INTREPID MINES LIMITED

TUJUH BUKIT PROJECT
REPORT ON MINERAL RESOURCES,
LOCATED IN EAST JAVA,
INDONESIA

TECHNICAL REPORT

FOR
INTREPID MINES LIMITED
LEVEL 1, 490 UPPER EDWARD ST.
SPRING HILL, QLD 4004
AUSTRALIA

FEBRUARY, 2009

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3. SUMMARY

Property
The Tujuh Bukit Project comprises a single exploration tenement (KP-Explorasi) of 11,621.45 hectares. The property is located approximately 205 kilometres southeast of Surabaya, the capital of the province of East Java, Indonesia and 60 kilometres southwest of the regional centre of Banyuwangi. The property is centred near 8° 35’ 20.6” S and 114° 01’ 08” N and is bound within UTM co-ordinates 163,000-179,000 E and 9042000-9055000 N.

Location
The property is located approximately 205 kilometres southeast of Surabaya, the capital of the province of East Java, Indonesia and 60 kilometres southwest of the regional centre of Banyuwangi. The property is centred near 8° 35’ 20.6” S and 114° 01’ 08” N and is bound within UTM co-ordinates 163,000-179,000 E and 9042000-9055000 N.

Ownership
The KP-Explorasi (Kuasa Pertambangan or exploration mining permit) was granted to PT. Indo Multi Niaga on 16 February 2007 by the Bupati of Banyuwangi (Regional Administrator, Banyuwangi, East Java) under decree number 188/05/KP/429.012/2007.

Intrepid Mines and PT IMN have signed a Joint Venture agreement enabling Intrepid to hold an 80% economic interest in the Tujuh Bukit Project.

Geology and Mineralisation
The principal styles of mineralisation that are the focus of exploration and delineation drilling on the Tujuh Bukit Project are high-sulphidation epithermal Cu-Au-Ag mineralisation and porphyry Cu-Au mineralisation. The rocks within the porphyry environment become intensely altered by the passage of hot saline fluids of varying Ph and by the late descent of cool oxidised ground-waters that are out of equilibrium with the host rocks.

These areas of rock alteration are typically zoned at the district-scale, a feature that can provide vectors to porphyry Cu-Au ore in magmatic-related hydrothermal systems. Porphyry deposits contain the vast majority of the copper resources of the Pacific island arcs and significant amounts of gold, silver and molybdenum. Porphyry copper-gold deposits tend to be large, fairly uniformly mineralized and relatively low-grade deposits with great vertical extent.

Exploration Concept
The project is of an advanced nature, with well understood geological potential and an Inferred Resource. It will progress by infill drilling, step-out drilling, drilling to depth and follow-up of geophysical (eg IP) and geochemical targets around the immediate area of identified mineralisation.

Status of exploration
Resource delineation and step-out drilling.
Development and Operations
None as yet.

Qualified Person’s Conclusions and Recommendations
A confirmatory phase of drill testing a gold-silver enriched cap over high-sulfidation epithermal Cu-Au-Ag and porphyry-style mineralisation has resulted in the delineation of an Inferred Resource in Zone A, at one of several zones of known oxide and sulphide gold-silver mineralisation within the Tumpangpitu Prospect. Zone A remains open in all directions. Resource estimation has been confined to the oxidised portions of Zone A.

<table>
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<th>Cut-off (AuEq)</th>
<th>Tonnes</th>
<th>AuEq (g/t)</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Oz AuEq (m)</th>
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<td>0.50</td>
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<td>0.62</td>
<td>27.8</td>
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<td>0.75</td>
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<td>1.00</td>
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<td>1.67</td>
<td>1.08</td>
<td>38.1</td>
<td>0.88</td>
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Data are shown for Gold Equivalent (AuEq) cut-off values. The gold equivalent ratio for silver has been set at 65:1 based on US$650/oz gold and US$10/oz silver. Historical bottle roll tests at Tumpangpitu have shown recoveries of 83% Au and 84.5% Ag, supporting a 65:1 ratio. Additional metallurgical testing is currently underway and results are expected in September, 2008. Recently received CIL metallurgical testing results from Zone C achieved recoveries of approximately 90% for both Au and Ag at a grind of 80% passing 75 um. More than 95% of these estimates fall within a preliminary open pit.

A staged drilling program that is guided by inputs from metallurgy, pit optimisation and mine planning, geotechnical data and resource estimation along with exploration geophysics and geochemistry is recommended to proceed as budgets allow.
4. INTRODUCTION

This technical report is prepared by P. Hellman, an Independent Consultant to Intrepid Mines Limited (Intrepid) to comply with NI 43-101 reporting guidelines.

Technical information and data contained in the report or used in its preparation are sourced from reports compiled by previous workers of the property together with internal reports of the current tenement holders as well as the authors own observations whilst visiting the site and working with data from the site generated by others.

This report documents the expansion of the inferred resource at the Tumpangpitu Au-Ag-Cu prospect in East Java, Indonesia, and increases the resource that was reported in September 2008.

The property was visited by the Author over a period of three days from 20 to 22 November, 2007 during which time the Author observed the progress of the drilling program in the Zone C area at the Tumpangpitu prospect, visited the site office at Pulau Merah and provided advice on sampling, QA/QC, geological logging, geotechnical data acquisition and general data handling protocols. A further visit of 3 days was made in October 2008 during which time the author observed the progress of the drilling program in the Zone A area at the Tumpangpitu prospect, visited the site office at Pulau Merah and provided follow-up advice on sampling, QA/QC, geological logging, geotechnical data acquisition and general data handling protocols.

5. RELIANCE ON OTHER EXPERTS

The author of this report is an Independent Qualified Person and has relied on various datasets and reports that were provided by Intrepid Mines Limited, and project consultants to support the interpretation of exploration results discussed in this report on mineral resources. The data that was provided to the author was deemed to be in good stead, and is considered to be reliable. The author is not aware of any critical data that has been omitted so as to be detrimental to the objectives of this report. There was sufficient data provided to enable credible and well constrained interpretations to be made in respect of data.

6. PROPERTY DESCRIPTION AND LOCATION


The main mineralised prospect, Tumpangpitu, is located in the southeast corner of the tenement and covers an area of about 2.5 by 1.5 kilometres. This report is the second to report on mineral resource estimates from this property.

7. ACCESS, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

8. HISTORY


The project area was first explored by PT. Hakman Platina Metalindo and its JV partner, Golden Valley Mines of Australia. Golden Valley Mines identified the potential of the Tumpangpitu and Salakan areas as prospective targets for porphyry copper type mineralisation following a regional (1:50,000) drainage and rock-chip geochemical sampling programme conducted during December 1997 – May 1998. Subsequently, a rapid detailed surface geochemical sampling programme was conducted over Gunung Tumpangpitu resulting in seven targets being identified for drilling. An initial drilling programme of 5 diamond drill holes – GT-001 to GT-005 – was conducted during March – June 1999.

In February 2000 Placer Dome Inc. (Placer) entered into a Joint Venture with Golden Valley Mines to earn 51% of the project and assumed operational control of the exploration programme. In order to better define targets for follow-up drilling on Tumpangpitu 32.75 kilometres of grid-based geochemical and IP surveys were completed between April-May 2000. Anomalous bedrock geochemistry demonstrated marked consistency with prominent ridges or topographic highs, trending to the northwest, consisting dominantly of vuggy silica altered breccia.

The results of the IP survey demonstrated strong correlation between the near-surface resistivity anomalies and the outcropping vuggy silica zones. Deeper chargeability anomalies (>200-400 m below surface) were recorded in the northern portion of the grid. Placer targeted the shallow resistivity anomalies for high sulphidation style Au-Ag mineralisation with a further 10 diamond drill holes – GT-006 to GT-014.

On the basis of the results from the second drilling programme a further 14 holes were designed (2,700m). However, Placer withdrew from the project possibly due to the combined influences of the relatively low metal prices at the time (i.e., the project did not appear to meet corporate thresholds of size and grade) together with an unstable economic and political climate across much of south-east Asia (the Asian Financial Crisis).

There is no report or record of further work being conducted on the project by Placer-GVM and the area became vacant by the time PT. IMN applied for a KP General Survey in 2006 over the project area.

In June 2006 Hellman and Schofield Pty Ltd (“H&S”, an independent geological consulting group from Australia) assisted in assembling exploration data and designing a drilling programme aimed at advancing the Tumpangpitu prospect in order to report resource estimates according to the JORC Code and Guidelines.

H&S were able to provide an indication of Potential Mineralisation within the variably oxidised gold-silver enriched zone above the deeper copper mineralisation by using the limited available drilling data along with soil sample geochemical results. This study
suggested that approximately 3m oz AuEq ("AuEq" was based on $650/Oz Au and $10/Oz Ag) was a reasonable amalgamated target size in prospects (Zones) A, B & C.

Overall indications of potential may be expressed using cautionary language and with grade and tonnage ranges (see Clauses 16 & 17 of the JORC Code and Section 2.3 (2) of NI43-101). It should never be assumed that suggested grades and tonnages from these types of studies will be realized, they are solely used in the context of understanding the types of drilling targets and broad scale of mineralisation.

On March 30, 2007 a Term Sheet was signed between Emperor Mines Ltd. (later to become Intrepid Mines Ltd. through the merger of Emperor Mines and Intrepid Mines) and PT IMN, which was followed by an Alliance Agreement in August 2007. Drilling on the project by PT IMN and Intrepid Mines commenced in September 2007 with hole GTD-07-015.

Additional historical drill hole assays became available between February and August 2007 enabling a more quantitative view of the geological potential. The September 2007 Hellman & Schofield study of Geological Potential used Ordinary Block Kriging of 2m composited AuEq data within polygon extrusions.

