,498	15,805
Operating Cost	– 5%	4,740	6,047	7,347	8,645	9,937	11,227	12,525	13,833	15,139
ZAR 690.83 per tonne	0%	4,076	5,388	6,694	7,992	9,290	10,579	11,869	13,167	14,474
	5%	3,399	4,728	6,036	7,339	8,637	9,935	11,221	12,512	13,809
	10%	2,714	4,058	5,376	6,684	7,984	9,282	10,577	11,863	13,154
	15%	2,012	3,381	4,715	6,024	7,331	8,629	9,927	11,220	12,505
	20%	1,288	2,694	4,040	5,365	6,672	7,976	9,274	10,573	11,862

18.9 Pothole Risk

A recent detailed study into the UG2 and Merensky Reef pothole characteristics, and their potential predictability, at the eight shafts comprising Rustenburg Platinum Mine (Chitiyo *et al.,* 2008) is used to illustrate the potential material risk in the LPM ore bodies and therefore Resource Statements. It is noted that this study was conducted at the Western Limb of the Bushveld Complex. The applicability of the findings to the Eastern Limb has yet to be confirmed.

Three pothole populations were defined, that relate to one magmatic process and are referred to as Populations A, B and C (in order of increasing size and from young to old). Population A, the smaller potholes, are randomly distributed, but clustered. Population B is evenly randomly distributed. Population C is represented by the very large potholes, such as the Brakspruit Depression, and are interpreted as having formed by erosion during new magma influx, associated with regional erosion. These three pothole populations are associated with the Merensky Reef, and possibly the UG2.

Pothole distribution patterns were quantitatively developed. This allows the grouping of differing pothole characteristics into specific facies. In addition, clustering and contouring of reef characteristics such as gradient, reef thickness and grade (coded) were performed.

No proxy for pothole density could be established for the UG2. Prediction of pothole density can consequently only be performed through statistical extrapolation of known pothole density (Figure 18.34).

The 1st derivative of the slope of the Merensky Reef (Figure 18.35), in contrast to the UG2, serves as a suitable proxy for pothole density (Figure 18.36). This facilitates enhanced prediction.

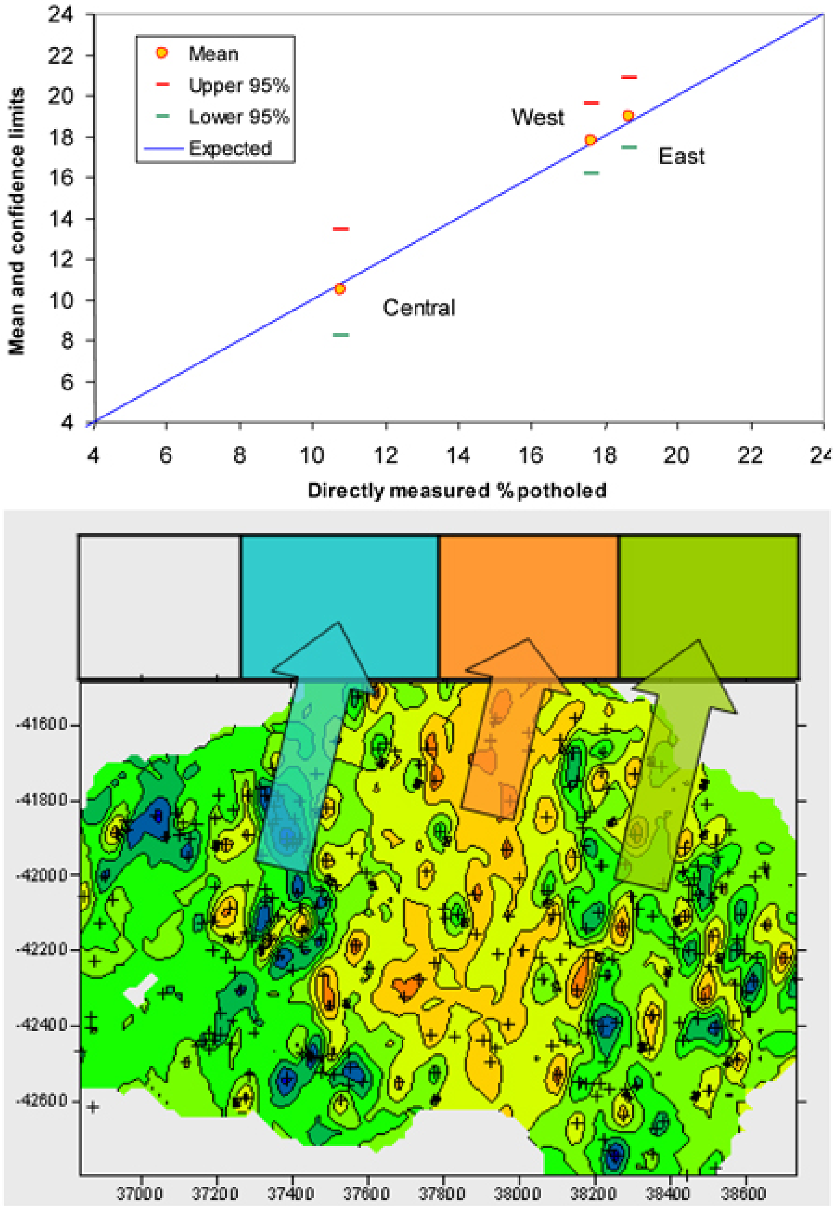


Figure 18.34: UG2 pothole prediction through extrapolation of known pothole density statistics, Bleskop Shaft (Chitiyo *et al.*, 2008).

