Salobo Copper-Gold Mine
Carajás, Pará State, Brazil

Technical Report

Neil Burns, P.Geo.
Chris Davis, P.Geo
Cassio Diedrich, AusIMM-CP(Min)
Maurice Tagami, P.Eng.

Effective date:
December 31, 2017
CERTIFICATE OF QUALIFIED PERSON

I, Neil Burns, M.Sc., P.Geo., am employed as Vice President, Technical Services, Wheaton Precious Metals Corp. (Wheaton).

This certificate applies to the technical report titled “Salobo Copper-Gold Mine Carajás, Pará State, Brazil – Technical Report” that has an effective date of December 31, 2017 (the “technical report”).

I am a professional geologist with over 20 years of exploration, mining and resource geology experience in precious and base metals. I graduated from Dalhousie University with a B.Sc in 1995 and from Queen’s University with a M.Sc. in 2003. I have practiced professionally since graduation in 1995. In that time I have been directly involved in generation of, and review of, mineral tenure, surface and other property rights, geological, mineralization, exploration and drilling data, geological models, sampling, sample preparation, assaying, quality assurance-quality control databases, mineral resource estimation, risk analyses, mine geology, reconciliation, preliminary economic assessment, pre-feasibility and feasibility studies, and due diligence studies in Canada, USA, Central and South America, Europe, Eurasia, Africa and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have visited the Salobo Operations.

I am responsible for Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 25 and 26 of the technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

March 29, 2018

“Signed and Stamped”

Neil Burns, M.Sc., P.Geo.
CERTIFICATE OF QUALIFIED PERSON

I, Christopher Davis, M.Sc., P. Geo., and employed as the Director Resource Management Group, Vale Base Metals.

This certificate applies to the technical report titled “Salobo Copper-Gold Mine Carajás, Pará State, Brazil – Technical Report” that has an effective date of December 31, 2017 (the “technical report”).

I am a professional geologist with over 30 years of geology and mining experience in gold and base metals, graduating from McMaster University with a Hons. B. Sc. and McGill University with a M. Sc. I am member of the Association of Professional Geoscientists of Ontario and the Canadian Securities Administrators Mining Technical Advisory and Monitoring Committee. I have practiced professionally since graduation in 1985. In that time I have been directly involved in generation of, and review of, mineral tenure, surface and other property rights, geological, mineralization, exploration and drilling data, geological models, sampling, sample preparation, assaying and other mineral resource-estimation related analyses, quality assurance-quality control databases, mineral resource estimates, risk analyses, preliminary economic assessment, pre-feasibility, and feasibility studies, and due diligence studies in Canada, Indonesia, New Caledonia and Brazil.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have visited the Salobo Operations a minimum of annually since 2013.

I am responsible for Sections 7, 8, 9, 10, 11, 12, 25 and 26 of the technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

March 29, 2018

“Signed and Stamped”

Christopher Davis, M. Sc., P. Geo.
I, Cassio Diedrich, AusIMM-CP(Min) am employed as General Manager of Technical Services, Vale S/A.

This certificate applies to the technical report titled “Salobo Copper-Gold Mine Carajás, Pará State, Brazil – Technical Report” that has an effective date of December 31, 2017 (the “technical report”).

I am a member of the Brazilian National Engineering Council (CREA) and AusIMM (Australasian Institute of Mining and Metallurgy). I am graduated as Mining Engineer by the Federal University of Rio Grande do Sul (UFRGS) in 2008 and I have completed a Master’s Scholarship Graduation on geostatistics and mining engineering in 2012. I have practiced professionally since graduation in 2008. In that time I have been directly involved in generation of, and review of mineral tenure, surface and other property rights, geological, mineralization, exploration and drilling data, geological models, sampling, sample preparation, assaying and other resource-estimation related analyses, quality assurance-quality control databases, resource estimates, risk analyses, pit optimization, mining scheduling, fleet dimensioning, MRMR, preliminary economic assessment, pre-feasibility, feasibility studies, and due diligence and operational studies related to Salobo Operations in Brazil.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have constantly visited the Salobo Operations.

I am responsible for Sections 14, 15, 16, 18 19, 20, 21, 25, 26 of the technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

DATED March 29, 2018

“Signed and Stamped”

Cassio Diedrich, AusIMM-CP(Min), Member No 312603.
CERTIFICATE OF QUALIFIED PERSON

I, Maurice Tagami, P. Eng., am employed as Vice President, Mining Operations, Wheaton Precious Metals Corp. (Wheaton).

This certificate applies to the technical report titled “Salobo Copper-Gold Mine Carajás, Pará State, Brazil – Technical Report” that has an effective date of December 31, 2017 (the “technical report”).

I am a professional engineer with over 35 years of metallurgical, mineral processing, mining operations and project development experience in both base and precious metals. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia and hold a Bachelor of Applied Science degree in Metallurgical Engineering from the University of British Columbia. I have practiced professionally since graduation in 1981. In that time, I have been directly involved in generation of, and review of, preliminary economic assessment, pre-feasibility, and feasibility studies, and due diligence studies in Canada, Brasil, Tanzania, Ghana and Portugal.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have visited the Salobo Operations a minimum of annually from 2013.

I am responsible for Sections 13, 17, 25 and 26 of the technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

March 29, 2018

“Signed and Stamped”

Maurice Tagami, P. Eng.
## Table of Contents

1. Summary .................................................................................................................. 9
2. Introduction ................................................................. 11
   2.1. Terms of Reference ................................................................. 11
   2.2. Qualified Persons............................................................................ 11
   2.3. Site Visits .......................................................................................... 12
   2.4. Previous Technical Reports ......................................................... 12
3. Reliance on Experts ............................................................................................... 12
4. Property Description and Location ......................................................................... 13
   4.1. Location ................................................................................................ 13
   4.2. Regulatory ............................................................................................ 14
   4.3. Mineral Tenure .................................................................................... 14
   4.4. Royalties/Mining Taxes ......................................................................... 15
   4.5. Social License ........................................................................................ 16
   4.6. Comments on Section 4 ........................................................................... 17
5. Accessibility, Climate, Local Resources, Infrastructure and Physiography ............ 17
   5.1. Accessibility ......................................................................................... 17
   5.2. Climate ................................................................................................. 17
   5.3. Local Resources and Infrastructure ...................................................... 18
   5.4. Physiography ......................................................................................... 18
   5.5. Comments on Section 5 ........................................................................... 18
6. History .................................................................................................................... 19
   6.1. Production History ................................................................................... 21
7. Geological Setting and Mineralization .................................................................... 22
   7.1. Regional Geology ................................................................................. 22
   7.2. Property Geology .................................................................................... 23
   7.3. Tectonic Setting ........................................................................................ 28
   7.4. Metamorphism ......................................................................................... 29
   7.5. Alteration ................................................................................................ 29
   7.6. Mineralization .......................................................................................... 29
   7.7. Comments on Section 7 ........................................................................... 31
8. Deposit Types ........................................................................................................ 31
9. Exploration ............................................................................................................. 32
   9.1. Geological Mapping ................................................................................... 32
9.2. Airborne Gravity Survey ................................................................................ 32
9.3. Comments on Section 9 ................................................................................ 34

10. Drilling ................................................................................................................. 34
10.1. Drill Methods ................................................................................................. 36
10.2. Core Reception, Handling and Storage ........................................................ 36
10.3. Geological logging ....................................................................................... 36
10.4. Recovery ....................................................................................................... 37
10.5. Collar Surveys .............................................................................................. 37
10.6. Downhole Surveys ........................................................................................ 37
10.7. Specific Gravity Determination ...................................................................... 38
10.8. Comments on Section 10 .............................................................................. 39

11. Sample Preparation, Analyses and Security ....................................................... 39
11.1. Sampling Methods ........................................................................................ 39
11.1.1. Drill Core ................................................................................................ 39
11.1.2. Blast Holes ............................................................................................. 39
11.2. Sample Preparation ...................................................................................... 40
11.2.1. Exploration ............................................................................................. 40
11.2.2. Grade Control ........................................................................................ 41
11.3. Sample Analysis ........................................................................................... 42
11.3.1. Exploration ............................................................................................. 42
11.3.2. Grade Control ........................................................................................ 42
11.4. Quality Assurance and Quality Control ......................................................... 43
11.4.1. Blanks .................................................................................................... 45
11.4.2. Duplicates .............................................................................................. 45
11.4.3. Quality Control for Blast-holes ............................................................... 46
11.5. Security ......................................................................................................... 46
11.6. Comments on Section 11 .............................................................................. 46

12. Data Verification .................................................................................................. 46
12.1. Major Mining Studies .................................................................................... 46
12.2. External Audits and Reviews ........................................................................ 47
12.3. Comments on Section 12 .............................................................................. 48

13.1. Metallurgical Testwork .................................................................................. 48
13.1.1. Variability Tests ...................................................................................... 49
<table>
<thead>
<tr>
<th>Section</th>
<th>Title</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>13.1.3.</td>
<td>Mixed Ore Zone Copper Recovery Testwork</td>
<td>53</td>
</tr>
<tr>
<td>13.2.</td>
<td>Recovery Estimates</td>
<td>56</td>
</tr>
<tr>
<td>13.3.</td>
<td>Metallurgical Variability</td>
<td>57</td>
</tr>
<tr>
<td>13.4.</td>
<td>Deleterious Elements</td>
<td>57</td>
</tr>
<tr>
<td>13.5.</td>
<td>Actual Plant Results versus Budgeted Projections</td>
<td>58</td>
</tr>
<tr>
<td>13.5.1.</td>
<td>Historical Metallurgical Results</td>
<td>58</td>
</tr>
<tr>
<td>13.5.2.</td>
<td>Historical Plant Availability and Utilization</td>
<td>62</td>
</tr>
<tr>
<td>13.5.3.</td>
<td>Recovery Projections</td>
<td>63</td>
</tr>
<tr>
<td>13.6.</td>
<td>Process Plant Optimization</td>
<td>64</td>
</tr>
<tr>
<td>13.7.</td>
<td>Comments on Section 13</td>
<td>67</td>
</tr>
<tr>
<td>14.1.</td>
<td>Introduction</td>
<td>67</td>
</tr>
<tr>
<td>14.2.</td>
<td>Geological Interpretation</td>
<td>68</td>
</tr>
<tr>
<td>14.3.</td>
<td>Domaining</td>
<td>70</td>
</tr>
<tr>
<td>14.4.</td>
<td>Statistical Analysis</td>
<td>73</td>
</tr>
<tr>
<td>14.4.1.</td>
<td>Raw Assay Statistics</td>
<td>73</td>
</tr>
<tr>
<td>14.4.2.</td>
<td>Compositing</td>
<td>74</td>
</tr>
<tr>
<td>14.4.3.</td>
<td>Domained Composite Statistics</td>
<td>74</td>
</tr>
<tr>
<td>14.4.4.</td>
<td>Outlier Analysis</td>
<td>74</td>
</tr>
<tr>
<td>14.5.</td>
<td>Continuity Analysis</td>
<td>75</td>
</tr>
<tr>
<td>14.6.</td>
<td>Block Modeling</td>
<td>75</td>
</tr>
<tr>
<td>14.6.1.</td>
<td>Dimensions</td>
<td>75</td>
</tr>
<tr>
<td>14.6.2.</td>
<td>Boundary Conditions</td>
<td>76</td>
</tr>
<tr>
<td>14.6.3.</td>
<td>Block Estimation</td>
<td>76</td>
</tr>
<tr>
<td>14.6.4.</td>
<td>Classification Coding</td>
<td>77</td>
</tr>
<tr>
<td>14.6.5.</td>
<td>Model Validation</td>
<td>77</td>
</tr>
<tr>
<td>14.7.</td>
<td>Resource Model Pit Optimization</td>
<td>79</td>
</tr>
<tr>
<td>14.8.</td>
<td>Classification of Mineral Resources</td>
<td>81</td>
</tr>
<tr>
<td>14.10.</td>
<td>Comments on Section 14</td>
<td>83</td>
</tr>
<tr>
<td>15.</td>
<td>Mineral Reserve Estimates</td>
<td>84</td>
</tr>
<tr>
<td>15.1.</td>
<td>Mineability and Dilution</td>
<td>84</td>
</tr>
<tr>
<td>15.2.</td>
<td>Pit Optimization</td>
<td>84</td>
</tr>
</tbody>
</table>
Figure 2 - Mineral Tenure Layout Plan ................................................................. 15
Figure 3 - Regional Geology of the Carajás Province ........................................ 23
Figure 4 - Major Lithological Units – Plan View ................................................. 26
Figure 5 - Major Lithological Units – Vertical Section View ............................. 27
Figure 6 – Core Photos of Major Lithological Units ......................................... 27
Figure 7 - Tectonic Setting of the Carajás Region .............................................. 28
Figure 8 - Copper Mineralization Styles at Salobo .......................................... 31
Figure 9 - Coincident Magnetic and Gravimetric Anomalies ............................ 33
Figure 10 - 3D Gravity Inversion with Current >0.5% Cu Block Model Outline ... 34
Figure 11 - Plan View – Drill Hole Traces ............................................................ 35
Figure 12 - Blast-Hole Sampling Pattern ............................................................ 40
Figure 13 - Sample Preparation Flowchart ....................................................... 41
Figure 14 - Copper Recoveries in 2003–2004 Variability Testwork .................... 50
Figure 15 - Gold Recoveries from Variability and LCT Testwork Programs ....... 51
Figure 16 - Derivation of Copper Recovery Projection, 2003–2004 Variability Testwork ................................................................. 51
Figure 17 - Derivation of Gold Recovery Projection from Variability Testwork ...... 52
Figure 18 - Testwork with Mixed Ore – Effect of Dispersant and Collector Dosage 54
Figure 19 - Expected Metallurgy of Various Sulphide-Mixed Ore Blends ........... 55
Figure 20 - Test Results with Fresh and Mixed Ores with Modified Reagent Scheme . 56
Figure 21 - Actual versus Projected Monthly Plant Copper Recovery ............... 59
Figure 22 - Actual versus Projected Monthly Plant Gold Recovery .................... 59
Figure 23 - Historical Plant Performance – Cu Recovery ................................... 60
Figure 24 - Historical Plant Performance – Au Recovery ................................. 60
Figure 25 - Historical Plant Performance – Concentrate Cu Grade .................. 61
Figure 26 - Historical Plant Performance – Concentrate Au Grade ..................... 61
Figure 27 - Salobo Plant Historical Availability ............................................... 62
Figure 28 - Salobo Plant Historical Operational Utilization .............................. 63
Figure 29 - Salobo Mineral Processing Facility ............................................... 65
Figure 30 - Salobo Flow Sheet ......................................................................... 66
Figure 31 - Salobo Deposit Sectors ................................................................. 69
Figure 32 - Cu Grade Shell Model – Section 1100SE (looking west) ................. 70
Figure 33 - Domain Definition for Copper ....................................................... 72
Figure 34 - Domain Definition for Gold ............................................................ 72
Figure 35 - Domain 1203 Swath Plot - Copper ............................................... 79
Figure 36 - Domain 1203 Swath Plot - Gold .................................................... 79
Figure 37 – Mineral Resource Pit – Isometric View NNE .................................. 81
Figure 38 - Revised Wall Designs ................................................................. 87
Figure 39 – Geotechnical Design Sectors ...................................................... 88
Figure 40- 2017 MRMR Longitudinal Section (looking southwest) .................. 92
Figure 41 - Production Reconciliation Schematic ............................................ 93
Figure 42 – Salobo Phases ........................................................................... 97
Figure 43 - Life of Mine Plan (Mineral Reserves Only) ................................... 98
Figure 44 - Simplified Process Flowsheet ....................................................... 104
Figure 45 - Process Plant Single Line Equipment .......................................... 108
Figure 46 - Concentrate Load Out, Parauapebas ............................................ 111
Table of Tables

Table 1 - December 31, 2017 Mineral Reserves & Mineral Resources ......................... 10
Table 2 - Salobo Production .......................................................................................... 21
Table 3 - Exploration Summary 1978–2003 ................................................................. 32
Table 4 - Drill Hole Summary Table ............................................................................. 35
Table 5 - Blast-Hole Sample Analysis ......................................................................... 43
Table 6 - Adjustment for Copper Assays for pre-2002 Drilling Programs ..................... 44
Table 7 - Adjustment for Gold Assays for pre-2002 Drilling Programs ......................... 45
Table 8 - Summary Duplicate Analysis ........................................................................ 46
Table 9 - Processing Recovery Assumptions (2018-2022) ......................................... 63
Table 10 - Processing Recovery Assumptions (2023-LoMP) ....................................... 64
Table 11 - December 31, 2017 Mineral Resource Estimates ......................................... 68
Table 12 – Block Model Domains Codes .................................................................... 71
Table 13 - Block Model Zone Codes ............................................................................. 71
Table 14 - Copper Assay Statistics by Lithology .......................................................... 73
Table 15 - Gold Assay Statistics by Lithology ............................................................... 73
Table 16 - Grade Capping Levels ................................................................................. 75
Table 17 – Outlier Restriction ....................................................................................... 75
Table 18 – Salobo Variography .................................................................................... 75
Table 19 - Block Model Origin ..................................................................................... 76
Table 20 - Boundary Conditions .................................................................................. 76
Table 21 - Search Ellipse Dimensions - Copper ............................................................ 77
Table 22 - Search Ellipse Dimensions – Gold ............................................................... 77
Table 23 - Global Mean Analysis ................................................................................. 78
Table 24 - Mineral Resource Open Pit Optimization Assumptions .............................. 80
Table 25 - Mineral Resources from 2016 to 2017 ....................................................... 82
Table 26 - Changes to Mineral Resources from 2016 to 2017 ..................................... 83
Table 27 – Geotechnical Design Sectors for Salobo Mine .......................................... 88
Table 28 – 2017 Cutoff Calculation Parameters ........................................................... 89
Table 29 - December 31, 2017 Mineral Reserves ....................................................... 90
Table 30 - Summary of Mineral Reserves from 2016 to 2017 ..................................... 91
Table 31 - Changes to Mineral Reserves from 2016 to 2017 ....................................... 91
Table 32 - 2017 Mineral Reserves Estimate by Phase ................................................. 91
Table 33 - 2016/2017 Calculated Reconciliation Factors ............................................. 94
Table 34 - Mill versus Production ............................................................................... 96
Table 35 - December 31, 2017 Mineral Reserve Estimate by Phase ............................ 97
Table 36 - 5 Year Plan ................................................................................................. 98
Table 37 - Actual Production Mining Fleet .................................................................. 100
Table 38 - Waste Dump & Stockpile Design Parameters ............................................. 101
Table 39 - Major Process Equipment ......................................................................... 107
Table 40 - Processing Recovery Assumptions (2018-2022) ........................................ 108
Table 41 - Processing Recovery Assumptions (2023-LoMP) ........................................ 108
Table 42 - Processing Plant Performance Forecasts ..................................................... 109
Table 43 - Actual and Forecasted Processing Recoveries............................................. 109
Table 44 - Metal Sale Price and Exchange Rate Assumptions ..................................... 115
Table 45 - Historical Capital Expenditures (US$ M) ................................................... 121
Table 46 – Capital Development (US$ M) .................................................................. 121
Table 47 - Mine Operating Costs .............................................................................. 122
Table 48 - Process Operating Costs ........................................................................... 122
Table 49 - Other Costs ............................................................................................. 122
1. Summary

The Salobo Operations comprise a large open pit mine and concentrator facility located in the Carajás Mining District in northern Pará State, Brazil. The mine is 100% owned by Vale S.A.

This technical report (the Report) summarizes the supporting information for the December 31, 2017 Mineral Reserve and Mineral Resource estimates for the Salobo Operations and outlines any important changes since the 2016 Amec Foster Wheeler (AmecFW) technical report.

The Report has been prepared in compliance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43–101F1.

All estimates are supported by applicable scientific and technical information. Unless otherwise indicated, all financial values are reported in US currency and metric as the units of measure.

The Salobo deposit is hosted in the Carajás Mining District within Carajás Province, a sigmoidal-shaped, west–northwest to east–southeast-trending late Archean basin. The basin contains a basement assemblage that is dominated by granite–tonalitic orthogneisses of the Pium Complex, and amphibolite, gneisses and migmatites of the Xingu Complex. The basement rocks are overlain by volcanic and sedimentary rocks of the Itacaiúnas Supergroup, which includes the Igarapé Salobo Group, the Igarapé Pojuca Group, Grão Pará Group and the Igarapé Bahia Group.

The Itacaiúnas Supergroup hosts all the Carajás iron deposits as well as Salobo. Salobo is considered to be an example of an iron oxide–copper–gold (IOCG) deposit. Global examples include Olympic Dam in Australia, Candelaria–Punta del Cobre in Chile, and Sossego in Brazil.

The major host units are biotite and magnetite schists. The Salobo hydrothermal system has a core of massive magnetite that is surrounded by less intensely altered rocks. Away from the massive magnetite, the magnetite content gradually diminishes, giving way to biotite–garnet schist and / or garnet–grunerite schist. Sulphide mineralization typically consists of assemblages of magnetite–chalcopyrite–bornite and magnetite–bornite–chalcosite.

Copper mineralization was discovered by a Vale predecessor company in 1974 and detailed exploration commenced in 1977. Initial exploration efforts included stream sediment sampling, reconnaissance exploration, and ground induced polarization (IP) and magnetometer geophysical surveys. Follow-up work in 1978 identified the presence of copper sulphides in an outcrop of magnetite schists at Salobo. Core drilling commenced in 1978 and was conducted through to 2003 in five different drilling
campaigns. An infill drilling program was initiated in 2017, the first core drilling since 2003.

A scoping study was completed in 1981, and pilot studies ran from 1985 to 1987, culminating in the grant of a mining concession. A prefeasibility study was concluded in 1988, an initial feasibility study was conducted in 1998, updates to the feasibility study were undertaken in 2001 and 2002, and a final study was completed in 2004.

The Salobo Mine commenced pre-stripping in 2009. Project ramp-up for Phase I (1 Mtpa) of the Salobo Operations was completed three years later and the first concentrate was shipped in September 2012. Phase II, doubling the nameplate capacity, was completed in 2014.

Table 1 details the Salobo Operations Mineral Reserves and Mineral Resources as of December 31, 2017.

<table>
<thead>
<tr>
<th>Item</th>
<th>Classification</th>
<th>M Tonnes</th>
<th>Cu %</th>
<th>Au g/t</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Mineral Reserves</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven</td>
<td></td>
<td>526.3</td>
<td>0.68</td>
<td>0.38</td>
</tr>
<tr>
<td>Probable</td>
<td></td>
<td>549.3</td>
<td>0.57</td>
<td>0.29</td>
</tr>
<tr>
<td>Stockpiles (Proven)</td>
<td></td>
<td>117.8</td>
<td>0.44</td>
<td>0.19</td>
</tr>
<tr>
<td><strong>P&amp;P</strong></td>
<td></td>
<td>1,193.4</td>
<td>0.61</td>
<td>0.32</td>
</tr>
<tr>
<td><strong>Mineral Resources within pit</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Measured</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Indicated</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M&amp;I</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Inferred</td>
<td></td>
<td>33.3</td>
<td>0.50</td>
<td>0.24</td>
</tr>
<tr>
<td><strong>Mineral Resources adjacent to pit</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Measured</td>
<td></td>
<td>33.0</td>
<td>0.72</td>
<td>0.42</td>
</tr>
<tr>
<td>Indicated</td>
<td></td>
<td>171.1</td>
<td>0.62</td>
<td>0.31</td>
</tr>
<tr>
<td>M&amp;I</td>
<td></td>
<td>204.1</td>
<td>0.64</td>
<td>0.33</td>
</tr>
<tr>
<td>Inferred</td>
<td></td>
<td>142.4</td>
<td>0.56</td>
<td>0.29</td>
</tr>
<tr>
<td><strong>Total Mineral Resources</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Measured</td>
<td></td>
<td>33.0</td>
<td>0.72</td>
<td>0.42</td>
</tr>
<tr>
<td>Indicated</td>
<td></td>
<td>171.1</td>
<td>0.62</td>
<td>0.31</td>
</tr>
<tr>
<td>M&amp;I</td>
<td></td>
<td>204.1</td>
<td>0.64</td>
<td>0.33</td>
</tr>
<tr>
<td>Inferred</td>
<td></td>
<td>175.7</td>
<td>0.55</td>
<td>0.28</td>
</tr>
</tbody>
</table>

Notes:
1. Mineral Resource estimates were prepared by Mr. Joao Dirk V. Reuwsaat and Mineral Reserve estimates by Mr. Wellington F. de Paula, both Vale employees. The Qualified Person for the Mineral Resource and Mineral Reserve estimates is Mr. Cassio Diedrich, AusIMM-CP(Min), Technical Services General Manager, Vale Base Metals.
2. Mineral Resources are exclusive of Mineral Reserves
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
4. Mineral Reserves and Mineral Resources are reported above a copper equivalent cutoff of 0.253%, assuming $1,200 per ounce gold and $2.86 per pound copper

In the opinion of the QPs, the exploration and diamond drilling data was completed according to 2003 CIM Best Practice Guidelines and the Mineral Resources and Mineral Reserves have been estimated according to 2014 CIM Definition Standards. The Salobo Operations is a fully developed mine site with all the required permits and infrastructure. The mine has a large Mineral Reserve base and strong economic margins which result in forecasted mining until the year 2045 and then the processing of stockpiled material until 2067.
2. Introduction

Wheaton Precious Metals Corp (Wheaton) in collaboration with Vale SA, has elected to prepare an updated technical report (the Report), in compliance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43–101F1, on the Salobo open pit mining operations (Salobo Operations), located in northern Brazil, in the southeastern portion of Pará State.

The corporate entity that conducts the mining operations is Salobo Metais SA (SMSA), an indirectly wholly-owned subsidiary of Vale SA. For the purposes of this Report, unless otherwise noted, Vale SA and Salobo Metais SA will be referred to interchangeably as Vale.

Wheaton’s interest in the Salobo Operations is restricted to a metal streaming agreement that applies to 75% of the gold produced as a byproduct at the Salobo Operations for the life of the mine (the streaming agreement).

The Salobo Operations consist of an operating copper–gold open pit mine, currently producing at a rate of 24 Mt/a through a conventional crush–grind–float processing plant, producing copper concentrates.

2.1. Terms of Reference

The Report was prepared to support scientific and technical disclosure on the Salobo Operations in Wheaton’s Annual Information Form for the year ending 31 December, 2017.

Mineral resources and Mineral reserves are reported with reference to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2003; 2003 CIM Best Practice Guidelines).

All measurement units used in this Report are metric units and currency is expressed in US dollars (US$), unless stated otherwise. The Brazilian currency is the real (BRL R$ or R$). The Report uses Canadian English.

2.2. Qualified Persons

The following Qualified Persons (QPs), as defined in NI 43-101 authored this Report:

Wheaton:

- Neil Burns, P.Geo., Vice President, Technical Services, Wheaton
- Maurice Tagami, P.Eng., Vice President, Mining Operations, Wheaton
2.3. Site Visits

Mr. Neil Burns and Mr. Maurice Tagami make annual visits to the Salobo Operations to review developments. Mr. Burns and Mr. Tagami have been to site numerous times since 2013 when the first Wheaton streaming deal was completed, most recently in December, 2017.

Since 2013, Mr. Chris Davis has made at a minimum, annual visits to the Salobo Operations to review all matters pertaining to Mineral Resources and Mineral Reserves, exploration and mining. Mr. Davis’ most recent site visit was in May, 2017.

Mr. Cassio Diedrich has actively participated in the technical reviews/evaluations of the Salobo Project and Operations since 2008 relative to Mineral Resources and Mineral Reserves, exploration and mining subjects and visits the Salobo Operations regularly.

2.4. Previous Technical Reports

Wheaton has previously filed the following reports on the Salobo Operations:


3. Reliance on Experts

The QPs have relied upon the following Vale technical staff, which provided information regarding mineral rights, surface rights, property agreements, royalties, taxation and marketing sections of this Report:

- Wellington F. de Paula, Responsible Person, Long Term Mining Engineer
- João Dirk Reuwsaat, Responsible Person, Long Term Geostatistician
- Marcio Medeiros, Manager of Mining Legal Counsel
4. Property Description and Location

4.1. Location

The Salobo operation is located along the southern margin of the Amazon Basin, northern central Brazil, in the southeastern part of the State of Pará (Figure 1). It is also located in the Parauapebas micro-region in the municipality of Marabá and is part of the Carajás Mineral Province. Geographic coordinates for the operation are 5°47’25” S latitude and 50°32’5” W longitude.

In addition to major iron mines, the Carajás Mineral Province also hosts manganese deposits, low-cost gold mines, copper sulphide and nickel laterite resources. The Carajás area has excellent infrastructure including the all-weather commercial airport at Carajás, which operates with 737 Boeing type aircraft, electrical power derived from the Tucuruí Dam, abundant water, good roads, and social institutions.