This report documents the drilling completed by PT. IMN and Intrepid during 2008 in Zone A, and the subsequent work by Hellman and Schofield in estimating a mineral resource for the Zone A oxide mineralisation.

9. GEOLOGICAL SETTING


10. DEPOSIT TYPES


11. MINERALISATION

An overview of high-sulphidation mineralization at Tumpangpitu was given in Section 11.1 of the September 2008 NI43-101 report (Hellman 2008).

Details of mineralisation at Zone C at Tumpangpitu are provided in Technical Report dated September 2008 (Sections 11.2 to 11.7 inclusive).

Details of lithology, alteration and mineralisation at Zone A at Tumpangpitu are presented below.
11.1 Zone A Lithology and Alteration

Drilling at Zone A was conducted on six (6) cross-sections based on a local grid oriented at 050-230° and spaced at 80 metre intervals as depicted in Figure 1 below (10960mN, 11040mN, 11120mN, 11200mN, 11280mN and 11360mN). Zone A lies over the peak of Gunung (Mt) Tumpangpitu.

![Figure 1. Location of drill cross-sections at Zone A, Tumpangpitu prospect.](image)

Locations are shown relative to topography (contours are 10m and 100m).

Plotted in Figures 2 to 13 are a series of lithology and alteration cross-sections through Zone A. Also plotted on both these sections are the base of complete oxidation (BOCO), the base of semi-oxidation (BOSO), assay histograms for Au and Cu, and a line graph of Ag assays.

Review of the lithology cross-sections (Figures 2, 4, 6, 8, 10 and 12) allow the following observations pertaining to lithology:

1) The principal lithological unit identified on all cross-sections comprises a thick and poorly differentiated to undifferentiated unit of lithic-crystal tuffs (Plates 1 and 2), though local variants comprise crystal tuffs (Plate 3). These tuffs exhibit some local
variation in lithic clast size and are invariably altered with varying intensity of advanced argillic and argillic alteration (Plate 4). The tuffaceous matrix to these lithic tuffs tends to be crystal-rich, imparting a matrix-supported texture to the volcanic breccias.

Because of the massive nature of the lithic tuffs, their intense alteration and the presence of rare and very narrow finer-grained crystal-rich variants, no correlation between drill holes has been possible to date other than identification of the thick altered package of volcanic breccias in the upper parts of each hole. The dip of this volcanic breccia sequence has not been determined from logging of drill holes. However, their apparent dip on section might be inferred from the dip of alteration zones identified within the package, assuming hydrothermal solutions have migrated parallel to volcanic bedding. These internal alteration zones are more clearly defined and suggest an apparent dip to the southwest on section.

Statistical data presented in Section 19 (Table 12) indicate that the lithic tuff (LtU) unit has the highest average Au values of all the sampled lithological units (excluding TBx – tuffisite breccia which has only 2 data-points) and it has the greatest number of assay intervals by approximately an order of magnitude (64.7% of assay intervals). This data indicates that the lithic tuff sequence hosts the bulk of the Au mineralization at Zone A.
Plate 2 – Matrix-supported lithic-crystal tuff from GTD-34 (Zone A)
*With strong alignment of flattened flame-like pyroclasts.*

Plate 3 – Oxidised and silica-clay altered crystal tuff in GTD-49 (19.0m) - Zone A.
Plate 4 – Progressive stages of advanced argillic alteration (and oxidation) of lithic-crystal tuffs in GTD-49 (Zone A).

Left-to-right: Hcy-si > Hsi-cy > vu-Hsi-cy > vu-Hsi.

2) An andesite body is encountered at depth in hole GTD-46 on section 11200mN (Figure 9). This andesite may be part of the original volcanic package that may have comprised alternating sequences of andesitic flows and lithic-crystal tuffs (volcanic breccias).

3) Two major types of intrusive bodies are identified on the Zone A cross-sections. A dominant series of feldspar porphyry intrusives that are identified on 4 sections (11040mN, 11200mN, 11280mN and 11360mN) and a subordinate dacitic intrusion that is only identified on section 11040mN. However, hydrothermal breccias associated with the upper parts of these intrusions are seen on the remaining two cross-sections.

4) The drill holes on cross-section 11200mN span a larger lateral area and greater vertical depth. Hence this section encountered the more extensive roots of these intrusions and had a greater proportion of feldspar-porphyry (fP) intrusions on section.

5) The feldspar-porphyry intrusions characteristically comprise coarse feldspar phenocrysts that are set in a microcrystalline matrix. Locally they contain quartz
phenocrysts of equivalent size to the feldspar phenocrysts, though these tend to be rare. There appears to be a paucity of coarse grained phenocrysts of mafic composition in the feldspar porphyry intrusions, so the subordinate mafic component in these felsic intrusions is likely to reside in the groundmass. These intrusions may equate to the tonalitic rocks described by Anthony Coote of Applied Petrographic Services (APS) from earlier petrographic studies of rocks from Zone C, and from holes to the immediate south. These “tonalites” were identified by APS as being one of a series of high-level intrusive bodies on the Tumpangpitu prospect with have varying composition.

6) The feldspar porphyry intrusions have intruded the primary volcanic sequence of lithic-crystal tuffs and subordinate andesite. These intrusions predate the main phase of acid alteration since they are commonly overprinted by varying intensities of acid alteration. Whilst alteration of these coarse grained feldspar porphyry intrusions is commonly propylitic (due to lower inherent permeability to later acidic fluids), there are zones of Hcy-si and Hsi-cy alteration that cross-cut and overprint the feldspar porphyry intrusions. Examples of these alteration overprints can be seen on all sections that contain the feldspar porphyry bodies (11040mN, 11200mN, 11280mN and 11360mN; cf Figures 4-5, 8-9, 10-11 and 12-13).

7) A series of matrix-supported hydrothermal breccias (ms-HBX) have been logged in many of the drill holes in Zone A and one area of intrusion breccia (Ibx) has been identified in GTD-1B (Figure 8). Interpretation of the cross-sections has revealed that these breccias tend to lie consistently along the upper contacts or carapace zone of the feldspar porphyry and dacite porphyry intrusions. This is quite evident on section 11040mN (Figure 4) and 11200mN (Figure 8). It is likely that the feldspar porphyry and dacite porphyry intrusions are transitional upward to zones of intrusive breccia around the carapace where magma is injected into the roof of the intrusions, and these inturn are transitional upward to fluidized matrix-supported hydrothermal breccias where exsolving fluids from the intrusion were forcefully injected along fractures and channels into the overlying lithic tuff sequence. It is worth noting that intrusion breccias have been identified in some drill holes (Coote 2007a).

The presence of matrix-supported hydrothermal breccias at depth on section 11120mN (Figure 6) suggests the proximity of the bottom of the holes to underlying intrusions.

Matrix-supported hydrothermal breccias have been logged in drill core near surface on section 11280 mN in holes GTD-36, GTD-37 and GTD-38 and display apparent continuity between holes along a SW-dipping zone (Figure 10). This unit may have been miss-logged as a hydrothermal breccia when it is plausibly a distinctive facies of the lithic-crystal tuff sequence. If so, then this may be tentative evidence of a dip direction for the lithic tuffs that controlled the upflow and outflow of hydrothermal fluids, as defined by the parallel zoned alteration assemblages.
Figure 2 (top) & 3 (below). Lithology and Alteration section 10960mN (local grid).
Figure 4 (top) & 5 (below). Lithology and Alteration section 11040mN (local grid).
Figure 6 (top) & 7 (below). Lithology and Alteration section 11120mN (local grid).
Figure 8 (top) & 9 (below). Lithology and Alteration section 11200mN (local grid).
Figure 10 (top) & 11 (below). Lithology and Alteration section 11280mN (local grid).
Figure 12 (top) & 13 (below). Lithology and Alteration section 11360mN (local grid).
Review of the alteration cross-sections (Figures 3, 5, 7, 9, 11 and 13) permit the following observations:

1) The alteration zones on the cross-sections are interpreted to have a tabular morphology, with an apparent dip of ~ 20-30° to the southwest. The tabular or planar morphology of the alteration zones is consistent with a model of acidic fluids migrating up-dip along permeable stratigraphic bedding (lithic tuffs).

2) The alteration is broadly zoned (from deep SW to shallow NE) from cores of massive silica (Hsi) and silica-alunite (Hsi-al) progressively upward and outward to silica-clay-alunite (Hsi-cy-al), silica-clay (Hsi-cy), clay-silica (Hcy-si), hydrothermal clay (Hcy) and then to clay-chlorite (CC) or propylitic alteration (PRO).

The silica and alunite-rich alteration end-members (Hsi, Hsi-al, Hsi-cy-al and Hsi-cy) reflect increased rock interaction with hotter and more acidic fluids whilst the clay-rich and/or chlorite-bearing end-members (Hcy-si, Hcy, CC and PRO) are increasingly neutral alteration assemblages that formed from less acidic or increasingly neutral fluids that have partially cooled, diluted and/or partly neutralised by rock interaction.

The alteration zones show trends from acid assemblages at depth to the southwest of the cross-sections and are transitional to increasingly neutral assemblages at shallower levels to the northeast. These trends, together with the dominantly propylitic and chlorite-clay assemblages that lie at depth on the northeast-side of the sections, suggest a fluid source that emanated at depth to the southwest of Zone A. The alteration trends interpreted at Zone A are a near mirror image of those identified at Zone C (Rohrlach 2008), both of which establish vectors to a higher grade high-sulphidation system and a local porphyry source located at depth between Zones A and C (see Figure 14).