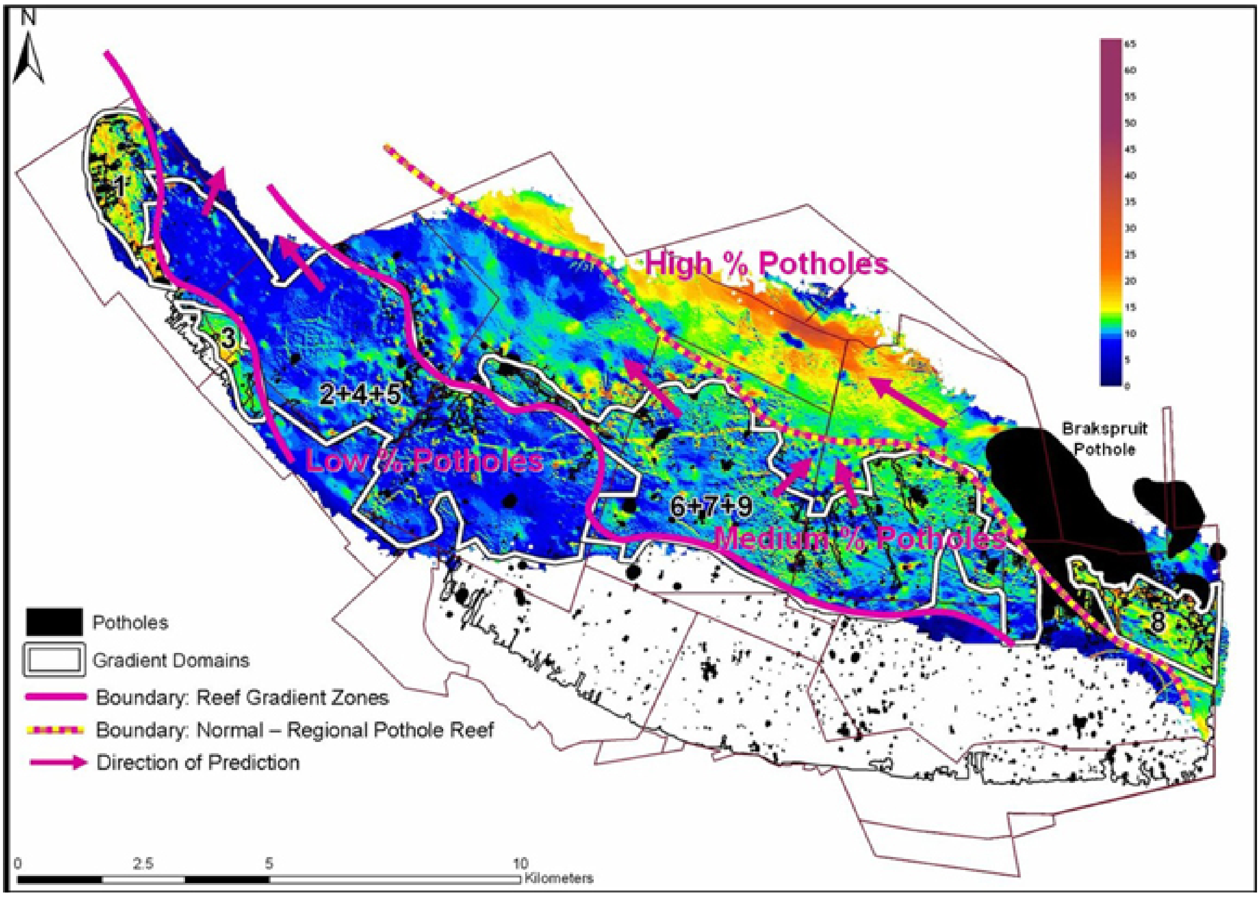


Figure 18.35: Subdivision of 1st derivative of the slope of the Merensky Reef into nine visually semi-homogeneous colour domains (after Chitiyo *et al*., 2008).

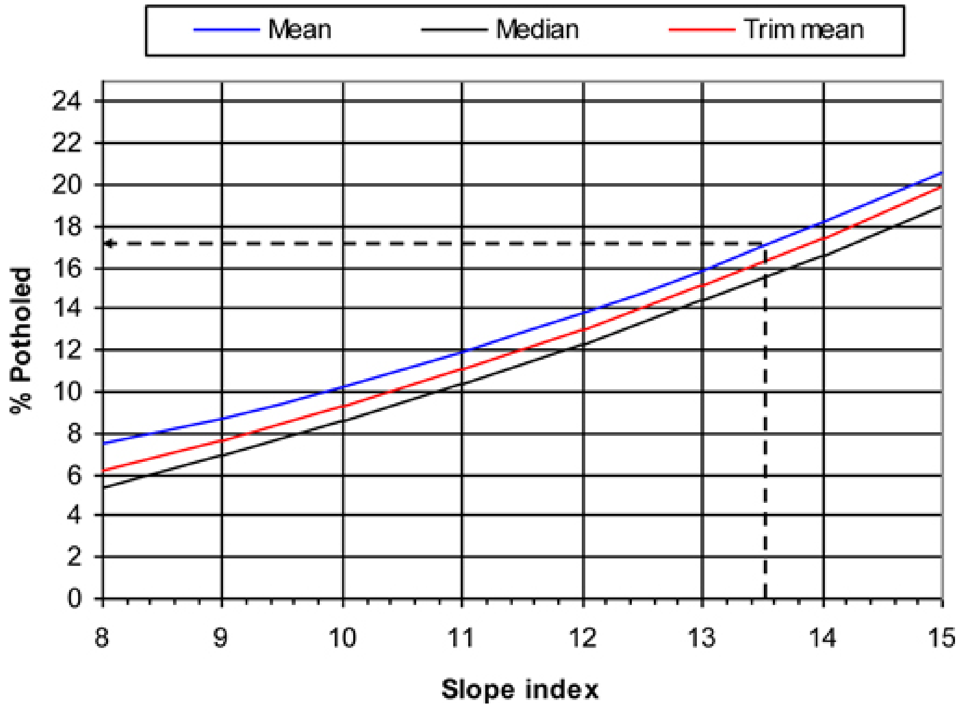


Figure 18.36: Upper 95% confidence limits for prediction derived from the domain data (Chitiyo *et al*., 2008).

There appears to be a similarity between the Brakfontein pothole (LPM – Figure 18.37) with its' associated disturbances and the Brakspruit pothole (Rustenburg Platinum – Figure 18.35). Note the pothole domain in which the Brakspruit pothole occurs has a high percentage of potholes and can be predicated into the unmined areas using the 1st derivative of the slope of the Merensky Reef. It is recommended that the mine implements these predictive techniques in order to mitigate the risk of incurring unplanned additional losses which could be as high as 35% of estimated Resource tonnages. Increased percentages of potholes are evident at the limits of the current mining faces toward the south (Figure 18.38). In addition the UG2 chromitite is associated with the same risk, with historically mined areas not representative of future planned mining areas (Figure 18.39). In the case of the UG2 chomitite the down dip extensions of the orebody appears to have larger potholes (Figure 18.39) and since this is related to the "new magma" – cumulate temperature differential the percentage and aerial extent should be predictable.

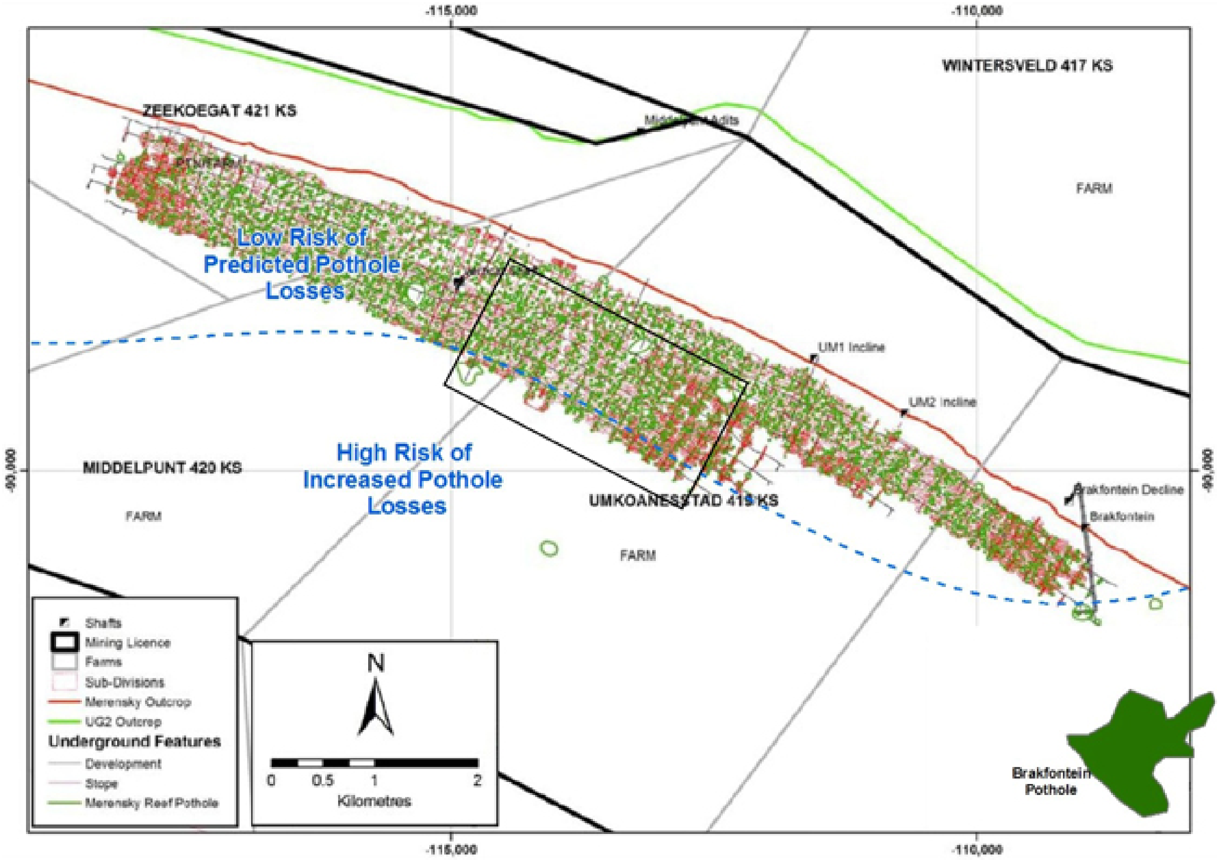


Figure 18.37: Merensky Reef potholes showing high risk potential of large losses due to potholes.