The Salobo Operations is a copper-gold deposit located approximately 80 km northwest of Carajás, Pará State in northern Brazil. The area is well-served by railroads and highways that connect the villages and cities. Air service is available at the Carajás airport, which is approximately 70 km from Salobo and is capable of receiving commercial aircrafts and it is served by two daily flights to Belém (Pará state major’s city) and to the main Brazilian cities. Marabá is approximately 240 km from Salobo Operations by highway.
4.2. **Regulatory**

During the course of Vale Base Metals operations at Salobo, Vale is subject to routine claims and litigation incidental to Vale’s business as well as various environmental proceedings. For greater certainty, none of these ongoing legal issues are considered to pose any “moderate”, “major” or “catastrophic” legal risk to the company’s ability to exploit the Mineral Reserves /Mineral Resources reported in this Report. The operation has secured all material licenses and permits; the company has secured all requisite mineral rights and surface rights; there are no material issues of non-compliance that may impact the company’s ability to exploit the resource; and there is no litigation that may impact the company’s ability to exploit the resource.

Vale holds clear mineral title to the deposit areas and has all the necessary permits for operation of the mine.

4.3. **Mineral Tenure**

The Salobo Operations tenement title is 100% owned by Vale S.A. The Salobo Operations are located on one claim. The area named Salobo (copper ore, DNPM 807.426/74) refers to Exploration Permit no. 1121 that is dated July 14, 1987, and defined as a polygon of 9,180.61 ha (Figure 2). There was no change to the land tenement status in 2017.
Brazilian legislation separates the ownership of the surface rights from mineral ownership. A mining company can operate a mine even if does not own the surface, provided it owns the minerals. In this case it is necessary to pay a royalty to the surface owner. The royalty is calculated as 50% of the CFEM (Compensation for Financial Exploitation of Mineral Resources), which is paid to the government. The mining concessions are updated every year on presentation by Vale of the annual report of mining production to the DNPM.

Figure 2 - Mineral Tenure Layout Plan

Note: Figure courtesy Vale, 2015.

4.4. Royalties/Mining Taxes

The Compensação Financeira pela Exploração de Recursos Minerais (CFEM) was enacted by legislation in 1989 and is based on a percentage of the holder’s net profit. The value of CFEM varies from 0.2 to 3.0% of the net sales of mineral products:

- 3.0%: aluminum ore, manganese, rock salt and potassium
- 2.0%: iron ore, fertilizer, coal and other substances
- 0.2%: precious stones, colored gemstones, carbonates and noble metals
• 1.0%: gold.

The majority of minerals incur the 2.0% royalty.

Of the amount collected, 65% is paid to the municipalities where production is to take place, 23% is paid to the host state, and 12% to the Federal government.

Since 2013, Wheaton has entered into the following three different life of mine gold stream agreements on Salobo with Vale, each for 25%, for a total of 75%. In each of the agreements Wheaton agreed to ongoing payments of the lesser of $400 (subject to a 1% annual inflation adjustment now commencing in 2019 on the entire 75% stream) and the prevailing market price for each ounce of gold delivered under the agreement.

• February 2013 - Vale entered into an agreement with Wheaton to sell 25% of the gold produced at the Salobo Operations for the life of the mine. In exchange, Vale received an initial cash payment of $1.33 billion and 10 million share purchase warrants exercisable at a strike price of $65 per common share.

• March 2015 - Wheaton acquired an additional 25% of the gold production, increasing the gold stream to 50%. Under the amended 2015 streaming agreement, Wheaton paid Vale a cash consideration of $900 million for the new gold stream.

• August 2016 – Wheaton agreed to acquire an additional 25% of the life of mine gold production from the Salobo Operations. This acquisition was in addition to the 50% of the Salobo gold production that Wheaton was entitled to. Wheaton paid upfront cash consideration of $800 million for the increased gold stream and the 10 million Wheaton common share purchase warrants previously issued to a subsidiary of Vale were amended to reduce the strike price from US$65.00 to US$43.75 per common share.

4.5. Social License

Areas reserved for indigenous populations are designated as “restricted access” or “prohibited” access for mining. The Brazilian Constitution requires that any mining activities in indigenous areas requires prior approval of the Brazilian National Congress. Indigenous communities have the right to receive royalties from any mining in their areas.

In addition to the indigenous communities, there are other communities (Quilombolas) that have Constitutional rights to own and occupy specific lands. Mining is permitted in these areas; however, the communities are entitled to compensation, and if the community needs to be relocated for mining purposes, the community must be relocated to land that has similar characteristics to the area that was previously occupied or be fairly compensated.
4.6. **Comments on Section 4**

In the opinion of the QPs, the information discussed in this section supports the declaration of Mineral Resources and Mineral Reserves, based on the following:

- Vale’s mining tenure held is valid and is sufficient to support estimation of Mineral Resources and Mineral Reserves.
- Vale holds sufficient surface rights in the Project area to support the mining operations envisaged in the life-of-mine plans, including access and power line easements.
- Vale currently holds the appropriate permits under local, Provincial and Federal laws to allow mining operations (refer to Section 20). Some permits will require renewal over the course of the planned life-of-mine.
- The appropriate environmental permits have been granted (refer to Section 20).
- At the effective date of this Report, environmental liabilities are typical of an operating open pit mining area (refer to Section 20).
- Vale is not aware of any significant environmental, social or permitting issues that would prevent continued exploitation other than those discussed in the Report.
- There is no active artisanal mining on or near the property.
- To the extent known, there are no other significant factors and risks known to Vale that may affect access, title, or the right or ability to perform work at Salobo Operations.

5. **Accessibility, Climate, Local Resources, Infrastructure and Physiography**

5.1. **Accessibility**

Mining is the primary industry in the area. The Salobo Operations are connected via an all-weather road network to the cities of Parauapebas (80 km), Marabá (240 km), and the commercial airport at Carajás. The Carajás airport can accommodate large aircraft and is served by daily flights to Belém (Pará State major’s city) and other major Brazilian cities.

Rail lines carry Salobo and Sossego copper concentrate and iron ore from Carajás to the port city of São Luís.

The mine concentrate is hauled by trucks to a rail-loading site in north of Parauapebas. Concentrate is then transported, approximately 870 km, by train to the port of São Luís.

5.2. **Climate**

The operations are located in the Carajás mountain range in the eastern Amazon humid tropical rainforest. Temperatures range from 20.8°C to 37.8°C with an average relative
humidity of 80.5%. Mean annual rainfall is 1,920 mm and evaporation is 1,500 mm. Winds are predominantly from the north and west.

Mining operations are conducted year-round.

5.3. Local Resources and Infrastructure

Mining is the primary industry of the area. As well as Salobo, Vale also operates the Sossego copper mine, located 136 km by road to the south of Salobo and the Carajás iron ore mine.

Local housing is available for employees within the communities surrounding the mine. There are adequate schools, medical services and businesses to support the work force. The mine site has medical facilities to handle emergencies. In addition, medical facilities are available in Carajás to support the mine’s needs.

Vale has invested significantly in infrastructure at Carajás, building a 130 km paved road to Parauapebas and a 20 km sewage system, together with a school, hospital, and day care center.

Project infrastructure and the infrastructure layout are discussed in detail in Section 18 of the Report.

5.4. Physiography

Salobo is in the northwest of the Carajás Reserve within the 190,000 ha Flona de Tapirapé–Aquiri forest. The area is heavily forested, and dominated by relative dense trees with substantial underbrush.

In the mine area, the topography is fairly steep, varying between 190 to 520 m in elevation. The ridge where the Salobo deposit is located has a nominal slope of 2.5H:1.0V. The site is lower than the Carajás Ridge, which is 850 m above sea level.

The two drainages on either side of the Salobo Ridge are the Cinzento and Salobo Igarapés (small rivers) which flow into the Itacaiúnas River. The Itacaiúnas River flows into the Tocantins River close to Marabá City. The long-term average unit runoff for the Project site is 13.5 L/s/km².

5.5. Comments on Section 5

In the opinion of the QPs:

- All necessary infrastructure has been built on site, is operational, and is sufficient for the projected LoMP (see also Section 18).
There is sufficient suitable land available within the mineral tenure held by Vale for tailings disposal, mine waste disposal, and installations such as the process plant and related mine infrastructure (see also Section 18).

6. History

- 1974 - CVRD (Companhia Vale do Rio Doce, a predecessor company to Vale) discovered copper mineralization in the Igarapé Salobo region, and commenced detailed exploration in 1977. Work completed included stream sediment sampling, reconnaissance exploration, and ground induced polarization (IP) and magnetometer geophysical surveys. As a result, various targets were identified.

- 1978 - The 1974 Salobo exploration targets were revisited and the presence of copper sulphides in an outcrop of magnetite schists at the Salobo 3 Alfa target was noted. Drilling of this target followed in conjunction with the development of two exploration adits. The Salobo 3 Alfa target is now referred to as Salobo.

- 1978 to 1983 - Drilling was initially conducted on a 400 m by 200 m drill grid, subsequently reduced to 200 m by 200 m, and then to 200 m by 100 m. A total of 65 core drill holes (29,322 m) were drilled between March 1978 and May 1983.

- 1981 - A preliminary assessment of potential Project economics was performed in 1981, based on an initial resource estimate. The findings were encouraging, and the Carajás Copper Project team submitted an Exploitation Economical Plan for the Salobo deposit to the DNPM in June 1981.

- 1985 – 1987 - A pilot-scale study was carried out from 1985 to 1987 to further define the mineralization style and geometry. This included additional drilling and an additional 1 km of exploration adits. A second drill campaign ran from January 1986 to June 1987. The grid spacing in the core of the deposit was reduced to 100 m by 100 m. Additional drilling was undertaken in the southeast of the deposit from the G-3 adit. This phase included 9,033 m of diamond drilling from 60 drill holes.

- 1987 - The MME granted CVRD mining rights through Ordinance No. 1121.

- 1988 - A prefeasibility study was completed by Bechtel.

- 1993 - Salobo Metais S.A. was incorporated on 29 June 1993 as a joint-venture vehicle between CVRD and Morro Velho Mining (a subsidiary of Anglo American Brasil Ltda. AABL). A third drill campaign was initiated. The primary objective was to investigate the best probable location in the deposit in which to commence mining and to optimize the first five years of production, as well as to investigate mineralized continuity at depth.

- 1993 to 1994 - A total of 64 drill holes (14,585 m) were completed.
1997 - A fourth drilling campaign was conducted, resulting in 25,491 m in 88 holes. Mineral Resources Development Inc. (MRDI) audited the drilling information that year.

1998 - A feasibility study was undertaken by Minorco.

2001 – The feasibility study was revised and updated by Kvaerner in 2001.

2002 - AMEC audited the drilling, sampling, assaying and databases that supported the Kvaerner study.

- Changes were made to the Exploitation Economic Plan allowing Salobo Metais to extract silver and gold were approved by DNPM. The original authorization had been for copper only.

- In June 2002, the Brazilian Council for Economic Defense (Conselho Administrativo de Defesa Econômica) approved the acquisition by CVRD of the 50% of Salobo Metais that was held by AABL. CVRD thus became the largest shareholder in Salobo Metais.

- A fifth drilling campaign drilled 133 drill holes (66,243 m)

2003 - A further 2,047 m of drilling was completed and some areas were drilled at a closer spacing of 50 m x 50 m, including the area around the G3 adit.

2006 – Final Pre-Feasibility Study and Installation Licence Granting.

2007 – Final Feasibility Study and construction start-up of Salobo I (12Mt/a).

2009 - Commenced pre-stripping.

2010 – Construction start-up of Salobo II (24Mt/a).

2012 - Project ramp-up for Phase I of the Salobo Operations was completed and the first concentrate was shipped in September 2012.

2013 – The first Wheaton streaming deal was completed for 25% of the life of mine gold production

- December 2013, the plant processed 898,000 t of ore, which represented 90% of the Phase I nameplate capacity (1 Mt run-of-mine (ROM) per month).

2014 - Phase II, intended to double the nameplate capacity and was completed.
- 2015 – The second Wheaton stream deal completed for an additional 25% of the life of mine gold production, increasing the total stream to 50%.

- 2016 – The third Wheaton stream deal completed for an additional 25% of the life of mine gold production, increasing the total stream to 75%.

- 2017 - During 2017, the following important changes occurred at the Salobo Operations:
  
  o The production data reconciliation process was revised and updated.
  
  o A medium range definition diamond drilling campaign was started.
  
  o A deep exploration drill hole was started to investigate the orebody below the final pit design.
  
  o The mine and plant quality control (sampling, etc.) process was externally audited.
  
  o A short-term deleterious estimation process for carbon, uranium, fluorine, sulphur and chlorine was started.
  
  o The phases/pushback design were modified together with the mining plan revision, changing from seven to eight phases.

6.1. Production History

Production since mine start-up in 2012 is summarized in the following table:

<table>
<thead>
<tr>
<th>Year</th>
<th>Feed</th>
<th>Concentrate</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnage (kt)</td>
<td>Cu (%)</td>
</tr>
<tr>
<td>2012</td>
<td>1,823</td>
<td>1.13</td>
</tr>
<tr>
<td>2013</td>
<td>7,366</td>
<td>1.09</td>
</tr>
<tr>
<td>2014</td>
<td>12,474</td>
<td>0.97</td>
</tr>
<tr>
<td>2015</td>
<td>20,290</td>
<td>0.88</td>
</tr>
<tr>
<td>2016</td>
<td>21,401</td>
<td>0.94</td>
</tr>
<tr>
<td>2017</td>
<td>23,650</td>
<td>0.95</td>
</tr>
</tbody>
</table>
7. Geological Setting and Mineralization

7.1. Regional Geology

The Carajás Mining District, located in the southeast of Pará State, lies between the Xingu and Tocantins/Araguaia Rivers, and covers an area of about 300 km x 100 km. It is hosted in the Carajás Province, forming a sigmoidal-shaped, west–northwest to east–southeast-trending late Archean basin (Figure 3).

The Archean basin contains a basement assemblage that is dominated by granite–tonalitic ortho-gneisses of the Pium Complex, and amphibolite, gneisses and migmatites of the Xingu Complex. The basement assemblage defines a broad, steeply dipping, east–west trending ductile shear zone (Itacaiúnas shear zone) that experienced multiple episodes of reactivation during the Archean and Paleoproterozoic.

The metamorphic rocks are cut by Archean-age intrusions, including the calc-alkaline Plaquê Suite (2.73 Ga), and the alkaline Salobo and Estrela granites (2.57 Ga and 2.76 Ga respectively).

The basement rocks are overlain by volcanic and sedimentary rocks of the Itacaiúnas Supergroup (2.56 Ga to 2.77 Ga). The Itacaiúnas Supergroup is informally sub-divided as follows (oldest to youngest):

- The Igarapé Salobo Group: iron-rich sediments, quartzites and gneisses, metamorphosed to amphibolite facies; associated with copper–gold and copper–gold–silver mineralization, e.g. Salobo.

- Igarapé Pojuca Group: basic to intermediate volcanic rocks (frequently with cordierite–anthophyllite alteration), amphibolites, gneisses and chemical sediments (cherts), banded iron formation (BIF), and chert; associated with copper–zinc deposits, e.g. Pojuca.

- Grão Pará Group: basal Parauapebas Formation, comprising bimodal volcanic rocks with various degrees of hydrothermal alteration, metamorphism and deformation; upper Carajás Formation, associated with various iron deposits, including the Carajás deposits.

- Igarapé Bahia Group: mafic volcanics (lavas, tuffs and breccias), meta-sediments and BIF, associated with copper, copper–iron, copper–gold–silver deposits, e.g. Igarapé Bahia, Alemão/Bahia and Serra Pelada.

The Itacaiúnas Supergroup hosts all the Carajás iron ore–copper–gold (IOCG) deposits, including Salobo and Sossego, and is thought to have been deposited in a marine rift environment. The metamorphism and deformation has been attributed to the development of a sinistral strike-slip ductile shear zone (the 2.7 Ga Itacaiúnas Shear Zone) and to sinistral, ductile–brittle to brittle transcurrent fault systems.
The Itacaiúnas Supergroup is overlain by an extensive succession of Archean marine to fluvial sandstones and siltstones known as the Rio Fresco Group or the Águas Claras Formation (2.68 Ga to 2.78 Ga). The non-deformed, Proterozoic Gorotire Formation, consisting of coarse arkoses and conglomerates with quartz, BIF, and basic rock clasts, overlies the older lithological units (Matos da Costa, 2012).

A Proterozoic suite (1.88 Ga) of anorogenic, alkaline granites, the Serra dos Carajás, the Cigano and the Pojuca granites, as well as several generations of younger mafic dykes, cross-cut the entire sequence.

**Figure 3 - Regional Geology of the Carajás Province**

Note: Figure modified by AmecFW after SMSA/CVRD, 2003a. The Salobo and Sossego operations (indicated in red) are held by Vale, as are the 118 and Cristalino prospects/deposits.

**7.2. Property Geology**

Mineralization at the Salobo deposit is hosted by upper greenschist to lower amphibolite metamorphosed rocks of the Igarapé Salobo Group. The group thickness varies from 300–600 m in the Project area, and may be weathered to depths of 30–100 m. The rocks strike approximately N70°W and have a subvertical dip.

The major host units are biotite (BDX) and magnetite schists (XMT). Granitic intrusions (GR) occur adjacent to the north and southern sides of the BDX and XMT, and a series of much younger diorite dykes (DB) cross-cut the mineralization forming barren zones.
Lithological descriptions of the major units are as follows and as shown in the plan and section views in Figure 4 and Figure 5 respectively and core photos in Figure 6:

**Magnetite Schist (XMT)**

XMT is represented by massive, foliated and banded rocks, with predominant magnetite, fayalite, grunerite, almandine and secondary biotite. Granoblastic textures with polygonal contacts in magnetite and fayalite are common. The presence of fayalite is marked by the replacement of grunerite and greenalite and transformation into magnetite and other sulphides. Iron-potassic alteration is common, creating schistosity in biotite units.

The southeast portion of the deposit hosts hastingsite, replaced partially by actinolite, grunerite and sulphide minerals. Fluorite, apatite, graphite and uranium oxides are associated with this assemblage, Fe-silicate minerals and alteration products of fayalite.

**Garnet-Grunerite Schist (DGRX)**

These are massive rocks with local development of schistosity. The rocks with significant almandine and grunerite content have isotropic texture or very few schistosity structures, with nematoblastic and granoblastic texture. The main mineralogical composition consists of almandine and cummingtonite-grunerite, with magnetite, hematite, ilmenite, biotite, quartz, chlorite, tourmaline and subordinate allanite. Fluorite and uraninite generally occur in veinlets related to stilpnomelane, calcite and grunerite.

**Biotite Schist (BDX)**

This unit is the most common lithology at Salobo and consists of medium to coarse-grained material with anastomosed foliation. The mineral assembly is characterized by biotite (responsible for the foliation observed within the rocks), garnet, quartz, magnetite and chlorite. The assemblage with garnet, magnetite, grunerite and biotite is partially replaced by a second generation of biotite and magnetite with chlorite, K feldspar, quartz, hematite and sulphides. Tourmaline, apatite, allanite, graphite and fluorite generally occur throughout this unit.

**Feldspar-Chlorite Mylonite (ML)**

The feldspar-chlorite-quartz mylonite is characterized by mylonitic foliation, produced by the orientation of rims of chloritized deformed biotite, hastingsite, elongated quartz and saussuritized plagioclase (K-feldspar, epidote and muscovite alteration). Porphyroblastic garnet is partially or totally replaced by chlorite and epidote. Allanite and apatite generally occur throughout this lithology.
Metavolcanic Basic (MTB)

This group of massive coarse-grained rocks is characterized by Fe-hastingsite and/or hornblende and plagioclase with chlorite alteration. It occurs irregularly in the system, but is concordant with other lithotypes in abrupt contacts, probably hydrothermally altered intrusive basic relicts within the package of volcanic rocks.

Quartz Mylonites (QML)

Quartz mylonite is grey or white in colour, passing through green to red. Where present, Fe-oxides are medium to fine grained, foliated and composed predominantly of quartz, muscovite, sericite, sillimanite and chlorite. Accessories, such as biotite, feldspar, magnetite, almandine, tourmaline, zircon and allanite are common. It is possible to differentiate: (a) red quartz-feldspathic rocks formed by K-feldspar and quartz and which may be a product of shearing between the gneissic basement and the supracrustal rocks; and (b) chlorite schists, mainly composed of chlorite and quartz, that represent intense hydrothermal alteration. This unit is found near the southern border of the deposits, close to important brittle shear zones, which may be interpreted as conduits for hydrothermal fluids.

Old Salobo Granite (GR)

The Old Salobo Granite occurs as a stockwork of approximately 2,573 ±2 Ma. The rocks appear colorless-pink to grey, coarse grained and with mylonitization in some areas. The main mineralogy is composed of K-feldspar (orthoclase-microcline), oligoclase, quartz, augite, hornblende, chlorite and, rarely, magnetite. There is no evidence of contact metamorphism with the host rocks. The mylonitic aspects that appear both in granite and host rocks are likely to have formed during the deformation phase.

Young Salobo Granite (GR)

The Young Salobo Granite occurs as small northwest-trending sills, hosted by the supracrustal sequence and by the gneisses of basement. It corresponds to the youngest granitic intrusion detected by drilling in the Salobo area. In some porphyritic portions, the matrix is aphanitic, containing a porphyry of red albite (Fe-oxide in micro-fractures) and chlorite pseudomorphed by biotite. This mineral assemblage is composed of fine to medium grained, equigranular, hypidiomorphic grains of albite/oligoclase, orthoclase, quartz, chlorite, with minor epidote, zircon, fluorite, magnetite, chalcopyrite and pyrite. Deformation was not observed, and the structure is isotropic. Age dating indicates an age of 1,880 ±80 Ma.

Diabase (DB)

Diabase is located in southeast of the deposit, striking at approximately N70°E, while in the northwest of the deposit striking near to N20°W. The predominant
minerals comprising the rock type are augite, plagioclase, magnetite, ilmenite and quartz. The fine-grained diabase has an age of 553 ±32 Ma, while the more granular margins are dated at 561 ±16 Ma. This unit represent the last magmatic event of the area. The dykes are set within shear/fault lateral geometries to (N70°E) and frontal geometries (N20°W), probably developed before the intrusions, in a compressional regime modified by an extensive regime.

Rhyolite (RIO)

Rhyolite dykes are grey-reddish in colour, porphyritic in texture within an aphanitic matrix. The majority are composed of K-feldspars, plagioclase, quartz, amphibole in a matrix cut by quartz veinlets. In drill holes the occurrence is rare or an ultimate phase.
7.3. Tectonic Setting

As depicted in Figure 7, the Salobo deposit is situated within the Cinzento strike-slip system which has been described as a set of Archean alignments that forms the Salobo transpressive duplex (or Salobo sidewall rip-out). This system post-dates the formation of the Itacaiúnas shear zone and was developed under ductile–brittle to brittle conditions.

The tectonic evolution of the Salobo area includes sinistral, transpressive, ductile deformation that developed under upper-amphibolite-facies conditions, followed by sinistral, transtensive, ductile–brittle-to-brittle shear deformation.

Shear zones are characterized by a mylonitic, penetrative foliation that generates a compositional banding. Where deformation is more intense, S-C foliations are parallel, and a lenticular pattern develops.

The ductile deformation along the Itacaiúnas shear zone, which has affected the basement rocks and rocks of the Salobo Group, produced widespread, subvertical, northwest–southeast schistosity, which affects all lithologies in the deposit, except the Young Salobo Granite and the diabase dykes.

The transtensive deformation along the Cinzento strike-slip fault system reactivated old structures and formed a subparallel ductile–brittle shear zone in the northern part of the deposit and a brittle shear zone in the south.

Brittle–ductile shear zone deformation has resulted in lenticular-shaped ore shoots that characteristically show close associations between copper mineralization and magnetite content.
7.4. Metamorphism

Two phases of metamorphism have been recognized in the Salobo area:

- Initial phase: associated with progressive amphibolite-facies metamorphism developed under ductile conditions of high temperature (650˚C), low pressure (2–3 kbar), and oxygen fugacities of -20 and -18. This caused partial substitution of chalcopyrite by bornite and chalcocite, accompanied by intense K-metasomatism.

- Retrograde phase: developed under greenschist facies, with an average temperature of 340˚C; characterized by intense chloritization and partial substitution of bornite by chalcocite.

7.5. Alteration

The Salobo hydrothermal system has a core of massive magnetite that is surrounded by less intensely altered rocks. Within the massive magnetite body there are small veins and irregular masses of secondary biotite. Garnet is completely replaced by magnetite, forming pseudomorphs. Away from the massive magnetite, the magnetite content gradually diminishes, giving way to biotite–garnet schist and/or garnet–grunerite schist. Alkali-metasomatism of the amphibolite facies rocks is expressed by weak sodium with intense, superimposed potassium alteration (≤4.6 wt% of K₂O).

K-feldspar, biotite and oligoclase are the main alteration minerals. A significant increase in the FeO content (≤35 wt%) accompanied the potassium alteration in amphibolite, and was marked by the replacement of calcium-amphibole (mostly magnesium-hornblende and hastingsite) by iron–magnesium amphibole (cummingtonite), and by formation of biotite and magnetite.

The chemistry of the meta-greywackes at the deposit indicates that they also underwent significant iron and potassium alteration. Alteration assemblages are characterized by almandine, garnet, biotite and grunerite, subordinate tourmaline and minor magnetite. The better-mineralized zones, located in the central part of the deposit, correspond to the most altered areas.

7.6. Mineralization

The Salobo deposit extends over an area of approximately 4 km along strike (west–northwest), is 100–600 m wide, and has been recognized to depths of 750 m below the surface.

The sulphide mineralization typically consists of assemblages of magnetite–chalcopyrite–bornite and magnetite–bornite–chalcocite. Accessory minerals include hematite, molybdenite, ilmenite, uraninite, graphite, digenite, covellite, and sulphosalts.
The mineral assemblages can be found in a number of styles: forming disseminations, stringers, stockworks, massive accumulations, filling fractures, or in veins associated with local concentrations of magnetite and/or garnet filling the cleavages of amphiboles and platy minerals, and remobilized in shear zones (Figure 8).

There is a positive relationship between copper minerals and magnetite. Copper content is typically >0.8% in XMT and BIF, whereas in gneisses and schists it is <0.8%. A positive correlation between copper content and uranium contents has also been established.

Chalcopyrite, bornite, and chalcocite occur interstitially to silicate minerals. These sulphide minerals are commonly found filling cleavage planes of biotite and grunerite. Hematite is rare, but in places it can reach as much as 4% by volume. It exhibits tabular textures (specularite), with infilling bornite, and partial replacement by magnetite.

Native gold occurs as grains smaller than 10 μm in cobaltite, safflorite, magnetite and copper sulphides, or interstitial to magnetite and chalcopyrite grains. Native gold grains contain up to 10 wt% Cu, with subordinate silver, arsenic, and iron.

Molybdenite occurs interstitial to magnetite and shows cleavage planes filled with chalcopyrite and bornite. In mylonitic samples, molybdenite forms kinked stringers.

Magnetite occurs mainly as idiomorphic to sub-idiomorphic grains, interstitial to silicate minerals or in fractures, or forms bands in mylonitic rocks.

The gangue minerals are almandine garnet, grunerite, and tourmaline, reflecting the intense iron-metasomatism. Minor amounts of fayalite and hastingsite are pseudomorphed by grunerite and magnetite. Tourmaline, with a dominant schörlitic (black-tourmaline) composition, occurs as idiomorphic crystals preferentially oriented parallel to mylonitic foliation, in association with biotite, garnet and grunerite. Ilmenite, uraninite, allanite, fluorite and apatite occur as accessory minerals.

Biotite sub-idiomorphic crystals, commonly kinked, are associated with potassic alteration, and spatially related to the copper–gold mineralization. Uraninite and zircon inclusions may be locally abundant in biotite.

Quartz is associated with biotite in ore-grade samples, and forms concordant veins within the host rocks.

Textural relationships indicate that mineralization was developed firstly as an oxide stage, with a second, subsequent, sulphide stage.
7.7. Comments on Section 7

In the opinion of the QPs, the knowledge of the deposit settings, lithologies, mineralization style and setting, ore controls, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

8. Deposit Types

There appear to be two classes of copper–gold deposits in the Carajás region. The first group includes Cu–Au–(W–Bi–Sn) deposits which contain quartz veins, and may or may not have associated iron oxides and are genetically related to the cooling of Palaeoproterozoic (ca. 1.88 Ga) granites. The second group includes iron oxide Cu–Au (±U–rare earth elements) deposits (e.g., Salobo, Sossego, Cristalino, 118 and Igarapé Bahia) that may be related to more alkaline rocks, including the ca. 2.57 Ga alkaline complexes of the Carajás belt (e.g., Estrela Complex, Old Salobo Granite) and the base metal mineralization-associated 1.88 Ga intrusives. The second group of deposits are commonly referred to as iron oxide copper gold deposits (IOCG). Global examples of IOCG deposits include Olympic Dam in Australia, Candelaria–Punta del Cobre in Chile, and Sossego in Brazil.
9. Exploration

The discovery of the Salobo copper deposit occurred during a systematic program of geochemical, geophysical and geological exploration in the Carajás region, initiated by CVRD/Docegeo in 1974. Since then, the area has been the subject of exploration and development activities and a considerable information database has developed as result of both exploration and mining activities. Table 3 summarizes the exploration activities from 1978 to 2003.