3) All of the alteration cross-sections at Zone A show some preferential development of alunite near the surface within a silica-clay-alunite alteration facies (Hsi-cy-al). This is quite evident on section 11120mN and 11200mN. These surficial zones may reflect local formation of alunite by weathering processes (supergene alunite).

4) Figure 7 illustrates that the highly significant Au intersection in GTD-49 coincides with an area of massive silica alteration that lies above the base of complete oxidation (BOCO). Similar correlation between Au and Hsi units that lie above BOCO occur in hole GTD-47 (Figure 5) and at the base of GTD-50 (Figure 9). Statistical data presented in Section 19 of this report highlights the effect of oxidation in upgrading primary Au grades at Zone A, corroborating interpretations in the model section shown in the September NI43-101 report and updated in Figure 14 below.
11.2 Zone A Mineralization

Au-Ag mineralization at Zone A occurs predominantly within the oxide and semi-oxide environment, hosted within lithic-crystal tuffs. Oxide mineralization is associated with oxidised limonitic and goethitic veins and fractures that cross-cut the lithic-crystal tuff sequence and also within vugs generated by acid leaching of lithic clasts and feldspar crystal fragments within this volcanic breccia sequence.

Plates 5 to 11 below illustrate the morphology of the oxide mineralization at Zone A, as typified by the textures observed in drill hole GTD-49 which was particularly well mineralized (166-410m: 244m @ 2.04 g/t Au, 47.33 g/t Ag [2.77 g/t Au equiv]).
Plate 5 – Oxidised limonite-goethite ex-sulphide veins cross-cutting acid-leached and silicified vuggy lithic-crystal tuffs (GTD-49; 75.8m).
Grade: 1.71 g/t Au (Interval 74-76m).

Plate 6 – Oxidised gossanous massive goethite vein cross-cutting acid-leached and silicified vuggy lithic-crystal tuffs (GTD-49; 83.15m).
Grade: 1.37 g/t Au (Interval 82-84m).
Plate 7 – Part of an 80cm-wide zone of chaotic limonite and low-temperature cherty silica veining, with silica locally rich in very fine grained sulphide (GTD-49; 132.30m).
Grade: 3.24 g/t Au (Interval 132-134m).

Plate 8 – Vuggy massive silica (vu-Hsi) altered lithic tuff with ghosted cherty quartz stringers & silicified zones at ~15° to core axis (GTD-49; 235.60m).
Grade: 4.63 g/t Au (Interval 234-236m).
Plate 9 – Vuggy massive silica (vu-Hsi) altered lithic tuff with ghosted cherty quartz stringers & silicified zones at ~15° to core axis (GTD-49; 242.00m).
Grade: 4.52 g/t Au/55 g/t Ag and 3.1 g/t Au/79 g/t Ag (Interval 240-242m and 242-244m).

Plate 10 – Vuggy massive silica (vu-Hsi) alteration of lithic tuff cross-cut by a dense network of oxidised, limonitic, crackle breccia veins (GTD-49; 303-305m).
Grade: 5.56 g/t Au/0.10 % Cu and 5.08 g/t Au/0.12 % Cu (Interval 302-304m and 304-306m).
**Plate 11 – Massive limonite-goethite vein formed by oxidation of a high-sulphidation stage massive sulphide vein (GTD-49; 330.20m).**
Grade: 2.43 g/t Au / 23 g/t Ag (Interval 330-332m).

### 12. EXPLORATION


Since involvement of PT. IMN in the Tujuh Bukit project (2006-2008) and the involvement of Intrepid Mining Ltd (formerly Emperor Mines Ltd) in 2007-2008 the following exploration programs have been undertaken over the Tumpangpitu prospect:

1) Re-establishment of the Tumpangpitu grid (initially established by Placer).

2) Completion of 475 soil grid samples at a density of 200m x 25m over the Tumpangpitu prospect (Figure 16). The soil samples were acquired along 17 cross-lines oriented at 050°-230° magnetic. Soil samples were analysed for Au, Cu, Pb, Zn, Ag, As, Sb, Mo and Ba.

3) Regional rock-chip sampling:

   2006-2007: Tumpangpitu Southeast (26 samples).
   Tumpangpitu (33 samples).
Salakan (94 samples).
Rajeg Besi (4 samples).
Lampon (7 samples).

2008:
Tumpangpitu (38 samples).
Salakan (118 samples).

4) Reconnaissance lithological and alteration mapping at Salakan. Field reconnaissance visits to areas on the Tujuh Bukit property.

5) Preparation for and subsequent completion of diamond drilling of the first-phase resource-definition programme at Tumpangpitu, as proposed by Hellman and Schofield. Eighteen (18) diamond drillholes were completed in 2007-2008 at the Zone C area of Tumpangpitu; (Section 13). A total of 7074 metres of drilling were completed at Zone C in 2007 and 2008. This involved preparation of drill pads, developing logistic supply lines and procedures, organising accounting procedures and designing the database to process the various data associated with the programme. The programme involved PT. IMN professional personnel (geologists, logistic managers and accountants) as well as local labour employed on a daily basis.

An Inferred Resource was calculated by Hellman and Schofield on the basis of this drilling at Zone C and was reported in the NI. 43-101 technical report dated September 2008. The inaugural Inferred Resource estimated for Zone C by Hellman and Schofield was:

Table 1. Summary of Zone C Inferred Resource as reported in September 2008.

<table>
<thead>
<tr>
<th>Cut-Off (Au-equiv)</th>
<th>Tonnes (million)</th>
<th>Au-Equiv (g/t)</th>
<th>Cu (%)</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Oz Eq (million)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5</td>
<td>36.5</td>
<td>0.92</td>
<td>0.10</td>
<td>0.53</td>
<td>25</td>
<td>1.1</td>
</tr>
<tr>
<td>1.00</td>
<td>10.0</td>
<td>1.54</td>
<td>0.12</td>
<td>0.79</td>
<td>49</td>
<td>0.5</td>
</tr>
<tr>
<td>1.50</td>
<td>3.74</td>
<td>2.10</td>
<td>0.05</td>
<td>0.90</td>
<td>78</td>
<td>0.3</td>
</tr>
</tbody>
</table>

The Au-equivalent ratio for Ag is set at 65:1 based on US$650/oz gold and US$10/oz Ag.

6) Preparation for and subsequent completion of diamond drilling of the second-phase resource-definition programme at Tumpangpitu. Eighteen (18) diamond drill holes were completed in 2008 at the Zone A area of Tumpangpitu; (Section 13). This involved preparation of drill pads, modifying logistic supply lines and procedures, and reviewing the database to process the various data associated with the programme. The programme involved PT. IMN professional personnel (geologists, logistic managers and accountants) as well as local labour employed on a daily basis.

The aim of the second phase diamond drilling programme was to test the northwest strike potential of the Au-enriched oxide zone along the advanced argillic altered ridge that trends through Zone A (Figure 1). Zone A was the second area to be tested following
completion of the first pass of drilling at Zone C. Historical drilling at Zone A had identified gold – silver mineralisation in holes GTD-1, GTD-10 and GTD-12.

A second inferred resource was calculated by Hellman and Schofield on the basis of this subsequent phase of drilling at Zone A, and is presented in Section 19 of this NI.43-101 technical report. The inferred resource that was calculated for Zone A by Hellman and Schofield was:

Table 2. Summary of Zone A Inferred Resource as reported in Section 19.

<table>
<thead>
<tr>
<th>Cut-Off (Au-equiv)</th>
<th>Tonnes</th>
<th>Au-Equiv (g/t)</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Oz Eq (million)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5</td>
<td>43,556,919</td>
<td>1.05</td>
<td>0.62</td>
<td>27.8</td>
<td>1.47</td>
</tr>
<tr>
<td>0.75</td>
<td>24,349,305</td>
<td>1.40</td>
<td>0.86</td>
<td>34.8</td>
<td>1.10</td>
</tr>
<tr>
<td>1.00</td>
<td>16,325,249</td>
<td>1.67</td>
<td>1.08</td>
<td>38.1</td>
<td>0.88</td>
</tr>
</tbody>
</table>

The Au-equivalent ratio for Ag is set at 65:1 based on US$650/oz gold and US$10/oz Ag.

7) Reprocessing and 3-D inversion modelling of existing aeromagnetic data over the property (400m-spaced flight-line data; Figure 15), plus acquisition and processing of ground magnetic data over the southern portion of the Tumpangpitu prospect was undertaken in 2008. The aeromagnetic data and the ground magnetic data were levelled and merged into a single image over the Tumpangpitu magnetic batholith (Figure 15). Reprocessing of existing IP data that was acquired by Placer in 2000 was also undertaken and is ongoing.
Figure 15. Plan of processed aeromagnetic data (analytical signal) merged with reprocessed ground magnetic data over areas of drilling.

This image covers the Tumpangpitu Prospect located in the south-eastern part of the Tujuh Bukit project. Prospects for oxide gold-silver mineralisation are shown in yellow, as Zones A, B & C. Collar positions for deep drill holes targeting copper-gold mineralisation are shown in red.

Following completion of the first-pass drilling at Zone C on the 8th May 2008, drilling commenced at the Zone A area (Figure 16 and 17), initially with pairs of holes drilled with azimuths of 230° on six cross-sections centered on the historical hole GTD-10 (135m of 0.84 g/t Au). Following completion of first pass drilling at Zone A and updating of the resource calculation, drilling moved to the Zone B area to test the first of two soil gold anomalies which lie south of Zone A (Figure 16).