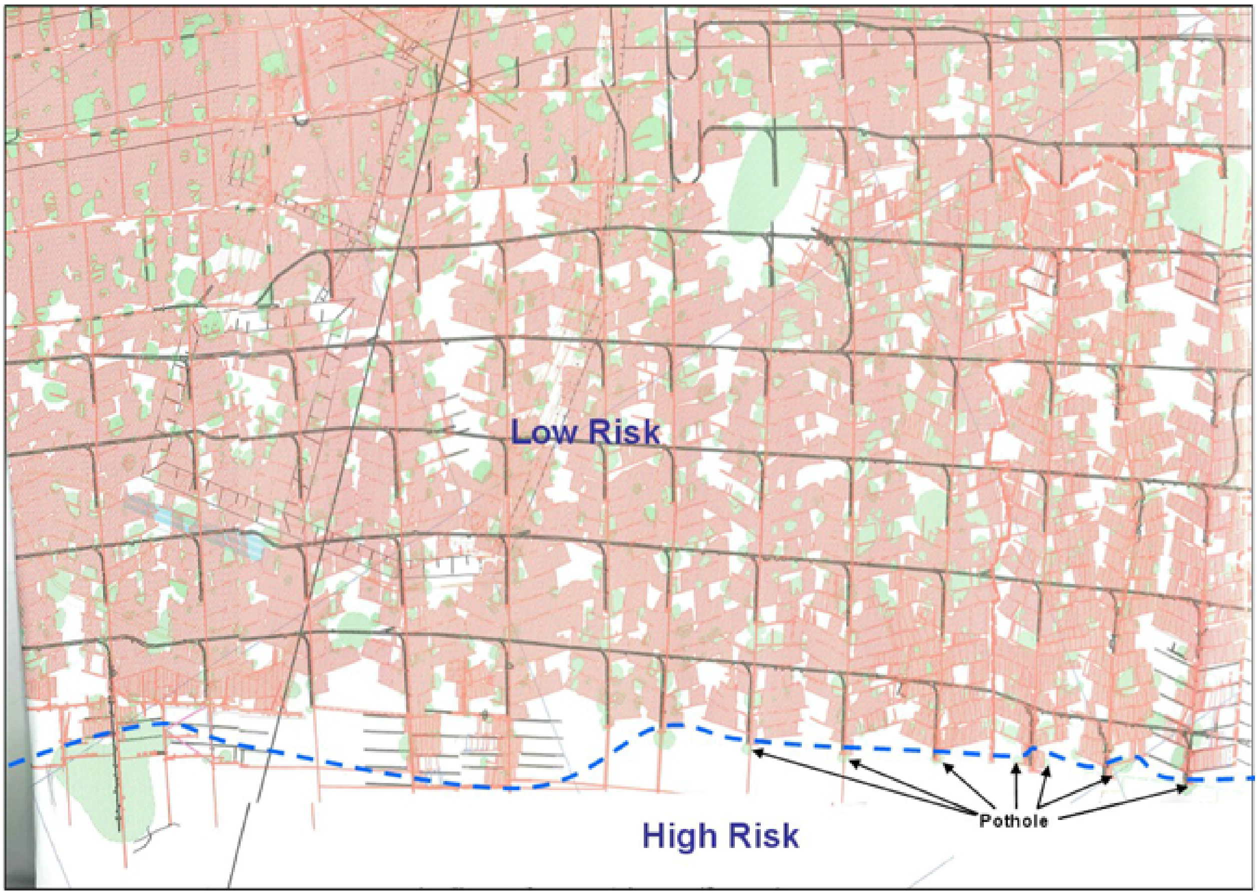


Figure 18.38: A portion of the Merensky Reef pothole plan illustrating the potential for higher pothole losses in the unmined area. Area outlined in previous figure.

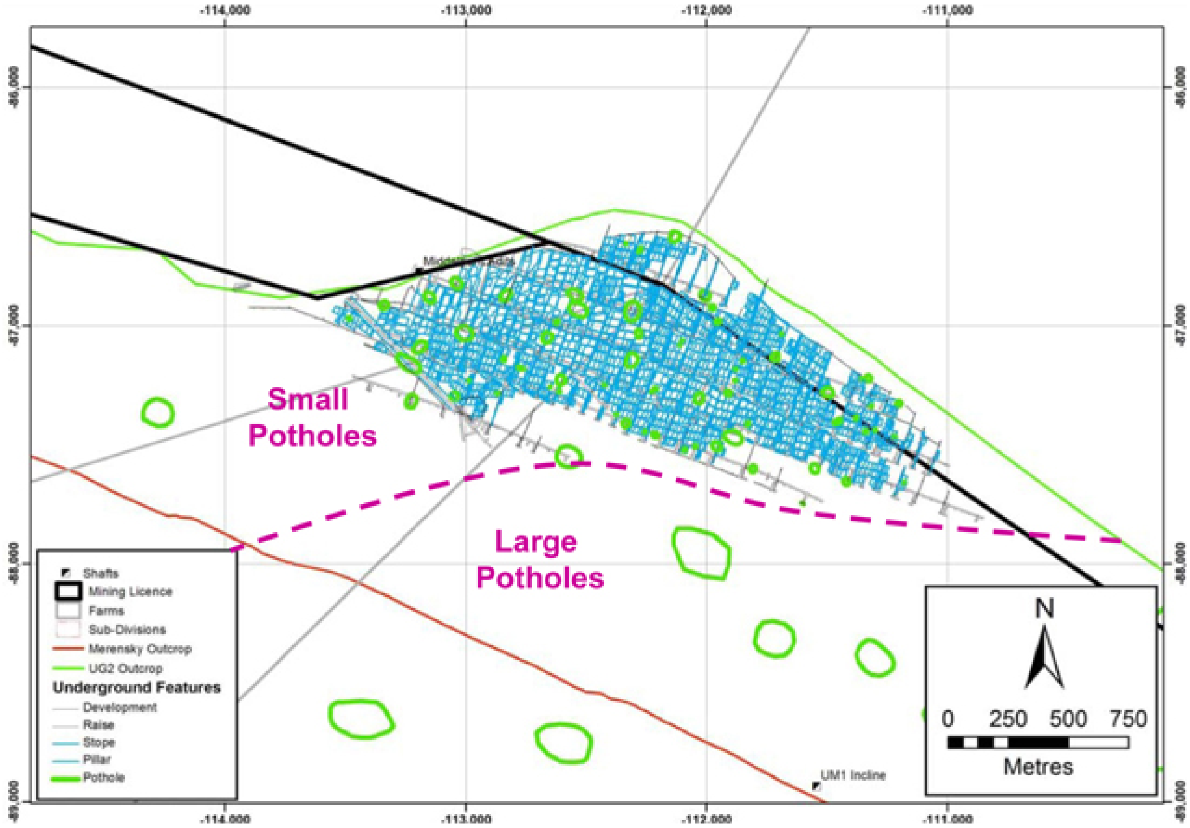


Figure 18.39: UG2 Chromitite potholes subdivided into domains of large and small potholes.

18.9.1 Bifurcated UG2 Reef

A significant number of boreholes have intersected bifurcated UG2 (Figure 18.40). Mining of bifurcated UG2 results in higher dilution and, in cases where the UG2 exceeds 1.20m in thickness, the reef is only partially extracted. It is therefore proposed that the potential of delineating areas of bifurcated UG2 is investigated. These areas should then be treated separately during Resource and Reserve estimation, with the appropriate dilution and geological loss factors.