Table 3 - Exploration Summary 1978–2003

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Surveying</td>
<td>Area (ha)</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>3,091</td>
</tr>
<tr>
<td></td>
<td>Lines (km)</td>
<td>258.2</td>
<td>21.3</td>
<td>-</td>
<td>52.3</td>
<td>-</td>
<td>3,091</td>
</tr>
<tr>
<td>Geochemistry</td>
<td>Lines (km)</td>
<td>3,230</td>
<td>138</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>3,368</td>
</tr>
<tr>
<td></td>
<td>Samples</td>
<td>3,433</td>
<td>2,616</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>6,049</td>
</tr>
<tr>
<td>Geophysics</td>
<td>I.P. (km)</td>
<td>26.3</td>
<td>165.9</td>
<td>-</td>
<td>-</td>
<td>180.6</td>
<td>180.6</td>
</tr>
<tr>
<td></td>
<td>Magnetometry (km)</td>
<td>76</td>
<td>171.6</td>
<td>-</td>
<td>43.5</td>
<td>212.7</td>
<td>256.2</td>
</tr>
<tr>
<td></td>
<td>Scintillometry (km)</td>
<td>52.3</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>52.3</td>
</tr>
<tr>
<td></td>
<td>TEM (loops)</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>26.3</td>
<td>214.5</td>
<td>240.8</td>
</tr>
<tr>
<td></td>
<td>Gamma-spectrometry (km)</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>750</td>
<td>750</td>
</tr>
<tr>
<td>Shafts</td>
<td>Amount</td>
<td>18</td>
<td>23</td>
<td>6</td>
<td>-</td>
<td>-</td>
<td>47</td>
</tr>
<tr>
<td></td>
<td>Length (m)</td>
<td>54</td>
<td>377.2</td>
<td>93.5</td>
<td>-</td>
<td>-</td>
<td>524.7</td>
</tr>
<tr>
<td>Adits</td>
<td>Amount</td>
<td>2</td>
<td>1</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Length (km)</td>
<td>450</td>
<td>950</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1400</td>
</tr>
<tr>
<td>Mapping</td>
<td>Lines (km)</td>
<td>427.6</td>
<td>221.3</td>
<td>1.8</td>
<td>43.5</td>
<td>-</td>
<td>694.2</td>
</tr>
</tbody>
</table>

No exploration occurred at Salobo between 2003 and 2011. In 2012, a regional airborne gravity survey was completed. The survey identified a potential continuation of the Salobo orebody at depth. In 2017, a deep drilling campaign was initiated exploring the deep extension and potential for underground mining. At the time of the Report the first hole was still in progress and no results were available.

9.1. Geological Mapping

Geological mapping at different scales was conducted over the Salobo during the initial campaigns, usually following survey traverses. However, due to the fact that nearly 80% of the rocks in the Carajás regions are poorly exposed, most direct observations were made along access roads for drill sites and were complemented with additional information such as interpretation of air-photo images, geophysical and geochemical maps, and correlation on surface of core logging data.

Current pit mapping is conducted twice a month. A geologist loads the long-term geologic map over the updated topographic map and establishes the actual position of the geological contacts in such points where access is possible.

9.2. Airborne Gravity Survey

A regional airborne gravity gradiometer survey was completed in 2012 over a portion of the Carajás Region, including the Salobo Operations area. It was designed to explore for
new shallow copper–gold targets. The survey flight is typically at an altitude of 80 m or greater with a line spacing dependent on the target of investigation. The 2012 survey was flown on 100 and 200 m spaced lines.

The aero-gravity gradiometer method measures acceleration of gravity and gradients of the acceleration of gravity respectively. It is not a precise tool and is not robust below 500 m depth.

There are a number of viable interpretations of the gravity data, one of which is that there could be a vertical extension of the Salobo mineralization below the current planned open pit, as the geological data have already indicated. In 2017, a deep drilling campaign was initiated exploring this potential orebody extension. At the time of this report, the first hole was still in progress and no results were available.

Figure 9 shows the gravity gradiometer survey results in relation to the magnetic survey in the Salobo Operations area. Figure 10 shows the gravity model, which has the 2015 block model outline at a 0.5% Cu cutoff, superimposed.

Note: Figure courtesy Vale, 2015. White outline is the current extend of the Salobo mineralization, projected to surface.
9.3. Comments on Section 9

In the opinion of the QPs, the Salobo deposit has been explored using appropriate techniques. Geophysical surveys recently completed by the Exploration department at the Salobo Operations have identified a significant gravity anomaly below the current Salobo open pit. Drilling is required to determine what the exploration potential at depth and help to identify additional Mineral Resources for supporting future potential operations and projects expansions.

10. Drilling

Diamond drill hole core is the majority sample type for geological modelling and mineral resource estimation at Salobo. Blast holes have been drilled since 2009 but are used only for grade control and short-term planning.

Core drilling commenced in 1978 and was conducted through to 2003 in five different drilling campaigns, for a total of 416 holes (146,674 m) completed for exploration purposes, and an additional 14 drill holes (7,590 m) for geotechnical purposes (Table 4). Most drill holes were vertical or oriented to the south–southwest, the latter with dips usually ranging from 60° to 70°. However, one campaign included holes with a north–
northwest orientation and similar dips. Various holes were also drilled from an adit. No exploration core drilling occurred between 2003 and 2016. In 2017, an infill drill program was initiated at the mine, however the results of this drilling were not available at the time of the Report. The following table summarizes the drilling campaigns completed at the Salobo Operations and Figure 11 is plan view map of the drill hole traces.

<table>
<thead>
<tr>
<th>Campaign/Period</th>
<th>Purpose</th>
<th>Drill Hole ID</th>
<th>Total Meterage Drilled (m)</th>
<th>Percentage of total drilling (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1978</td>
<td>Exploration</td>
<td>SAL-2ALF-FD001 to SAL-3ALF-FD 065</td>
<td>29,275</td>
<td>19%</td>
</tr>
<tr>
<td>1986</td>
<td>Exploration</td>
<td>SAL-SALF-FD066 to SAL-3ALF-FD 125</td>
<td>9,033</td>
<td>6%</td>
</tr>
<tr>
<td>1993</td>
<td>Exploration</td>
<td>SAL-3ALF-FD126 to SAL-3ALF-FD 189</td>
<td>14,585</td>
<td>9%</td>
</tr>
<tr>
<td>1997</td>
<td>Exploration</td>
<td>SAL-3ALF-FD190 to SAL-3ALF-FD 277</td>
<td>25,491</td>
<td>17%</td>
</tr>
<tr>
<td>2002</td>
<td>Exploration</td>
<td>SAL-3ALF-FD278 to SAL-3ALF-FD 410</td>
<td>66,243</td>
<td>43%</td>
</tr>
<tr>
<td>2003</td>
<td>Exploration</td>
<td>SAL-3ALF-FD411 to SAL-3ALF-FD 416</td>
<td>2,047</td>
<td>1%</td>
</tr>
<tr>
<td><strong>Total exploration</strong></td>
<td></td>
<td></td>
<td><strong>416</strong></td>
<td><strong>146,674</strong></td>
</tr>
<tr>
<td>1997</td>
<td>Geotechnical</td>
<td>SAL-3ALF-FG001 to SAL-3ALF-FG 007</td>
<td>3,847</td>
<td>2%</td>
</tr>
<tr>
<td>2003</td>
<td>Geotechnical</td>
<td>SAL-3ALF-FG008 to SAL-3ALF-FG 013</td>
<td>3,743</td>
<td>2%</td>
</tr>
<tr>
<td><strong>Total geotechnical</strong></td>
<td></td>
<td></td>
<td><strong>14</strong></td>
<td><strong>7,590</strong></td>
</tr>
<tr>
<td><strong>Grand Total</strong></td>
<td></td>
<td></td>
<td></td>
<td><strong>154,264</strong></td>
</tr>
</tbody>
</table>

Figure 11 - Plan View – Drill Hole Traces
10.1. **Drill Methods**

All boreholes drilled in support of resources were planned and laid out according to the Vale Salobo Guidelines and Reporting Standards. Compliance with these guidelines is checked on a quarterly basis to ensure guidelines and procedures are being followed.

Surface drilling was typically initiated with HQ diameter (63.5 mm) core and reduced to NQ diameter (47.6 mm). The minimum diameters were BX (36.6 mm) and BQ (36.5 mm). The underground drilling utilized BX diameter rods.

Drill hole collar locations are determined with a total station survey instrument. Collar verification is completed by plotting drill hole locations on plan and in cross section and comparing with the topographic surface. Down hole surveys are verified against the original survey data and on cross section plots.

10.2. **Core Reception, Handling and Storage**

The diamond drill core is collected and placed in wooden boxes. Core is delivered by the drilling contractor to the core logging / storage area where geological and geotechnical logging is carried out. At the core logging facility, core recovery and physical properties are measured and recorded. Geologic logs are prepared, and sample intervals are marked. Sample intervals average 1 m in mineralization and 2 or 4 m in barren zones. Sample lengths vary from these standards to honor significant geologic boundaries.

Before logging is completed pictures are taken of all drill core. After logging, the mineralized core is split in half using an electric saw, with one half being retained for further studies and audit purposes, whereas the other half is submitted for sample preparation and analysis. The position where the core is to be split is marked by Vale geologists.

Facilities for drill core storage consist of warehouse facilities special for this purpose and are located at the project site. Drill cores are stored in wooden trays and labelled with metallic plates. Pulps are stored in paper envelopes grouped in plastic bags, while the coarse rejects are stored in plastic bags. Both are organized in properly identifies boxes.

10.3. **Geological logging**

No written details are available on the logging procedures for the drilling campaigns prior to 1997.

During drilling campaigns from 1997 onward, core was collected in 1 m wooden boxes, and photographed in sets of two boxes each after transportation to the core shack. Logging was completed after sampling, and consisted of describing each individual lithologic package, as well as mineralogical variations within each one, the textures and structures, the ore minerals (including a visual assessment of volume percentage), the
presence of deleterious minerals (mainly fluorite), the visible structures and the foliation angle with respect to the core axis.

As part of the logging procedure, magnetic susceptibility was measured using Scintrex K2 and KT5C kappameters, with readings every 20 cm. This information was recorded in paper format (De Souza and Vieira, 1998).

In some of the early campaigns, uranium, thorium and potassium were directly determined in core using an Exploratium GR-320 gamma-spectrometer; however, this method was soon discarded. During the 2002–2003 campaign, uranium was chemically determined on 2 m intervals, which allowed this deleterious element to be modelled during Mineral Resource estimation.

Geotechnical logging of drill core was conducted by geologists following the guidance of geotechnical engineers. Logging included simple descriptions of the weathered zones and the weathering and fracturing degrees of the mineralized schists, as well as visual determination of the rock-quality designation (RQD) and rock resistance, and descriptions of the fracture types. Point-load tests (PLT) were also conducted every 20 m.

10.4. Recovery

Micon (2013) noted that core recoveries of 80% in weathered rock and 90% in fresh rock were achieved by the drilling companies during the campaigns. The average core recovery of the 2002 campaign (drill holes SAL-3ALF-FD 278 to 410) was 97.6%.

AmecFW (2016) reviewed the recovery data for holes where the information was recorded and supported Micon’s assessment of overall good recoveries.

10.5. Collar Surveys

During the 1997 campaign, drill-hole collars were placed and resurveyed after completion using WILD T1 stations.

During the 2002–2003 campaign, drill sites were placed and collar coordinates measured using total station equipment (before and after hole completion). The survey team also oriented the drill rigs, and provided proper initial alignment and inclination to the drilling rods. Collar verification was completed by plotting drill hole locations on plan and in cross-section and comparing with the topographic surface.

Current collar surveying of grade-control holes is conducted by company surveyors using high-precision, differential GPS equipment.

10.6. Downhole Surveys

No written details on the down-hole survey procedure in place prior to the 1997 campaign were available to AmecFW for their 2016 technical report.
During the 1997 campaign, down-hole survey readings taken on average every 3 m were conducted using the Reflex DDI (dip and direction pointer) and Maxibor units, to prevent errors in azimuth readings due to the influence of magnetite in the host rocks.

During the 2002–2003 campaign, down-hole survey measurements were conducted every 3 m using Reflex Maxibor and gyroscopic instruments.

10.7. Specific Gravity Determination

During the 1997 campaign, bulk density determinations were made with the water-displacement method. Tests were conducted on 20 cm to 40 cm long saprolite and bedrock core samples within intervals of approximately 10 m length.

Wet and dry bulk densities were determined on saprolite samples, which were weighed in air prior to and after drying (respectively), then coated with a thin plastic film, and submerged in water in a PVC recipient with a discharge opening. The sample volume was determined by measuring the water displaced through the discharge into a graduated cylinder. Core samples were assumed to be dried, so only dry density was determined. The bulk density (D) was determined as \( D = \frac{P}{V} \), where \( P \) is the dry (or wet) weight, and \( V \) is the volume of displaced water.

During the 2002–2003 campaign, the specific gravity (SG) was determined on representative fragments from all sampling intervals using a standard procedure. Hard-rock samples were cleaned and dried in air, and then weighed in air and in water. Saprolite samples were dried using an oven, then coated with paraffin prior to submerging them in water.

SG was then estimated as follows:

\[
SG = \frac{x_A}{x_A - x_W}
\]

Where:

\( x_A = \text{weight of core in air} \)

\( x_W = \text{weight of core completely submerged in water} \)

At Salobo, SG was measured on approximately 79,000 samples collected across the entire deposit. Values for weathered waste rock and unweathered bedrock were categorized separately due to differences in permeability and porosity caused by weathering.
10.8. Comments on Section 10

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs during the 1997 and later campaigns are sufficient to support Mineral Resource and Mineral Reserve estimation.

11. Sample Preparation, Analyses and Security

11.1. Sampling Methods

11.1.1. Drill Core

The core sampling procedure was similar during the 1997 and 2002–2003 campaigns. Sample intervals averaged 1 m in mineralized zones, and between 2 m and 4 m in barren zones. Sample intervals are constrained to geologic boundaries, lithological / mineralogical changes, and faults and shear zones.

Core is halved using diamond saws with one half bagged and submitted to the mine laboratory for analysis, and the remaining half retained as backup in the same original boxes.

11.1.2. Blast Holes

Blastholes are drilled on a 5 m x 5 m or 5 m x 7 m grid with a hole diameter of 12¼ inches. All blastholes located in ore zones are sampled; however, as the blasthole reaches the barren zones, the proportion of sampled holes decreases (one in two or even less), and the grade-control geologist determines which waste blastholes are sampled to ensure mineralization matches the interpretation in the geological model.

The sampling pattern depends on the shape of the cone. If it is well formed, then four channels are cut across the cone at 90° (Figure 12) using a small mattock, and the sample is collected using a jar from bottom to top of the inner channel wall.

If the cone has been partially damaged, then three channels are cut; however, if it is seriously damaged then the cone is not sampled. The average sample weight is 2 kg.

Tags are being inserted into the ore after blasting for tracking purposes to understand the differences between the samples estimated recovery and processing plant actual recovery. The tags are detected at specific points in the processing plant.
11.2. Sample Preparation

11.2.1. Exploration

Sample preparation details prior to 2002 are unknown. During 2002 – 2003, sample preparation was conducted by Lakefield / GEOSOL laboratory at a local facility built at the Project site, and consisted of the steps detailed in Figure 13.
11.2.2. Grade Control

Blast-hole samples are prepared and assayed at the Salobo Operations laboratory which has separate areas for the preparation of concentrate, tailings and blast-hole samples to avoid contamination. The preparation laboratory is well organized, and has modern equipment including ESSA jaw crushers, rotary splitters, puck-and-bowl pulverizers and Mettler-Toledo precision scales. A special, separated, scale room is used only for gold assays. The dust-extraction system is in place to reduce the chances of sample contamination.

The preparation procedure implemented for blast-hole samples is as follows:

- Drying in an electric oven at 105°C
- Jaw-crushing to >95% passing -3 mm size; granulometric tests are carried to check particle size on one in 20 samples
- Homogenization and splitting using rotary splitters to obtain 500 g splits
Pulverization using puck-and-bowl pulverizers to >95% passing 0.105 mm; granulometric and mass-loss checks are carried out on one in 20 samples on 100 g subsamples that are later discarded. The pulverized material is bagged and submitted for chemical assay.

11.3. Sample Analysis

11.3.1. Exploration

During the 1978 campaign, samples were assayed at the Docegeo laboratory in Belém, Pará, and at the SUTEC laboratory in Santa Luzia, Minas Gerais. Copper was assayed on 0.5 g aliquots by multi-acid digestion and atomic absorption spectrometry (AAS). Iron, molybdenum, and silver were also determined using this method. Gold was assayed by aqua regia leaching, with solvent extraction (MIBX) and AAS determination.

During the 1986 campaign, CVRD assayed samples at the Docegeo laboratory in Belém and at the pilot plant laboratory on the mine site, using the same analytical methods as in the previous campaign.

During the 1993 campaign, SML used the Mineração Morro Velho (MMV) laboratory. Copper was assayed with multi-acid digestion and AAS reading on 0.5 g aliquots (0.002% detection limit), and gold was determined using the fire-assay (FA) method with gravimetric finish on 100 g aliquots (0.05 g/t detection limit). In addition, samples were assayed for sulphur and carbon by LECO, and fluorine by alkaline fusion with sodium carbonate and potassium nitrate, followed by ion-selective electrode determination.

SMSA used the same analytical procedures during the 1997 campaign.

During the 2002–2003 campaign, Lakefield GEOSOL was used for the routine analysis of copper, gold, and silver, while Acme analyzed for molybdenum, uranium, fluorine, sulphur, and carbon. Chemical analysis was by atomic absorption spectrometry (AAS) for copper, silver and fluorine (on a 0.5 g aliquots and multi-acid digestion), while gold was assayed by FA with AAS finish on 20 g aliquots. Sample rejects are currently kept stored at the mine core shack.

In the early stages of the exploration program platinum, palladium, nickel, molybdenum and uranium were also analyzed; however, these elements were later excluded from the analytical package.

11.3.2. Grade Control

Blast-hole samples are assayed at the Salobo Operations analytical laboratory for copper, gold, silver iron, carbon, sulphur, fluorine, choline and soluble copper. Table 5 details the various analytical methods used for each of the elements and their detection limits.
Table 5 - Blast-Hole Sample Analysis

<table>
<thead>
<tr>
<th>Element</th>
<th>Aliquot (g)</th>
<th>Method</th>
<th>Detection Limit</th>
<th>Qualification Limit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu (%)</td>
<td>0.25</td>
<td>ARD-AAS</td>
<td>0.005</td>
<td>0.012</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>30</td>
<td>FA-AAS</td>
<td>0.023</td>
<td>0.063</td>
</tr>
<tr>
<td>Ag (g/t)</td>
<td>10</td>
<td>MAD-AAS</td>
<td>0.078</td>
<td>0.202</td>
</tr>
<tr>
<td>Fe (%)</td>
<td>0.25</td>
<td>ARD-AAS</td>
<td>0.003</td>
<td>0.008</td>
</tr>
<tr>
<td>C (%)</td>
<td>0.5</td>
<td>LECO</td>
<td>0.003</td>
<td>0.01</td>
</tr>
<tr>
<td>S (%)</td>
<td>0.5</td>
<td>LECO</td>
<td>0.001</td>
<td>0.003</td>
</tr>
<tr>
<td>F</td>
<td>NA</td>
<td>AF-I SE</td>
<td>NA</td>
<td>NA</td>
</tr>
<tr>
<td>Cl</td>
<td>NA</td>
<td>SAL-SNT</td>
<td>NA</td>
<td>NA</td>
</tr>
<tr>
<td>CuSol (%)</td>
<td>1</td>
<td>AcAL</td>
<td>0.001</td>
<td>0.003</td>
</tr>
</tbody>
</table>

Note: ARD: aqua-regia digestion; AAS: atomic absorption spectrometry; FA: fire assay; MAD: multi-acid digestion; AcAL: acetic acid leach; AF-I SE: boric-acid/sodium-carbonate fusion and ion-selective electrode determination; SAL-SNT: sulphuric acid leach and silver-nitrate titration; NA: not available; CuSol – acid soluble copper.

Precision scales and assay instruments are linked to a laboratory information management system (LIMS) to ensure the assay data are digitally transferred into the mine database. The LIMS is programmed to determine when readings comply with the required quality-control thresholds. Turnaround time is usually less than 24 hours for most elements, and four to five days for fluorine and chlorine.

Coarse and pulp rejects are stored for an unspecified time, after which they are discarded.

Assay batches are usually organized in 25 samples, not including the internal control samples. The lab’s quality control (QC) protocol includes the insertion of one reference material, one reactive blank (consisting of pure solution or flux in the case of FA), one coarse duplicate, and one pulp duplicate per batch.

11.4. Quality Assurance and Quality Control

The quality control (QC) programs at the Salobo Operations varied considerably over time, depending on the primary analytical laboratory used for assaying (AmecFW, 2016).

- 1986 – A total of 402 samples were resubmitted to alternative laboratories for external checks with GEOSOL acting as secondary laboratory for the Docegeo laboratory for copper and gold assays, the pilot plant laboratory as secondary laboratory for Docegeo on copper assays and Docegeo as secondary laboratory for the pilot plant laboratory for gold assays.
  - Results on copper assays indicated good correlation between the three laboratories; however, poor correlation was obtained between GEOSOL and Docegeo on the gold assays.

- 1993 - The QC program included external checks of 5% of the samples at the Nomos laboratory (for Cu) and at Fazenda Brasileira (for Au), using the FA method. In total, copper checks were conducted on 664 samples, and gold checks on 2,168 samples. For both elements, the correlation between laboratories was assessed as good.
- 1997 - SMSA implemented a QC program consisting of the insertion of 574 coarse duplicates and 14 reference materials, and the submission of 750 check samples to the Label laboratory for external checks.

- 2002 - Due to the lack of appropriate QC results for the drilling campaigns prior to 2002, a re-assay campaign was initiated to validate the available analytical data, thus a total of 51,768 of the original 75,577 samples drilled prior to 2002 were re-assayed to corroborate the original results.
  
  o Vale concluded that the external assay check review revealed bias for copper and gold assay results obtained by Nomos and Gamik laboratories. Based on the results obtained, Vale applied an adjustment factor to original sample grades (Table 6 and Table 7).

- 2002-2003 - In-house Standard Reference Material (SRMs) samples used during the 2002–2003 campaign (a total of nine) were derived from both the sulphide and oxide mineralization and incorporate a significant spread in the copper and gold grades. The recommended values for SRMs were established from a set of analytical results provided by three laboratories (the former Bondar Clegg laboratory, Gamik and Lakefield / GEOSOL). Each laboratory analyzed 10 aliquots of each SRM.
  
  o Two internal SRM samples were also prepared; however, they became available only at the end of the drilling program. As a result, a total of 1,500 samples from the 2002–2003 drilling program were selected for re-assaying in order to validate the 2002–2003 assay data. A total of 76 samples of two internal, project-derived SRMs were randomly inserted in the batch (5% frequency).

- AmecFW reviewed the QC data reported by CVRD (2003) and concluded that copper and gold check assays did not reveal significant biases, and that precision was within acceptable limits. Bongarcon (2003) also reviewed the 2002–2003 QC data, and concluded similarly that the special lot assays validated the 2002–2003 data for use in Mineral Resource estimation.

**Table 6 - Adjustment for Copper Assays for pre-2002 Drilling Programs**

<table>
<thead>
<tr>
<th>Holes</th>
<th>Number</th>
<th>Outliers</th>
<th>Interval</th>
<th>Regression</th>
<th>Correlation Coefficient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Docegeo D-001 to 065</td>
<td>10,833</td>
<td>126</td>
<td>1%</td>
<td>Cu adj = (1.029 * Cu) + 0.007</td>
<td>98%</td>
</tr>
<tr>
<td>CVRD D-066 to 125</td>
<td>3,609</td>
<td>113</td>
<td>3%</td>
<td>Cu adj = (1.068 * Cu) – 0.02</td>
<td>98%</td>
</tr>
<tr>
<td>SML D-126 to 189</td>
<td>400</td>
<td>46</td>
<td>12%</td>
<td>Cu adj = (0.98 * Cu) + 0.023</td>
<td>97%</td>
</tr>
<tr>
<td>SMSA D-190 to 277</td>
<td>12,453</td>
<td>489</td>
<td>4%</td>
<td>Cu adj = (1.014 * Cu) – 0.005</td>
<td>97%</td>
</tr>
<tr>
<td>Lakefield Geosol D-278 to 410</td>
<td>1,440</td>
<td>33</td>
<td>2%</td>
<td>Cu adj2 = (0.997 * Cu adj) – 0.003</td>
<td>99%</td>
</tr>
</tbody>
</table>
Table 7 - Adjustment for Gold Assays for pre-2002 Drilling Programs

<table>
<thead>
<tr>
<th>Holes</th>
<th>Number</th>
<th>Outliers</th>
<th>Interval</th>
<th>Regression</th>
<th>Correlation Coefficient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Docegeo + CVRD + SML</td>
<td>D-001 to 189</td>
<td>26,760</td>
<td>522</td>
<td>&gt;0.01</td>
<td>Au adj = (1.027 – Au) + 0.008</td>
</tr>
<tr>
<td>SMSA</td>
<td>D-190 to 277</td>
<td>11,519</td>
<td>257</td>
<td>&gt;0.01</td>
<td>Au adj = (1.018 – Au) + 0.005</td>
</tr>
</tbody>
</table>

11.4.1. Blanks

Golder (2010) reviewed the result of preparation blank samples submitted with the regular sample stream from 1999 to 2005. Although generally acceptable, some of the blank samples showed anomalously high content of copper which could have been caused by either cross-contamination in the sample preparation stage or sample mix-ups. Vale have confirmed that all blank exceedances were followed-up with the laboratory and re-assayed.

11.4.2. Duplicates

Three types of pulp duplicates were generated during the various Salobo QC programs including:

- Pulp duplicates, as part of the internal QC program of Lakefield Geosol.
- Inter-laboratory duplicates (external assay checks), part of the Salobo QAQC program. The majority of the pulp duplicates were sent as a special batch to a secondary laboratory (GAMIK) at the end of the program.
- A set of 1,500 samples from 2002-2003 drilling program, along with inserted standard samples, were re-analyzed to corroborate the original assay results by the primary laboratory.

The duplicate sample precision results are summarized in Table 8. For copper, the results demonstrate acceptable precision with average half absolute relative difference (HARD) values between 3% - 7%. This value is consistent with precision observed at 83.4% where values greater than 20% are indicating moderate precision. For gold the results indicate marginal to poor precision with average HARD values greater than 10%. This is consistent with precision observed at 83.4% where values are greater than 30%, although this may be expected due to its mineralizing pattern (nugget or fines) and therefore the levels of precision observed may be considered as acceptable. Bias is measured in terms of averaged HARD values. The results indicate no obvious bias for copper while for gold a slightly negative bias was detected.
Table 8 - Summary Duplicate Analysis

<table>
<thead>
<tr>
<th>Element</th>
<th>Laboratories</th>
<th>Type</th>
<th># Samples</th>
<th>Avg HARD (%)</th>
<th>Avg HARD (%)</th>
<th>Avg Bias</th>
<th>Precision (at 83.4%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>Lakefield vs Lakefield</td>
<td>Blind</td>
<td>515</td>
<td>7.37</td>
<td>0.06</td>
<td>0.01</td>
<td>28.7</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Blind</td>
<td>724</td>
<td>6.49</td>
<td>0.49</td>
<td>0.02</td>
<td>24.4</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Lakefield</td>
<td>Non blind</td>
<td>429</td>
<td>3.16</td>
<td>-1.00</td>
<td>-0.01</td>
<td>21.2</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Non blind</td>
<td>667</td>
<td>6.32</td>
<td>1.28</td>
<td>0.00</td>
<td>23.7</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Special lot</td>
<td>1,502</td>
<td>6.47</td>
<td>-0.46</td>
<td>0.01</td>
<td>28.7</td>
</tr>
<tr>
<td>Au</td>
<td>Lakefield vs Lakefield</td>
<td>Blind</td>
<td>515</td>
<td>16.98</td>
<td>5.97</td>
<td>-0.01</td>
<td>46.9</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Blind</td>
<td>537</td>
<td>15.92</td>
<td>-9.72</td>
<td>-0.05</td>
<td>41.3</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Lakefield</td>
<td>Non blind</td>
<td>367</td>
<td>8.45</td>
<td>-0.68</td>
<td>0.00</td>
<td>32.2</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Non blind</td>
<td>355</td>
<td>21.46</td>
<td>-17.92</td>
<td>-0.09</td>
<td>46.2</td>
</tr>
<tr>
<td></td>
<td>Lakefield vs Gamik</td>
<td>Special lot</td>
<td>1,417</td>
<td>13.88</td>
<td>-2.31</td>
<td>-0.04</td>
<td>42.8</td>
</tr>
</tbody>
</table>

11.4.3. Quality Control for Blast-holes

A QC program has been implemented to monitor blast-hole sampling quality. This program includes the insertion of 5% twin samples (obtained from repeating the sample process), 5% field duplicates (Jones splits of the same original sample that are assayed separately) and 5% SRMs.