Concomitant with the diamond drilling program at Zone A, several deep diamond drill holes were completed to test for deep hypogene porphyry Cu-Au mineralisation and higher-grade primary high-sulphidation mineralization beneath the oxidised silica cap at depths of 400-800m. These holes [GTD-35, GTD-40 (failed hole), GTD-42, GTD-46 and GTD-53; Figure 15] were targeted to test deep-seated IP chargeability anomalies and magnetic features beneath the shallow resistivity anomalies at Tumpangpitu.
13. DRILLING

Intrepid Mines Ltd and their Joint Venture partner Indo Multi Niaga (IMN), conducted a diamond drilling program in the Zone A area of the Tumpangpitu prospect during 2008. Eighteen (18) diamond drillholes were completed by Intrepid-IMN between 24th April 2008 and 31st October 2008 (holes GTD-07-33 to 34, 36 to 41 and 43 to 52). The total drill meterage by Intrepid-IMN was 7328.45m for these Zone A holes. These holes are in addition to the 18 holes drilled by Intrepid Mines Ltd in the Zone C area that were reported in September 2008. The location of these drill holes is shown in Figure 17.

The drill holes by Intrepid-IMN at Zone C and at Zone A were designed to test the extent and continuity of oxidised high-sulphidation epithermal mineralisation (Au-Ag) at shallow levels of the Tumpangpitu prospect. In positioning the series of drill holes in the Zone C and Zone A areas, Intrepid-IMN reviewed all existing data, including surface alteration data from prior mapping by Placer, previous drilling results of Golden Valley Mines and Placer (7 holes; GT-004, GTD-11, GT-001 (A & B), GT-002, GT-010, GT-012 and GTD-
14), chargeability and resistivity anomalies from a prospect-scale IP survey conducted by Placer, and the results of repeat and follow-up soil sampling over the Tumpangpitu prospect conducted by IMN in early 2007. The drill holes in the Zone A area were targeted primarily around intersections encountered in the three Golden Valley Mines and Placer drill holes (GT-001, GT-010 and GT-012) in conjunction with grid soil Au anomalies in the area that were defined by the IMN survey.

![Figure 17. Location of drill holes, Zone A.](image)

Details of the drilling program are outlined below. Collar coordinates and hole depths are listed in Table 3 and in Appendix 1.
Table 3. Drill hole details for holes at Zone A, Tumpangpitu.

<table>
<thead>
<tr>
<th>Core Hole ID</th>
<th>Company</th>
<th>Collar Easting</th>
<th>Collar Northing</th>
<th>Collar RL</th>
<th>Azimuth (UTM)</th>
<th>Dip</th>
<th>EOH Depth</th>
</tr>
</thead>
<tbody>
<tr>
<td>GT-001A</td>
<td>GVM</td>
<td>174536.488</td>
<td>9046858.059</td>
<td>336.762</td>
<td>244.5</td>
<td>-45</td>
<td>72.30</td>
</tr>
<tr>
<td>GT-001B</td>
<td>GVM</td>
<td>174536.488</td>
<td>9046858.059</td>
<td>336.762</td>
<td>244.5</td>
<td>-45</td>
<td>500.50</td>
</tr>
<tr>
<td>GTD-010</td>
<td>Placer</td>
<td>174314.574</td>
<td>9046734.893</td>
<td>481.637</td>
<td>229.5</td>
<td>-80</td>
<td>329.50</td>
</tr>
<tr>
<td>GTD-012</td>
<td>Placer</td>
<td>174493.993</td>
<td>9046596.733</td>
<td>441.516</td>
<td>229.5</td>
<td>-50</td>
<td>318.10</td>
</tr>
<tr>
<td>GTD-33</td>
<td>Intrepid-IMN JV</td>
<td>174361.095</td>
<td>9046765.625</td>
<td>465.716</td>
<td>229.5</td>
<td>-60</td>
<td>360.10</td>
</tr>
<tr>
<td>GTD-34</td>
<td>Intrepid-IMN JV</td>
<td>174360.809</td>
<td>9046765.565</td>
<td>465.736</td>
<td>229.5</td>
<td>-80</td>
<td>274.50</td>
</tr>
<tr>
<td>GTD-36</td>
<td>Intrepid-IMN JV</td>
<td>174266.363</td>
<td>9046799.091</td>
<td>429.468</td>
<td>229.5</td>
<td>-60</td>
<td>433.20</td>
</tr>
<tr>
<td>GTD-37</td>
<td>Intrepid-IMN JV</td>
<td>174316.578</td>
<td>9046846.398</td>
<td>417.138</td>
<td>229.5</td>
<td>-60</td>
<td>436.85</td>
</tr>
<tr>
<td>GTD-38</td>
<td>Intrepid-IMN JV</td>
<td>174214.325</td>
<td>9046857.268</td>
<td>399.134</td>
<td>229.5</td>
<td>-60</td>
<td>401.85</td>
</tr>
<tr>
<td>GTD-39</td>
<td>Intrepid-IMN JV</td>
<td>174259.238</td>
<td>9046893.481</td>
<td>390.407</td>
<td>229.5</td>
<td>-60</td>
<td>373.65</td>
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<tr>
<td>GTD-40</td>
<td>Intrepid-IMN JV</td>
<td>174081.028</td>
<td>9046550.240</td>
<td>256.885</td>
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<td>-60</td>
<td>220.55</td>
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<tr>
<td>GTD-41</td>
<td>Intrepid-IMN JV</td>
<td>174365.250</td>
<td>9046670.284</td>
<td>485.241</td>
<td>229.5</td>
<td>-60</td>
<td>432.30</td>
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<tr>
<td>GTD-43</td>
<td>Intrepid-IMN JV</td>
<td>174414.715</td>
<td>9046708.435</td>
<td>471.475</td>
<td>229.5</td>
<td>-60</td>
<td>439.70</td>
</tr>
<tr>
<td>GTD-44</td>
<td>Intrepid-IMN JV</td>
<td>174474.331</td>
<td>9046649.836</td>
<td>454.868</td>
<td>229.5</td>
<td>-60</td>
<td>443.30</td>
</tr>
<tr>
<td>GTD-45</td>
<td>Intrepid-IMN JV</td>
<td>174475.401</td>
<td>9046650.199</td>
<td>454.438</td>
<td>49.5</td>
<td>-60</td>
<td>435.80</td>
</tr>
<tr>
<td>GTD-46</td>
<td>Intrepid-IMN JV</td>
<td>174511.976</td>
<td>9046871.592</td>
<td>338.149</td>
<td>229.5</td>
<td>-70</td>
<td>843.15</td>
</tr>
<tr>
<td>GTD-47</td>
<td>Intrepid-IMN JV</td>
<td>174429.043</td>
<td>9046611.389</td>
<td>457.763</td>
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<td>-60</td>
<td>435.15</td>
</tr>
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<td>GTD-48</td>
<td>Intrepid-IMN JV</td>
<td>174314.798</td>
<td>9046846.414</td>
<td>417.116</td>
<td>49.5</td>
<td>-60</td>
<td>411.75</td>
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<tr>
<td>GTD-49</td>
<td>Intrepid-IMN JV</td>
<td>174364.698</td>
<td>9046668.696</td>
<td>485.382</td>
<td>229.5</td>
<td>-50</td>
<td>487.80</td>
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<td>GTD-50</td>
<td>Intrepid-IMN JV</td>
<td>174316.177</td>
<td>9046734.477</td>
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<td>229.5</td>
<td>-50</td>
<td>322.55</td>
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<tr>
<td>GTD-51</td>
<td>Intrepid-IMN JV</td>
<td>174372.237</td>
<td>9046767.220</td>
<td>465.161</td>
<td>49.5</td>
<td>-70</td>
<td>301.40</td>
</tr>
<tr>
<td>GTD-52</td>
<td>Intrepid-IMN JV</td>
<td>174485.986</td>
<td>9046564.648</td>
<td>435.890</td>
<td>229.5</td>
<td>-66</td>
<td>274.85</td>
</tr>
</tbody>
</table>

13.1 Drilling Contractors and Drilling Statistics

The drilling contractor used during the drilling program conducted by Intrepid-IMN was PT. Maxidrill. PT. Maxidrill is based in Jakarta.

The company address of the drilling contractor is:

PT. Maxidrill Indonesia.
Jl. Gatot Subroto Km. 8.
Jatake, Tangerang, Banten 15137
Telephone: 62-21 5913583-6
Facsimile: 62-21 5918780
e-mail:  info@maxidrill.net
website:  www.maxidrill.net

Table 4 shows some relevant statistics of the drilling program conducted by PT. Maxidrill for Intrepid-IMN at Zone A, Tumpangpitu. The depths of the diamond holes ranged between 274.5m and 487.8m, with an average depth of 381.49m (excluding hole GTD-46 which had an end-of-hole depth of 843.15m).
Table 4. Number of core samples assayed per core size (Zone A; 2008 drill holes).

<table>
<thead>
<tr>
<th>Core Size</th>
<th>Meterage</th>
<th>Production / Shift</th>
<th>No. Samples Assayed</th>
</tr>
</thead>
<tbody>
<tr>
<td>PQ</td>
<td>1712.4</td>
<td>13.17m / 8 hr shift</td>
<td>845</td>
</tr>
<tr>
<td>HQ</td>
<td>2925.8</td>
<td>12.67m / 8 hr shift</td>
<td>1473</td>
</tr>
<tr>
<td>NQ</td>
<td>2423.2</td>
<td>8.75m / 8 hr shift</td>
<td>1212</td>
</tr>
<tr>
<td>BQ</td>
<td>267.05</td>
<td>7.42m / 8 hr shift</td>
<td>133</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>7328.45</strong></td>
<td><strong>m / 8 hr shift</strong></td>
<td><strong>3663</strong></td>
</tr>
</tbody>
</table>

The total drill meterage used in the resource calculation for Zone A, which also include four holes drilled by Golden valley Mines and Placer (GT-001A, GT-001B, GTD-010 and GTD-012) is 8548.85m.