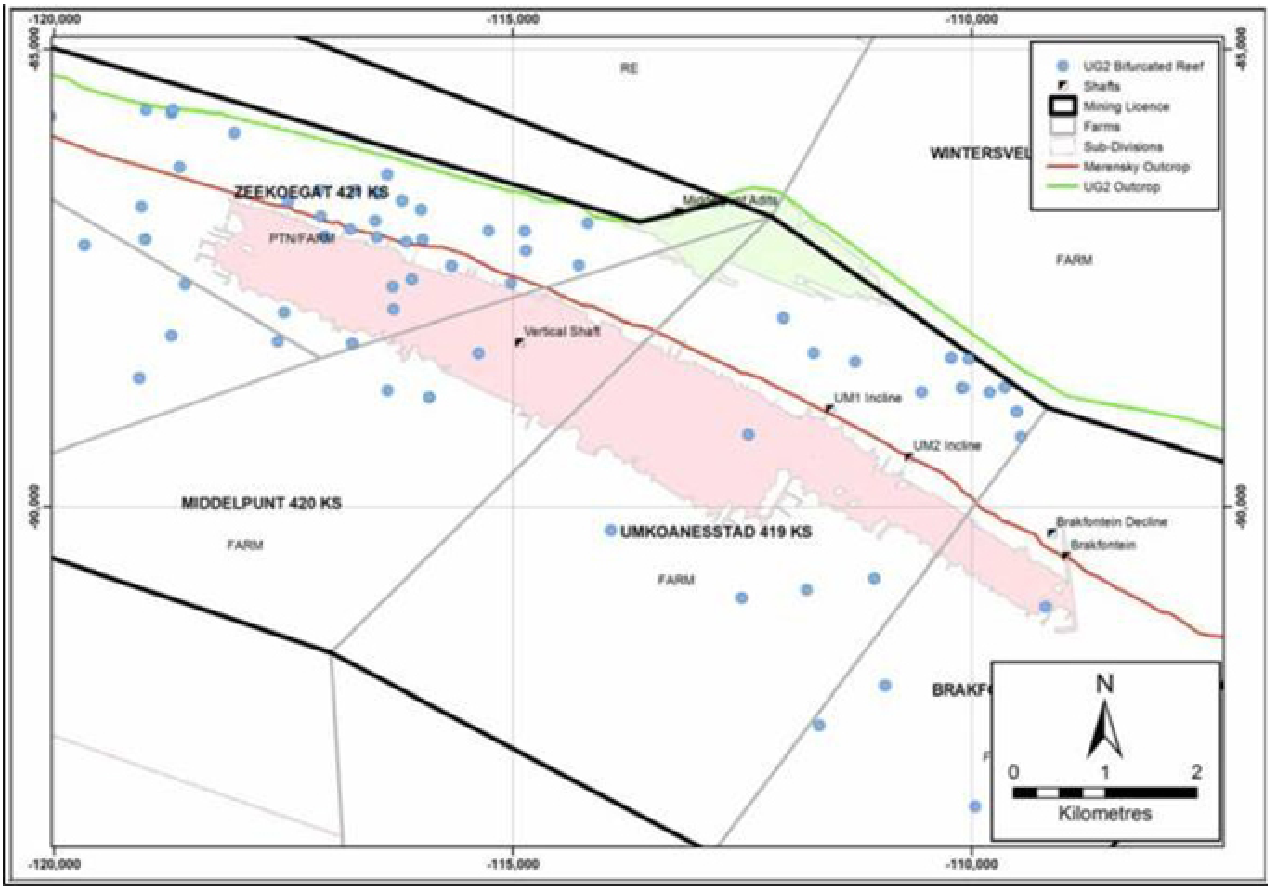


Figure 18.40: High risk of losses from bifurcated UG2 Reef (blue dots).

19 Interpretation and Conclusions

Resources and Reserves

Deloitte has reviewed the relevant information provided by Anglo Platinum and is satisfied that the quality and quantity of geological information, and the estimation methodology, to be sufficient for the declaration of Measured, Indicated (and Inferred) Mineral Resources.

Deloitte considers that the Mineral Resources are classified in accordance with the July 2007 SAMREC Code and in accordance with the *Definition Standards on Mineral Resources and Mineral Reserves* of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) and adopted by CIM Council.

LPM has significant additional Inferred Mineral Resources, both Merensky and UG2 for future classification to Measured and Indicated status with the appropriate follow-up drilling, development and sampling. Both economic horizons extend from outcrop and are continuous to depths beyond 2,000 m.

The UG2 and Merensky Reef 2007 and 2008 Reserve estimates were determined by applying modifying factors based on historical insights and experience, independent from the mining plan and scheduling process. This approach, according to Deloitte's interpretation of the SAMREC Code, may invalidate code compliancy, as the SAMREC Code requires Reserves for operating mines to be based on a Life of Mine plan. National Instrument 43-101, which follows the CIM Definition Standards, requires a pre-feasibility study as a minimum basis on which to base reserve estimation.

The total 2008 declared UG2 Reserve is 41.2 Mt, however the mining schedule generates a 44.9 Mt reserve. The total 2008 declared Merensky Reserve is 27.1 Mt, as compared to a scheduled output of 25.8 Mt. These discrepancies could be attributed to the non-adherence to concurrent reserve estimation and mine planning.

Grades are fairly evenly distributed in the two orebodies, but scope for domaining exists. A significant margin exists between cutoff grades and paylimits, pointing to the inherent economic viability of the orebodies under consideration.

Considering the financial erosion of the *in-situ* value of the two orebodies, and the depreciation of the rand against the US dollar, the UG2 shows significantly more deterioration when compared to the Merensky Reef. This is attributed to the differing prill splits of the orebodies, with the UG2 being characterised with higher rhodium contents.

LOM Plan

Deloitte concurs with Anglo Platinum that the current operations at LPM are incongruous with the extensive resource base.

Anooraq has a conceptual long term strategy for LPM, supported by a financial plan. This strategy is being developed into a long-term mine plan, which will be finalised as the 2009 LOM plan. This will be incorporated into the 2009 Resource and Reserve Statement for LPM.

Key production and financial indicators

Anooraq's financial model, as presented here, is not based on the current Life of Mine plan, but on a statement of strategic intent.

According to this financial model, considering LPM's anticipated 35-year LOM some 92.7 Mt of ore (66.9 Mt UG2, 25.8 Mt Merensky) will be processed, producing some 13.68 MOz's of 4E PGM's including 6.5 MOz's of platinum, 5.8 MOz's of palladium, 1.0 MOz's of rhodium and 320 kOz's of gold plus 50 kt of copper, 73.7 kt of nickel and 1.6 kt of cobalt.

The planned tonnage of UG2 in the current financial model (66.9 Mt) exceeds the UG2 Proven and Probable Reserves (41.2 Mt) by 25.7 Mt. The Measured and Indicated UG2 Resources available for conversion to Reserves are 180.4 Mt.

The DCF model indicates, in real terms, revenue of ZAR126,749 M and a net profit of ZAR33,974 M at a 24.7 % average margin.

The NPV at a discount rate of 7.5 % is ZAR9,290 M.

An overall payback period is not applicable as Lebowa is an operating mine.

Concentrators

No major problems or flaws with the basic process flow sheets of either the Merensky or UG2 Concentrators at LPM are evident. By modern technological standards, the concentrators employ older technology, but nevertheless maintain high operating efficiencies and availability. In general, the plants operate well with few major equipment breakdowns. The most common reasons for current plant stoppages are due to lack of ore from the shafts and planned maintenance. Both concentrators can operate at utilization efficiencies near 90%.

The Merensky Concentrator recoveries of the 4E'S are good, due to the accommodating mineralogy of the Lebowa ore-body. Typically, PGM's are associated with base metal sulphides, and clearly, liberation is very good at the current plant grinds.

The average 4E's recovery over the past 5 year period (2004 to 2008) is 91.9 %, over a range 90.6 % to 93.0% (annual recoveries). Deloitte is of the opinion that the Merensky Concentrator metal recovery performance is good.

Similarly, the UG2 PGM recoveries are good with a five year (2004 to 2008) average of 88.1 %, over a range 86.9 to 89.3 (annual recoveries). The concentrate grades here too are acceptable for smelting. Deloitte is of the opinion that the UG2 Concentrator performance is good.

In addition, it is noted that as UG2 production increases, cognisance has to be taken of the increased overall chrome content for smelting. The ratio of Merensky and UG2 concentrates, in terms of the chrome content, will be affected and may necessitate reducing the chrome content in the UG2 concentrate. This may be at the expense of 4E recovery from the UG2 plant. This aspect would have to be considered when the chrome content specifications and penalties are defined with Polokwane Smelter.