11.5. Security

During drill campaigns all drill core was brought from the drill sites, at the end of shift, to a dedicated logging and storage facility. All drill core is stored in wooden boxes with proper numbering to indicate the drill hole number and meterage. The core storage and logging facility is kept locked when unoccupied. Unshipped samples are also stored in a secure facility at the same location.

Since August 2010, the evaluation of drilling and mine information has been uploaded to a Geovia Gems SQL database. This provides the geologists and mine engineers with a secure and more efficient access to information.

11.6. Comments on Section 11

In the opinion of the QPs, the 1997 and 2002–2003 sampling, sample preparation, assay and density data are suitable to support Mineral Resource and Mineral Reserve estimation.

12. Data Verification

12.1. Major Mining Studies

Prior to commencement of production the following studies were completed:

- Bechtel, 1988: Prefeasibility study
- Minorco, 1998: Feasibility study
12.2. **External Audits and Reviews**

Vale and its predecessor companies have commissioned a number of audits and third-party reviews of block models and Mineral Resources and Mineral Reserve estimates and processes:

- **Aker Kvaerner, 2001**: Updated feasibility study
- **Fluor JPS, 2004**: Feasibility study.

Vale and its predecessor companies have commissioned a number of audits and third-party reviews of block models and Mineral Resources and Mineral Reserve estimates and processes:

- **MRDI, 1997**: Mineral Resource estimate audit
- **AMEC, 2002**: Mineral Resource estimate audit
- **AMEC, 2004**: Mineral Resource estimate audit
- **Pincock, Allen and Holt, 2007**: Mineral reserve estimate audit
- **Pincock, Allen and Holt, 2007**: Due diligence audit
- **Pincock, Allen and Holt, 2008**: Mineral reserve estimate audit
- **Snowden, 2009**: Mineral Resource estimate audit

Third-party reviews of data in support of a technical report prepared for Wheaton was completed in 2013 and 2016:


As part of AmecFW work preparing their 2016 report, they undertook various data verification procedures including the following:

- **Review of drill-hole folders**: Vale keeps ordered folders for each drill hole in the mine office. AmecFW reviewed 14 folders, corresponding to 10% of the drill holes from the 2002–2003 campaign. All reviewed folders were well organized, and included collar survey data (but not original documents), drill reports, down-hole survey data, original geological logs and copies of original assay certificates. Most folders also included geotechnical logs, and density, magnetic susceptibility and PLT measurements. Some folders corresponding to earlier campaigns (1981, 1986) only included original logs.
- **Review of down-hole survey and assay data**: AmecFW compared 2,245 down-hole survey data from original paper records with the digital records in the database and did not identify any errors. Spot checks on assay data corresponding to the reviewed folders did not reveal any differences between assay certificates and the digital database records.
- **Interpretation of geology and mineralization**: AmecFW reviewed the geometry of the interpreted polygons in all geological and mineralization vertical cross-sections, in order to, assess the spatial continuity and correlation to individual drill holes.
holes. The lithology cross-sections were represented at the 1:7,500 scale, and the mineralization sections were represented at the 1:2,000 scale. The sections were spaced at 50 m to 100 m. AmecFW also reviewed horizontal plans with 50 m spacing. Drill holes were represented with the projected trace, and included lithology polygons in the lithology cross-sections, and ore-zone polygons and copper and gold grades in the ore cross-sections. During the review, AmecFW did not find significant discrepancies.

- AmecFW recognized that the interpretation generally respects the data recorded in the logs, plans and cross-sections, as well as the interpretation from adjoining plans and sections, and is consistent with the known characteristics of this deposit type. The lithological and mineralization models were diligently constructed in conformance to industry standard practices.

- Core review: AmecFW reviewed selected core sections of three drill holes (FD-280, FD-296 and FD-360), and observed that the core was properly cut. The observed contacts between major units approximately matched the logged depths, although it was apparent that the boundaries between units was usually established mainly on the basis of a visual appreciation of the proportion of magnetite and garnets. Recovery in the reviewed core was excellent.

12.3. Comments on Section 12

In the opinion of the QPs, data verification has been extensively conducted since 1988 by Vale employees and numerous consultants. No material issues have been identified by those programs. In addition, Vale continues to use various procedures to verify the quality of the data.

The QPs conclude that the drilling logging and sampling procedures are appropriate for the Salobo mineralization styles, that the assay data are reliable, and that the database is reasonably free of errors. Therefore, the data is suitable to support Mineral Resource and Mineral Reserve estimation and can be used for mine planning purposes.


13.1. Metallurgical Testwork

Five distinct phases of testwork have been completed:

- CVRD from 1978–1981
- CVRD and Anglo American from 1986–1987
- SMSA from 1993–1998, including a pilot-plant campaign carried out at the CVRD Research Centre (CRC)
- Locked-cycle flotation tests, flotation variability and grinding studies from 2003–2004; and
- Trade-off study using high-pressure grinding rolls (HPGR) for tertiary crushing as an alternative to conventional semi-autogenous grinding (SAG), from 2005–2006.

The following sections summarize the most relevant programs indicated above, as they provided the basis of the plant design criteria and/or its metallurgical performance projections.

13.1.1. Variability Tests

For the 2004 variability test programs, rougher flotation tests were conducted on 251 samples obtained from drill core representing a wide range of ore grades and lithologies. Approximately 47% were classified as XMT and 41% BDX, which are the two most important lithologies.

In addition, 59 locked cycle tests were carried out at Minas Gerais Technological Centre (Cetec) in Belo Horizonte, involving 30 BDX samples and 16 XMT samples.

Two major metallurgical improvements were incorporated into these tests. Firstly, a reagent scheme was adopted using a blend of two collectors, A350 (potassium amyl xanthate) and A3477 (sodium di-isobutyl dithiophosphate). This resulted in improved metallurgy and stable flotation conditions. During the 1994 pilot plant trials only A350 was used, resulting in unstable flotation conditions and the evaluation of a two-stage grinding circuit. The addition of sodium sulphide during rougher/scavenger flotation was demonstrated as being important for the effective flotation of bornite, which tends to oxidize and tarnish quickly and requires higher collector addition.

U.I. Minerals (Uimin) consolidated the 1994 plant trial results and the variability study results in December 2003. During this process, a number of filters were applied to the results, leaving 71 tests carried out on what was then deemed to be representative samples of the lithologies.

From this exercise, an average metallurgical recovery for copper is 90.7% and a mass recovery of 18.2% was projected. A total of 177 samples were analyzed with grades above 0.4% Cu. Results showed 87.6% of the samples with a copper recovery above 90%, 10.7% of samples had a recovery between 85 and 90%, and only 1.7% of samples were anomalous with recoveries less than 85%.

Copper recoveries for the 251 variability rougher tests are summarized in Figure 14.

There was a direct correlation between the copper recovery and mass recovery for each lithology. XMT samples had higher mass recoveries, which could be due to either higher grades and/or the presence of magnetite in the concentrates.

The average gold recovery for the deposit is 67.4% with a standard deviation of 14.4%. Approximately 64% of samples with initial gold grades above 0.4 g/t have gold recoveries of as up to 70%.
Gold recoveries for all the 251 variability rougher tests are summarized in Figure 15.

Based on the consolidated results, equations for predicting copper and gold recoveries were developed by UMIN, for use in mine planning and production forecasts.

- \[ \text{Rec Cu} \% = -0.023 / [\%\text{Cu in feed}] + 0.9023 \]  \{Equation 1\}
- \[ \text{Rec Au} \% = 0.0256 * [\text{g/t Au}] + 0.6485 \]  \{Equation 2\}

UMin commented that “it should be stated that the curve fit is rather poor” for Equation 1, as exhibited by the spread of data in Figure 14.

A further comment that Equation 1 is applicable mostly in the 0.6–1.5% Cu range, as head grade, was made given the test data fell outside this range.

Equation 1 is derived from using the data points represented by the yellow crosses in Figure 16 and subtracting 6.5% from these values. This deduction represents the losses expected in the cleaner-scavenger tails relative to what is recovered in the rougher-scavenger. The resulting overall recovery trend appears as the orange crosses in Figure 16.

The gold recovery projection represented by Equation 2 is illustrated by the red line in Figure 17 as the “best fit” for the locked-cycle tests as symbolized by the red circles. These are the equations used in the Project justification studies and the Geology and Mine Operations departments use them for projecting recoveries, across all lithologies, even though the pre-production testwork data had already shown differential responses for the XMT, BDX and DGRX.

---

Note: Figure from Umin, 2004. X-axis label is the initial Cu head grade in %; Y-axis is the rougher Cu recovery in percent, SE is the southeast sector, NW is the northwest sector, and the red line is the estimated recovery curve.
Figure 15 - Gold Recoveries from Variability and LCT Testwork Programs

Note: Figure from Uimin, 2004. X-axis label is the initial Au head grade in g/t (ppm); Y-axis is the Au recovery in percent, SE is the southeast sector, NW is the northwest sector.

Figure 16 - Derivation of Copper Recovery Projection, 2003–2004 Variability Testwork

Note: Figure courtesy Vale, 2014

The feasibility study conducted by Fluor Daniel in 2004 incorporated a conventional primary crushing circuit, a standard SAG mill/ball mill grinding circuit and a conventional copper flotation circuit.

However, several unique problems with Salobo ore led to the evaluation of an alternative to standard SAG mill grinding:

- Firstly, the high magnetite content (potentially exceeding 20% at times) presents a difficulty in the SAG mill circuit due to the need to remove and crush the critical size pebbles in a pebble crusher. The use of a magnet to remove tramp steel ahead of the crusher would invariably remove magnetite pebbles with a resulting loss of the associated copper and gold values (the higher grade copper is associated with the magnetite schists). There would then be additional design and cost implications to re-handle and process the magnetite pebbles.

- Secondly, significant variations in hardness and density were predicted for Salobo ore and conventional SAG milling circuits are sensitive to such variations, resulting in potential significant variability in mill throughput and performance.

Because of these concerns, an extensive evaluation of an alternative comminution circuit was conducted that included primary crushing, secondary cone crushing and tertiary HPGR crushing followed by conventional ball milling.
Polysius conducted two separate HPGR evaluations, in 2005 and 2006. The 2005 program tested pilot ore from the G3 adit at a top feed size of 32 mm and the 2006 program tested two samples to represent typical ore for the first five years of mining and hard ore. Top feed sizes tested were 25 mm and 32 mm. The Bond ball mill work index for the hard ore sample was 21.4 kWh/t.

General observations from this testwork program were that there was a decline in specific throughput as the roll speed increased and as the feed moisture content was increased. For the first five-year sample, there was an 18% reduction in specific throughput when the feed moisture content was increased from 0.1% to 4.0%. Abrasion testing and specific wear rates on all samples indicate that Salobo ore has low abrasion characteristics.

Grindability tests were conducted on samples of HPGR product at <6 mm and conventionally crushed material at <6 mm of the pilot ore sample from the 2005 program. The results indicated a very similar Bond ball mill work index for both samples (19.4 kWh/t and 19.2 kWh/t, respectively), indicating no micro-fracturing of the rock and therefore no grindability advantage was attributed to HPGR.

SMCC was retained as an independent reviewer of the Polysius test program and to size both the HPGR and ball mill units. Finally, Aker Kvaerner conducted a trade-off study using the results of the Polysius testwork programs and the SMCC review in 2006.

After reviewing all the work, Vale decided to implement the HPGR option based on the technical and economic benefits compared to conventional SAG.

13.1.3. Mixed Ore Zone Copper Recovery Testwork

A copper recovery study for the mixed ore stockpiled at the mine was commissioned in 2014. Figure 18 is indicative of the improved metallurgical response brought by much increased addition rates of the collectors currently used in the rougher flotation, as accompanied with the inclusion of sodium silicate, used as a dispersant (e.g. viscosity modifier).

Another study involved the transition ore, at the boundary between the oxide cap and the sulphide ore. The objective is to find the proportion of oxidized mineralization that can be added to the fresh, sulphide-bearing bedrock without impacting the overall copper recovery in the plant. As presented in Figure 19, such work provided an indication that a mixed ore component of up to 30% could be tolerated with limited impact on the results expected with fresh material only.

Work mostly carried out by the Vale Sheridan Park Research Centre (SPRC) in recent years resulted in development of a projected recovery curve for copper. SPRC also was involved in providing support towards the eventual processing of the mixed ore stockpile.
material. Figure 20 shows the recovery projection model, built by SPRC, and the results of testwork realized with fresh (ROM in the figure legend) and stockpiled ore samples.

The equation underlying the recovery projection model is expressed as follows:

- \( \text{Rec Cu}(\text{@}38\% \text{Cu}) = 88.5 \times (1 - \exp(-3.5 \times [\text{Cu in feed}])) \) \{Equation 3\}

Where \([\text{Cu in feed}]\) is the copper feed grade and the resulting projected recovery is based on a standardized concentrate grade target of 38% Cu.

The testwork program completed included modified reagent schemes, relative to the plant operations (changing the xanthate used from potassium amyl xanthate (PAX) to sodium isopropyl xanthate (SIPX), removing the sodium sulphide as modifier), as well as testing the addition of a desliming stage, with a cyclone, of the mixed stockpile material in an attempt to remove the most oxidized component and reduce reagent consumptions.

Figure 18 - Testwork with Mixed Ore – Effect of Dispersant and Collector Dosage

Note: Figure courtesy Vale, 2015. X-axis label is addition rate of dispersant (dispersante; as sodium silicate, Na\(_2\)SiO\(_3\)), in g/t (ppm); Y-axis is the addition rate of the sulphide collectors (coletor; as a potassium amyl xanthate and a sodium dithiophosphate, in undisclosed proportions), in g/t (ppm). Legend is showing colour legend for achieved Cu rougher recovery ranges in sensitivity area plot.
As illustrated in Figure 20, the ROM sample response under the modified (e.g. without sodium sulphide) or SIPX processing scheme did not present a marked advantage from the design (e.g. current plant operations) scenario. The figure is also indicative of the large degradation in the metallurgical results that would result from processing the stockpile material with the design reagent scheme and then the marginal improvement expected under the approach where the flotation feed would have been first deslimed by cycloning.

On the basis of the current testwork knowledge, the approach retained would be not to provide the plant with more than 30% of its total feed tonnage from the mixed material accumulated on the stockpile.
13.2. Recovery Estimates

Recovery projections for copper and gold (see Section 13.1.3) are based on Equation 1 and 2, respectively. These are underlying a fixed target copper grade in concentrate of 38% Cu. These equations are currently used to project these metals’ recoveries in the Mineral Reserve estimate, cutoff grade calculations, and the life-of-mine financial model.

Silver recovery was not tracked as diligently as copper and gold during the testwork phases.

Pre-production testwork, especially the large variability testwork program of 2003–2004, provided indications that the copper metallurgical response was variable, not only per the feed grade but also per the lithology of the ore. The adoption of a single equation to predict recovery, instead of drafting discrete equations for each lithology and using these to match the expected relative proportions provided in the mine plan as plant feed, is a simplification that may create daily discrepancies between expectations and actual results. However, this approach is likely, over monthly periods for instance, to provide a valid approximation of potential results. The shortfalls suffered during the commissioning periods of each plant are discounted. These would be the absence of sodium hydrosulphide (NaHS) as a modifier during the first six months of the Salobo I plant operations, the lack of a lime preparation system and associated safety concern related...
to the addition of NaHS in a low pH environment, and the high frequency of individual line shutdowns still encountered in the plant.

Figure 21 and Figure 22 in Section 13.5 present the actual monthly plant results against the projections from 2013 to 2017.

13.3. Metallurgical Variability

As discussed in Section 13.2, some variability in the metallurgical results can be expected as the mixture of lithologies found in the plant feed change. Over monthly periods, the resulting blend is more likely to approach the Mineral Reserves profile and thus mitigate the variability that may be detected on a daily basis, versus projections.

Introduction of mixed material above a proportion of 30% of plant feed has been shown to lead to a degradation of the flotation results. Proper blending of such material, albeit representing only 1% of the Mineral Reserves, will be required.

13.4. Deleterious Elements

There are three deleterious elements of potential concern in the copper concentrate, namely fluorine, chlorine and uranium. Of these, fluorine is the most significant. In general, smelters will tend to reject concentrates with high fluorine content due to problems that result in the smelter’s sulphuric acid plants.

Testwork was conducted as part of the 1994 pilot plant campaign and continued by CVRD in 1995 to evaluate the potential for acid leaching of the concentrate to reduce fluorine levels. This was apparently unsuccessful due to insufficient removal of fluorine, high dissolution of copper and difficulty in filtering the leach residue.

Mineralogical examination of the ore lithologies and concentrate samples have indicated a tendency for this element to be concentrated in fluorine and silicates, in particular the biotite. These gangue minerals are partially reporting to the concentrate stream mostly through partial liberation from sulphide-bearing mineral grains and mechanical entrainment in the froth phase of the flotation process. Regrinding of the scavenger concentrate, for incremental liberation, as well as cleaning in the flotation circuit with flotation columns, instead of conventional mechanically-agitated cells, was adopted to enhance liberation and provide a means for more effective froth washing, with water, of the final concentrate stream. These two approaches demonstrated a capability to improve rejection of the deleterious elements at levels acceptable to some smelters, albeit while still attracting penalties on fluorine and chlorine, for their high residual contents.

Vale has secured contracts with smelters able to accept the copper concentrate, with an average fluorine content of about 2,000 ppm, and a maximum content of 4,000 ppm. Penalties are charged though starting below the actual content.
These smelters also placed the maximum acceptable chlorine content at 1,200 ppm, but with a penalty drawn at the 550–650 ppm.

Uranium, is present in the copper concentrate. Recent annual averages of uranium levels are between 35 to 50 ppm. The specification limit is 50 ppm. There is a strong correlation between uranium in the concentrate to uranium in the feed. Operational procedures are being implemented to accurately forecast the uranium grade in the long term and short term models to enable blending of plant feed from the mine. Since concentrate lots are segregated by grade (lower, medium and high grades) at the Parauapebas transfer shed and at the port of São Luis, blending of out-of-specification concentrate is possible, should it ever be necessary.

13.5. Actual Plant Results versus Budgeted Projections

Considering the accumulated historical data from the plant operations, it is relevant to assess how current results match expectations from the design stage and how they are integrated in the prediction of future results, both from the aspects of the metallurgy and of the throughput capability of the plant.

13.5.1. Historical Metallurgical Results

Figure 21 and Figure 22 are indicative of the monthly plant performance to December 2017 in relation to the predicted recoveries based on testwork data and Equation 1 and Equation 2. Section 13.2 offered an overview of some of the events that may have contributed to the differentials shown in these figures over specific periods, with copper having apparently suffered more shortfalls than gold as a result of these.

Figure 23 to Figure 26 present in more detail the recent metallurgical results such as the 2015 monthly data.

The 2017 copper recovery target was 87.9% in the budget, and the actual copper recovery in 2017 was 86.5% (Figure 21). Using Equation 1, the actual copper grade in 2017 of 0.946% would have predicted a copper recovery of 87.8% so the actual copper recovery was 98.5% of the predicted value.

The 2015-2017 results for copper recovery represent a marked improvement both from 2014, which may have been negatively influenced by the commissioning of Salobo II, and from 2013, at 81.4% and 80.9%, respectively (Figure 23).

The 2017 gold recovery target is was 68.7% in the budget, in comparison to 69.4% achieved (refer to Figure 23). Using Equation 2, the 0.63 g/t Au head grade budgeted for the full year would have called for a projected recovery of 66.5%. The actual gold recovery of 68.0% is 102% of the predicted value.
Gold recovery from 2013-2017 has been consistent at 65-69%, generally above budget (Figure 22 and Figure 24) with the lower recoveries the result of frequent process plant stoppages during the production ramp-up of both Salobo I and II with higher grades in 2013-2014 offsetting these process interruptions.
Figure 23 - Historical Plant Performance – Cu Recovery

Copper Recovery

Year
2013
2014
2015
2016
January
80.9%
81.4%
86.8%
87.5%
February
86.4%
87.3%
86.8%
87.3%
March
87.8%
87.3%
87.3%
87.3%
April
86.8%
86.7%
85.7%
85.6%
May
87.0%
87.7%
87.7%
87.3%
June
87.3%
87.7%
87.7%
87.3%
July
87.3%
87.7%
87.7%
87.3%
August
85.9%
85.6%
86.0%
86.1%
September
86.5%
86.5%
86.5%
86.5%
October
2016
2017

Figure 24 - Historical Plant Performance – Au Recovery

Gold Recovery

Year
2013
2014
2015
2016
January
65.2%
64.2%
65.3%
66.6%
February
67.5%
69.3%
67.8%
66.6%
March
69.1%
64.1%
67.8%
62.3%
April
62.3%
61.4%
61.4%
61.4%
May
61.4%
61.4%
61.4%
61.4%
June
62.3%
64.1%
67.8%
67.8%
July
67.8%
67.8%
67.8%
67.8%
August
67.8%
67.8%
67.8%
67.8%
September
72.8%
74.3%
72.8%
72.8%
October
71.5%
71.5%
71.5%
71.5%
November
70.0%
68.0%
68.0%
68.0%
December
2017
Equation 1 was derived on the premise of achieving a final copper concentrate grade of 38% Cu. This value is used in the budget and financial model, and by the operations personnel to decide whether to pull the flotation circuit harder or not. As shown in Figure 25, a very good performance from the plant has been achieved in this regard, and should be able to be upheld in the future. The gold grade in the copper concentrate has been
consistently between 19 and 24 g/t with variations being attributable to different gold to copper ratios in the feed.

13.5.2. Historical Plant Availability and Utilization

The throughput capability of the plant has been increasing over time through improvements in both the plant availability and utilization. These have both been improving over the initial few years of operation when ramp-up and commissioning of the Salobo I and Salobo II circuits was occurring. The improvements are noted in Figure 27 and Figure 28.

Figure 27 - Salobo Plant Historical Availability

![Plant Availability (%)](image-url)
The onus for achieving incremental processed tonnage is thus mostly placed on the maintenance group, as it is tasked with generating the availability improvement expected from the equipment, and on the process control group, with control tuning and strategic modifications. The 2020 production plan requires that the availability target gradually increases to 88.9% by 2020. With the 2017 achieved availability of 89.3%, it has been demonstrated that the plant can run at the required availability.

### 13.5.3. Recovery Projections

The estimates used for 2018 and 2018-2022 plant recoveries of copper and gold have been based on recent plant operating performance and are shown in Table 9.

<table>
<thead>
<tr>
<th>Metal</th>
<th>Total Recovery to All Concentrates (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper</td>
<td>((-3.0427 \times %\text{Cu in feed} + 91.0894)</td>
</tr>
<tr>
<td>Gold</td>
<td>(1/(-0.0025 \times \text{Rec Cu% + 0.23395}))</td>
</tr>
</tbody>
</table>

Estimates of plant recoveries for 2023-remaining LoMP are shown in Table 10 below.
Table 10 - Processing Recovery Assumptions (2023-LoMP)

<table>
<thead>
<tr>
<th>Metal</th>
<th>Total Recovery to All Concentrates (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper</td>
<td>(-2.5362*1/(%Cu in feed)) + 90.674</td>
</tr>
<tr>
<td>Gold</td>
<td>(1.0173*RecCu) - 20.357</td>
</tr>
</tbody>
</table>

Equations to project copper and gold recoveries used a large dataset from testwork. Within the 0.6–1.5% Cu range of the data points retained for a regression analysis, the resulting equations are fairly insensitive to the actual feed grade encountered.

The “noisy” set of testwork data may have been divided by lithologies for analysis, in order to use this additional information as a projection tool. More recent testwork confirmed a better copper response for XMT than for BDX, for example. The potential variability of gold, as it relates to lithology, is less clear.

The recovery projection equations may therefore be sufficient for predicting results over longer-term periods (e.g. yearly, maybe monthly), but may not be adequate for applying a daily target to the plant operations since variations in the lithological make-up of the plant feed over such a short period may have called for different target recoveries than indicated by the equations.

A geometallurgical program initiated in June 2014 is expected to allow for building a database from which refinements to the projection tools, if deemed valid, could be made. Results achieved to date from operations indicate that metallurgical targets are reasonable and fairly aligned with the projection tools in place, once consideration is made for ramp-up effects.

The geometallurgical routine process was improved in 2017 based on the following actions:

- Short-term analyses are currently being executed for copper, gold, and deleterious variables and the flotation performance results are delivered one day before the liberation of blasting polygons by geology team. This is providing improved information for the short-range block model estimation process, increasing its predictability.

- Tags are being inserted into the ore after blasting for tracking purposes to understand the differences between the samples estimated recovery and processing plant actual recovery. The tags are detected at specific points in the processing plant.

13.6. Process Plant Optimization

Figure 29 and Figure 30 show the as constructed Salobo Mineral Processing Facility and process flowsheet.
Figure 29 - Salobo Mineral Processing Facility
Changes and accomplishments at the Salobo processing plant in 2017 include the following:

- The flotation reagents (frother and collectors) were changed, generating a reduction of 20% in reagent consumption.

- Modification to the lime dosage system, reducing the plugging occurrences in the dosage pipes.

- Coagulant dosage in the plant tailings in order to reduce the tailings dam turbidity.

- The wash water system was replaced by a more effective system (“arandela water”) in all flotation columns in the cleaner circuit in order to reduce the deleterious content at the final concentrate.

- The rougher circuit was modified to regrind the rougher 1 concentrate in the vertical mills, allowing better liberation and contributing to decrease the deleterious content at the final concentrate.

Some actions were carried out to increase the mill rate, as follows:

- Improvements in the tramp metal extraction system (guard magnets), increasing the HPGR’s rolls lifespan.

- Reducing the aperture size at the primary screening.
Increasing the vortex outlet diameter of the hydro-cyclones, reducing the circulating load in the ball mill circuit.

13.7. Comments on Section 13

In the QPs opinion, the testwork is appropriate to be the basis for the Salobo concentrator design. This is evidenced by the recent metallurgical performance of the commercial plant being very close to that predicted by models derived from the results of the extensive testwork.


Mineral Resource estimation is completed by João Dirk under the supervision of Cassio Diedrich, both Vale employees. The estimates are prepared according to the 2014 CIM Definitions Standards and the 2003 CIM Best Practice Guidelines.

Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

There has been insufficient exploration to classify the Inferred Mineral Resources as an Indicated or Measured Mineral Resource. The extent to which further exploration may result in upgrading them to an Indicated or Measured Mineral Resource category is uncertain at this time. Infill drilling was completed in 2017 and continued in 2018, targeting areas of lower density drilling with the intent of upgrading Inferred resources.

In 2017, a 5-year medium-range diamond drilling program was started. The program plans to drill 10,000 m per year in an effort to improve the long-range geological model contours, grade estimation, uncertainty, predictability and classification. The impact of the new drilling information is expected to be realized in the 2019 mine plan, as the 2017 drilling information will be available for use at the end of 2018.

The last diamond drilling campaign at the Salobo Operations was executed in 2010 (three drill holes) and the last major drilling campaign was in 2003 (142 drill holes). Since then all updates of the long-range Mineral Resource model are based on the short-range production reconciliation results. Blasthole information is used to update long-range geological model contours in the mined-out zone.

A major review of the Mineral Resource model was carried out in 2017 including a detailed assessment of the Mineral Resource estimate and procedures, ensuring that the applied method complies with industry best practices. No new diamond drill hole information was used in the 2017 revision.

14.1. Introduction

Mineral Resource modeling for Salobo utilizes drilling data, enhanced knowledge of metallurgical processing, geology and mineralization, and refined interpolation
parameters. The geologic and Mineral Resource models were constructed using GEMS™ and Isatis® software. The estimated Mineral Resources are then converted to Mineral Reserves using long term mine planning techniques and quoted above a cutoff grade of 0.253% Cu equivalent (CuEq).

Only diamond drill hole composites form the database and are considered in building the Mineral Resource model for the Salobo deposit.

Mineral resources were classified as Measured, Indicated and Inferred in accordance with 2014 CIM Definition Standards. Vale’s geologic and block models have been peer reviewed via external audits. No inferred resources are converted to Mineral Reserves.

Table 11 summarizes the estimated Mineral Resources at the Salobo Operations as of 31 December, 2017.

<table>
<thead>
<tr>
<th>Classification</th>
<th>M Tonnes</th>
<th>Grades Cu %</th>
<th>Au g/t</th>
<th>Contained Metal Cu (M lb)</th>
<th>Au (M oz)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured</td>
<td>33.0</td>
<td>0.72</td>
<td>0.42</td>
<td>524</td>
<td>0.4</td>
</tr>
<tr>
<td>Indicated</td>
<td>171.1</td>
<td>0.62</td>
<td>0.31</td>
<td>2,339</td>
<td>1.7</td>
</tr>
<tr>
<td>M&amp;I</td>
<td>204.1</td>
<td>0.64</td>
<td>0.33</td>
<td>2,863</td>
<td>2.2</td>
</tr>
<tr>
<td>Inferred</td>
<td>175.7</td>
<td>0.55</td>
<td>0.28</td>
<td>2,130</td>
<td>1.6</td>
</tr>
</tbody>
</table>

Notes:
1. Mineral Resource estimates were prepared by Mr. Joao Dirk V. Reuwsaat, a Vale employee. The Qualified Person for the Mineral Resource estimates is Mr. Cassio Diedrich, AusIMM-CP(Min), Technical Services General Manager, Vale Base Metals.
2. Mineral Resources are classified as Measure, Indicated and Inferred Mineral Resources and are based on the 2014 CIM Definition Standards.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. Mineral Resources are reported above a copper equivalent cutoff of 0.253%, assuming $1,200 per ounce gold and $2.86 per pound copper.
6. Tonnages are rounded to the nearest 100,000 tonnes; grades are rounded to two decimal places.
7. Contained copper is reported as Imperial pound units and contained gold as troy ounces.