13.2 Drilling Equipment

PT. Maxidrill used two drill-rigs during the Zone A drilling program in 2008. They are the MD-400 rig (Plates 12 and 13; shown drilling on the Zone A and Zone C areas) and the MD-420 drill rig which were manufactured by Maxidrill in Indonesia. Both drill rigs are skid-mounted and man-portable, breaking down into pieces that can be hauled manually between drill sites. The drill rigs are moved by a team of around 80-100 haulers, taking around a day or two to move site, depending on the distance to the next drill site. The MD-400 drill-rig has been able to drill to between 400-450m consistently at Tumpangpitu using NQ after reducing from PQ and HQ higher in the holes. The MD-420 drill-rig is rated by PT Maxidrill as being capable to drill to 0-150m (PQ), 0-450m (HQ) and 0-700m (NQ). The MD-420 drill-rig has recently drilled to 849.20 m depth (after reducing to BQ) on a drill-hole outside of the Zone A drill-grid area.

The cores were retrieved using triple-tube sampling and core sizes drilled were PQ-3 (83 mm diameter) from surface, with reduction to HQ-3 (61.7 mm) and NQ-3 (45 mm) at depth.
Plate 12.  PT. Maxidrill rig MD-400 at drill-site GTD-38 at Zone A (15th June 2008).

Plate 13.  PT. Maxidrill rig MD-400 at drill-site GTD-23 at Zone C (29th January 2008).
13.3 Production Rates

Diamond drilling at Zone A at Tumpangpitu was conducted on three (3) 8-hour shifts per 24 hour day. Table 5 below lists the average production rates per 8-hour shift for PQ, HQ and NQ core sizes for the Zone A drilling program, and the average total production rate for these core sizes per hole is also listed. The production rates exclude setting up and pulling out time at the start and end of each hole. The average production rate of diamond drilling at Zone A is 13.17m / 8-hour shift for PQ, 12.67m / 8-hour shift for HQ, 8.75m / 8-hour shift for NQ core and 7.42m / 8-hour shift for BQ core.

Table 5. Production statistics for Zone A core drilling at Tumpangpitu (2008).

<table>
<thead>
<tr>
<th>Rig</th>
<th>Start Date</th>
<th>End Date</th>
<th>PQ (m)</th>
<th>HQ (m)</th>
<th>NQ (m)</th>
<th>BQ (m)</th>
<th>PQ (m/shift)</th>
<th>HQ (m/shift)</th>
<th>NQ (m/shift)</th>
<th>BQ (m/shift)</th>
<th>Average Production (m/8hr shift)</th>
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<td></td>
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<td>12.67</td>
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<td>10.87m/shift</td>
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</tbody>
</table>

Production statistics exclude pull-out time at end of drill hole. Shifts are 8-hour shifts.

13.4 Downhole Surveys

A total of 153 down-hole surveys were conducted in all 22 drill holes that were used in the Zone A resource calculation (including historical holes GT-001A, GT-001B, GTD-10 and GTD-12). Down-hole survey data existed for the historical holes GT-001A, GT-001B, GTD-010 and GTD-012 although it is not known what type of survey tool was used for these 4 holes (it is assumed that the survey data were recorded using the widely used Eastman single-shot system). All drill holes drilled at Zone A by Intrepid in 2008 were
surveyed using a REFLEX EZ-Shot™ down-hole survey instrument which recorded azimuth, inclination, roll-face angle, magnetic field strength and bore-hole temperature. All down-hole survey data are listed in Appendix 2.

13.5 Drill Hole Collar Survey and Topographic Survey

The collar position of drill-holes at Zone A and the surface topography in the Zone A area were surveyed by PT. GEOINDO GRI JAYA, whose contact details are listed below.

PT. GEOINDO GRI JAYA  
Jl. Batununggal Indah IV No.83  
Bandung 40266 – Indonesia  
Telp : 022 7513168, 7538775,  
Fax : 022 7513776  
Contacts : Mr. Robert Bacciarelli and Mr. Darwis Legawa.

Three survey teams were involved with surveying of hole collars and topography between August 2008 and December 2008 in conjunction with other survey tasks.

The first team leader was supported by 4 topographic surveyors equipped with three Leica TC 803 and one TC 1100 total station units and were mobilised to carry out survey work on the 30\textsuperscript{th} August 2008. The second team leader was supported by 3 GPS surveyors equipped with two GPS Leica 1200 units and one GPS Leica SR 501 unit, and were mobilised on the 2\textsuperscript{nd} November 2008 to carry out investigation and audit of reference control points and to establish a larger and more suitable geodetic control network for the survey area. Following the work of an investigation team, a third team was mobilised on the 23\textsuperscript{nd} November 2008 for re-survey work with 4 topographic surveyors and 1 data processing surveyor, equipped with three Leica 803 total station units.

Ten new benchmarks were installed and surveyed as geodetic control (Appendix 9). One new Benchmark BM INT 08 located in the concession area and was tied to BM Bakosurtanal N1.0232 and was used as a reference point for the entire survey area. Static Differential DGPS method was carried out over a two hour period to determine coordinates and elevation. Benchmark N1.0232 is located at Grajagan, about 20 km from the concession area.

Subsequently, 9 new benchmarks were established and surveyed with a Static Differential Global Positioning System (DGPS) network method to tie in to existing benchmark BM INT 08. Coordinates and elevations of Benchmarks are listed in Table 6. A Leica DGPS system 1200 with dual frequency and 12 channel accuracy was used for the survey.

The new benchmarks were made permanent by cementing a concrete-filled piece of 4 inch diameter PVC into a 30x30cm concrete block that extends 60cm below and 40cm above ground surface. A 17-inch bolt with an inscribed bolt was cemented into the concrete-filled PVC tube of the benchmarks (see Appendix 9).
Technical specification of equipment that was used is as follows:
Model: Leica System 1200
Sensor Type: Dual frequency, 12 channel
Accuracy: 2 mm + 0.5 ppm
Data was processed using our Static Kinematics GPS program Leica Geo Office V 1.1.

A traverse survey was also carried out by total station to determine the coordinates of control points that were used as reference points for the topographic survey and which in turn were tied to the 2 reference benchmarks established by DGPS and tied to Bakosurtanal reference datum.

A closed loop traverse was carried out and tied in to a minimum of two GPS control points. Redundant observations were carried out to obtain angle and distance data. A minimum 2 sets of angle and distance data in 1st face and 2nd face were recorded. All data observation was automatically recorded on PCMCIA cards built into the total station units to avoid manual recording errors.

The Bowditch / Transit method was applied to obtain definitive coordinates.

Technical specification of the equipment used is as follows:
Model: Leica TC 1100 and TC 803 series
Angle Standard Deviation: 3” both vertical and horizontal
Distance: 2 mm + 2 ppm
Recording media: PCMCIA Card 8 MB

Survey parameters that were used for control benchmarks and the latest topographic survey were as follows:

• Reference Ellipsoid : WGS84
• Projection : UTM Zone 50 South of Hemisphere
• False Northing : 10000000
• False Easting : 500000
• Central Meridian (CM) : 117º East
• Scale Factor on CM : 0.9996
• Units : Meters

<table>
<thead>
<tr>
<th>Point ID</th>
<th>Geographic Coordinates</th>
<th>UTM Coordinates</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
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<td></td>
<td>Latitude</td>
<td>Longitude</td>
<td>Ellips. Height (m)</td>
</tr>
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Note:
- Ellipsoid Reference: WGS-84
- Projection: UTM
- Zone: 50 South

Four survey teams carried out the topographic survey over an initial area of about 107 Ha. This area for topographic surveying was increased by Intrepid Mines Ltd to an additional 215 Ha by using the old Local Grid baseline and cross-lines as reference points whilst they were being surveyed. The total area surveyed for topography was 322 Ha.

The topographic survey was carried out by using the cross-section method, with 100m between cross-sections and with 50m interval points. The area mainly consisted of dense vegetation and forest and steep undulating terrain.

The detailed topographic survey was carried out using one TC 1100 and three TC 803 total station units. Technical specification of the equipment which was used is as follows:

- Model: Leica TC/TCA 1100 and 803 series
- Angle Standard: 2” both vertical and horizontal
- Distance: 2 mm + 2 ppm
- Recording Media: PCMCIA Card 512 kb and internal memory

The positions and elevations of 22 drilled holes in the Zone A area were surveyed using total station survey equipment and tied to the established geodetic network with an accuracy of 1-cm in both position and elevation. The coordinates of these drill holes are shown in 7.
Table 7. Drill hole coordinates for Zone A as surveyed by PT. GEOINDO.

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Plates 14 and 15. GEOINDO Benchmarks.
Left Benchmark INT-8 which is part of the Geodetic network established on the Tumpangpitu prospect site by GEOINDO surveyors. Right - GEOINDO conducting a traverse survey from hole GTD-14, immediately south of Zone A.

13.6 Summary Results of Drilling

Summary intercepts of the drill holes from the Zone A area are provided in Table 8. Note: These results show the cumulative thickness and grade above a cut-off for each drill-hole, so no spatial continuity distribution is implied in the table. However, due to the style of mineralisation, the relationship of true thickness to apparent thickness is less critical than for strongly directional mineralisation.