20 Recommendations

Deloitte raises the following recommendations. These elaborate, but are also in addition, to those noted in the main body of this TR:

a) LPM should run their mining plan and scheduling concurrent with the Reserve estimation in order to ensure compliance with the SAMREC code.

b) The Mine Call Factor for the UG2 Reserve estimate should be 94 %.

c) It is recommended that LPM continue with the process of updating the LOM plan in accordance with the strategy outlined in Anooraq's financial model.

d) Scope exists to enhance the mine's geological model, which is also indicated by the variability of reef characteristics for both the UG2 and the Merensky Reef. Model improvement can be achieved through more detailed domaining using orebody characteristics such as pothole characteristics and distribution, grade, channel width or the bifurcated nature of the UG2. Once these domains have been defined these should be geostatistically validated and the Resource estimation should be re-run accordingly.

e) Future studies should be conducted at LPM aiming at the quantification of the geological loss associated with bifurcated UG2 reef, as well as geological losses due to potholing below a depth of 650 m (cf. Section 18.9). In addition, special mining conditions, including support, may need to be considered where bifurcated UG2 exceeds 1.20 m.

f) Due to the significant travelling and transport distances at Vertical Shaft, dynamic logistic simulation should be performed. This will indicate whether the planned tonnages can be hoisted, and will also identify potential bottlenecks.

g) Due to the volatile nature of the current financial environment, the input parameters for the DCF, such as commodity prices, exchange rates and CPI, should be analysed at shorter intervals to establish the impact of changing conditions.

h) It is recommended to upgrade the UG2 Measured and Indicated Resources to Reserve status to cover the existing shortfall in the current financial model.

i) The MRM system should be upgraded in order to cater for storage of prill split data for the underground sampling.

j) Increasing the frequency of underground sampling should be investigated.

k) The Department of Mineral and Energy Affairs has opted to temporarily close operations in the case of major accidents or fatality(ies). It is recommended that this "Stop and Think" approach is considered during mine planning.

l) Borehole logs do not always identify potholes or the edge of potholes. This should be rectified through detailed logging and recording.

m) Further work is necessary to understand the sporadic Merensky Reef footwall mineralisation so that the mining cut Resource extraction can be optimised.

n) A pothole study similar to the one performed by Chitiyo *et al.* (2008) should be considered by LPM. This could potentially lead to the confident predication of percentage potholed, and could also identify areas of significant potholing, which will impact on the geological loss factor to be applied.

21 Glossary

	Units and Abbreviations
3E	Platinum plus palladium plus gold
4E	Platinum plus palladium plus rhodium plus gold
AAS	Atomic Absorption Spectroscopy
AR	Anglo Research – previously known as Anglo American Research Laboratories
ASG	Advance strike gulley
BM	Base metals
BRK	Brakfontein
°C	Degrees centigrade (unit of temperature)
CRM	Certified Reference Material
DCF	Discounted cash-flow
g/t	Grams per tonne
ICP	Inductively Coupled Plasma. (See Technical Terms)
kt	Kilo tonnes (1000 tonnes)
ktpm	Kilo tonnes per month
LOM	Life of Mine
LTP	Long Term Plan
m	Metres, a unit of length. Can be used for length, height, depth, etc.
Ma	Million years (unit denoting geological age)
Merensky	Merensky Pyroxenite reef
MF2	Mill-float, Mill-float
MF3	Mill-float, Mill-float, Mill-float
Minerals Act	Minerals Act, Act No.50 of 1991 (South Africa)
mm	Millimetres, as defined by the International System of Units (SI)
MPH	Middelpunt Hill
Mt	Million tonnes

Mtpa	Million tonnes per year
MPRDA	Minerals and Petroleum Resources Development Act No. 28 of 2002 (South Africa)
Moz's	Million Ounces (see Oz)
NI 43-101	Canadian National Instrument 43-101 (See Technical Terms)
NPV	Net Present Value
NUM	National Union of Mineworkers (South Africa)
Oz	Troy ounce, equivalent to 31.1035 grams
pa	Per annum
PDA	Personal Digital Assistant
PGE / PGM	Platinum Group Metal / Platinum Group Element (See Technical Terms)
PLC	Programmed Logic Control
Pr.Eng	A suitably qualified and experienced professional registered as a *Professional Engineer* with the Engineering Council of South Africa (ECSA)
Pr.Sci.Nat	A suitably qualified and experienced professional registered as a *Professional Natural Scientist* with the South African Council for Natural Scientific Professions (SACNASP)
QA/QC	Quality Assurance / Quality Control
RPM	Rustenburg Platinum Mines Limited
SIB	Stay-in-business
SPECDENS	Spectral density
tpm	Metric tonnes per month
TR	Technical Report
UG2	Upper Group 2 Chromitite Layer
US$	The United States currency, the Dollar
XRF	X-Ray Fluorescence
ZAR	The South African currency, the Rand
%	Percentage

	Technical Terms
Adit	A horizontal or nearly horizontal passage driven from the surface for the working or dewatering of a mine. If driven through the hill or mountain to the surface on the opposite side, it would be a tunnel.
Adjacent Property (NI43-101)	A property (a) in which the issuer does not have an interest; (b) that has a boundary reasonably proximate to the property being reported on; (c) that has geological characteristics similar to those of the property being reported on.
Advance strike gulley (ASG)	A narrow, near horizontal excavation in the direction of strike on the reef horizon that is used to scrape broken ore from the working face back to the original raise.
Base Metals	A classification of metals usually considered to be of low value and higher chemical activity when compared with the noble metals (gold, silver, platinum, etc.). This nonspecific term generally refers to the high-volume, low-value metals copper, lead, tin, and zinc.
CIM	"*CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines*," developed by the CIM Standing Committee on Reserve Definitions, Canadian Institute of Mining, Metallurgy and Petroleum.
Definition Standards	The CIM Definition Standards establish definitions and guidelines for the reporting of exploration information, mineral resources and mineral reserves in Canada. The Mineral Resource and Mineral Reserve definitions were incorporated, by reference, in Canadian National Instrument 43-101.
Competent Person (SAMREC)	A 'Competent Person' is a person who is registered with SACNASP, ECSA or PLATO, or is a Member or Fellow of the SAIMM, the GSSA or a Recognized Overseas Professional Organisation (ROPO). The Competent Person must comply with the provisions of the relevant promulgated Acts. A Competent Person must have a minimum of five years' experience relevant to the style of mineralization and type of deposit or class of deposit under consideration and to the activity he or she is undertaking. Persons being called upon to sign as a Competent Person must be clearly satisfied in their own minds that they are able to face their peers and demonstrate competence in the commodity, type of deposit and situation under consideration.
Concentrate	The clean product recovered from froth flotation, bearing payable metals in a comparatively high concentration. The metals are recovered by smelting and refining.
Concentrator	A plant where ore is separated into values (concentrates) and rejects (tails). An appliance in such a plant, e.g., flotation cell, jig, electromagnet, shaking table. Also called mill; reduction works; cleaning plant. Syn: concentrating plant
Cross-cut	A horizontal underground mining tunnel or excavation that provides access to the

	mining horizon (reef) from the footwall haulage way.
Data verification (NI43-101)	The process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used.
Development	Work of driving openings to and in a proved orebody to prepare it for mining and transporting the ore.
Development Property (NI43-101)	A property that is being prepared for mineral production and for which economic viability has been demonstrated by a feasibility study. This implies that Mineral Reserves can be or have been declared for this property.
Diamond Drilling	The act or process of drilling boreholes using a rotary-type drill machine which uses equipment and tools designed to recover rock samples in the form of cylindrical cores from rocks penetrated by the boreholes.
Dilution / Contamination	Waste material that is mined during the course of mining operations, and thereby forms part of the Reserve.
Disclosure (NI43-101)	Any oral statement or written disclosure made by or on behalf of an issuer and intended to be, or reasonably likely to be, made available to the public in a jurisdiction of Canada, whether or not filed under securities legislation, but does not include written disclosure that is made available to the public only by reason of having been filed with a government or agency of government pursuant to a requirement of law other than securities legislation.
Drilling	The operation of making deep holes with a drill for the purposes of prospecting, exploration, or valuation.
Economically Mineable	Extraction of the Mineral Reserve has been demonstrated viable and justifiable under a defined set of realistically assumed modifying factors. The existence of a mine plan is implied.
Feasibility Study (SAMREC)	A comprehensive design and costing study of the selected option for the development of a mineral project in which appropriate assessments have been made of realistically assumed geological, mining, metallurgical, economic, marketing, legal, environmental, social, governmental, engineering, operational and all other modifying factors, which are considered in sufficient detail to demonstrate at the time of reporting that extraction is reasonably justified (economically mineable) and the factors reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The overall confidence of the study should be stated.
Feasibility Study (NI43-101)	A comprehensive study of a mineral deposit in which all geological, engineering, legal, operating, economic, social, environmental and other relevant factors are considered in sufficient detail that it could reasonably serve as the basis for a final decision by a financial institution to finance the development of the deposit for mineral production.