14.2. Geological Interpretation

Mineral resource estimates are based on a three-dimensional computer block model utilizing GEMS™ software. Horizontal and vertical block sizes were chosen to adequately model the geometry of mineralized zones, and to approximate the selective mining unit (SMU) based on the proposed mining fleet. The resource block model was estimated using a regular block model with block sizes of 30 m by 30 m by 15 m.

The geologic models for lithology and mineralization were produced by Vale geologists and were based on their experience of the geological features of the deposit, including structure, hydrothermal alteration minerals, lithologies and mineralization.

The following two zones were interpreted:
- Low grade grading between 0.2 to 0.6% Cu, corresponding to the structurally controlled alteration halo consisting primarily of the BDX unit; and
- High grade above 0.6% Cu, consisting primarily of the XMT unit.
In general, a minimum drill-intercept width of 8.0 m was used to define the two mineralized zones and internal barren or weakly mineralized zones. Isolated intervals below 0.2% Cu are included in mineralized zones to provide continuity of geometry from hole to hole and section to section. Narrow isolated intervals grading above 0.2% Cu are generally not interpreted as mineralized zones.

Other than the intersection of the diabase dyke, little or minimal structural disturbance is observed within the Salobo deposit and for the purpose of mineral envelope modelling has not been considered. The presence of any faulting is noted but due to the orientation of these faults the operation has taken the decision not to model these at this time as they are not thought to materially impact the estimation model.

Mineral Resource estimation was undertaken by applying a knowledge of the deposit and understanding of local variations within each domain that control the spatial grade variation. This is further investigated by testing the search parameters to arrive at the most robust estimate. The Ordinary Kriging (OK) method was used to estimate block grades. Estimation for all variables was performed using a three pass OK approach by estimation domain. One additional pass was performed for domains to allow for the estimation of all blocks. A block discretization of 5 m by 5 m by 5 m was adopted for the 15 m block bench height.

Salobo was divided in two sectors from west to east to account for changes in the orientation and style of the mineralized zone along strike (Figure 31). Polygonal shapes were used to create solids to code the other sectors, based on level plan views of the mineralized zones.

Note: Figure courtesy Vale, 2014.
Triangulated solid models were also created for each of the waste rock types using generalized geologic sections and level plans. In the case of late-stage, unmineralized units such as mafic dykes, the solids are used to overwrite mineralization codes in the block model. Figure 32 shows the grade shells used in the block modeling in vertical section view.

14.3. Domaining

The estimation domains are based on units defined for total copper and gold and are the result of a combination of sector, ore code and weathering variables. The subdivision between NW and SE sectors is defined by difference in deformation and hydrothermal alteration. The existence of a diabase dyke with strike N70°E defines the border between sectors. Another relevant aspect is the dip of the lithological units. In the SE sector, the dip is subvertical to southwest and in the NW sector the dip is subvertical to northeast.
Block model domain and zone codes as they relate to sectors and grade groups are shown in Table 12 and Table 13.

### Table 12 – Block Model Domains Codes

<table>
<thead>
<tr>
<th>Sector</th>
<th>Oxide Description</th>
<th>Code</th>
<th>Sulphide Description</th>
<th>Domain</th>
</tr>
</thead>
<tbody>
<tr>
<td>Southeast</td>
<td>Low Grade - Saprolite</td>
<td>1101</td>
<td>Low Grade - Fresh Rock</td>
<td>1103</td>
</tr>
<tr>
<td></td>
<td>Low Grade - Semi-Weathered</td>
<td>1102</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>High Grade - Saprolite</td>
<td>1201</td>
<td>High Grade - Fresh Rock</td>
<td>1203</td>
</tr>
<tr>
<td></td>
<td>High Grade - Semi-Weathered</td>
<td>1202</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Northwest</td>
<td>Low Grade - Saprolite</td>
<td>2101</td>
<td>Low Grade - Fresh Rock</td>
<td>2103</td>
</tr>
<tr>
<td></td>
<td>Low Grade - Semi-Weathered</td>
<td>2102</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>High Grade - Saprolite</td>
<td>2201</td>
<td>High Grade - Fresh Rock</td>
<td>2203</td>
</tr>
<tr>
<td></td>
<td>High Grade - Semi-Weathered</td>
<td>2202</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Table 13 - Block Model Zone Codes

<table>
<thead>
<tr>
<th>Code</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Oxidized (SAP)</td>
</tr>
<tr>
<td>2</td>
<td>Transition Ore (ZTR)</td>
</tr>
<tr>
<td>3</td>
<td>Sulphide Ore (RFR)</td>
</tr>
</tbody>
</table>

Figure 33 and Figure 34 plot mean grades versus standard deviation for copper and gold respectively (Micon, 2013). The plots show the High Grade Domains (1203 and 2203) and Low Grade Domains (1103 and 2103) having very distinct populations for both copper and gold which supports the domaining strategy.
Figure 33 - Domain Definition for Copper

Figure 34 - Domain Definition for Gold
14.4. Statistical Analysis

14.4.1. Raw Assay Statistics

Table 14 and Table 15 show the raw assay statistics by lithological units for copper and gold respectively. The coefficient of variation (CV) is a ratio of the standard deviation to the mean, and is useful measurement of relative grade variability. Higher CV values indicate higher variability. The BDX, DGRX and XMT are the most important lithologies in the deposit economics. CVs for copper for these three lithologies are all quite low being below 1.5. Gold CVs for the BDX and DGRX lithologies are relatively high (>3.0) but the XMT is below 1.5.

Table 14 - Copper Assay Statistics by Lithology

<table>
<thead>
<tr>
<th>Litho</th>
<th>Description</th>
<th># Samples</th>
<th>Min</th>
<th>Max</th>
<th>Mean</th>
<th>Median</th>
<th>St Dev</th>
<th>CV</th>
</tr>
</thead>
<tbody>
<tr>
<td>All</td>
<td>All</td>
<td>82,888</td>
<td>0.02</td>
<td>30.00</td>
<td>0.63</td>
<td>0.35</td>
<td>0.85</td>
<td>1.35</td>
</tr>
<tr>
<td>BDX</td>
<td>Biotite-Garnet-Schist</td>
<td>32,967</td>
<td>0.02</td>
<td>30.00</td>
<td>0.51</td>
<td>0.32</td>
<td>0.69</td>
<td>1.35</td>
</tr>
<tr>
<td>COB</td>
<td>Cover</td>
<td>1,186</td>
<td>0.02</td>
<td>2.30</td>
<td>0.26</td>
<td>0.22</td>
<td>0.22</td>
<td>0.84</td>
</tr>
<tr>
<td>CX</td>
<td>Chlorite-Schist</td>
<td>31</td>
<td>0.02</td>
<td>8.50</td>
<td>0.74</td>
<td>0.22</td>
<td>1.53</td>
<td>2.07</td>
</tr>
<tr>
<td>DB</td>
<td>Diabase</td>
<td>540</td>
<td>0.02</td>
<td>2.85</td>
<td>0.19</td>
<td>0.04</td>
<td>0.34</td>
<td>0.76</td>
</tr>
<tr>
<td>DGRX</td>
<td>Garnet-Grunerite Schist</td>
<td>5,091</td>
<td>0.02</td>
<td>10.00</td>
<td>0.53</td>
<td>0.29</td>
<td>0.66</td>
<td>1.26</td>
</tr>
<tr>
<td>GM</td>
<td>Granitoid</td>
<td>2,932</td>
<td>0.02</td>
<td>8.90</td>
<td>0.21</td>
<td>0.08</td>
<td>0.48</td>
<td>2.23</td>
</tr>
<tr>
<td>GR</td>
<td>Granite</td>
<td>67</td>
<td>0.02</td>
<td>2.30</td>
<td>0.11</td>
<td>0.03</td>
<td>0.31</td>
<td>2.70</td>
</tr>
<tr>
<td>HD</td>
<td>Hydrothermalite</td>
<td>3,827</td>
<td>0.02</td>
<td>25.50</td>
<td>0.46</td>
<td>0.21</td>
<td>0.84</td>
<td>1.84</td>
</tr>
<tr>
<td>ML</td>
<td>Mylonite</td>
<td>4,553</td>
<td>0.02</td>
<td>14.80</td>
<td>0.31</td>
<td>0.17</td>
<td>0.49</td>
<td>1.57</td>
</tr>
<tr>
<td>MTB</td>
<td>Metavolcanic Basic</td>
<td>502</td>
<td>0.02</td>
<td>9.40</td>
<td>0.24</td>
<td>0.13</td>
<td>0.50</td>
<td>2.11</td>
</tr>
<tr>
<td>QML</td>
<td>Quartz-Mylonite</td>
<td>3,085</td>
<td>0.02</td>
<td>8.90</td>
<td>0.39</td>
<td>0.25</td>
<td>0.52</td>
<td>1.33</td>
</tr>
<tr>
<td>QZ</td>
<td>Quartzite</td>
<td>28</td>
<td>0.08</td>
<td>1.80</td>
<td>0.31</td>
<td>0.24</td>
<td>0.32</td>
<td>1.04</td>
</tr>
<tr>
<td>RIO</td>
<td>Rhyolite</td>
<td>2</td>
<td>0.07</td>
<td>0.16</td>
<td>0.12</td>
<td>0.16</td>
<td>0.06</td>
<td>0.55</td>
</tr>
<tr>
<td>SP</td>
<td>Saprolite</td>
<td>13,942</td>
<td>0.02</td>
<td>14.70</td>
<td>0.63</td>
<td>0.42</td>
<td>0.68</td>
<td>1.09</td>
</tr>
<tr>
<td>XMT</td>
<td>Magnetite Schist</td>
<td>11,196</td>
<td>0.02</td>
<td>18.30</td>
<td>1.51</td>
<td>1.20</td>
<td>1.25</td>
<td>0.83</td>
</tr>
<tr>
<td>ZT</td>
<td>Semi-Weathered Rock</td>
<td>2,939</td>
<td>0.02</td>
<td>15.00</td>
<td>0.60</td>
<td>0.39</td>
<td>0.84</td>
<td>1.39</td>
</tr>
</tbody>
</table>

Table 15 - Gold Assay Statistics by Lithology

<table>
<thead>
<tr>
<th>Litho</th>
<th>Description</th>
<th># Samples</th>
<th>Min</th>
<th>Max</th>
<th>Mean</th>
<th>Median</th>
<th>St Dev</th>
<th>CV</th>
</tr>
</thead>
<tbody>
<tr>
<td>All</td>
<td>All</td>
<td>75,079</td>
<td>0.02</td>
<td>67.27</td>
<td>0.38</td>
<td>0.14</td>
<td>1.10</td>
<td>2.88</td>
</tr>
<tr>
<td>BDX</td>
<td>Biotite-Garnet-Schist</td>
<td>30,368</td>
<td>0.02</td>
<td>58.37</td>
<td>0.28</td>
<td>0.13</td>
<td>0.92</td>
<td>3.30</td>
</tr>
<tr>
<td>COB</td>
<td>Cover</td>
<td>1,124</td>
<td>0.02</td>
<td>53.00</td>
<td>0.62</td>
<td>0.30</td>
<td>1.62</td>
<td>2.93</td>
</tr>
<tr>
<td>CX</td>
<td>Chlorite-Schist</td>
<td>20</td>
<td>0.03</td>
<td>1.20</td>
<td>0.33</td>
<td>0.32</td>
<td>0.29</td>
<td>0.88</td>
</tr>
<tr>
<td>DB</td>
<td>Diabase</td>
<td>675</td>
<td>0.02</td>
<td>1.46</td>
<td>0.05</td>
<td>0.02</td>
<td>0.13</td>
<td>2.58</td>
</tr>
<tr>
<td>DGRX</td>
<td>Garnet-Grunerite Schist</td>
<td>4,710</td>
<td>0.02</td>
<td>50.25</td>
<td>0.24</td>
<td>0.10</td>
<td>0.97</td>
<td>3.97</td>
</tr>
<tr>
<td>GM</td>
<td>Granitoid</td>
<td>2,395</td>
<td>0.02</td>
<td>26.55</td>
<td>0.12</td>
<td>0.05</td>
<td>0.58</td>
<td>4.94</td>
</tr>
<tr>
<td>GR</td>
<td>Granite</td>
<td>49</td>
<td>0.02</td>
<td>1.35</td>
<td>0.06</td>
<td>0.02</td>
<td>0.20</td>
<td>3.36</td>
</tr>
<tr>
<td>HD</td>
<td>Hydrothermalite</td>
<td>3,812</td>
<td>0.02</td>
<td>67.27</td>
<td>0.19</td>
<td>0.06</td>
<td>1.25</td>
<td>6.53</td>
</tr>
<tr>
<td>ML</td>
<td>Mylonite</td>
<td>3,533</td>
<td>0.02</td>
<td>33.46</td>
<td>0.21</td>
<td>0.08</td>
<td>1.02</td>
<td>4.86</td>
</tr>
<tr>
<td>MTB</td>
<td>Metavolcanic Basic</td>
<td>390</td>
<td>0.02</td>
<td>2.69</td>
<td>0.07</td>
<td>0.03</td>
<td>0.16</td>
<td>2.33</td>
</tr>
<tr>
<td>QML</td>
<td>Quartz-Mylonite</td>
<td>2,521</td>
<td>0.02</td>
<td>6.31</td>
<td>0.11</td>
<td>0.05</td>
<td>0.23</td>
<td>2.07</td>
</tr>
<tr>
<td>QZ</td>
<td>Quartzite</td>
<td>18</td>
<td>0.02</td>
<td>0.13</td>
<td>0.04</td>
<td>0.03</td>
<td>0.03</td>
<td>0.76</td>
</tr>
<tr>
<td>RIO</td>
<td>Rhyolite</td>
<td>2</td>
<td>0.02</td>
<td>0.08</td>
<td>0.05</td>
<td>0.08</td>
<td>0.04</td>
<td>0.85</td>
</tr>
<tr>
<td>SP</td>
<td>Saprolite</td>
<td>11,832</td>
<td>0.02</td>
<td>53.00</td>
<td>0.36</td>
<td>0.15</td>
<td>0.98</td>
<td>2.70</td>
</tr>
<tr>
<td>XMT</td>
<td>Magnetite Schist</td>
<td>11,086</td>
<td>0.02</td>
<td>54.65</td>
<td>1.00</td>
<td>0.61</td>
<td>1.48</td>
<td>1.48</td>
</tr>
<tr>
<td>ZT</td>
<td>Semi-Weathered Rock</td>
<td>2,544</td>
<td>0.02</td>
<td>44.33</td>
<td>0.31</td>
<td>0.11</td>
<td>1.40</td>
<td>4.48</td>
</tr>
</tbody>
</table>
14.4.2. Compositing

The sample intervals are generally 1.0 m and adhere to breaks such as geological contacts, faults and obvious changes in metal content. Two metre down-hole composites were created for statistical and geostatistical analysis and block grade interpolation. This composite length was chosen to provide the greatest amount of detail for estimating mineralized zones and to provide greater flexibility in dilution control and number of composite samples used for interpolation.

Samples were composited in Isatis® to 2.0 m using the Regularization tool. The compositing process considered breaks in the presence of non-assayed intervals or in ore/waste contacts. If a sample interval is not assayed, it is not used in the calculations of the composite value.

14.4.3. Domained Composite Statistics

Subdividing a deposit into the domains segregates areas of common grade populations which normally reduces grade variability. Compositing also reduces variability by smoothing the effects of any short intervals of anomalous grades over a wider common interval. Table 16 details the composite statistics for copper and gold grouped according to the interpolation domains described above. CV values for both copper and gold are reduced from the raw assay statistics shown in Table 14 and Table 15. The level of grade variability indicated by these CVs supports the use of OK for block grade interpolation with proper attention to the handling of outliers.

14.4.4. Outlier Analysis

Outlier analysis of the original assays included a statistical review of the grade populations and a visual review of the location high grades outliers. The statistical review examined CVs as discussed above and plotting of scatter and histograms. The plots are used to identify the point at which high grade tails separate from the main populations through the upper percentiles. The grades deemed as outliers were spatially reviewed to determine if they were truly outliers or pockets of anomalously high grades. In some cases, the core was examined to better understand the local context of the high grades.

The resulting outlier strategy included both grade capping and outlier restriction and was done in two steps: 1) grade capping of assays during compositing (Table 16); and 2) outlier restriction on the composites during block grade estimation (Table 17).
Table 16 - Grade Capping Levels

<table>
<thead>
<tr>
<th>Domain</th>
<th>Copper</th>
<th>Gold</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Threshold (%)</td>
<td>Cumul Freq</td>
</tr>
<tr>
<td>1103</td>
<td>7.40</td>
<td>&gt; 99%</td>
</tr>
<tr>
<td>1203</td>
<td>10.50</td>
<td>&gt; 99%</td>
</tr>
<tr>
<td>2103</td>
<td>4.70</td>
<td>&gt; 99%</td>
</tr>
<tr>
<td>2203</td>
<td>6.90</td>
<td>&gt; 99%</td>
</tr>
</tbody>
</table>

Table 17 – Outlier Restriction

<table>
<thead>
<tr>
<th>Passes 1 &amp; 2</th>
<th>Passes 3 &amp; 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Domain</td>
<td>Copper</td>
</tr>
<tr>
<td></td>
<td>Threshold (%)</td>
</tr>
<tr>
<td>1103</td>
<td>2.58</td>
</tr>
<tr>
<td>1203</td>
<td>2.40</td>
</tr>
<tr>
<td>2103</td>
<td>2.50</td>
</tr>
<tr>
<td>2203</td>
<td>4.50</td>
</tr>
</tbody>
</table>

14.5. Continuity Analysis

Experimental grade correlograms were modelled from the composited drill hole data for copper, gold, specific gravity, silver, carbon, sulphur, molybdenum, fluorine and uranium for the Low and High Grade domains. The nugget effect was obtained using “down the hole” correlograms.

The Low and High Grade domains were combined to produce larger datasets for analysis. Table 18 shows the resulting correlogram models for copper and gold.

Table 18 – Salobo Variography

<table>
<thead>
<tr>
<th>Element</th>
<th>Domain</th>
<th>Direction</th>
<th>Rotation ADA</th>
<th>Nugget</th>
<th>S1</th>
<th>R1</th>
<th>S2</th>
<th>R2</th>
<th>S3</th>
<th>R3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>1103 &amp; 1203</td>
<td>1</td>
<td>140</td>
<td>0.29</td>
<td>0.5</td>
<td>30</td>
<td>0.11</td>
<td>70</td>
<td>0.1</td>
<td>650</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2</td>
<td>-90</td>
<td></td>
<td></td>
<td>35</td>
<td></td>
<td>140</td>
<td></td>
<td>700</td>
</tr>
<tr>
<td></td>
<td></td>
<td>3</td>
<td>50</td>
<td></td>
<td></td>
<td>15</td>
<td></td>
<td>50</td>
<td></td>
<td>70</td>
</tr>
<tr>
<td>Cu</td>
<td>2103 &amp; 2203</td>
<td>1</td>
<td>110</td>
<td>0.29</td>
<td>0.3</td>
<td>30</td>
<td>0.32</td>
<td>60</td>
<td>0.09</td>
<td>250</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2</td>
<td>-90</td>
<td></td>
<td></td>
<td>10</td>
<td></td>
<td>45</td>
<td></td>
<td>400</td>
</tr>
<tr>
<td></td>
<td></td>
<td>3</td>
<td>20</td>
<td></td>
<td></td>
<td>10</td>
<td></td>
<td>50</td>
<td></td>
<td>70</td>
</tr>
<tr>
<td>Au</td>
<td>1103 &amp; 1203</td>
<td>1</td>
<td>140</td>
<td>0.49</td>
<td>0.2</td>
<td>20</td>
<td>0.21</td>
<td>40</td>
<td>0.1</td>
<td>400</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2</td>
<td>-90</td>
<td></td>
<td></td>
<td>20</td>
<td></td>
<td>60</td>
<td></td>
<td>700</td>
</tr>
<tr>
<td></td>
<td></td>
<td>3</td>
<td>50</td>
<td></td>
<td></td>
<td>10</td>
<td></td>
<td>20</td>
<td></td>
<td>85</td>
</tr>
<tr>
<td>Au</td>
<td>2103 &amp; 2203</td>
<td>1</td>
<td>110</td>
<td>0.49</td>
<td>0.35</td>
<td>30</td>
<td>0.11</td>
<td>100</td>
<td>0.05</td>
<td>300</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2</td>
<td>-90</td>
<td></td>
<td></td>
<td>15</td>
<td></td>
<td>60</td>
<td></td>
<td>500</td>
</tr>
<tr>
<td></td>
<td></td>
<td>3</td>
<td>20</td>
<td></td>
<td></td>
<td>25</td>
<td></td>
<td>60</td>
<td></td>
<td>80</td>
</tr>
</tbody>
</table>

Note: ADA = azimuth dip azimuth, S = sill and R = range

14.6. Block Modeling

14.6.1. Dimensions

A partial (percent) block model was generated in GEMS™ with the dimensions outlined in Table 19. The model is rotated 21.27º clockwise so that block model X axis lies along
the general strike of N111.27E. Blocks were assigned percent volumes using the four domain wireframes (1103, 2103, 1203 and 2203).

Table 19 - Block Model Origin

<table>
<thead>
<tr>
<th>Axis</th>
<th>Origin*</th>
<th>Block size (m)</th>
<th>No of Blocks</th>
<th>Model Extension (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>X</td>
<td>548,540</td>
<td>30</td>
<td>174</td>
<td>5,220</td>
</tr>
<tr>
<td>Y</td>
<td>9,359,800</td>
<td>30</td>
<td>75</td>
<td>2,250</td>
</tr>
<tr>
<td>Z</td>
<td>547.5</td>
<td>15</td>
<td>67</td>
<td>1,005</td>
</tr>
</tbody>
</table>

*Origin in GEMS is defined as top left corner of the block model

14.6.2. Boundary Conditions

Table 20 details the boundary conditions that were applied to the model with respect to the sharing of composites between domains. Between the Low Grade domains (1103 and 2103) soft boundaries were used during the estimation of Cu, Au, Ag and S grades and hard boundaries for density and all other elements. Hard boundaries were used between High Grade domains 1203 and 2203 in estimating density and all grades.

Table 20 - Boundary Conditions

<table>
<thead>
<tr>
<th>Domain</th>
<th>Cu</th>
<th>Au</th>
<th>Ag</th>
<th>S</th>
<th>Density</th>
<th>C</th>
<th>F</th>
<th>Mo</th>
<th>U</th>
</tr>
</thead>
<tbody>
<tr>
<td>1103</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1103</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2103</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2103</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1203</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1203</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2203</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2203</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

14.6.3. Block Estimation

Block model interpolation of all variables was done using the Isatis® software applying the various parameters described above. Block grades were estimated using the OK algorithm and the parameters described in the sections above. Blocks were estimated during three successive passes of OK and then a final fourth pass was done to estimate blocks that were not informed during the first three passes. Blocks estimated during the fourth pass were not included in the classified resources but rather are intended for use in defining exploration targets during future drill programs. Block discretization was set to 5 x 5 x 5 discretization points for the 30 m x 30 m x 15 m blocks. The resulting block model was then imported into GEMS™.

The nuggets, nested sills and ranges shown in Table 18 were applied in the OK interpolation, as well as, the outlier restrictions and boundary conditions described above. The rotation angles and dimensions of the search ellipses were based on the correlograms. The same angles were used for all four OK interpolation passes but increasingly larger search ellipse dimensions were used with each pass. Table 21 details the search ellipse dimensions for copper and Table 22 for gold.
Density values were assigned according to lithology for the blocks outside of the Low and High Grade domains. The average density for each lithology was based on the mean of the SG measurements for each specific lithology. Block density within the Low and High Grade Domains was estimated by OK of the SG measurements.

14.6.4. **Classification Coding**

Classification of blocks was initially assigned according to the pass in which the blocks were estimated. Blocks estimated during pass 1 were coded as Measured, pass 2 as Indicated and pass 3 as Inferred. Subsequently, this automated classification was adjusted to recode any anomalous blocks situated in areas of common category.

14.6.5. **Model Validation**

The following methods were used to validate the block grade estimates:

- Global mean comparison of mean composite, OK and Nearest Neighbor (NN) block grade estimates
• Visual inspection of the composite and block model grades
• Swath plots of OK versus NN block grades on a series of sections and plans throughout the deposit

Table 23 shows a global comparison mean, standard deviation and CV between the composites, OK block grades and NN block grades. For both copper and gold the OK block grades compare very well to the composites and NN block grades. For domains 1203 and 2203 the OK copper and gold grades are within 2% of the NN grades.

Table 23 - Global Mean Analysis

<table>
<thead>
<tr>
<th>Element</th>
<th>Domain</th>
<th>Count Data</th>
<th>Composite Mean</th>
<th>St Dev</th>
<th>CV</th>
<th>OK block grades</th>
<th>Count Mean</th>
<th>St Dev</th>
<th>CV</th>
<th>NN block grades</th>
<th>Count Mean</th>
<th>St Dev</th>
<th>CV</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu %</td>
<td>1103</td>
<td>17,730</td>
<td>0.44</td>
<td>0.35</td>
<td>0.8</td>
<td>41,469</td>
<td>0.48</td>
<td>0.16</td>
<td>0.3</td>
<td>41,469</td>
<td>0.46</td>
<td>0.20</td>
<td>0.4</td>
</tr>
<tr>
<td></td>
<td>1203</td>
<td>11,504</td>
<td>1.31</td>
<td>0.89</td>
<td>0.7</td>
<td>19,976</td>
<td>1.20</td>
<td>0.28</td>
<td>0.2</td>
<td>19,976</td>
<td>1.21</td>
<td>0.46</td>
<td>0.4</td>
</tr>
<tr>
<td></td>
<td>2103</td>
<td>6,295</td>
<td>0.46</td>
<td>0.41</td>
<td>0.9</td>
<td>21,020</td>
<td>0.47</td>
<td>0.14</td>
<td>0.3</td>
<td>21,020</td>
<td>0.45</td>
<td>0.22</td>
<td>0.5</td>
</tr>
<tr>
<td></td>
<td>2203</td>
<td>4,511</td>
<td>1.31</td>
<td>0.86</td>
<td>0.7</td>
<td>10,772</td>
<td>1.27</td>
<td>0.32</td>
<td>0.3</td>
<td>10,772</td>
<td>1.26</td>
<td>0.51</td>
<td>0.4</td>
</tr>
<tr>
<td>Au ppm</td>
<td>1103</td>
<td>17,732</td>
<td>0.20</td>
<td>0.25</td>
<td>1.3</td>
<td>41,469</td>
<td>0.20</td>
<td>0.10</td>
<td>0.5</td>
<td>41,469</td>
<td>0.20</td>
<td>0.14</td>
<td>0.7</td>
</tr>
<tr>
<td></td>
<td>1203</td>
<td>11,503</td>
<td>0.77</td>
<td>0.75</td>
<td>1.0</td>
<td>19,976</td>
<td>0.71</td>
<td>0.37</td>
<td>0.5</td>
<td>19,976</td>
<td>0.72</td>
<td>0.53</td>
<td>0.7</td>
</tr>
<tr>
<td></td>
<td>2103</td>
<td>6,296</td>
<td>0.17</td>
<td>0.24</td>
<td>1.5</td>
<td>21,020</td>
<td>0.18</td>
<td>0.12</td>
<td>0.7</td>
<td>21,020</td>
<td>0.16</td>
<td>0.14</td>
<td>0.9</td>
</tr>
<tr>
<td></td>
<td>2203</td>
<td>4,511</td>
<td>0.66</td>
<td>0.83</td>
<td>1.3</td>
<td>10,772</td>
<td>0.57</td>
<td>0.29</td>
<td>0.5</td>
<td>10,772</td>
<td>0.56</td>
<td>0.41</td>
<td>0.7</td>
</tr>
</tbody>
</table>

Visual inspection of block and composite grades on plans and sections showed good correlation between the input data and output values. No obvious discrepancies were noted.

To test the local estimation accuracy for each domain, swath plots were created comparing OK block grades versus NN block grades. These plots consist of narrow slices generated through the deposit along northing, easting and elevation directions. All domains show good correspondence between the OK and NN block estimates for both copper and gold with the OK grades being somewhat smoother as expected from the effects of the kriging interpolation. Portions of the graphs where the block grades deviate are generally associated with areas of low data. Figure 35 and Figure 36 are swath plots for copper and gold respectively for domain 1203 by elevation. Both figures show the OK grades corresponding very well, particularly through the areas of higher data density as indicated by the count bars.
14.7. Resource Model Pit Optimization

Mineral Resources exclusive of Mineral Reserves that are amenable to open pit mining methods at Salobo represent sulphide mineralization that is adjacent to the current
Mineral Reserve pit plus Inferred Mineral Resources within the Mineral Reserve pit. There are no oxide Mineral Resources.