There are areas in Zone A where significantly higher grades are observed. Table 9 tabulates some selected intervals of substantially higher than average Au grade.
### Accumulated Intercepts Exceeding 0.3 g/t Au

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<td>10</td>
<td>0.42</td>
<td>4</td>
</tr>
<tr>
<td>GTD-08-46</td>
<td>42</td>
<td>0.47</td>
<td>5.43</td>
</tr>
<tr>
<td>GTD-08-47</td>
<td>78</td>
<td>0.60</td>
<td>29.65</td>
</tr>
<tr>
<td>GTD-08-48</td>
<td>14</td>
<td>1.03</td>
<td>1.14</td>
</tr>
<tr>
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<td>349.8</td>
<td>1.79</td>
<td>34.84</td>
</tr>
<tr>
<td>GTD-08-50</td>
<td>176.55</td>
<td>0.94</td>
<td>7.45</td>
</tr>
<tr>
<td>GTD-08-51</td>
<td>54</td>
<td>0.90</td>
<td>6.41</td>
</tr>
<tr>
<td>GTD-08-52</td>
<td>22</td>
<td>0.85</td>
<td>79.14</td>
</tr>
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</table>

### Accumulated Intercepts Exceeding 20 g/t Ag

<table>
<thead>
<tr>
<th>Drill Hole</th>
<th>Length</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>GT-001A</td>
<td>2</td>
<td>0.1</td>
<td>131</td>
</tr>
<tr>
<td>GT-001B</td>
<td>7.3</td>
<td>0.17</td>
<td>29.78</td>
</tr>
<tr>
<td>GTD-010</td>
<td>82.5</td>
<td>0.71</td>
<td>40.85</td>
</tr>
<tr>
<td>GTD-012</td>
<td>48</td>
<td>0.15</td>
<td>107</td>
</tr>
<tr>
<td>GTD-08-33</td>
<td>112</td>
<td>0.37</td>
<td>50.70</td>
</tr>
<tr>
<td>GTD-08-34</td>
<td>44</td>
<td>1.08</td>
<td>33.23</td>
</tr>
<tr>
<td>GTD-08-36</td>
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<td>0.25</td>
<td>53.23</td>
</tr>
<tr>
<td>GTD-08-37</td>
<td>34</td>
<td>0.13</td>
<td>59.29</td>
</tr>
<tr>
<td>GTD-08-38</td>
<td>44</td>
<td>0.12</td>
<td>45.45</td>
</tr>
<tr>
<td>GTD-08-39</td>
<td>24</td>
<td>0.11</td>
<td>28.75</td>
</tr>
<tr>
<td>GTD-08-40</td>
<td>22.55</td>
<td>0.51</td>
<td>33.06</td>
</tr>
<tr>
<td>GTD-08-41</td>
<td>86</td>
<td>1.36</td>
<td>82.30</td>
</tr>
<tr>
<td>GTD-08-43</td>
<td>126</td>
<td>0.34</td>
<td>69.38</td>
</tr>
<tr>
<td>GTD-08-44</td>
<td>76</td>
<td>0.15</td>
<td>39.58</td>
</tr>
<tr>
<td>GTD-08-45</td>
<td>60</td>
<td>0.09</td>
<td>41.50</td>
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<td>GTD-08-46</td>
<td>6</td>
<td>0.27</td>
<td>27.67</td>
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<tr>
<td>GTD-08-47</td>
<td>73.15</td>
<td>0.21</td>
<td>51.09</td>
</tr>
<tr>
<td>GTD-08-48</td>
<td>13.15</td>
<td>0.17</td>
<td>32.22</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>139.8</td>
<td>2.10</td>
<td>80.66</td>
</tr>
<tr>
<td>GTD-08-50</td>
<td>22</td>
<td>0.48</td>
<td>48.64</td>
</tr>
<tr>
<td>GTD-08-51</td>
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<td>0.32</td>
<td>33</td>
</tr>
<tr>
<td>GTD-08-52</td>
<td>72</td>
<td>0.29</td>
<td>117.53</td>
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</table>
Table 9. Selection of higher grade Au intercepts at Zone A

<table>
<thead>
<tr>
<th>Hole</th>
<th>From</th>
<th>To</th>
<th>Length</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>GTD-010</td>
<td>2</td>
<td>6</td>
<td>4m</td>
<td>7.22</td>
<td>30.5</td>
</tr>
<tr>
<td>GTD-08-33</td>
<td>28</td>
<td>40</td>
<td>12m</td>
<td>4.93</td>
<td>12.0</td>
</tr>
<tr>
<td>GTD-08-34</td>
<td>38</td>
<td>56</td>
<td>18m</td>
<td>8.05</td>
<td>17.89</td>
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<tr>
<td>incls</td>
<td>40</td>
<td>44</td>
<td>4m</td>
<td>21.35</td>
<td>2.5</td>
</tr>
<tr>
<td>GTD-08-34</td>
<td>102</td>
<td>106</td>
<td>4m</td>
<td>5.50</td>
<td>11.0</td>
</tr>
<tr>
<td>GTD-08-41</td>
<td>168</td>
<td>180</td>
<td>12m</td>
<td>6.29</td>
<td>178</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>108</td>
<td>110</td>
<td>2m</td>
<td>6.45</td>
<td>20</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>166</td>
<td>172</td>
<td>6m</td>
<td>6.20</td>
<td>35</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>232</td>
<td>256</td>
<td>24m</td>
<td>4.13</td>
<td>76.58</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>300</td>
<td>308</td>
<td>8m</td>
<td>4.92</td>
<td>11.25</td>
</tr>
<tr>
<td>GTD-08-49</td>
<td>326</td>
<td>330</td>
<td>4m</td>
<td>5.28</td>
<td>45.5</td>
</tr>
<tr>
<td>GTD-08-50</td>
<td>2</td>
<td>4</td>
<td>2m</td>
<td>8.03</td>
<td>15</td>
</tr>
<tr>
<td>GTD-08-50</td>
<td>40</td>
<td>42</td>
<td>2m</td>
<td>7.88</td>
<td>18</td>
</tr>
</tbody>
</table>

14. SAMPLING METHOD AND APPROACH

All drill holes on the Tujuh Bukit project area conducted to date, by GVM-Placer and by Intrepid-IMN have been drilled by the diamond drilling method. Consequently two types of samples are collected for assay during the drill program at Tumpangpitu, half-core samples of PQ, HQ, NQ and BQ core and three-metre composite sludge samples.

14.1 Core Processing Protocols

The drill core is acquired in a triple-tube assembly which is used by both the MD-400 and the MD-420 drill rigs. Prior to sampling of the drill core, trained local core technicians measure the core recovery at the drill site (per drill run) and mark up the core trays before placing the core trays in sealed wooden boxes for carrying to the core processing facility located on the prospect. The drill-rig core technicians, trained by IMN, fill out a Field Geotech Form at the drill rig. This form records run depths and core recovery data.

The drill-rig-based core technicians accompany and rotate in tandem with the three daily drill shifts. The data recorded on the Field Geotech Form is oriented primarily to capture core recovery as soon as possible after core is drilled. Columns in the Field Geotech Form include the drill shift (I, II or III), the Run No. (sequential numbers 1→X), WSL, S/U, From, To, Recovery Drilled (= advancement meterage), Recovery (Actual), % Recovery and Comments.

The Field Geotech forms for each drill rig are delivered daily to the core shed supervisor by the rig-based core technician who has been rostered at the drill rig on the night shift.
(shift III). The forms are held at the core shed until drilling, sampling and processing of the drill hole has been completed, and then are dispatched to the site office at Pulau Merah for filing together with other relevant drill hole data at the site office.

Prior to transport of core from the drill rig to the core shed, the core trays are packed with plastic bag inserts to prevent core movement during manual transport from the drill site to the core shed. Core from some holes is carried up to a kilometre in distance.

When the core boxes arrive at the core shed, a core technician fills out a Tray List. This form records the Hole ID, the Tray number (1→X), From, To, Core Size (PQ, HQ, NQ, BQ) and a column to indicate if the core tray has been photographed. Following entry of details into the Tray Form, the core is carefully washed in situ and then each core tray is digitally photographed on a wooden frame. Typically 2 boxes are photographed in a single photographic frame. A label across the core box records the hole number, date and the from and to intervals for each core box.

Once the core box photos have been taken, the photo column in the Tray List is marked to indicate completion of core photography. The Tray List is held on site at the core shed until sampling of the drill hole has been completed, after which it is dispatched to the site office for data entry into IMN’s digital database. Digital core photographs are transferred to a USB memory module and also dispatched to the site office in Pulau Merah for archiving. Each photograph is given a file name that reflects the hole ID and the from-to interval of the photographed core in each image (e.g. GTD-08-27-39.50-45.03.jpg).

Following core photography, a Geotech Log is filled in by several trained core technicians (under guidance from PT. IMN geologists) at the core sampling facility. The Geotech Log in current use records the Hole ID, From and To intervals (per run), Drill_Int (= From minus To, or length of run), Recovery (m), RQD, Hardness (1-5), Fractures (fractures per run), CFA (core fracture angle), CFO, Fracture Style, Core Size and Comments. Fracture style is rarely recorded however is recorded in the detailed geology logs. The Geotech Logs are also held on site at the core shed until the drill hole has been completely sampled. The Geotech Logs are then sent down to the site office at Pulau Merah and are key-punched into hole-specific Excel spreadsheets and also into a composite Access database called “GeotechLog” by a data-entry clerk.

Following completion of the Geotech Log, PT. IMN geologists conduct detailed geological logging of the drillcore. Following completion of logging, the geologists mark up the core for sampling in conjunction with the core technicians. During this process the core is visually assessed to ensure that the half of the core marked for sampling is representative of the contained mineralisation.