Footwall	In metal mining, the part of the country rock that lies below the ore deposit. Also the underlying side of a fault, orebody, or mine working; esp. the wall rock beneath an inclined vein or fault.
Hangingwall	The overlying side of an orebody, fault, or mine working, esp. the wall rock above an inclined vein or fault.
Haulage / Haulageway	A horizontal underground mining tunnel or excavation that is used primarily for the transfer of workers, supplies, ore and waste rock between the mine workings and the shaft or adit. It is often located in the footwall.
Head grade	The average grade of ore fed into a mill or concentrator.
Incline	A shaft sunk at an inclination from the vertical, usually following the dip of a lode.
Independent (NI43-101)	A qualified person is independent of an issuer if there is no circumstance that could, in the opinion of a reasonable person aware of all relevant facts, interfere with the qualified person's judgment regarding the preparation of the technical report.
Indicated Mineral Resource (SAMREC)	An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on information from exploration, sampling and testing of material gathered from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological or grade continuity but are spaced closely enough for continuity to be assumed.
Indicated Mineral Resource (CIM)	An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
Inductively Coupled Plasma (ICP)	An analytical technique whereby Inductively Coupled Plasma is used to ionise the sample aliquot, with target concentrations determined by Optical Emission Spectroscopy (OES) or Mass Spectroscopy (MS).
Inferred Mineral Resource (SAMREC)	An 'Inferred Mineral Resource' is that part of a Mineral Resource for which volume or tonnage, grade and mineral content can be estimated with only a low level of confidence. It is inferred from geological evidence and sampling and assumed but not verified geologically or through analysis of grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that may be limited in scope or of uncertain quality and reliability.

Inferred Mineral Resource (CIM)	An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
JORC Code	The Australasian Code for Reporting of Mineral Resources and Ore Reserves prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia, as amended.
Level	A main underground roadway or passage driven along a level course to afford access to stopes or workings and to provide ventilation and a haulageway for the removal of coal or ore. Levels are commonly spaced at regular depth intervals and are either numbered from the surface or designated by their elevation below the top of the shaft.
Life of Mine Plan (LOM)	A design and costing study of an existing operation in which appropriate assessments have been made of realistically assumed geological, mining, metallurgical, economic, marketing, legal, environmental, social, governmental, engineering, operational and all other modifying factors, which are considered in sufficient detail to demonstrate at the time of reporting that extraction is reasonably justified.
Measured Mineral Resource (SAMREC)	A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable information from exploration, sampling and testing of material from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.
Measured Mineral Resource (CIM)	A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
Mine Design	A framework of mining components and processes taking into account such aspects as mining methods used, access to the ore body, personnel and material handling, ventilation, water, power, and other technical requirements, such that mine planning can be undertaken.
Mine Planning	Production planning and scheduling, within the Mine Design, taking into account such aspects as geological structures and mineralization and associated

	infrastructure and constraints.
Mineable	Those parts of the ore body, both economic and uneconomic, that can be extracted during the normal course of mining.
Mineral Deposit (SAMREC)	A deposit is a concentration (or occurrence) of material of possible economic interest, in or on the earth's crust, that may include mineralized material that cannot be estimated with sufficient confidence to be classified in the Inferred category. Portions of a deposit that do not have reasonable and realistic prospects for eventual economic extraction are not included in a Mineral Resource.
Mineral Project (NI43-101)	Any exploration, development or production activity, including a royalty interest or similar interest in these activities, in respect of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.
Mineral Resource (SAMREC)	A 'Mineral Resource' is a concentration or occurence of material of economic interest in or on the earth's crust in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. The location, quantity, grade, continuity and other geological characteristics of a Mineral Resource are known, or estimated from specific geological evidence, sampling and knowledge interpreted from an appropriately constrained and portrayed geological model. Mineral Resources are subdivided, and must be so reported, in order of increasing confidence in respect of geoscientific evidence, into Inferred, Indicated or Measured categories.
Mineral Resource (CIM)	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
Mineral Reserve (SAMREC)	A 'Mineral Reserve' is the economically mineable material derived from a Measured or Indicated Mineral Resource or both. It includes diluting and contaminating materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a Pre-Feasibility Study for a project and a Life of Mine Plan for an operation must have been completed, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors (the modifying factors). Such modifying factors must be disclosed.
Mineral Reserve (CIM)	A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting

	materials and allowances for losses that may occur when the material is mined.
Modifying Factors	"Modifying Factors' include mining, metallurgical, economic, marketing, legal, environmental, social and governmental considerations.
NI 43-101	Canadian National Instrument 43-101, *Standards of Disclosure for Mineral Projects*. Public disclosures by listed entities must comply to these standards.
Ore	The naturally occurring material from which a mineral or minerals of economic value can be extracted profitably or to satisfy social or political objectives. The term is generally but not always used to refer to metalliferous material, and is often modified by the names of the valuable constituent; e.g., iron ore.; ore mineral.
Orebody	A continuous, well-defined mass of material of sufficient ore content to make extraction economically feasible.
Ore pass	A vertical or inclined passage for the downward transfer of ore; equipped with gates or other appliances for controlling the flow. An ore pass is driven in ore or country rock and connects a level with the hoisting shaft or with a lower level.
Platinum Group Metal (PGM)	Any of the minerals native platinum, osmium, iridium, palladium, rhodium, ruthenium, and their alloys, such as osmiridium (Ir,Os), ruthenosmiridium (Ir,Os,Ru), rutheniridosmine (Os,Ir,Ru), and platiniridium (Ir,Pt).
Pre-feasibility Study (SAMREC)	A comprehensive study of the viability of a range of options for a mineral project that has advanced to a stage at which the preferred mining method in the case of underground mining or the pit configuration in the case of an open pit has been established and an effective method of mineral processing has been determined. It includes a financial analysis based on realistic assumptions of technical, engineering, operating, economic factors and the evaluation of other relevant factors that are sufficient for a Competent Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve. The overall confidence of the study should be stated. A Pre-feasibility Study is at a lower confidence level than a Feasibility Study.
Preliminary Assessment (NI43-101)	A study that includes an economic analysis of the potential viability of mineral resources taken at an early stage of the project prior to the completion of a preliminary feasibility study.
Preliminary Feasibility Study or Pre- feasibility Study (NI43-101)	A comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, has been established and an effective method of mineral processing has been determined, and includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be classified as a mineral reserve.
Probable	A 'Probable Mineral Reserve' is the economically mineable material derived from