The Mineral Resource estimates were prepared by Vale staff using the following design approach:

- Determine reasonableness of Mineral Resource pit extents, such as impact on planned mine infrastructure (waste lay down areas, processing facilities); distribution of deleterious mineral; adequateness of current waste storage capacity.
- Consider a cutoff grade of 0.253% copper equivalent (CuEq), consistent with Salobo's cutoff grade used for Mineral Reserves reported in 2017.
- Prepare a provisional extension of the LoMP production schedule to include material above this cutoff grade.
- Apply metal price and exchange rate assumptions to forecast cash flows, including appropriate provision for sustaining capital and operating costs.
- Scheduling waste and mineralized material.
- Determine if the Mineral Resource estimates demonstrate a positive cash flow.

External mining dilution and mine loss were not applied. Table 24 summarizes the technical and economic parameters used for optimizing the Mineral Resource pit. Figure 37 is an isometric view of the resulting resource pit.

Table 24 - Mineral Resource Open Pit Optimization Assumptions

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper sale price</td>
<td>$US/lb.</td>
<td>$2.86</td>
</tr>
<tr>
<td>Gold sale price</td>
<td>$US/oz</td>
<td>$1,200</td>
</tr>
<tr>
<td>Exchange rate</td>
<td>$BRL/$US</td>
<td>3.8</td>
</tr>
<tr>
<td>Mining method</td>
<td></td>
<td>Open Pit</td>
</tr>
<tr>
<td>Cutoff</td>
<td>%CuEq.</td>
<td>0.253</td>
</tr>
<tr>
<td>Mineability</td>
<td>%</td>
<td>100%</td>
</tr>
<tr>
<td>Dilution</td>
<td>%</td>
<td>3%</td>
</tr>
<tr>
<td>Mine production rate – ore</td>
<td>M tonnes/year</td>
<td>40</td>
</tr>
<tr>
<td>Mine production rate – waste</td>
<td>M tonnes/year</td>
<td>86</td>
</tr>
<tr>
<td>Mine full operating cost</td>
<td>$/tonne mined</td>
<td>3.37</td>
</tr>
<tr>
<td>Mine sustaining capital cost</td>
<td>$/tonne mined</td>
<td>0.56</td>
</tr>
<tr>
<td>Overall processing cost</td>
<td>$/tonne ore</td>
<td>7.91</td>
</tr>
<tr>
<td>Site G&amp;A</td>
<td>M $US/year</td>
<td>38</td>
</tr>
<tr>
<td>Overall processing Cu recovery</td>
<td>%</td>
<td>-0.023*(1/Cu)+0.9023</td>
</tr>
<tr>
<td>Overall processing Au recovery</td>
<td>%</td>
<td>(2.56*(Au)+64.9)/100</td>
</tr>
</tbody>
</table>
14.8. **Classification of Mineral Resources**

Mineral Resource model blocks are classified as Measured, Indicated or Inferred Mineral Resources, in accordance with CIM guidelines (CIM, 2014). Vale’s long-term mine planning and design process then converts Measured and Indicated Mineral Resources within the current LoMP open pit design into Proven and Probable Mineral Reserves, respectively.

Therefore, Mineral Resources at Salobo Operations are stated exclusive of Mineral Reserves. Mineral Resources at Salobo Operations thus comprise:

- Measured, Indicated and Inferred Mineral Resources outside, but adjacent to, the current LoMP open pit design, that Vale considers to have reasonable prospects for economic extraction based upon it’s analysis of an optimized pit shell as described above.

- Inferred Mineral Resources located within the current LoMP open pit design that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. In the LoMP open pit production schedule, this Inferred Resource material is planned as waste material. If during the mining process this material is confirmed as ore as per blast holes analysis results, this would be defined as ore for the short-term grade control purposes.

Mineral resource estimates (Table 11) represent in-situ tonnages and grades that take into account the minimum block size that can be selectively extracted. Mining recovery has not been applied to the Mineral Resource estimates but a factor for mining dilution is included in the reserve estimation. The Mineral Resources estimated in this statement are consistent with the requirements outlined in NI 43-101 and the Vale Base Metals 2017 Guidelines and Standards for MRMR Reporting.

A comparison of the Mineral Resources from 2016 to 2017 is presented in Table 25 and summary of the changes in Table 26. For Measured and Indicated resources, there was a 10% decrease in tonnes. However, the majority of these changes were due to conversion to Proven and Probable Mineral Reserves. Inferred Resource tonnes decreased by 9% as a result of re-evaluation of the Mineral Resource and Mineral Reserve pits.

<table>
<thead>
<tr>
<th></th>
<th>2016</th>
<th>2017</th>
<th>% Diff</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Mea</td>
<td>Ind</td>
<td>M&amp;I</td>
</tr>
<tr>
<td><strong>Within Current LoMP Open Pit</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M Tonnes</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Cu %</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td><strong>Adjacent to Current LoMP Open Pit</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M Tonnes</td>
<td>37.3</td>
<td>190.7</td>
<td>228.0</td>
</tr>
<tr>
<td>Cu %</td>
<td>0.75</td>
<td>0.62</td>
<td>0.64</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>0.44</td>
<td>0.31</td>
<td>0.33</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M Tonnes</td>
<td>37.3</td>
<td>191</td>
<td>228.0</td>
</tr>
<tr>
<td>Cu %</td>
<td>0.75</td>
<td>0.62</td>
<td>0.64</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>0.44</td>
<td>0.31</td>
<td>0.33</td>
</tr>
</tbody>
</table>
Table 26 - Changes to Mineral Resources from 2016 to 2017

<table>
<thead>
<tr>
<th></th>
<th>M Tonnes</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>2016 Measured Mineral Resources</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less mining (includes forecast to year end)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less Mineral Resources converted to proven Mineral Reserves</td>
<td>(4.3)</td>
<td>1.00</td>
<td>0.59</td>
</tr>
<tr>
<td>Less conversion to indicated Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to inferred Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less downgrade to exploration target</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less stockpile reclaim</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less sterilization</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Re-evaluation</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus Measured Mineral Resources reclassified from Mineral Reserves</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus upgrade from Inferred Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus new Measured Mineral Resources from drilling</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>33.0</td>
<td>0.72</td>
<td>0.42</td>
</tr>
<tr>
<td><strong>2016 Indicated Mineral Resources</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less mining (includes forecast to year end)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Proven Mineral Reserves</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Probable Mineral Reserves</td>
<td>(12.0)</td>
<td>0.55</td>
<td>0.24</td>
</tr>
<tr>
<td>Less upgrade to Measured Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Inferred Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less downgrade to exploration target</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less sterilization</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less re-evaluation</td>
<td>(7.6)</td>
<td>0.65</td>
<td>0.32</td>
</tr>
<tr>
<td>Plus reclassified from Mineral Reserves</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus re-categorized from Measured Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus upgrade to Indicated from Inferred Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus new Indicated Mineral Resources from drilling</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>171.1</td>
<td>0.62</td>
<td>0.31</td>
</tr>
<tr>
<td><strong>2017 Measured Mineral Resources</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less mining (includes forecast to year end)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Proven Mineral Reserves</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Probable Mineral Reserves</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less upgraded to Measured Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to Indicated Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less conversion to exploration target</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less sterilization</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Less re-evaluation</td>
<td>(16.4)</td>
<td>1.14</td>
<td>0.51</td>
</tr>
<tr>
<td>Plus re-categorized from M&amp;I Mineral Resources</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plus new Mineral Resources from drilling (upgrade from ET)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>175.7</td>
<td>0.5</td>
<td>0.2</td>
</tr>
</tbody>
</table>

14.10. **Comments on Section 14**

In the opinion of the QPs, the Mineral Resources have been estimated and classified in accordance with 2014 CIM Definition Standards.

To the extent known to the QPs, there are no known environmental, permitting, legal, title related, taxation, socio-political or marketing issues that could materially affect the Mineral Resource estimation that are not documented in this Report.

As detailed in Section 15.7, reconciliation work has shown that the block model estimates are accurately predicting production.
15. Mineral Reserve Estimates

15.1. Mineability and Dilution

Dilution and mineability have both been applied in the conversion of Mineral Resources to Mineral Reserves. Mining dilution is the unavoidable mixture of different quality materials during the operation and can be split in two forms: planned (or internal) mining dilution and unplanned (or external) mining dilution. Mining recovery is the failure to recover all predicted ore in the operation, which can be the result of mining dilution but also of several other factors.

Internal or planned dilution represents zones of mineralization below the cutoff grade that are unavoidably mined along with the mineralization above the cutoff grade due to the selectivity of the Selective Mining Unit (SMU). It is related to the production rate, equipment size, and, in open pit operations, to the size of the average-grade-polygons delivered to the operations to be mined. Planned dilution is included in the Mineral Resource tonnage and grade through the regularizing of the block model to the SMU size.

External or unplanned dilution is the result of introducing different quality material within the planned ore via blasting and equipment operation. External dilution has been incorporated into the conversion of resources to reserves by increasing tonnes by 4% with zero grade.

Mining recovery is a factor that decreases the ore tonnage predicted by the in-situ geological model, by failing to extract it as ore. In 2017, an average mining recovery factor of 100% in the conversion of Mineral Resource tonnage and grade to Mineral Reserve tonnage and grade was used.

In 2018, an action plan to fully understand the dilution-recovery issue will be concluded and new dilution and recovery factors will be calculated, preferentially by distinct grade ranges to be used in future mine plans.

15.2. Pit Optimization

The last pit optimization update was executed in 2016 and optimized shells were generated using Whittle Four-XTMv4.4. software and updated metal prices, recoveries, geotechnical information, and costs as described in the following sections. After detailed pit optimization the revenue factor 1.0 shell was chosen to guide the redesign of the ultimate pit. A pit optimization updated will be completed in 2018.

15.2.1. General Assumptions

The topographic surface was adjusted to account for the expected mining of the open pit phases and construction of waste dumps to December 31, 2017.
Optimized pits were not constrained by any surface infrastructure as it is all located beyond the economic pit limit.

A discount rate of 8% and an assumed mining decent rate of five benches per year were applied during the optimization process.

15.2.2. Mineral Resource Model

The Geology Department created an updated Mineral Resource model in 2017 as described in Section 14.6. The Mineral Resource model is a partial model that calculates the portion of the blocks within the low and high-grade domains. This partial model was then regularized to a whole block model. The whole block model uses a 30 m x 30 m x 15 m block size which adequately represents the amount of internal dilution currently experienced in mining the Salobo deposit.

The other modifying factor used is the incorporation of 4% unplanned mining dilution in the mining plan scheduling. Mining recovery of 100% is assumed during the pit optimization.

15.2.3. Mining Costs

Mining operating costs were updated based on current costs and exchange rates. For pit optimization the base mining costs at the 250 bench is $3.54/t mined for fresh rock and $3.20/t mined for saprolite, with an average overall mining cost of $3.37/t. The mining unit costs were increased by $0.00412/t for each 15 m bench above the 250 bench and increased by $0.0440/t for every bench below the 250 bench.

15.2.4. Processing Costs

Processing operating costs were updated by applying updated costs and exchange rates to the actual costs and consumption experienced in the mill. The process cost applied in pit optimization is a constant $7.91/t milled, including the processing sustaining costs.

The processing cost does not include any incremental drilling, blasting, loading, hauling, or ore control costs. This has the effect of not increasing the marginal cutoff grade. The processing cost does not include any tailings expansion or other sustaining capital costs.

15.2.5. Recovery

Copper and gold recovery is estimated based the empirical equations shown below:

Equation 1 - Copper Recovery

\[
\text{Copper Recovery} = 90.23\% - \frac{2.3\%}{\text{Copper Grade}}
\]
Equation 2 - Gold Recovery

\[ \text{Gold Recovery} = 64.9\% + 2.56\% \times \text{Gold Grade} \]

To calculate payable metal average payable factors of 96.7\% for copper and 93.94\% for gold were applied to the recovered metal.

15.2.6. Over Head Costs

General and administrative operating costs were based upon current operating costs and modeled at a constant rate of $1.60/t milled.

15.2.7. Refining, Freight, and Royalties

Concentrate was modeled with a copper grade of 38\%, 9.5\% moisture, and a 0.5\% loss in transit.

Refining costs inclusive of any penalties are based on current contracts and modeled as a cost of $0.57 / lb of copper and $0.52 / oz of gold.

Freight consists of road transport to Parauapebas, handling and load out at Parauapebas, rail transport to São Luis, handling and load out at São Luis, and overseas transport. The total freight cost is $0.103 / lb copper or $77.66 / wet tonne of concentrate.

There are no 3rd party royalties that are applicable to the property.

15.2.8. Sustaining Capital

Sustaining capital was divided between the mine, process plant, and G&A. The sustaining capital was based on the estimated capital requirements to execute the life of mine plan. The sustaining capital for the mine is $0.56/t mined, the sustaining capital for the mill is $0.53/t milled, and $0.02/t milled for the G&A. The sustaining capital was added to the mining, processing, and G&A operating costs for the Whittle optimizations.

15.2.9. Geotechnical Assumptions

The overall wall slopes used in the pit optimization are based on the sectors shown in Figure 39 but are flatter then shown in Table 27 as they include an allowance for the access ramps. The overall angles used range from 48 to 52 degrees.

15.3. Selective Mining Unit

The SMU was updated in 2016 to represent the actual selectivity achieved by the mine. The original SMU of 15 m x 15 m x 15 m was increased to 30 m x 30 m x 15 m. The
average blast hole pattern in ore is 4.5 m x 4.5 m, resulting in 44 holes per block. This provides good coverage of blast hole samples for grade control purposes.

15.4. Geotechnical Considerations

Salobo has been actively mining for over seven years and to date there has not been any significant wall failures. The pit walls have been monitored continually since 2014 by interferometry radar. When some movement has been detected preventative measures have been successfully implemented to avoid risks to the operators.

Operations have encountered significant spillage of rock down the temporary wall between a higher and lower phase leading to both operational and safety concerns. To improve conditions on the lower phase, the pit wall designs on Phases 4 and 5 have been modified to increase the catchment bench size. Where double benching with catch berms on 30 m vertical intervals had been designed the new design increases the catch berm from 12.5 m to 20 m wide. Where single benching had previously been designed the new design is for double benching with 20 m catch berms on 30 m vertical intervals instead of 8 m wide catch berms on 15 m vertical intervals as shown in Figure 38.

![Figure 38 - Revised Wall Designs](image)

Phases 6 through Phase 8 (the ultimate pit) use the original wall design as the phases are designed and sequenced to avoid one active mining area directly above another. The original wall designs used for Phases 6 through 8 are shown in show in Table 27 and Figure 39.
### Table 27 – Geotechnical Design Sectors for Salobo Mine

<table>
<thead>
<tr>
<th>Sector</th>
<th>Vertical Berm Spacing (m)</th>
<th>Face Angle (degrees)</th>
<th>Berm Width (m)</th>
<th>Inter Ramp Angle (degrees)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Saprolite and semi-weathered material</td>
<td>7.5 or 15</td>
<td>52</td>
<td>10</td>
<td>35</td>
</tr>
<tr>
<td>Sector I</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>I</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>IA</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>IB</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>IC</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>Sector II</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>II</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>IIA</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>IIB</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>Sector III</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>III</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>Sector IV</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>IV</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>Sector V</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>V</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>VA</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>Sector VI</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>VI</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>A</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>B</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>C</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
<tr>
<td>D</td>
<td>15</td>
<td>70</td>
<td>8</td>
<td>48.1</td>
</tr>
<tr>
<td>Sector VII</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>VII</td>
<td>30</td>
<td>70</td>
<td>12.5</td>
<td>52</td>
</tr>
</tbody>
</table>

### Figure 39 – Geotechnical Design Sectors

A review of the geomechanical model of the Salobo Pit is budgeted for 2018. The scope of work has been defined but a consulting firm has not yet been selected.

The construction of the waste dump and stockpiles are regularly monitored with particular attention to the placement of weathered rock and soil.
Geotechnical inspections and monitoring of the tailings dam and other water dams have been completed according to the new Brazilian Governmental Regulation (DNPM Act 70.389/17). There is an action plan to comply with all new dam regulations, including the review of Dam Safety Procedures (Safety Plan and Emergency Action Plan).

The first increase in height of the tailings dam was finished in December 2017. Regular monitoring of the tailings placement and water balance is conducted to ensure that the facility operates within design parameters.

15.5. **Cutoff Grade**

The marginal cutoff grade (GCu) is applied to a copper equivalent grade (CuEq). The method for calculating the copper equivalent was developed for the purpose of determining a mining cutoff grade strategy based on 2017 detailed costs update. The calculation is based on costs, prices and recoveries of the plant revised on Vale’s 2017 budget cycle. No material changes were verified to the cutoff grade value considering 2017 updated values. The general expression for the equivalent copper grade is shown in Equation 3 and the marginal cutoff grade for defining ore and waste is shown in Equation 4 using the parameters shown in Table 28.

**Equation 3– Copper Equivalent Calculation**

$$\text{CuEq} = \text{Cu} + \left( \frac{\text{Au} \times (\text{PrAu} - \text{CvAu}) \times \text{RCAu} \times \text{RFAu}}{(\text{PrCu} - \text{CvCu}) \times \text{RCCu} \times \text{RFCu}} \right) \times \frac{31.103}{2,204.62}$$

**Equation 4 - Break Even Cutoff Grade Calculation**

$$\text{GCu} = \frac{\text{CP} + \text{CG&A} + \text{CAM}}{(\text{PrCu} - \text{CvCu}) \times \text{RCCu} \times \text{RFCu} \times (100 - \text{PT}) \times 2,204.62}$$

**Table 28 – 2017 Cutoff Calculation Parameters**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>Copper Grade (%)</td>
<td>Variable</td>
</tr>
<tr>
<td>Au</td>
<td>Gold Grade (g/t)</td>
<td>Variable</td>
</tr>
<tr>
<td>PrCu</td>
<td>Copper Price ($/lb.)</td>
<td>$2.99</td>
</tr>
<tr>
<td>PrAu</td>
<td>Gold Price ($/oz.)</td>
<td>$1,275</td>
</tr>
<tr>
<td>CvCu</td>
<td>Copper Selling Cost ($/lb)</td>
<td>$0.57</td>
</tr>
<tr>
<td>CvAu</td>
<td>Gold Selling Cost ($/oz.)</td>
<td>$0.52</td>
</tr>
<tr>
<td>RocCu</td>
<td>Copper Flotation Recovery (%)</td>
<td>Calculated (see Equation 2)</td>
</tr>
<tr>
<td>RocAu</td>
<td>Gold Flotation Recovery (%)</td>
<td>Calculated (see Equation 1)</td>
</tr>
<tr>
<td>RfCu</td>
<td>Copper Smelting Recovery (%)</td>
<td>96.70%</td>
</tr>
<tr>
<td>RfAu</td>
<td>Gold Smelting Recovery (%)</td>
<td>93.94%</td>
</tr>
<tr>
<td>CP</td>
<td>Total Cost of Processing($/t)</td>
<td>$9.74</td>
</tr>
<tr>
<td>CG&amp;A</td>
<td>Total Cost of G&amp;A ($/t)</td>
<td>$1.26</td>
</tr>
<tr>
<td>CAM</td>
<td>Additional Cost of Mining Ore ($/t)</td>
<td>$0</td>
</tr>
<tr>
<td>PT</td>
<td>Loss of Concentrate in Transport</td>
<td>0.50%</td>
</tr>
<tr>
<td>31.103</td>
<td>Conversion Factor troy oz. to gram</td>
<td>31.13</td>
</tr>
<tr>
<td>2,204.62</td>
<td>Conversion Factor tonne to lb.</td>
<td>2,204.62</td>
</tr>
</tbody>
</table>

A marginal cutoff grade of 0.253% was calculated for copper and is applied in equivalent amounts of copper. This cutoff is used for the determination of Mineral Reserves and mining sequencing.
At the Salobo Operations, a strategy to stockpile lower grade ore was adopted to maximize the NPV of the project which results in an elevated cutoff being used until 2045. Currently 118 M tonnes of ore or around 5 years of mill feed is stored in the Low Grade Stockpile.

15.6. Mineral Reserve Tabulation

The long-term mine planning, pit optimization and design process converts the open pit measured and indicated Mineral Resources into proven and probable Mineral Reserves. All Inferred material contained within the ultimate pit is treated as waste.

The Vale 2017 block model forms the basis of Salobo’s Mineral Reserves and Mineral Resources. Mineral reserve estimates are derived from this block model by applying the appropriate technical and economic parameters, within the 2017 ultimate pit design. Key parameters are calculated separately for each discrete mining block, based on geometry and mining method, as detailed in this section of the technical report. Copper equivalent grades were calculated using metal prices of $2.99/lb for copper and $1,275/oz. for gold in 2017.

The cutoff grade of 0.253% CuEq applied to the 2017 block model reflects Vale’s forecasts of direct operating costs, recoveries and metal prices, etc. The Mineral Reserve includes planned mining dilution of 4% with 100% recovery. Table 29 details the 2017 Mineral Reserves for the Salobo Operations.

<table>
<thead>
<tr>
<th>Classification</th>
<th>M Tonnes</th>
<th>Grades</th>
<th>Contained Metal</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Cu %</td>
<td>Au g/t</td>
</tr>
<tr>
<td>Proven</td>
<td>526.3</td>
<td>0.68</td>
<td>0.38</td>
</tr>
<tr>
<td>Stockpiles (Proven)</td>
<td>117.8</td>
<td>0.44</td>
<td>0.19</td>
</tr>
<tr>
<td>Probable</td>
<td>549.3</td>
<td>0.57</td>
<td>0.29</td>
</tr>
<tr>
<td>P&amp;P</td>
<td>1,193.4</td>
<td>0.61</td>
<td>0.32</td>
</tr>
</tbody>
</table>

Notes:
1. Mineral Reserve estimates were prepared by Mr. Wellington F. de Paula, an employee of Vale. The Qualified Person for the Mineral Reserve estimates is Mr. Cassio Diedrich, AusIMM-CP(Min), Technical Services General Manager, Vale Base Metals.
2. Mineral Reserves are classified as Proven and Probable Mineral Reserves based on the 2014 CIM Definition Standards.
3. Mineral Resources are reported above a copper equivalent cutoff of 0.253%, assuming $1,200 per ounce gold and $2.86 per pound copper.
4. Tonnages are rounded to the nearest 100,000 tonnes and grades are rounded to two decimal places.
5. Contained copper is reported as Imperial pound units and contained gold as troy ounces.

A comparison of the 2017 to 2016 Mineral Reserves estimates is provided in Table 30 and the changes between the 2016 and 2017 Mineral Reserves are shown in Table 31.
### Table 30 - Summary of Mineral Reserves from 2016 to 2017

<table>
<thead>
<tr>
<th></th>
<th>M Tonnes</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>2016</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven</td>
<td>572.5</td>
<td>0.71</td>
<td>0.38</td>
</tr>
<tr>
<td>Prob</td>
<td>554.6</td>
<td>0.59</td>
<td>0.30</td>
</tr>
<tr>
<td>P&amp;P</td>
<td>1,127.1</td>
<td>0.65</td>
<td>0.34</td>
</tr>
<tr>
<td><strong>2017</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven</td>
<td>526.3</td>
<td>0.68</td>
<td>0.38</td>
</tr>
<tr>
<td>Prob</td>
<td>549.3</td>
<td>0.57</td>
<td>0.29</td>
</tr>
<tr>
<td>P&amp;P</td>
<td>1,075.6</td>
<td>0.63</td>
<td>0.33</td>
</tr>
<tr>
<td><strong>% Diff P&amp;P</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Salobo Pit</td>
<td></td>
<td>-5%</td>
<td>-3%</td>
</tr>
<tr>
<td><strong>Stockpiles</strong></td>
<td></td>
<td>130%</td>
<td>13%</td>
</tr>
<tr>
<td>M Tonnes</td>
<td>51.2</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Cu %</td>
<td>0.39</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>0.17</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>623.7</td>
<td>0.68</td>
<td>0.36</td>
</tr>
<tr>
<td>M Tonnes</td>
<td>554.6</td>
<td>0.59</td>
<td>0.30</td>
</tr>
<tr>
<td>Cu %</td>
<td>0.68</td>
<td>0.64</td>
<td>0.34</td>
</tr>
<tr>
<td>Au (g/t)</td>
<td>0.36</td>
<td>0.30</td>
<td>0.33</td>
</tr>
</tbody>
</table>

### Table 31 - Changes to Mineral Reserves from 2016 to 2017

<table>
<thead>
<tr>
<th>2016 Proven Mineral Reserves</th>
<th>M Tonnes</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Less mining (includes forecast to year end)</td>
<td>(23.60)</td>
<td>0.95</td>
<td>0.66</td>
</tr>
<tr>
<td>Less re-evaluation(^1)</td>
<td>(26.55)</td>
<td>0.91</td>
<td>0.30</td>
</tr>
<tr>
<td>Less re-categorize to Probable Mineral reserves</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Less reclassification to Measured Mineral Resources</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Less downgrade to exploration target</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Less stockpile reclaims</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Plus re-evaluation</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Plus upgrade probable to Proven Mineral Reserves</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Plus Measured Mineral Resources converted to Mineral Reserves</td>
<td>3.93</td>
<td>0.89</td>
<td>0.67</td>
</tr>
<tr>
<td>Plus new Mineral Reserves</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Plus stockpile additions</td>
<td>6.65</td>
<td>0.47</td>
<td>0.21</td>
</tr>
<tr>
<td>2016 Probable Mineral Reserves</td>
<td>554.6</td>
<td>0.59</td>
<td>0.30</td>
</tr>
</tbody>
</table>

| Less mining (includes forecast to year end) | (17.3) | 1.05 | 0.39 |
| Less upgrade of Probable to Proven Mineral Reserves | - | - | - |
| Less reclassification to Indicated Mineral Resources | - | - | - |
| Less downgrade to exploration target | - | - | - |
| Less stockpile reclaims \(^1\) | - | - | - |
| *Plus re-evaluation | - | - | - |
| Plus re-categorize Proven to Probable Mineral Reserves | - | - | - |
| Plus Indicated Mineral Resources converted to Mineral Reserves | 12.0 | 0.55 | 0.24 |
| Plus new Mineral Reserves | - | - | - |
| Plus stockpile additions | - | - | - |
| 2017 Probable Mineral Reserves | 549.3 | 0.57 | 0.29 |

### Table 32 details the Mineral Reserves by mining phases.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Phase</th>
<th>M Tonnes</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven &amp; Probable</td>
<td>3</td>
<td>58.9</td>
<td>0.75</td>
<td>0.45</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>137.2</td>
<td>0.64</td>
<td>0.35</td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>104.9</td>
<td>0.68</td>
<td>0.32</td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>249.6</td>
<td>0.57</td>
<td>0.33</td>
</tr>
<tr>
<td></td>
<td>7</td>
<td>205.3</td>
<td>0.62</td>
<td>0.32</td>
</tr>
<tr>
<td></td>
<td>8</td>
<td>319.7</td>
<td>0.63</td>
<td>0.31</td>
</tr>
<tr>
<td>Stockpiles</td>
<td></td>
<td>117.8</td>
<td>0.44</td>
<td>0.19</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td>1,193.4</td>
<td>0.61</td>
<td>0.32</td>
</tr>
</tbody>
</table>

* - considers run of mine mined material and geological model changes.

### Table 32 - 2017 Mineral Reserves Estimate by Phase

Figure 40 is a long section through the middle of the orebody depicting the location of Proven and Probable Mineral Reserves within the 2017 reserve pit (red) and the
Measured, Indicated and Inferred Resources between the 2017 Mineral Reserve and Mineral Resource (green) pits. The figure also shows the location of Inferred Mineral Resources within the reserve pit at the northwest and southeast ends of the orebody. Future drilling will target the reserve pit Inferred Mineral Resource areas for conversion to Mineral Reserves.

**Figure 40 - 2017 MRMR Longitudinal Section (looking southwest)**

![Salobo Mine Longitudinal Section - LG00](image)

15.7. **Reconciliation**

15.7.1. **Method**

Vale’s reconciliation process at Salobo consists of computing mining call factors that compare tonnage, grade and metal at different measurement points along the process. The key measurement points are: long-range model (LTM), short-range model (STM), polygon model (POL), production (PRD), total ore sent to crusher (TSC), and processed ore (PO). For all measurement points, tonnage, grade and metal are quoted on a dry basis.

The following reconciliation factors are calculated at the mine and Figure 41 shows the comparison rationale for each:
• F1 = STM / LTM  
• F2 = PRD / STM  
• F3 = PO / TSC  
• M1 = POL / STM  
• M2 = PRD / POL  
• F4 = F1 x F2 x F3

Figure 41 - Production Reconciliation Schematic

**F1 factor**
- Compares long-range and short-range models and is performed between topographic surfaces. It allows an assessment of the uncertainty of the geological model boundaries by the evaluation of tonnage differences between the mineralization envelopes.
- Conditional simulations are used in this process to provide a better understanding of grade variability, mining call factors variability and grade estimation related to long-range grade uncertainty.