A Sample Number Form is then filled in by the core shed supervisor. The form is used primarily to record the sample intervals and assigned sample numbers and is used for both drill core and sludge samples. The form records Hole ID, SNO (sample number), From, To, Interval (typically 2m), sample type (core or sludge), size core (PQ, HQ, NQ, BQ) and reason (e.g. GCA – geochemical analysis; Met – metallurgy and Dup – duplicate). IMN-designed sample ticket books are used. The position for insertion of analytical standards in
the sample string is recorded by the core shed supervisor on the Sample Number Form. Typically a standard is inserted for every 20 unknown samples. The Sample Number Form is held at the core shed until the entire hole has been sampled, then the forms are sent to the site office at Pulau Merah where the data are key-punched into a hole-specific Excel spreadsheet and also into a composite Access database called “SampleDrilling” for drill core samples and “Sludge_Sample_Numbers” for sludge samples.

14.2 Measurement of Specific Gravity

Prior to sampling, segments of core were measured for specific gravity (SG) at the prospect site core shed. The specific gravity data were typically acquired on 10cm-long segments of whole core prior to splitting. These drill core density measurements were made on site at Tumpangpitu by trained Indonesian geotechnicians employed by IMN. A total of 1258 SG determinations have been acquired to date from holes GTD-33 to 34, GTD-36 to 41 and GTD-43 to 52. This represents 34.3% of the total intervals assayed. The SG measurements were taken at near regular intervals of every 5 metres down-hole, equating to roughly one SG determination per tray of drill core. Where the rock interval was fractured and friable, the spacing of SG measurements was locally extended beyond 5 metre intervals.

All measurements of SG on drill core from Zone A were made by Intrepid-IMN using the waxed core method. Samples were first dried in a 1600-watt (220-240V) Kris Electric Oven with a 30 litre capacity for 4 hours at 100°C. SG data acquired by IMN were recorded on a Specific Gravity Form which recorded Hole ID, From (m), To (m), Interval (m; = From-To; typically 0.1m), Wt_Air (weight of unwaxed core in air), Wt_Waxed_Air (weight of waxed core in air), Wt_Waxed_Water (weight of waxed core in water), SG and Comments. The completed forms for each drill hole were dispatched to the site office where the data were keypunched into hole-specific Excel databases.

14.3 Sampling Intervals

The drill holes in the Zone A area that comprise the resource estimate that is the subject of this report were drilled on an approximately 80 x 80m grid along six (6) cross-sections that are oriented at 050-230° from magnetic north (Figure 17). The collars of the holes in Zone A lie within a northwest-trending area of approximately 400m x 80m (excluding the collars to holes GT-001A/B and GTD-46 which lie further northeast of the main Zone A drill grid area, and hole GTD-40 which lies southwest of the main Zone A drill grid area).

Of the 22 drill holes which have been drilled into, adjacent to and below the Zone A area, holes GT-001A and GT-001B were drilled by Golden Valley Mines (GVM) whilst holes GTD-10 and GTD-12 were drilled by Placer.

The entire length of hole GT-001A was sampled by GVM at variable intervals between 0.95m and 3m. Hole GT-001B was sampled at variable intervals ranging from 0.75m to 3.75m in length from 48m depth through to end-of-hole. The interval from surface to 48m depth was not sampled since representative sample was acquired from failed hole GT-001A which was collared at the same site. Drill-hole GTD-10 was sampled in entirety by
Placer at intervals of predominantly 2m although with some intervals ranging from 1.2m to 3.1m. Drill hole GTD-12 was also sampled in entirety, at 2m intervals.

Of the remaining 18 drill holes which were drilled by Maxidrill for IMN-Intrepid in 2008, the entire drill-holes were sampled at predominantly 2m intervals.

A total of 4174 assay intervals were used in the Zone A resource estimate (excluding hole GTD-35 which was also used in the resource). Of these, 511 assay intervals comprise assays from Golden Valley Mines and Placer (~12.2%) whilst 3663 assay intervals comprise assays generated by Intrepid-IMN (~87.8%).

Table 10 lists the number of samples assayed for each sampling interval employed in the Zone A drilling.

### Table 10. Number of core samples assayed per sampling interval (Zone A).

<table>
<thead>
<tr>
<th>Sampling Length</th>
<th>No. of Samples</th>
<th>% of Samples</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5 to &lt;1m</td>
<td>3</td>
<td>0.072</td>
</tr>
<tr>
<td>1m</td>
<td>2</td>
<td>0.048</td>
</tr>
<tr>
<td>&gt;1m and &lt; 2m</td>
<td>31</td>
<td>0.743</td>
</tr>
<tr>
<td>2m</td>
<td>3950</td>
<td>94.633</td>
</tr>
<tr>
<td>&gt;2m and &lt; 3m</td>
<td>38</td>
<td>0.910</td>
</tr>
<tr>
<td>3m</td>
<td>141</td>
<td>3.378</td>
</tr>
<tr>
<td>&gt; 3m and &lt; 4m</td>
<td>8</td>
<td>0.192</td>
</tr>
<tr>
<td>4m</td>
<td>1</td>
<td>0.024</td>
</tr>
<tr>
<td><strong>4174 samples</strong></td>
<td></td>
<td><strong>100%</strong></td>
</tr>
</tbody>
</table>

The core marked for splitting was cut lengthways down the middle (irrespective of size; PQ, HQ, NQ) using a diamond core saw, and half of the core was placed into a calico bag with respective sample number tag placed inside with the core and the sample number written on the outside of the calico bag. The other half of the cut core was left in the core box (also with a sample number tag stapled to the side of the box) as a permanent physical record of the drill core.

### 14.4 Core Recovery Data

Core recoveries during the diamond drilling program at Zone A are shown in Figure 18. The average core recovery over the duration of the Zone A drilling program in 2008 was high at 98.1%.
14.5 Comparison of Sludge Samples versus Core Samples

Comparison of assays from core versus sludge assays was undertaken to check for any bias that might be induced in the assays due to circulation of drilling fluids through porous, leached, friable and oxidized rock that might preferentially flush components of the core that have a higher or lower average grade.

The higher grade Au-Ag mineralisation in the Zone A area appears to be correlated with increasing intensity of oxidation within the vuggy silica, silica-clay, silica-alunite and silica-clay-alunite alteration zones. This oxidation creates hematitic and goethitic vug fills and porous vuggy limonite and goethite veins and fractures. Consequently it is possible that some mineralisation may be lost during core drilling, due to the flushing of circulating drilling fluids through porous zones of the oxidised rock.

In an attempt to gain some measure of this, samples of the sludges were collected in a sump that was designed to capture drill cuttings from the water return. Samples were collected at 3 metre intervals, coinciding with the drilling of each drill rod. This procedure is only effective as a measure of grade in the sludge material if constant water return is achieved during the drilling of the mineralized zone – a difficult task given the highly porous and fractured nature of the rock. Hence plots of sludge assays versus core assays for the same intervals can be expected to have significant variance. This scatter is also partly created by the unequal sampling intervals (3m for sludges and 1-2m for cores). Only core samples that are wholly contained within a given sludge sample interval are plotted in Figure 25. Nevertheless, the plots shown in Figure 19 provide some indication as to whether grade is being over-estimated or underestimated in the core. Because the weight of sludge samples is small relative to the core samples and because the grades of both are
typically low, the effective gain or loss of grade between core and sludge is not likely to be materially significant.

Figure 19. Plots of Au, Ag and Cu in core and in corresponding sludge samples for Zone A.
429 sample intervals from Zone A (log scale).

The Au values in core and in corresponding sludges from Zone A are dispersed about the 1:1 line, but with a slight bias to higher Au grades in the sludge samples. This suggests that the “insitu” drill core may be slightly higher grade than is indicated by assaying in the laboratory once the core has been drilled and sampled. However, the relatively small weight proportion of sludge relative to core would suggest that this effect is minor.

The Ag values tend to be slightly enriched in the sludges relative to core when the dataset is viewed as a whole, suggesting that if there is a bias, it is in the direction of inducing lower values in drill core once the rock has been drilled and sampled.

The Cu however shows more systematic deviations between sludges and core at grades below about 200ppm Cu (where Cu is enriched in the sludges) and at grades above about 0.1% Cu (where Cu is enriched in the core). At low grades below 200ppm Cu this deviation is not materially significant since the rocks are far from ore grade with respect to Cu. At grades above 0.1% Cu, the lower Cu values in sludges suggests that the sludges contain, on average, more Cu-deficient material than the average core, resulting in a very slight enrichment in Cu in the residual core. This variation is not likely to be materially significant given the low weight of sludge relative to core. The relationship between Cu grades in sludge and core samples for Zone A is identical to that observed in the Zone C dataset (Hellman, September 2008).

14.6 Controls on Mineralisation and Basis of Sampling Intervals

The major rock-types and alteration zones in the Zone A drill grid area were discussed in detail in Section 11 of this report. High- to intermediate-sulphidation epithermal mineralisation is hosted largely within a thick sequence of medium to coarse grained
volcanic lithic-crystal tuffs (pyroclastic breccias) and significantly lesser crystal tuffs and porphyry intrusives. The pyroclastic sequence may exert a strong stratigraphic control on the geometry and orientation of alteration zones within the area drilled. Zones of massive silica alteration (Hsi), silica-alunite (Hsi-al), silica-clay (Hsi-cy) and silica-clay-alunite (Hsi-cy + al) are associated with high-sulphidation epithermal Au-Ag mineralisation above the base of oxidation (BOCO) in the oxide zone, and also below BOCO but above the base of semi-oxidation (BOSO), in the transitional zone. These alteration zones are also associated with disseminated and fracture-controlled chalocite-bearing Au-Ag-Cu mineralisation below the base of semi-oxidation, in the sulphide zone.