Mineral Reserve (SAMREC)	a Measured or Indicated Mineral Resource or both. It is estimated with a lower level of confidence than a Proved Mineral Reserve. It includes diluting and contaminating materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a Pre-Feasibility Study for a project or a Life of Mine Plan for an operation must have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. Such modifying factors must be disclosed.
Probable Mineral Reserve (CIM)	A 'Probable Mineral Reserve' is the economically mineable part of an indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
Producing Issuer (NI43-101)	An issuer with annual audited financial statements that disclose (a) gross revenues, derived from mining operations, of at least $30 million for the issuer's most recently completed financial year; and (b) gross revenues, derived from mining operations, of at least $90 million in the aggregate for the issuer's three most recently completed financial years.
Professional Association (NI43-101)	A self-regulatory organization of engineers, geoscientists or both engineers and geoscientists that (a) is (i) given authority or recognition by statute in a jurisdiction of Canada, or (ii) a foreign association recognised in NI43-101; (b) admits individuals on the basis of their academic qualifications and experience; (c) requires compliance with the professional standards of competence and ethics established by the organization; and (d) has disciplinary powers, including the power to suspend or expel a member.
Proved Mineral Reserve (SAMREC)	A 'Proved Mineral Reserve' is the economically mineable material derived from a Measured Mineral Resource. It is estimated with a high level of confidence. It includes diluting and contaminating materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a Pre-Feasibility Study for a project or a Life of Mine Plan for an operation must have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. Such modifying factors must be disclosed.
Proven Mineral Reserve (CIM)	A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Public Report (SAMREC)	Public Reports are all those reports prepared for the purpose of informing investors or potential investors and their advisers and include but are not limited to companies' annual reports, quarterly reports and other reports included in JSE circulars, or as required by the Companies Act. The SAMREC Code also applies to the following reports if they have been prepared for the purposes described above: environmental statements; information memoranda; expert reports; technical papers; website postings; and public presentations.
Qualified Person (NI43-101)	An individual who (a) is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; (b) has experience relevant to the subject matter of the mineral project and the technical report; and (c) is in good standing with a professional association and, in the case of a foreign association recognised in NI43-101, has the required designation.
Raise	A vertical or inclined opening in a mine driven upward from a level to connect with the level above, or to explore the ground for a limited distance above one level. After two levels are connected, the connection may be a winze or a raise, depending upon which level is taken as the point of reference.
RC Drilling	Reverse Circulation drilling. A drilling method in which the bit-coolant and cuttings-removal liquids, drilling fluid, mud, air, or gas is circulated down the borehole outside the drill rods and upward inside the drill rods. This decreases the potential for contamination, as the drill cuttings are transported from the bottom of the borehole *inside* the drill rods.
Reef	A local term for a stratiform or stratabound metalliferous mineral deposit, e.g. Witwatersrand gold-bearing horizons, Bushveld PGE-bearing horizons.
Return airway	Any airway in which vapid air flows from the workings to the upcast shaft or fan.
SAMREC	The South African Mineral Resource Committee. This Committee is a joint Committee of the Southern African Institute of Mining and Metallurgy (SAIMM) and the Geological Society of South Africa (GSSA).
SAMREC Code	*The South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (2007 edition)*, prepared by the South African Mineral Resource Committee (SAMREC) Working Group under the joint auspices of the Southern African Institute of Mining and Metallurgy (SAIMM) and the Geological Society of South Africa (GSSA).
Shaft	An excavation of limited area compared with its depth; made for finding or mining ore or coal, raising water, ore, rock, or coal, hoisting and lowering workers and material, or ventilating underground workings. The term is often specifically applied to an approximately vertical shaft, as distinguished from an incline or inclined shaft. A shaft is provided with a hoisting engine at the top for handling workers, rock, and supplies; or it may be used only in connection with pumping or

	ventilating operations.
Stope	The working above and below a level where the mass of the orebody is broken. A stope is the very antithesis of a shaft, tunnel, drift, winze, or other similar excavation in a mine.
Stoping	The term stoping is loosely applied to any subterranean extraction of ore except that which is incidentally performed in sinking shafts, driving levels, etc., for the purpose of opening the mine.
Technical Report (NI43-101)	A report prepared and filed in accordance with Canadian National Instrument 43-101 and Form 43-101F1 Technical Report that does not omit any material scientific and technical information in respect of the subject property as of the date of the filing of the report.
Travelling way	An underground excavation, sometimes inclined, and used by stope workers to travel to and from the stope. Also a roadway used by miners for walking to and from the face; i.e., from the shaft bottom or main entry to the workings and back.
Trenching	The act of excavating a narrow, shallow ditch cut across a mineral deposit to obtain samples or to observe character.
Working places	Any area of development, usually restricted in meaning to apply to development and stoping areas.
Written Disclosure (NI43-101)	Includes any writing, picture, map or other printed representation whether produced, stored or disseminated on paper or electronically, including websites.

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23 Dates and signatures

Name of Report:

Lebowa Platinum Mine

Limpopo Province, South Africa

12 May 2009

Issued by:
Anooraq Resources Corporation

Dr. J Schweitzer	Date: 12 May 2009
G Güler	Date: 12 May 2009
P Kramers	Date: 12 May 2009
Prof. S de Waal	Date: 12 May 2009
T Naidoo	Date: 12 May 2009

24 Certificates

CERTIFICATE of QUALIFIED PERSON

(a) I, Dr J K Schweitzer – Associate of Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa, do here certify that:

(b) I am the co-author of the technical report titled Lebowa Platinum Mine and dated May 2009 (the Technical Report') prepared for Anooraq Resources Corporation.

(c) I graduated with a Ph.D. in Geology, on the Bushveld Complex, from the University of Pretoria and am a registered Pr. Sci. Nat. Registration No. 400050/04. I am geologist with more than 22 years experience in the South African mining industry, gained predominantly in Witwatersrand Gold and Bushveld Complex Platinum deposits. My experience includes almost 5 years of geological consultancy to the mining industry as a Director of Shango Solutions. Previous employment was with the CSIR, Division of Mining Technology, the last 3 years in a Senior Technical and Scientific Management position as Programme Manager – Mining Systems, which included technical and financial management of mining engineering, non-explosive rock breaking, engineering testing, orebody information, automation and backfill departments (about 47 employees; R30 Million budget). I represented the CSIR on the collaborative PlatMine Research initiative, which was guided by Anglo Platinum, Lonmin, Impala and Northam. Prior to this, I spent 4 years as Thrust Area Manager - Orebody Information (Geology and Geophysics), with the same organization, the CSIR. Responsibilities included management of about 10 professionals, marketing and securing of funding, financial and technical management, initiation of collaborative efforts with other companies working across disciplines within both the CSIR and the mining companies. Such other disciplines included rock engineering, mining engineering, geophysics, surveying, and mine planning. Before receiving this promotion, I spent 10 years as a geologist with the Chamber of Mines Research Unit which was incorporated into the CSIR. Most of the work concentrated on the Witwatersrand sediments and the Bushveld Igneous Complex but some work on the Karoo coal was also conducted. I have been involved in Geology, Mining and Resource Evaluation consulting practice for over 20 years, including more than 5 years in Bushveld Complex style of deposit mineralisation.

I am a Fellow of the Geological Society of South Africa (GSSA) and a Fellow of the South African Institute of Mining and Metallurgy (SAIMM). I have served on the Council of the former and am at present also on the Professional Affairs Committee of the GSSA.

I am a "qualified person" for the purposes of NI43_101.

(d) I visited the Lebowa Platinum Mine Property on the 18th and 19th December 2008.

(e) I am co-responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in Section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa, this 12th day of May 2009.

Dr J K Schweitzer – Associate -Deloitte Mining Advisory Services – Johannesburg.

CERTIFICATE of QUALIFIED PERSON

(a) I, S A de Waal, Professor Emeritus and Associate of Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa, do here certify that:

(b) I am the co-author of the technical report titled Lebowa Platinum Mine dated May 2009 (the 'Technical Report') prepared for Anooraq Resources Corporation.