**F2 factor**
- Evaluates the mining operation by comparing the predicted tonnage and grade from the short-range model to those measured by the truck and crusher weighing scales.
**F3 factor**
- Compares the production tonnage and grades with those measured by the plant.

**M1 factor**
- Compares the short-range model with the polygon model, which consists of the polygons with average grades that are assigned to the operation to be mined. This factor shows what losses may occur due to selectivity by delivering polygons to the operation instead of SMU blocks. It provides a measure of planned dilution.

**M2 factor**
- Compares the production data measured by the truck and crusher weighing scales to the polygons received by the operation to be mined. This gives an indication of the unplanned dilution and mining recovery occurring within the mining process.

### 15.7.2. Results

Table 33 shows the various reconciliation factors calculated for 2016 and 2017 and descriptions after the table explain the results.

<table>
<thead>
<tr>
<th>Factor</th>
<th>2016 Tonnage</th>
<th>Cu (%)</th>
<th>Cu (t)</th>
<th>Au (g/t)</th>
<th>Au (t)</th>
<th>2017 Tonnage</th>
<th>Cu (%)</th>
<th>Cu (t)</th>
<th>Au (g/t)</th>
<th>Au (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>F1</td>
<td>1.10</td>
<td>0.96</td>
<td>1.06</td>
<td>0.93</td>
<td>1.03</td>
<td>1.12</td>
<td>0.99</td>
<td>1.11</td>
<td>1.00</td>
<td>1.12</td>
</tr>
<tr>
<td>F2</td>
<td>0.86</td>
<td>1.01</td>
<td>0.86</td>
<td>1.06</td>
<td>0.91</td>
<td>0.89</td>
<td>0.98</td>
<td>0.88</td>
<td>0.99</td>
<td>0.88</td>
</tr>
<tr>
<td>M1</td>
<td>0.99</td>
<td>0.99</td>
<td>0.98</td>
<td>1.01</td>
<td>1.00</td>
<td>0.98</td>
<td>0.99</td>
<td>0.97</td>
<td>0.99</td>
<td>0.97</td>
</tr>
<tr>
<td>M2</td>
<td>0.86</td>
<td>1.02</td>
<td>0.88</td>
<td>1.05</td>
<td>0.91</td>
<td>0.91</td>
<td>0.99</td>
<td>0.90</td>
<td>1.00</td>
<td>0.91</td>
</tr>
<tr>
<td>F3</td>
<td>1.02</td>
<td>1.00</td>
<td>1.02</td>
<td>1.06</td>
<td>1.08</td>
<td>1.00</td>
<td>0.99</td>
<td>1.00</td>
<td>1.06</td>
<td>1.06</td>
</tr>
<tr>
<td>F4</td>
<td>0.96</td>
<td>0.97</td>
<td>0.93</td>
<td>1.05</td>
<td>1.01</td>
<td>1.00</td>
<td>0.96</td>
<td>0.98</td>
<td>1.05</td>
<td>1.04</td>
</tr>
</tbody>
</table>

**F1 factor**
- Shows that mining (short-range model) yields 10% more tonnes than indicated in the long-range model in 2016 and 12% more in 2017. These differences are due to the large volume of low-grade mineralization at the waste contact that is not predicted by the long-range model. This issue was identified at the end of 2014 and the long-range interpretation was remodeled. However, since the only new information available is the blast holes only a small portion of the long-range model is updated. The infill diamond drill program initiated in 2017 is intended to reduce variance.

**F2 factor**
- This was historically not well controlled at Salobo. Since the beginning of the operations the F2 showed 15% to 20% lower tonnage in the production data than the short-range model. Between 2012 and 2016 this issue was attributed to the learning curve for the operations to deal with data tracking, weighing scales calibration, dispatch system, increasing fleet, planned and unplanned dilution and so on. The differences were not assumed to be high unplanned dilution/mining recovery, but rather a data tracking issue with, for example crusher weighing scales not yet in use.
In 2016, two improvements were done regarding modelling and reconciliation.
  - The SMU was increased from 15 m x 15 m x 15 m to 30 m x 30 m x 15 m after a complete study of planned dilution and blasting polygon sizes.
  - The production reconciliation process was modified to include two more factors (M1 and M2) which are a split of the F2 factor, designed to better control the grade in the operations and identify the source of the F2 tonnage difference.

**M1 factor**
- The 2016 and 2017 factors show that the new SMU of the short-range model correctly represents the blasting polygons used for delivering the ore for the operations, accounting for the planned dilution. The difference from the short-range model to the blasting polygon model is within the expected values.

**M2 factor**
- The M2 is responsible for most of the difference the F2 factor presents. In 2016, M2 showed 14% tonnage difference from the polygon model to the production data. In 2017, after identifying the source of the tonnage difference, an action plan was started to reduce difference. The action plan, currently being implemented, consists of a thorough revision of the dispatch tracking data and control along with a precise determination of the geology polygon contact. The M2 and consequently the F2 factors improved in 2017.
  - The remaining difference could be due to failures in the truck weighing scales, topography surveys, truck destination control, and truck spillage resulting in higher average payload applied to trucks. These issues are under investigation.

**F3 factor**
- The trucks are all equipped with fully functioning calibrated scales. At the end of 2017 all five weighing scales and five of the six metallurgical samplers of the process plant were installed, but not all of them are functional. Two weighing scales after the primary crushers were still being tested, and the metallurgical samplers at the concentrate and tailings are not being used yet. Nevertheless, the F3 results indicate a good reconciliation.

Table 34 lists the overall copper production reconciliation between the milled production with the material sent to the crushers. The estimated tonnage and copper grade of material sent to the crusher is derived from estimates based on detailed drilling and grade control mapping. The table shows the improvement since 2012 with the 2017 reconciliation within 1%.
15.8. Comments on Section 15

In the opinion of the QPs, Mineral Reserves have been estimated according to 2014 CIM Definition Standards and estimates are based on the most current knowledge, permit status and engineering and operational constraints. Mineral Reserve declaration is supported by a positive cashflow.

Continuous improvements in the dispatch controls and follow up process, will result in further improvement of the F2 reconciliation (mining recovery and dilution).

Review of the operational pit slope angles through geotechnical examination of the pit wall operation, design of pushbacks, and further geotechnical studies may provide support for steepening of some of the pit walls. The current geomechanical sectors are based on a limited amount of information and this should be further analyzed.

16. Mining Methods

Salobo mine utilizes standard open pit methods, developed in 15 m benches, with trucks and shovels. After drilling and blasting the material, cable shovels, large front-end loaders and hydraulic excavators are used to load this material. A fleet of 240 t and 360 t trucks are used to haul the waste material to waste dumps proximal to the pit or ore material to the primary crusher. Lower grade ore is stockpiled for later processing.

The mine planning objective is to mine the ore sequentially in mining phases, considering the largest possible vertical spacing between phases. The plan is to provide an approximately steady annual production of 24.0 million tonnes to the mill. The overall site layout is shown in Figure 47.

16.1. Pit and Phase Designs

During 2017, the ultimate pit was redesigned based on the 2016 Whittle pit optimization results and incorporating the revised pit wall designs.

The internal phases were also redesigned in 2017 with an eighth phase added to improve the LoMP schedule.
The revised phases and ultimate pit are shown in Figure 42.

16.2. Production Schedule

After estimating Mineral Reserves, a practical and executable production schedule is developed by short and long term mine planning teams. The ultimate pit has been subdivided into eight phases two of which have been mined out the remaining six phases form the basis of the life of mine plan. The Mineral Reserves for the remaining phases are shown in Table 35.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Phase</th>
<th>M Tonnes</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven &amp; Probable</td>
<td>3</td>
<td>58.9</td>
<td>0.75</td>
<td>0.45</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>137.2</td>
<td>0.64</td>
<td>0.35</td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>104.9</td>
<td>0.68</td>
<td>0.32</td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>249.6</td>
<td>0.57</td>
<td>0.33</td>
</tr>
<tr>
<td></td>
<td>7</td>
<td>205.3</td>
<td>0.62</td>
<td>0.32</td>
</tr>
<tr>
<td></td>
<td>8</td>
<td>319.7</td>
<td>0.63</td>
<td>0.31</td>
</tr>
<tr>
<td>Stockpiles</td>
<td></td>
<td>117.8</td>
<td>0.44</td>
<td>0.19</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>1,193.4</td>
<td>0.61</td>
<td>0.32</td>
</tr>
</tbody>
</table>

In general, the phases have been sequentially scheduled with a maximum ore plus waste production rate of 126 million tonnes per year feeding 24.0 million tonnes of ore to the processing plant. The initial 5 years of the mine plan is shown in Table 36 and the entire schedule is shown in Figure 43.
### Table 36 - 5 Year Plan

<table>
<thead>
<tr>
<th>Period</th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
<th>2021</th>
<th>2022</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore (kt)</td>
<td>60,072</td>
<td>52,108</td>
<td>56,156</td>
<td>42,212</td>
<td>51,512</td>
</tr>
<tr>
<td>Waste (kt)</td>
<td>58,227</td>
<td>69,274</td>
<td>65,440</td>
<td>77,960</td>
<td>65,793</td>
</tr>
<tr>
<td>Mine Production (kt)</td>
<td>118,300</td>
<td>121,382</td>
<td>121,596</td>
<td>120,172</td>
<td>117,305</td>
</tr>
<tr>
<td>Other Movement (kt)</td>
<td>22,533</td>
<td>21,477</td>
<td>24,144</td>
<td>24,018</td>
<td>17,695</td>
</tr>
<tr>
<td>Total Movement (kt)</td>
<td>140,833</td>
<td>142,859</td>
<td>145,740</td>
<td>144,190</td>
<td>135,000</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td>1.0</td>
<td>1.3</td>
<td>1.2</td>
<td>1.8</td>
<td>1.3</td>
</tr>
<tr>
<td>Plant feed (kt)¹</td>
<td>23,647</td>
<td>23,637</td>
<td>23,703</td>
<td>23,637</td>
<td>23,637</td>
</tr>
<tr>
<td>Plant feed (Cu %)</td>
<td>0.92</td>
<td>0.94</td>
<td>0.96</td>
<td>0.96</td>
<td>0.95</td>
</tr>
<tr>
<td>Plant feed (Au g/t)</td>
<td>0.61</td>
<td>0.57</td>
<td>0.61</td>
<td>0.55</td>
<td>0.52</td>
</tr>
</tbody>
</table>

Notes: ¹ Dry metric tonnes; Source: Five-year plan production.

² Average Salobo II plan 2023 to 2035 (in accordance to the total movement, after 2035 the total movement decrease).

---

*Figure 43 - Life of Mine Plan (Mineral Reserves Only)*

The 2017 LoMP includes an increase in the total movement in the first five years compared to the 2016 LoMP. The total movement was increased to compensate for increased geological losses incorporated in the 2017 block model revision and to improve (de-risk) the mining operating process.

The open pit mine life is approximately 28 years, ending in 2045. However, the process plant will continue to operate by reclaiming stockpiled material until 2067. Phasing of the open pit development and application of the cutoff grade strategy allows higher grade ore (above 0.90% Cu) to be processed in the initial years of the operation.
16.3. Grade Control

Grade control at Salobo Mine uses samples of drill cuttings collected from blastholes and applies procedures developed at the nearby Vale Sossego mine. All ore blastholes are sampled, and the grade control geologist determines which waste blastholes are sampled to ensure mineralization matches the interpretation in the geological model.

Surveyors measure drill hole collar locations using high precision global positioning system (GPS) equipment.

As described in Section 11.1.2, samples of drill cuttings are collected from the entire length of the blasthole, including the subdrill. The sample is homogenized and reduced to 2 kg using a Jones splitter before it is bagged and numbered prior to dispatch to the analytical laboratory.

Blasthole samples are analyzed at the Salobo laboratory and the results are forwarded to the geology department to be entered into the ore control database.

Ore polygons are defined, based on the assay results and taking account of where the blasted material was thrown. This information is uploaded to the GPS units of the operating shovels and loaders to guide the mucking operations.

A dispatch system is used to control the activities of all mine equipment, and compliance to the mine plan is monitored on a monthly basis. Adherence to the mine plan is recognized as key to achieving the overall production forecast.

16.4. Mining Equipment

The Salobo bulk mining operations primarily utilize large electric (rope) shovels for ore and waste production. Hydraulic shovels are used for the oxide saprolite and transition material where a lower ground pressure is required. Wheel loaders are used for miscellaneous clean up jobs and for backup of the shovels when needed.

A fleet of off-road haul trucks are used to transport the material to either the waste dump or the primary crusher stockpiles. Low and medium grade ore is stockpiled near the open pit. Cycle times for haulage calculations are determined for each mining period using the Mine Haul software.

The track dozers are assigned to maintain the production areas, waste dumps and cleaning up the benches. Wheeled dozers, road graders and water trucks complete the remainder of the auxiliary equipment fleet. Table 37 provides a summary of the mining fleet at the Salobo Operations. The equipment listed is used to develop, drill, blast/muck and haul ore from the active mining levels.

The equipment consists of electrical and diesel powered drills of 12 1/4”, 10” and 6 1/2” diameters, cable shovels 42 yd³ and 63 yd³, hydraulic excavators 38 yd³ and wheel
loaders of $33 \text{ yd}^3$ capacity. The diesel drills and the wheel loaders are mainly used for ore exploitation that needs more mobility, and electrical drills and cable shovels are used for waste removal and bulk ore portions. The wheel loaders were considered in the fleet to support the larger units in narrow areas and in opening new accesses. Komatsu 830E-AC (240t), Caterpillar 793 (240 t) and 797 (360 t) trucks were selected to haul all material within the pit.

The auxiliary units are bulldozers (D475A-2, Cat D11, D375A, Cat D10 and D6R); this equipment is necessary to maintain the production areas, waste piles, and cleaning the material on the benches. Wheeled tractors, motor graders and water trucks complete the auxiliary equipment fleet. Table 37 details the current fleet.

16.5. MANPOWER

The mine operates on a continuous schedule with three shifts per day of 8 hours each. Approximately 10 days per year are planned as lost production delays due to poor weather conditions (i.e., rain and fog).

Forecast mine manpower utilization takes into account delays for training, blast moves and other operational delays.

16.6. Ore Stockpiles & Waste Disposal

Low-medium grade ore and waste rock from the mine are stored in three locations along the perimeter of the pit as shown in Figure 42. The main waste rock dump is to the west of the pit and contains both oxidized and fresh rock. Geotechnical investigations were conducted to develop the dump design parameters. A 35% swell factor is used to compute the required storage volumes in the stockpiles and waste dumps.

Material is end-dumped in 20 m high lifts with 10 m berms between lifts. The bench face angles range from 32 to 35 degrees, depending on the angle of repose for the material (see Table 38). Including the berms, the overall slope of the dumps ranges from 2H:1V.
to 2.5H:1V. The resulting slopes were shown to have an estimated 1.5 factor of safety against large scale circular slip failures.

The waste materials and the low-medium grade ore have been characterized as having low acid rock drainage potential (Brandt, 2003). However, there is a concern with fluorine leaching from the finer grained oxidized materials (saprolite). Accordingly, mineralized saprolite material is encapsulated within the waste rock dump to control infiltration of surface water and minimize resultant leaching.

The long-term storage of the medium and low grade material in a tropical environment may lead to some oxidation of contained sulphide minerals, impacting recovery of metals during eventual processing of the stockpiles.

16.7. Mine Services

Water management (including pit dewatering, and control of runoff within the open pit and surrounding area) requires additional attention during the rainy season. With over 1.92 m of rainfall each year, sumps and pumps need to be well managed to maintain the roads and pit working surfaces. The Operation recognizes this and has allocated appropriate resources to this task.

During the dry season, dust control is maintained through the use of water trucks. Evaluations are on-going to determine the effectiveness of additives, such as calcium chloride for dust control.

16.8. Comments on Section 16

In the option of the QPs, the mining methods, equipment, overall design and the production rate assumptions used to develop the LoMP and Mineral Reserves are reasonable and achievable.

A technical analyses of Mineral Resources below the Mineral Reserve pit should be studied to determine if the low-grade stockpile reclaiming in the later years of operation can be delayed.
17. Recovery Methods

17.1. Process Flowsheet

The process flowsheet has evolved through the various study phases of the Project, incorporating the additional knowledge gained from metallurgical testwork and the relative importance of the identified lithologies in the Mineral Resource and Mineral Reserve estimates. In particular, the following stages of Project development contributed to the evolution of the retained flowsheet.

- The CVRD and Anglo American testwork program, from 1986–1987, provided the basis for a prefeasibility study completed by Bechtel in 1988. At this stage, fluorine contamination of the concentrate was recognized.

- The SMSA testwork program, culminating in a pilot plant campaign at the CRC, performed between 1993 and 1998, provided additional data for a final feasibility study completed by Bechtel.

- Locked-cycle flotation tests, flotation variability, and grinding studies, completed in 2003 and 2004, were used by Fluor Daniel to complete a second feasibility study in 2004, which evaluated production scenarios at 12 M/ta and 24 Mt/a.

- A trade-off study using high-pressure grinding rolls (HPGR) for tertiary crushing as an alternative to conventional semi-autogenous grinding (SAG), conducted from 2005–2006. The data thus collected were used by Kvaerner to prepare a trade-off study, from which the HPGR approach was adopted.

HPGR were retained instead of SAG mills because of the high magnetite (and copper) content of critical-size pebbles that would have been removed with the magnet protecting the pebble crushers, and therefore requiring additional re-handling (per Vale’s experience at Sossego). In addition, the relatively high ore hardness and its expected variability as different mixtures of ore lithologies are introduced as plant feed, would have caused high-frequency variability in plant throughput in a typical SAG mill–ball mill–pebble crusher (SABC) circuit.

Phase I of the Salobo plant (Salobo I) was designed to process 12 Mt/a of ore, to produce approximately 100 kt of copper-in-concentrate annually. Production commenced in June 2012.

The Salobo II plant permitted a doubling of the nominal plant throughput, to 24 Mt/a, with an annualized copper-in-concentrate production of approximately 200 kt. The Salobo II plant was commissioned in June 2014 and is basically a mirror-image of Salobo I, i.e. essentially two identical, parallel, production lines.

Salobo I was designed to operate 365 days per year, 24 hours per day and with a targeted 90% of actual operating time, accounting for availability and utilization. Salobo II started
operation in June, 2014 and is designed for a targeted up-time of 90%. The overall simplified process flow diagram is illustrated in Figure 44.

Apart from the inclusion of HPGR for tertiary crushing duty, ahead of ball milling, the circuit is conventional, but with the flotation cleaning circuit making extensive use of flotation columns, to reduce entrainment of F-bearing non-sulphide gangue minerals such as fluorite and biotite.

17.2. Plant Design

The whole plant is extensively instrumented. All signals are provided to a distributed control system (DCS), allowing for the remote activation and stoppage of equipment, as well as the monitoring of the status of process equipment and of the metallurgical performance of the plant. A manned control room is used to implement changes to the circuit, with the instructions relayed from floor supervisors via radio.

Run-of-mine ore at 2.5 m top size is hauled in 240 t trucks and crushed in one of two 60" x 89" primary gyratory crusher (600 kW motor), rated for 1,826 t/h each, to a product size distribution with 80% of the mass passing 152 mm while operated with an open-side setting (OSS) of 140 mm. The dump pocket capacity is equivalent to the volume of 2.5 trucks. Primary crushed ore is conveyed to a common crushed ore stockpile which has a live capacity of approximately 24,800 t and a total capacity of 73,400 t.

Four coarse ore stockpile reclaim feeders are used to feed onto the primary screen feed conveyor which feeds two operating double-deck vibrating screens. The screens have a 100 mm aperture top deck and 55 mm aperture bottom deck to yield and underflow product sizing of 80% passing 38 mm. Screen oversize is crushed in two MP-1000 cone crushers (746 kW motors) in a standard closed circuit. A third screen and crusher were added to the original two units with the Salobo II plant. These units are typically on standby.

Secondary-crushed product is then conveyed in a 2 km long pipe conveyor running at a speed of 2.5 m/s to the secondary crushed ore stockpile. This stockpile has a total capacity of approximately 171,000 t and a live capacity of about 75,000 t.

Two parallel lines of four operating reclaim feeders each are then used to reclaim the crushed ore and deliver it to the HPGR circuit via the two stockpile reclaim conveyors merging into a single line of transfer conveyors leading to the HPGR silos feed conveyor, equipped with a shuttle head. This unit delivers ore into one of four concrete silos, providing approximately 20 min of surge at nominal capacity. A reversible feed belt conveyor and feed belt feeders then feed each of the four HPGR units.
Each HPG unit has a drum 2.0 m diameter by 1.5 m wide. The maximum feed size is 55 mm and the HPG product is exhibiting 80% passing 17 mm while operating with a 40 mm gap and at 150 bars of hydraulic pressure applied to the floating roll. The crushed HPG product is discharged via the product collection conveyor and is then screened at 8 mm on the bottom deck of banana screens, with the top deck aperture set at 15 mm. There are a total of eight operating screens, with half dedicated to the HPG of either Salobo I or Salobo II. The screen undersize, at 80% passing 6 mm, discharges directly into one dedicated ball mill discharge sump. The screen oversize is recirculated back via the screen oversize collection conveyor to the HPG silos feed conveyor for further crushing. The circulating load is typically 110% around this circuit.

Slurry in the ball mill discharge sump is pumped to a battery of ten 660 mm hydrocyclones, of which seven are typically operating. Hydrocyclone underflow is fed by gravity to an overflow ball mill of 7.9 m diameter by 12.2 m long, equipped with a 17 MW gearless motor. There are four ball mills operating in closed circuit, each with a dedicated hydrocyclone cluster. Ball mill discharge feeds into the discharge sump for recirculation to the hydrocyclones. The design grinding circuit product is set at 80% passing 106 μm. Hydrocyclone overflow advances to the Rougher 1 flotation circuit at 45% solids by weight. The ball mills were designed to operate at a 30–35% ball charge using 76 mm diameter steel balls and with a circulating load of approximately 300%. These conditions were adjusted by the operations, now showing use of a 30% ball charge. Under these conditions, 15 MW are drawn from the mill motors. A higher ball charge would reportedly require the addition of a retainer ring at the mill discharge. The circulating load is about 200%.

The flotation circuit is of conventional design but the cleaning circuit is making extensive use of column flotation, in order to improve rejection of gangue contaminants carrying
fluorine values. Lime is added at the front end of the circuit to raise the pH to about 10. Addition of NaHS is made ahead of roughing so as to clean the surfaces of the bornite and increase its recovery. PAX and a dithiophosphate are used as the primary and secondary collectors, respectively. Frothing is provided by propylene glycol and methyl isobutyl carbinol (MIBC).

Rougher 1 (e.g. rougher) flotation is carried out in four parallel lines (one for each ball mill) of two cells each. The cells are mechanically agitated units of 200 m³ capacity, providing six minutes of design retention time. The Rougher 1 concentrate advances to the cleaning circuit. The Rougher 1 tailings advance to the Rougher 2 (scavenger) circuit consisting of four lines, with each line containing six mechanically-agitated 200 m³ cells, for a nominal retention time of 39 min. Staged Flotation Reactors (SFR’s) have been installed on the rougher tailings. The concentrate from the SFR’s reports to concentrate regrinding.

SFR tailings gravitate to the tailings storage facility (TSF), while the concentrate advances to the regrinding circuit.

The cleaning circuit is divided into three upgrading stages and closed by a cleaner–scavenger bank of conventional agitated cells. The arrangement of each upgrading stage is typical, whereas the concentrate of one stage advances to the next one and the tailings are moved back to the previous stage. Exceptions are found with the Cleaner 1 tailings, proceeding to the cleaner–scavenger and Cleaner 3 concentrate, which is the final concentrate.

The Cleaner 1 circuit consists of 16 column cells, each 6 m diameter x 14 m height, arranged in four lines of four cells each. Design residence time is 39 min. The Cleaner 1 columns are fitted with a Microcel sparging system, introducing flotation air to recirculated slurry pumped through static mixers. All of the other columns only use more standard air spargers.

The concentrate from the Cleaner 1 circuit advances to the Cleaner 2 circuit, consisting of eight cells, in four lines of two columns each, of 4.3 m diameter x 14 m height, for a design retention time of 34 min. Concentrate from the Cleaner 2 circuit advances to the Cleaner 3 circuit, consisting of four cells, in four lines of one cell each, each column 4.3 m diameter x 14 m height for a design retention time of 39 min.

The tailings of Cleaner 1 are fed into the cleaner–scavenger section, made of four lines of four 200 m³ agitated cells each. The tailings of this stage join the Rougher 2 tailings to form the complete plant tailings stream, directed by gravity to the TSF. The cleaner–scavenger concentrate is combined with the Rougher 2 concentrate and undergoes regrinding in one of four vertical mills fitted with 1.1 MW motors. These mills, filled with 20 mm diameter steel grinding media, are operated in closed-circuit with one dedicated cyclone cluster per mill, ensuring a regrinding circuit product at 80% passing 20 µm.
The final concentrate exiting Cleaner 3 is pumped to one of two 15 m diameter high-capacity thickener, producing an underflow slurry at 65% solids. This slurry is transferred to a surge tank ahead of the concentrate filters.

The concentrate is dewatered further through the use of four pressure filters, each with a horizontal frame holding 50 plates of 1,500 mm x 1,500 mm. A typical filtration cycle lasts 18 minutes. The filtered concentrate has a residual moisture content of about 11%. It is stockpiled below the filters in a covered concentrate storage area holding 6,000 t.

Concentrate is reclaimed by front-end loader and loaded into trucks at a nominal rate of 1,500 wmt/d. The concentrate is weighed to about 27 wmt in the trucks using a static scale and delivered to a rail spur storage area at the town of Parauapebas, some 94 km away. The warehouse can hold 16 kt of concentrate, allowing for blending when required. The concentrate is reclaimed by front-end loader and loaded into 80–90 wmt railcars carrying it to the port of Itaqui, in São Luís, in trains of 100 railcars. The concentrate is stored there in an enclosure with a capacity of 50 kt, while awaiting loading into boats at a rate of 1,100 wmt/h. Sampling of the concentrate is carried out at the Port of Itaqui, in lots of 500 wmt, when the material is reclaimed by loader and placed on the conveyor system feeding it into ships. Shipment weights can vary from 13 kt to 45 kt, with two to three shipments completed per month.

The combined flotation circuit tailings (Rougher 2 and cleaner–scavenger tailings) flow by gravity from the plant to the TSF, located directly north of the processing plant. Tailings are dumped from a single-point discharge and create a beach on the south side of the dam. Over the mine life, several phases of dam raising with mine waste will be required to provide the required storage volume. Vertical pumps installed on pontoons pump recycled tailings water back to the process plant, accounting for over 95% of the total process water requirements.

A summary of the main process equipment is provided in Table 39 for Salobo I and Salobo II.


The plant is provided with electricity from the plant substation. Step-down transformers provide the various voltages used by the equipment.

The bulk of the process water needs are covered by the recirculation from the TSF. The consumption of fresh water is limited to systems requiring such a quality. Water is provided by vertical pumps installed in the Mirim and Salobo Creeks.

Reagent dosages, as budgeted for the 2015–2019 period, are 71 g/t for PAX, 60 g/t for dithiophosphate, 70 g/t for propylene glycol and 90 g/t for MIBC. NaHS and lime consumptions are at 120 g/t and 600 g/t, respectively.
The other major consumables are the grinding balls, with the ball mills calling for 600 g/t of the 76 mm balls, and the regrinding stage 50 g/t of 20 mm balls.

17.4. **Plant Equipment and Design Considerations**

The major process equipment installed in the Salobo I and Salobo II sections of the plant are shown in Table 39.

<table>
<thead>
<tr>
<th>Table 39 - Major Process Equipment</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Salobo I</strong></td>
</tr>
<tr>
<td>1 Gyratory crusher – 60 in. x 89 in.</td>
</tr>
<tr>
<td>2 (*) Cone crusher</td>
</tr>
<tr>
<td>2 (*) Vibrating screens 12 x 24 ft</td>
</tr>
<tr>
<td>1 Overland pipe conveyor (78.7 in.; 6,000 hp; 1,700 m length; 4,600 t/h capacity)</td>
</tr>
<tr>
<td>2 High pressure grinding rolls – Ø 2.0 x 1.5 m</td>
</tr>
<tr>
<td>2 Ball mills – Ø 26 ft x 40 ft</td>
</tr>
<tr>
<td>24 Flotation tank cells – 200 m³</td>
</tr>
<tr>
<td>14 Flotation columns – 14 m</td>
</tr>
<tr>
<td>4 Vertimills – 1,500 hp</td>
</tr>
<tr>
<td>1 Concentrate thickener – Ø 15 m</td>
</tr>
<tr>
<td>2 Pressure filters 1,500 x 1,500 / 50 chambers</td>
</tr>
</tbody>
</table>

*Note:* * One cone crusher and one vibrating screen are common for both Salobo I and II plants, as stand-by equipment; Ø = effective grinding length.