(c) I graduated with a D.Sc. in Geology from the University of Pretoria in 1967 and am a registered Pr. Sci. Nat. Registration No. 400598/83. I am an Honorary Fellow of the Geological Society of South Africa which I previously served in the capacities of Ordinary Member, Council Member, Vice-President, and President. I also served on the Publication and Activation Committees of the Society, and as Convenor of the Publications Committee. Since my retirement from the University of Pretoria I have been involved in several projects on the Bushveld Igneous Complex, comprising target generation and pothole investigations, as well as on Witwatersrand gold mines as a part time consultant with Shango Solutions. I had previously been Professor and Head of the Geology Department at the University of Pretoria, Professor of Geology at the Potchefstroom University for CHE and Senior Lecturer at Rand Afrikaans University and taught courses in Systematic and Process Mineralogy, Igneous Petrology, Metamorphic Petrology, Geochemistry and Geostatistics. Interspersed between the positions in academia I had worked as Director of Mines and Acting Director of Energy, promoted to Deputy Permanent Secretary in the Ministry of Mines and Energy, Republic of Namibia. Previously, I had worked at the National Institute for Metallurgy (now Council for Mineral Technology), firstly as a Scientist, promoted to Senior Scientist, Principal Scientist and finally to Chief Scientist in charge of research on the application of mineralogy in extraction metallurgy and projects on low grade nickeliferous and chromite ores of the Bushveld Complex. At the Council for Mineral Technology (MINTEK), I served as Chief Scientist, promoted to Assistant Director (Applied Mineralogy) where I researched supergene the alteration of sulphide ore and its effects on the floatation properties of these minerals.

I have been involved in Geology for over 46 years, have worked on many mineral deposits including the Bushveld Complex, and produced ca. 300 technical reports and publications on different aspects of geology and mineral sciences.

I am a “qualified person” for the purposes of NI 43-101.

(d) I have not visited the property recently.

(e) I am co-responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in Section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa, this 12th day of May 2009.

S A de Waal, Associate -Deloitte Mining Advisory Services – Johannesburg.

CERTIFICATE of QUALIFIED PERSON

(a) I, G Güler – Associate of Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa do here certify that:

(b) I am the co-author of the technical report titled Lebowa Platinum Mine and dated May 2009 (the 'Technical Report') prepared for Anooraq Resources Corporation.

(c) I graduated with an M.Sc. in Mining Engineering from the University of the Witwatersrand, Johannesburg, in 1997 and completed a Programme in Engineering Management at the University of Pretoria, Pretoria, South Africa. I am a registered Professional Engineer with the Engineering Council of South Africa (ECSA) – Registration No. 970017 and a Fellow the South African Institute of Mining and Metallurgy (SAIMM) as well as a Member of the Australian Institute of Mining and Metallurgy (AusIMM).

Since September 2008, I have been with the consultancy firm Shango Solutions as a consulting Mining Engineer. Previously I worked at TWP Consulting (Pty) Ltd for a year, finishing up as Senior Mining Engineer – Project Manager. Before joining TWP, I spent 14 years working at CSIR, Division of Mining Technology, the last 3 years in a Senior Technical and Scientific Management position as Contract Manager: Mining. I have also worked as a mining engineer on Witwatersrand and Turkish mines.

Included in my responsibilities have been managing various consultancy projects such as optimization of underground mining operations, pre-feasibility and feasibility studies and preparation of mining plans for open pit mining operations.

I am a "qualified person" relating to my contribution on the mining engineering section of the report for the purposes of NI 43-101.

(d) I have visited the Lebowa Platinum Mine Property on 18-19 December 2008, 20-21 January 2009 and 17 March 2009.

(e) I am responsible for the preparation of the sections of the Technical Report related to mining engineering as detailed in Table 2.1.

(f) I am independent of the issuer as defined in Section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa, this 12th day of May 2009.

G Güler – Associate -Deloitte Mining Advisory Services – Johannesburg.

CERTIFICATE of QUALIFIED PERSON

(a) I, C P Kramers, Associate of Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa do here certify that:

(b) I am a co-author of the Technical Report titled Lebowa Platinum Mine dated May 2009 (the 'Technical Report'), prepared for Anooraq Resources Corporation.

(c) I graduated with a B. Sc. Eng (Metallurgy) in 1971 from the University of the Witwatersrand, Johannesburg, South Africa. I am a Fellow of the Southern African Institute of Mining and Metallurgy, and am registered as a Professional Engineer in terms of the Engineering Professions Act, 2000 (Act No. 46 of 2000), Registration No. 20050235, with the Engineering Council of South Africa (ECSA). I have worked as a Metallurgical Engineer and Manager in the South African mining industry continuously for over 30 years – including the platinum, gold and base metal mining industry, since graduating from university. The metallurgical processing section of the Technical Report has been prepared in compliance with the Instrument and Form 43-101F1. I have been involved in the mining consulting profession for more than 15 years and have done peer review, audit and due diligence studies, accident investigations, design and feasibility studies and valuations on various projects in the platinum, gold and base metal mining industry during this period.

I am a "qualified person" relating to my contribution on the metallurgical processing section of the report for the purposes of NI 43-101.

(d) I visited the metallurgical processing section of Lebowa Platinum Mine from 22nd to 23rd January, 2009.

(e) I am co-responsible for the preparation of the metallurgical processing section of the Technical Report.

(f) As of the date of this Technical Report, to the best of my knowledge, information and belief, the metallurgical processing section of the report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa, this 12th day of May 2009.

C P Kramers, Associate -Deloitte Mining Advisory Services – Johannesburg.

CERTIFICATE of CO-AUTHOR

(a) I, P E Lambert, Associate of Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa do here certify that:

(b) I am the co-author of the technical report titled Lebowa Platinum Mine and dated May 2009 (the 'Technical Report') prepared for Anooraq Resources Corporation.

(c) I graduated with an M.Sc. (Eng) Mineral Economics – University of the Witwatersrand (part-time 1997 – 1998), Johannesburg, South Africa. Since 2006 I have worked as a Geological Consultant to the mining industry through Dunrose Trading 186 (PTY) Ltd, t/a Shango Solutions. Most of the work comprised projects on both the Witwatersrand gold mines and the Bushveld Complex platinum deposits. During 2003, I worked as an Independent Consultant to the CSIR/Miningtek, conducting due diligences, advising on geological modelling and reserve and resource estimations. Since graduating in 1982, I have spent 20 years as a geologist with Goldfields, in capacities ranging from Mine Geologist on both Witwatersrand gold and Bushveld platinum mines, to Consulting Geologist in Head Office. Four years were spent as Regional Geologist responsible for African exploration.

I have been involved in the mining and consulting profession and have conducted feasibility studies and valuations on various platinum projects in the past.

(d) I visited the Lebowa Platinum Mine Property on the 18th and 19th December 2008.

(e) I am co-responsible for the preparation of the Technical Report under the supervision of a 'qualified person' for the sections as detailed in Table 2.1.

(f) I am independent of the issuer as defined in Section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa, this 12th day of May 2009.

P E Lambert - Associate -Deloitte Mining Advisory Services – Johannesburg.

CERTIFICATE of CO-AUTHOR

(a) I, T Naidoo, Senior Geologist – Deloitte Mining Advisory Services of Deloitte Consulting Pty Ltd., C/O Building 4, Deloitte Place, The Woodlands, Woodlands Drive, Woodmead, Sandton, Johannesburg, South Africa, do hereby certify that:

(b) I am the co-author of the technical report titled Lebowa Platinum Mine and dated May 2009 (the 'Technical Report') prepared for Anooraq Resources Corporation.

(c) I graduated with a MSc degree in Exploration Geology in 2005 from Rhodes University, Grahamstown, South Africa and a BSc (Hons.) degree in Geology in 1999 from the University of Natal, Durban, South Africa.

I am registered as a Professional Natural Scientist (Pr.Sci.Nat) with the South African Council of Natural Scientific Professions (SACNASP), registration number **400262/05.**

I am a Member of the Geological Society of South Africa. I currently serve on its Council, and am a member of its Professional Affairs Committee. I have worked as a Geologist and Consultant in the South African mining and exploration industry for over 10 years.

I have been involved in the mining consulting profession for more than 4 years and have worked on various platinum projects in that time.

I am a "qualified person" for the purposes of NI43-101.

(d) I made a current visit to the Lebowa Platinum Mine from 17 to 19 December 2008 and 20 to 21 January 2009.

(e) I am co-responsible for the preparation of the Technical Report under the supervision of a 'qualified person' for the sections as detailed in Table 2.1.

(f) I am independent of the issuer as defined in Section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg, South Africa this day, the 12th of May 2009

T Naidoo – Senior Geologist – Deloitte Mining Advisory Services - Johannesburg