Salobo I relied on a series of conveyors that were oversized for the design capacity of the 12 Mt/a plant, but were planned to cover the eventual addition of Salobo II and the total 24 Mt/a expanded capacity. The resulting processing facilities therefore have a series of critical items that are not duplicated, the failure of which can curtail the complete plant operations if the failure is located after the plant stockpile where no more significant surge capacity exists.

Figure 45 shows the process lines and highlights in grey the elements found as single-line items in the flowsheet. It shows in particular that 12 conveyors, including two with shuttle heads, are forming a critical path after the last available stockpile. This presents a risk for interruptions in plant operation however the improvements in plant availability and utilization demonstrate that these items have been addressed.
17.5. Process Plant Performance Projections

The copper and gold recoveries applied from 2018 to 2022 production plans were defined based on historical plant performance data and the plant performance projections are shown in Table 42 with recovery assumptions from Table 40. From 2022 onward, the processing recoveries used in LoMP are shown in Table 41.

Table 40 - Processing Recovery Assumptions (2018-2022)

<table>
<thead>
<tr>
<th>Metal</th>
<th>Total Recovery to All Concentrates (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper</td>
<td>(-3.0427*1/%Cu in feed + 91.0894)</td>
</tr>
<tr>
<td>Gold</td>
<td>1/(-0.0025*Rec Cu% + 0.23395)</td>
</tr>
</tbody>
</table>

Table 41 - Processing Recovery Assumptions (2023-LoMP)

<table>
<thead>
<tr>
<th>Metal</th>
<th>Total Recovery to All Concentrates (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper</td>
<td>(-2.5362*1/%Cu in feed) + 90.674</td>
</tr>
<tr>
<td>Gold</td>
<td>(1.0173*Rec Cu)-20.357</td>
</tr>
</tbody>
</table>
### Table 42 - Processing Plant Performance Forecasts

<table>
<thead>
<tr>
<th></th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
<th>2021</th>
<th>2022</th>
</tr>
</thead>
<tbody>
<tr>
<td>ROP (wmt)</td>
<td>24,146,508</td>
<td>24,119,159</td>
<td>24,186,286</td>
<td>24,119,159</td>
<td>24,119,159</td>
</tr>
<tr>
<td>Cu Feeding Grade (%)</td>
<td>0.92</td>
<td>0.94</td>
<td>0.96</td>
<td>0.96</td>
<td>0.95</td>
</tr>
<tr>
<td>ROP (dmt)</td>
<td>23,663,577</td>
<td>23,636,776</td>
<td>23,702,560</td>
<td>23,636,776</td>
<td>23,636,776</td>
</tr>
<tr>
<td>Au Feeding Grade (g/t)</td>
<td>0.61</td>
<td>0.57</td>
<td>0.61</td>
<td>0.55</td>
<td>0.52</td>
</tr>
<tr>
<td>Concentrate Production (dmt)</td>
<td>505,403</td>
<td>513,331</td>
<td>527,733</td>
<td>524,736</td>
<td>522,093</td>
</tr>
<tr>
<td>Concentrate Production (wmt)</td>
<td>564,697</td>
<td>573,554</td>
<td>589,646</td>
<td>586,297</td>
<td>583,344</td>
</tr>
<tr>
<td>Concentrate Moisture (%)</td>
<td>10.5</td>
<td>10.5</td>
<td>10.5</td>
<td>10.5</td>
<td>10.5</td>
</tr>
<tr>
<td>Concentrate Cu Grade (%)</td>
<td>38</td>
<td>38</td>
<td>38</td>
<td>38</td>
<td>38</td>
</tr>
<tr>
<td>Contained Cu in Concentrate (t)</td>
<td>192,053</td>
<td>195,066</td>
<td>200,539</td>
<td>199,400</td>
<td>198,395</td>
</tr>
<tr>
<td>Concentrate Au Grade (g/t)</td>
<td>19.8</td>
<td>18.4</td>
<td>19.3</td>
<td>17.6</td>
<td>16.8</td>
</tr>
</tbody>
</table>

The 2017 operational unit cost forecast is 1.4% higher than the actual of 2016 (Actual 2016: US$3,182/tonne vs. Forecast 2017: US$3,108/tonne), mainly due to exchange rate impact and energy price, despite higher production volumes and cost reduction plans that were applied on labor, material and services.

Table 43 presents the actual recent historical operating recoveries and a comparison to the forecasted recovery assumptions.

### Table 43 - Actual and Forecasted Processing Recoveries

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Mill Feed kWMT</td>
<td>20,696</td>
<td>21,829</td>
<td>24,123</td>
<td>24,147</td>
<td>24,119</td>
<td>24,186</td>
<td>24,119</td>
<td>24,119</td>
</tr>
<tr>
<td>Mill Feed kDMT</td>
<td>20,290</td>
<td>21,401</td>
<td>23,650</td>
<td>23,664</td>
<td>23,637</td>
<td>23,703</td>
<td>23,637</td>
<td>23,637</td>
</tr>
<tr>
<td>Mill Feed DMT/hr</td>
<td>2,316</td>
<td>2,443</td>
<td>2,700</td>
<td>2,701</td>
<td>2,698</td>
<td>2,706</td>
<td>2,698</td>
<td>2,698</td>
</tr>
<tr>
<td>Mass Recovery %</td>
<td>2.00%</td>
<td>2.10%</td>
<td>2.10%</td>
<td>2.10%</td>
<td>2.20%</td>
<td>2.20%</td>
<td>2.20%</td>
<td>2.20%</td>
</tr>
<tr>
<td>Cu Recovery %</td>
<td>86.80%</td>
<td>87.50%</td>
<td>86.50%</td>
<td>87.80%</td>
<td>87.90%</td>
<td>87.90%</td>
<td>87.90%</td>
<td>87.90%</td>
</tr>
<tr>
<td>Au Recovery %</td>
<td>67.50%</td>
<td>69.30%</td>
<td>69.40%</td>
<td>69.30%</td>
<td>69.80%</td>
<td>70.80%</td>
<td>70.70%</td>
<td>70.80%</td>
</tr>
<tr>
<td>Cu Concentrate kDMT</td>
<td>403</td>
<td>445</td>
<td>498</td>
<td>505</td>
<td>513</td>
<td>528</td>
<td>525</td>
<td>522</td>
</tr>
<tr>
<td>Cu Produced kt</td>
<td>155</td>
<td>176</td>
<td>193</td>
<td>192</td>
<td>195</td>
<td>201</td>
<td>199</td>
<td>198</td>
</tr>
<tr>
<td>Au Produced t.Ozs</td>
<td>251,250</td>
<td>317,461</td>
<td>352,875</td>
<td>321,934</td>
<td>303,551</td>
<td>328,224</td>
<td>297,332</td>
<td>281,702</td>
</tr>
</tbody>
</table>

### 17.6. Comments on Section 17

In the QPs' opinion, the Salobo concentrator is currently operating at throughput and performance levels upon which it was designed. Plant availability, utilization and throughput have been at design levels over the past 12 months. Metallurgical performance for copper and gold have improved and are currently near or at design levels.
The operating team have made continual modifications and improvement to the plant which has resulted in the improvements to availability and performance with the installation of the SFRs as an example of this. Efforts are ongoing to utilize a geometallurgical program to further optimize plant recoveries.

The QPs agree with the use of dedicated metallurgical samplers in the reject and concentrate lines of the plant. This initiative is currently being implemented and should result in an improvement in the metallurgical balance.

18. Project Infrastructure

18.1. Roads and Logistics

The area is well-served by railroads and highways that connect the towns and cities. Regularly scheduled air service is available in Marabá approximately 225 km from Salobo by highway. Most flights connect to the capital, Brasilia.

Vale has a contract with a transportation company to transport all proper employees and contractors in the Carajás and Parauapebas city to the Salobo Operations site. The traffic routes are elaborate and supervised by Vale (infrastructure area managing this contract).

Employees also use a fleet of company owned vehicles for transportation. For safety, all the DIMB vehicles have a system that registers speed, abrupt stops, and rpm. The system signals the driver when the vehicle is travelling above the allowed speed.

Copper concentrate is shipped by truck (40t) from the Salobo Operations to a rail carried out facility near the city of Parauapebas, 85 km from the mine. From there it is transported by rail to the Ponta da Madeira Marine Terminal at São Luís, a distance of approximately 870 km, for shipment.
18.2. Stockpiles and Waste Rock Storage Facilities

Low-grade ore and waste rock from the mine are stored in three locations along the perimeter of the pit (Figure 47). The main WRF is to the west of the pit, and contains both oxidized and fresh rock. Additional information is included in Section 20.
18.3. Site Infrastructure

Surface facilities include (see Figure 47 for the locations of the numbers referred to):

- Central Administrative Facilities, (24), which includes administrative offices, restaurant, change rooms, training centre and a medical clinic
- Central Maintenance Facilities, (26), which includes a mine heavy equipment workshop including tyre changing facility, a light vehicle maintenance shop, a plant maintenance shop for component overhaul and repair, a warehouse and maintenance offices
- Mine facilities, (13), which includes mine operations change rooms and mine operations offices
- Mine heavy equipment fuelling facilities, (10), which are located next to the primary crushers
- Main substation (36)
- Small vehicle fueling station (25)
- Recycle centre (16)
- Security/ access control gate (20)
18.4. Tailings Storage Facility

The Salobo tailings storage facility (TSF), comprising an earth dam and concrete-lined spillway, was designed for Vale by Brazilian engineering company BVP Engineering to withstand a 1 in 10,000 year event. The current design was completed in July, 2010.

The TSF, when completed to an elevation of 285 m, will have sufficient capacity to store the entire forecasted tails from the processing of the Mineral Reserve planned over the life of the mine. Potentially the TSF site could also store the forecasted tails from the material presently identified as Mineral Resources.

During 2017 the dam was raised from 220 m elevation to an intermediate design height of 227.6 m elevation. The Tailings Storage Facility is shown in Figure 48 and its storage capacity is shown in Figure 49.
18.5. **Water Supply**

Process make-up water comprises runoff and direct precipitation within the tailings storage basin. This raw water is pumped to the plant together with return water from tailings deposited in the storage facility. If the plant requires additional makeup water, this can be extracted from Igarapé Mamão (Pawpaw creek) via a floating intake within the project site, using vertical pumps.

18.6. **Power and Electrical**

In accordance with legislation governing the Brazilian electrical power sector, the Salobo Operations are supplied by the Eletronorte division of Eletrobras, responsible for the northern region of Brazil, operating and maintaining the system on behalf of the National Operator of the Electrical System (NOS).

Electrical energy is supplied from Tucurui, an 8,370 MW hydroelectric generating station on the Tocantins River, 200 km north of Marabá, and 250 km due north of Parauapebas. The 150MW of power required by the Salobo Operations is transmitted 87 km by an overhead 230 kV transmission line.

18.7. **Communications**

Telephone communications are available over land-lines, and via a cellular network. Internet communications are also available at the mine site.

18.8. **Housing**

Local housing is available for employees within the Carajás urban centre and Parauapebas. There are adequate schools, medical services and businesses to support the work force. The mine sites have medical facilities to handle certain emergencies. In addition, medical facilities are available in to support the mine’s additional needs.

Vale has invested significantly in the town’s infrastructure, building a 130 km paved road to Parauapebas and a 20 km sewage system, together with a school, hospital, and day care centre.

18.9. **Comments on Section 18**

The Salobo mine has been in operation since 2012, and at has been producing at a 24 Mt/a capacity since 2014. The infrastructure is well established and adequate for this production level and the LoMP.
19. Market Studies and Contracts

Vale has agreements at typical copper concentrate industry benchmark terms for metal payables, treatment charges and refining charges for concentrates produced. Treatment costs and refining costs vary depending on the concentrate type and the destination smelter. For all of Vale’s sales contracts, the risk of the concentrates transfers either at the load port or discharge port according the standard International Commercial Terms (Incoterms); whereas the title to the concentrates transfers either at the load port or discharge port according the standard Incoterms or upon payment.

The terms contained within the sales contracts are typical and consistent with standard industry practice, and are similar to contracts for the supply of copper concentrate throughout the world. Depending on the specific contract, the terms for the copper concentrate sale are either annually negotiated, benchmark-based treatment and refining charges, or in the case of spot agreements are based on fixed treatment and refining charges based on market terms negotiated at the time of sale. The differences between the individual contracts are generally in relative quantity of concentrates that are covered under annually-negotiated treatment and refining charges.

The typical grade of copper, gold and silver in the final product is approximately 38%, 17 g/t and 30 g/t, respectively. The copper concentrate is used in copper smelting and refining operations to produce copper cathode and precious metals.

Market for copper concentrates is well developed with a large number of custom smelters located around the world who use the copper concentrate as feed. Higher levels of fluorine, higher copper grade and other specificities limit some of the processing options for the Salobo concentrate. Customers for Salobo concentrate have been well established.

The Wheaton streaming agreement is discussed in Section 4.4.

Metal price and exchange rate assumptions are shown in Table 44.

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
<th>2021</th>
<th>2022</th>
<th>Long Term</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Price</td>
<td>US$/tonne Cu</td>
<td>5,600</td>
<td>5,700</td>
<td>6,100</td>
<td>6,200</td>
<td>6,600</td>
<td>6,600</td>
</tr>
<tr>
<td>Gold Price</td>
<td>US$/oz. Au</td>
<td>1,265</td>
<td>1,265</td>
<td>1,300</td>
<td>1,300</td>
<td>1,275</td>
<td>1,275</td>
</tr>
<tr>
<td>Exchange Rate</td>
<td>Real$/US$</td>
<td>3.35</td>
<td>3.35</td>
<td>3.35</td>
<td>3.35</td>
<td>3.35</td>
<td>3.35</td>
</tr>
</tbody>
</table>

19.1. Comments on Section 19

It is the QP’s understanding that the Salobo concentrates have been successfully marketed to custom smelters located around the world.

A definition drilling campaign was initiated to increase the confidence of the Cu and Au grade variability and grade distribution of deleterious elements (uranium, fluorine, and
chlorine) and their impact on the copper concentrate quality will also be used for silver evaluation and its applicability in the long-term model and mine planning. This drilling will assist in improving the confidence of the reserves and ability to deliver the LoMP grades.

20. Environmental Studies, Permitting and Social or Community Impact

Environmental and social baseline study areas were defined to characterize the current conditions in the areas potentially affected by mine components or activities.

The project is located in the Carajás mountain range in the eastern Amazon humid tropical rainforest. Temperatures range from 20.8°C to 37.8°C with an average relative humidity of 80.5%. Mean annual rainfall is 1,920 mm and evaporation is 1,500 mm. Winds are predominantly from the north and west.

The project lies in part of Salobo Creek and the Cinzento River basins which are tributary to the Itacaiúnas River. The long-term average unit runoff for the project site is 13.5 L/s/km².

The Tapirapé–Aquiri National Forest has a registered area of 190,000 ha. The Tapirapé Biological Reserve, which covers an area of 103,000 ha, borders the National Forest (and mine area) to the north. The mine site is within the Tapirapé–Aquiri National Forest and the access road crosses the Carajás National Forest and lies adjacent to the Igarapé Gelado Protected Area. Figure 50 shows the location of the mine in relation to the forest areas.
As a requirement of the mine installation licence, an agreement was signed between the Chico Mendes Biodiversity Conservation Institute and the Salobo Operations to provide payment and support towards management of the Tapirapé–Aquiri National Forest (ICMBio, 2007).

The protected areas have distinct management categories that were established by Decree N° 97,720 dated 5 May 1989. Within these areas, a regular polygon outlining the mining zone Special Use Area was defined by the National Department of Mineral Production of Brazil. The polygon encompasses the mine area, roads, and supporting infrastructure, and incorporates a 100 m buffer zone. A second 10 km buffer surrounds the Special Use Area polygon.

Within the Special Use Area, Vale controls access to the area and the mine site, and access to the Tapirapé–Aquiri National Forest along the eastern boundary of the Special Use Area with the forest.

To the northwest of the Special Use Area is the Lindoeste settlement, developed on land in the São Felix do Xingu region, which currently covers about 120 ha; the mine site has no influence over forest access by this community.

The Salobo Operations also have a commitment to offset effects by planting seedlings in the Igarapé Gelado Protected Area (National Press, 2007).
20.1. Environmental Management

The Salobo Operations have an Environmental Control Plan (Brandt, 2003) that includes the following components:

- Project Description
- Environmental Management System
- Vegetation Clearing and Stripping
- Erosion Control
- Water and Effluent Management
- Waste Management
- Atmospheric Emissions
- Noise and Vibration
- Environmental Emergencies
- Disease Control
- Archaeology Protection and Salvage
- Rehabilitation Plan
- Environmental Compensation and Social Inclusion
- Environmental Education
- Environmental Monitoring
- Closure Plan

These social and environmental management plans detail best practices and Brazilian legislation to prevent and mitigate potential impacts and manage compliance specifically for the Salobo Operations.

20.2. Permitting

Brazil is a federal republic, and its legal system is based on Civil Law tradition, characterized by codification of legal requirements. The Federal Constitution (October, 1988) is the basis of the legal system.

Key applicable legislation for construction, operation and closure of the Project includes the following:

- Mining Code (Decree-Law No. 227, 28 February, 1967) and its Regulations (Decree No. 62934, 2 July, 1968)
- Forest Code (Law No. 4771, 15 September, 1965)
- National Environmental Policy Law (Law No. 6938, 31 August, 1981)
- CONAMA (National Environment Council) Resolutions Nos. 1/86, 23/86, 9/90, 10/90 and 237/97; and

The Salobo Operations began production in 2010 and received its first Operating Licence No. 1096/2012 on November 5, 2012 (valid for 4 years). The current license refers to the research, mining and mineral processing of 24 Mtpa as well as all administrative and support facilities, including workshops, the central material disposal area and warehouse, dining hall, transportation, storage and shipment of copper concentrate and it was valid until November 5, 2016.

The actual Operating License was renewed within the validity period, therefore, according to Brazilian laws, the license is automatically renewed until the pronouncement of the government responsible department (protocol No. 02001.005679/2016-14).

The Installation License for the expansion of the feed stockpile from Salobo processing plant to 24 Mtpa (No. 1046/2015) valid until February 26, 2016 was renewed within the validity period (protocol No. 02001.020492/2015-60).

Regarding vegetal removal, Salobo has four valid licenses: No. 1104/2016 valid until March 23, 2019; No. 1181/2016 (rectification) valid until November 22, 2021; No. 1188/2017 valid until February 21, 2021; No. 1001/2015 valid until November 05, 2016, which was renewed within the validity period (protocol No. 02001.014.542/2016-51).

There is also the surface water capture and discharge concession (No. 1896/2017) granted in October 9, 2017 and valid until October 9, 2027, and the underground water capture concession for explosive factoring (No. 2519/2016) granted on June 17, 2016 and valid until May 16, 2020.

The Salobo Operations currently hold all required permits to operate. The mine has a robust control and monitoring system to ensure that permits remain current, and to ensure that the requirements of each permit are monitored to comply with the relevant regulatory conditions imposed.

20.3. Social and Community Impact

The Salobo Operations area of influence is located in the southeast Paraense mesoregion, in the municipalities of Marabá and Parauapebas. These regions are considered to have moderate human development indices for the level of health, education and living conditions, based on data from 2000. The extractive industry accounts for 23.5% of the economic activity in the state of Pará, with 17.9% other industrial activities, 52.0% services and 6.6% farming and ranching based on 2010 data (IBGE, 2013).

The Project is not located on indigenous lands. The nearest indigenous lands include the river Tapirapé Tuere, Trincheira Bacaja and Xicrin do Cateté, all located 25 km or more...
from the Project. The Xikrin indigenous peoples traditionally use the Project area for food collection.

CVRD signed an agreement with the Xicrin do Cateté indigenous community in 1989 (Convenio No. 453/89; FUNAI, 1989).

In 2001, a forest management program was implemented between the indigenous communities and government associations to sustainably harvest the forest in the Project area in a manner that benefitted the indigenous community in capacity building and financial resources.

Vale currently maintains a Communication Plan that commits to continued communication with the local indigenous to maintain community health and safety, cultural preservation, transparency of activities and harmony between the workers and the indigenous community.

There are a number of social management plans carried out by the Social Communications Department. The Environmental Compensation and Social Inclusion plan objectives are to support sustainable development by capitalizing on the positive effects of project development and minimizing the potential negative effects. In addition, this plan is supported by a Social Communications program that facilitates information exchange and works to improve relations between the Salobo Operations and the diverse social segments of the surrounding communities.

An Environmental Education program was developed between the Department of Environment and Sustainable Development (DIAM), Vale Education and the municipality of Parauapebas. The program seeks to spread the principles of sustainability recognized as environmental, social and economic responsibility through educational activities geared towards Vale’s employees and contractors and the surrounding community. The program aims to strengthen and expand environmental education in the municipal education program and the community.

20.4. Closure Plan

The mine Closure Plan assumes that there will be partial recovery of infrastructure for use by educational activities, research and tourism. The closure plan is included in the Environmental Control Plan and rehabilitation and re-vegetation work is ongoing during operations. The Closure Plan (SETE, 2015) outlines the steps to be taken for the progressive rehabilitation and ultimate closure of the open pit and concentrator facilities and the auxiliary components of the operation and all associated infrastructure and equipment. The overall objective is to return the Project area to a natural condition to support the local vegetation and wildlife biodiversity of the Tapirapé–Aquiri National Forest.

There are no reclamation bonds required for the mine. Rehabilitation and re-vegetation work is ongoing during operations.
Closure costs have been estimated by Vale at approximately US$201.9 million. Closure costs are to be reviewed annually and are included as indirect costs in each operational centre for budgeting, expenditure tracking and financial planning (Vale, 2017).

20.5. Comments on Section 20

It is the QPs understanding that the Salobo Operations have all required permits for operating and that appropriate environmental management systems are in place. Also, that an approved closure plan is in place designed to return the site to a natural condition to support the local vegetation and wildlife biodiversity of the Tapirapé–Aquiri National Forest.

21. Capital and Operating Costs


The construction of the Salobo Project II processing plant expansion is complete, increasing process throughput from 12 Mtpa to 24 Mtpa. A total of US$508M will be invested in sustaining capital in the 5 year plan for mine and processing plant improvement and upgrades (equipment, materials, spare parts, etc.), health, safety, and environmental sustaining expenditures relating to dam works.

Table 45 and Table 46 present the historical and estimated sustaining capital estimates, respectively.

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Growth</td>
<td>55</td>
<td>223</td>
<td>438</td>
<td>782</td>
<td>853</td>
<td>700</td>
<td>446</td>
<td>369</td>
<td>53</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Sustaining</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>20</td>
<td>68</td>
<td>59</td>
<td>55</td>
<td>150</td>
<td>150</td>
<td>98</td>
</tr>
</tbody>
</table>

Table 46 – Capital Development (US$ M)

<table>
<thead>
<tr>
<th>Type</th>
<th>2017</th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
<th>2021</th>
</tr>
</thead>
<tbody>
<tr>
<td>sustaining</td>
<td>98</td>
<td>116</td>
<td>98</td>
<td>111</td>
<td>88</td>
</tr>
</tbody>
</table>

21.2. Operation Cost Estimates

21.2.1. Mine Operating Costs

The operating cost estimation is performed in conjunction with the mobile equipment fleet selection and mine planning. In addition to the equipment direct operating costs, the other key factors include labour, salaries, energy, and fuel costs.
Annual mining operating costs, on a per unit basis, stayed the same between 2016 and 2017 mainly due to FX impact being offset by cost reductions achieved.

Table 47 lists the forecasted LoM unit and operating costs. Total annual costs are divided by total mine movement tonnes to determine the operating unit costs as an output.

Table 47 - Mine Operating Costs

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Cash Cost (US$ M)</td>
<td>$305</td>
<td>$312</td>
<td>$325</td>
<td>$340</td>
<td>$351</td>
<td>$354</td>
<td>$349</td>
</tr>
<tr>
<td>Mined Material (wet M t)</td>
<td>114</td>
<td>130</td>
<td>135</td>
<td>141</td>
<td>143</td>
<td>146</td>
<td>144</td>
</tr>
<tr>
<td>Unit Cost (US$/tonne)</td>
<td>$2.68</td>
<td>$2.40</td>
<td>$2.40</td>
<td>$2.41</td>
<td>$2.46</td>
<td>$2.43</td>
<td>$2.42</td>
</tr>
</tbody>
</table>

21.2.2. Process Operating Costs

Annual unit processing costs was reduced in the actual 2017 LoMP compared to the 2016 LoMP mainly due to electric energy price decrease and FX variation.

Table 48 lists the estimated process plant operating costs.

Table 48 - Process Operating Costs

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Cash Cost (US$ M)</td>
<td>$200</td>
<td>$183</td>
<td>$204</td>
<td>$208</td>
<td>$210</td>
<td>$215</td>
<td>$221</td>
</tr>
<tr>
<td>Processed Material (dry Mt)</td>
<td>20</td>
<td>21</td>
<td>24</td>
<td>24</td>
<td>24</td>
<td>24</td>
<td>24</td>
</tr>
<tr>
<td>Unit Cost (US$/t)</td>
<td>$9.83</td>
<td>$8.56</td>
<td>$8.67</td>
<td>$8.81</td>
<td>$8.89</td>
<td>$9.07</td>
<td>$9.34</td>
</tr>
</tbody>
</table>

21.2.3. Other Operating Costs

The Royalties (CFEM – Federal Royalty – Mineral Exploitation Levy) are calculated on the net revenue obtained at the time the mineral product is sold. For copper the CFEM rate is 2%. For calculation purposes, net revenue is considered as the sale value of the mineral product after deduction of the taxes levied on its commercialization, transport and insurance.

Table 49 lists the estimated other costs.

Table 49 - Other Costs

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Cash Cost (US$ M)</td>
<td>$72</td>
<td>$64</td>
<td>$74</td>
<td>$79</td>
<td>$81</td>
<td>$84</td>
<td>$83</td>
</tr>
</tbody>
</table>
21.3. **Comments on Section 21**

In the QPs opinion, projected operating and capital costs are reasonable as they are in line with historical actuals.

22. **Economic Analysis**

According to Item 22, for technical reports on properties currently in production, issuers may exclude economic analysis information.

The Mineral Reserve declaration is supported by a positive cashflow.

23. **Adjacent Properties**

This section is not relevant to this Report.

24. **Other Relevant Information**

There is no other relevant data and information to disclose.

25. **Interpretation and Conclusions**

The Salobo Operations have been in operation since 2009 (pre-stripping) and has successfully ramped up to design capacity of 24 M tonnes per year and designed process recoveries. Vale operates the mine according to high standards with respect to safety, operating practices and the environment.

Mineral Resources and Mineral Reserves have been prepared according to the 2014 CIM Definition Standards and this Report according to the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects and Form 43–101F1.

The mine has a large Mineral Reserve base and strong economic margins which result in forecasted mining until the year 2045 and then the processing of stockpiled material until 2067.

In order to maintain strong operating performance, the established production reconciliation control and operational mining dilution studies must be continued (selective mining unit and equipment operations) to ensure that best criteria are being applied to the Mineral Resources and Mineral Reserves definition.

Significant improvements in the mining sequencing were realized from 2016 to 2017.
The areas that have the highest impact on production are mining dilution and recovery, as well as, the challenge to optimize mining operations. Risks include meeting the material movement targets and drilling/production equipment efficiency.

26. Recommendations

At all mining operations there are areas to improve efficiencies and reduce costs. The following recommendations are presented as items that could have a positive impact to the mine economics:

1. A technical analyses of Mineral Resources below the Mineral Reserve pit should be studied to determine if they can delay the low-grade stockpiles reclaimed in the later years of operation.

2. Geophysical surveys recently completed by the Exploration department at the Salobo Operations have identified a significant gravity anomaly below the current Salobo open pit. Drilling is required to determine what the anomaly source is and help to identify additional Mineral Resources for supporting future potential operations and projects expansions.

3. A definition drilling campaign was initiated to increase the confidence of the Cu and Au grade variability and grade distribution of deleterious elements (uranium, fluorine, and chlorine) and their impact on the copper concentrate quality will also be used for silver evaluation and its applicability in the long-term model and mine planning. This drilling will assist in improving the confidence of the reserves and ability to deliver the LoMP grades.

4. Review of the operational pit slope angles through geotechnical examination of the pit wall operation, design of pushbacks, and further geotechnical studies may provide support for steepening of some of the pit walls. The current geomechanical sectors are based on a limited amount of information and this should be further analyzed.

5. Gold recoveries are currently higher than predicted in the model. However, increases in recovery predictions require support from a full metallurgical balance process, and the sampling processes to provide data for such a balance are not fully implemented. If higher recoveries can be sustained or improved it will provide some economic upside to the project.

6. Continuous improvements in the dispatch controls and follow up process, will result in further improvement of the F2 reconciliation (mining recovery and dilution).

7. The QPs agree with the use of dedicated metallurgical samplers in the reject and concentrate lines of the plant and the utilization of the primary crusher weighing scales to evaluate the dispatch system truck data measurement and calibration.
are of high importance. This should result in an improvement in the metallurgical balance.

27. References