There appear to be two main local geological controls on the distribution of mineralisation identified within drilling in the Zone A area to date:

Surface oxidation has resulted in a substantial upgrade of originally low-grade hypogene mineralisation, producing Au + Ag enrichment and Cu depletion.

The advanced argillic alteration sequence dips moderately to the southwest and is zoned outward from tabular and stratigraphic-controlled inner zones of silica and silica-alunite alteration, to increasing clay components (silica-clay, silica-clay-alunite and clay-silica). This zonation likely reflects decreasing degrees of acid-leaching and increasing dilution of acidic magmatic vapour condensates during fluid mixing on the margins of stratigraphic-controlled upflow-outflow zones.

Mineralisation within the oxide zone, transitional zone and within the sulphide zone tends to be of relatively consistent grade within each of these zones respectively. The mineralized intervals within these zones tend to span lengths that often exceed 50 metres in apparent width. Because of the relative consistency of the mineralisation grade over extensive intervals, as well as the disseminated nature of much of the mineralisation (as disseminated vug fill and as widespread fine fractures and micro-veinlets), the sampling was mostly at regular 2 metre intervals. Demonstration of the wide nature of the mineralized zones and their good grade continuity supports the choice of a 2m sampling interval.

Whilst many of the fractures and veinlets are pervasive and widespread through the core in the mineralized intervals, there are local and narrow (1-2m) zones of semi-massive (15-30%) pyrite-chalcocite veins within the sulphide zone at depth. These more massive veins tend to be rare and isolated, with most veins occurring as narrow micro-veinlets and sulphide-lined fractures. Thus in the sulphide zone there are some local intersections of higher Cu grade that contrast with the surrounding lower-grade hypogene Cu mineralisation however these are relatively isolated and minor. In this respect, the style of mineralisation encountered in Zone A is broadly similar to that encountered previously in Zone C.
15. SAMPLING PREPARATION, ANALYSES AND SECURITY

15.1 Sample Splitting, Packaging and Labelling

During the sampling procedure, the diamond drill core is initially cut using an electric-powered, water-cooled diamond-bladed core cutter located at the core storage facility at the Tumpangpitu prospect (Plate 17). All core was halved for assay. During the cutting of core, intervals of significantly broken core were initially wrapped in plastic and sealed with tape prior to cutting on the core saw to minimize breakage and to prevent parts of the sample being washed away during core cutting. Intervals of core which were extremely clay-rich and broken or friable were sampled by a spatula and spoon.

Split core was sampled into calico sample bags, the sample number was written with permanent marker on the outside of the sample bag and the sample number ticket-stubs were inserted into the calico bags used for sampling. The sample numbers were recorded on the Sample Number Form for core (or sludges). The bagged split-core samples were subsequently packed into rice-sacks and manually hauled to the Pulau Merah site office at the end of each day (Plate 20).

The beginning and end of each 2-metre sample interval in the core trays are recorded by stapling one of the three ticket stubs against the intervening partitions in the core tray, and were labeled according to sample number and depth.

(Left) Black polyurethane core boxes (from Zone A drilling) arrayed on logging racks in the core yard at the Tumpangpitu prospect. (Right) Core sawing at the core shed site at the Tumpangpitu prospect.
Plate 18. Sampling of split core into numbered calico bags.

Plate 19. Example of sample stub booklet
with pre-numbered sample stubs that are labelled and placed in calico sample bags with the split core samples.
15.2 Procedures Employed to Ensure Sample Integrity

The following are some of the procedures employed by Intrepid-IMN to ensure sample integrity during the diamond drilling program:

- Drill-rig geotechnicians were assigned to each coring rig, on every shift, to record core recoveries and to ensure that core was appropriately handled and packed into the core boxes after each core run, and to ensure that the core boxes were appropriately labelled. They oversaw the retrieval of drill core from the core tubes, placement of core in core boxes, security strapping of the core boxes and they organised manual transport of the core to the core yard.

- Diamond core boxes were packed with plastic inserts during manual transport from the drill rig to the core yard to minimize breakage of the core prior to logging and sampling.

- All diamond core trays were photographed as routine documentation of the core samples. In the most recent drill holes, the core is photographed both in the dry state and after it has been wetted.

- Diamond drill core that was broken or friable was cut only when the core had been wrapped tightly in plastic and tape to ensure fragments were not lost during core splitting.
• Drill cores were stored in sturdy black polyurethane core boxes marked with permanent markers.

• IMN sample number stubs were used to label each drill sample (Plate 19).

• The core yard on the prospect site has 24 hour security, with two local employees assigned to secure the core yard from 5pm to 7am each day. Core samples that were not on the core racks were stored in a lockable building in the core storage facility.

• Internal sample dispatch log books were used to track samples that were sent from the core yard on the prospect site to the Pulau Merah site office.

• Prior to sending samples to the Intertek Laboratory, all sample bags and number strings are checked for continuity and sample bag integrity.

15.3 Use of IMN Employees in Sampling Procedures

Trained PT.IMN employees were involved at all stages of the sampling, sample packaging and sample transportation process. During the diamond drilling program, an IMN employee was based full-time at the drill rig site to supervise the core handling procedures and to document core recoveries. During the diamond drilling program one PT. IMN geologist would visit each drill rig approximately once per day during the course of the Zone A drilling program. A number of haulers were employed to assist with transporting drill core from the drill rig to the core yard. All core handling procedures in the core yard were undertaken by trained geotechnicians employed by PT. IMN and supervised by PT. IMN geologists.

15.4 Sample Security and Transport

Split core samples that are carried down from the prospect (Plate 20) were received at the sample storage and dispatch area at the site office in Pulau Merah and were signed into a log-book by IMN employees to ensure complete transfer.

1. The core sample receiving and dispatch area at Pulau Merah was kept under lock during evening hours and there were always IMN staff present during daylight hours.

2. When an entire set of samples from a single drill hole had accumulated in the storage area and the drilling contractors supply truck was due to backload samples to Jakarta (Plate 21), the samples in storage were sorted and checked for completeness. The rice sacks, used to transport the calico bags of split core from the core yard on the prospect area to the sample receiving area at Pulau Merah, were opened and the calico bags (labelled with sample numbers) were laid out in numerical order (Plate 22). The samples were checked for completeness and integrity of the sample number labeling and calico bag condition. Sets of 4-6 adjacent samples were weighed (Plate 25) and then repacked in new empty rice sacks (Plate 26). The total weight of the re-packed sacks, the sack number, and the
sample numbers of samples within each sack were recorded in a sample dispatch log (Plate 27). Certified reference standards and analytical blanks were inserted where appropriate (Plate 24). The rice sacks were sealed with heavy gauge wire twists as security (Plate 28). The rice sacks that contain the core samples were pre-labeled with the sack number as well as the To and From interval and the address of the laboratory (Plate 23).

3. Samples, either drill core samples or sludge samples, were sent as whole drill-hole batches to Intertek with an Intertek Sample Submission Form.

4. Sealed samples were then transported to Jakarta in a Mitsubishi truck owned by the drill contractor, PT. Maxidrill, at approximately 2 week intervals as the truck was routinely back-loaded to Jakarta after delivering drilling consumables to site. A PT. IMN employee accompanies the samples on each trip from Pulau Merah to the Intertek Laboratory to ensure security of the samples en-route to the laboratory.

5. Intertek routinely email IMN-Intrepid a Sample Receipt Confirmation note for every Sample Submission Batch that they receive at their laboratory in Jakarta, confirming receipt of samples and noting any irregularities in the received samples.

Plate 21 and 22. Core dispatch

*Left* - Maxidrill truck used to transport samples from Pulau Merah to the Intertek laboratory in Jakarta. *Right* - Sorting samples at Pulau Merah before dispatch to Intertek.
Plate 23 and 24. Labelling
Left - Labelling of rice sacks that are used to transport split core samples in calico bags to the Intertek laboratory. Right - Example of a certified reference standard that is inserted into a plastic bag which in turn is inserted into a labelled calico bag with an assigned dummy sample number that is contiguous with the core sample number strings.

Plate 25 and 26. Sample weighing and checking.
Left - Weighing of checked samples prior to repacking. Right - re-packing of samples into labelled rice-sacks ready for sealing with metal twist ties.
Plate 27 and 28. Sample recording and despatch.
Left - Sample dispatch log book to monitor details of samples sent to Intertek. Right - Applying security wire twist-ties to sample bags prior to dispatch to Intertek.

15.5 Analytical Laboratories

Three analytical laboratories were used for analysis of samples generated by the drilling programs at Tumpangpitu.

The principal laboratory was: PT. Intertek Utama Services (Jakarta). The Intertek laboratory generated all of the primary assay data pertinent to the drill programs that are the subject of this report. The address of the Intertek laboratory in Jakarta is:

PT. Intertek Utama Services.
Cilandak Commercial Estate 103E,
JI Cilandak KKO,
Jakarta 12560.
Tel: (632) 819-5841 to 48.
Contact – Ms Becky Torre.

PT. Intertek Utama Services is accredited for chemical testing under ISO 17025:2005 (General requirements for the competence of testing and calibration laboratories) by the Komite Akreditasi National (KAN). Their Accreditation Number is LP-130-IDN (renewed on 30 April 2007) and is equivalent to the NATA certification in Australia.

Two secondary laboratories were used as independent checks on the Intertek laboratory for the Zones A and C diamond drilling programs.

The first check laboratory was McPhar Geoservices (Philippines) Inc. located in Manila. McPhar is an ISO-9001:2000 accredited laboratory and has been servicing the Philippines
Mining Industry since 1971. The principal chemist in charge is Art del Mundo (lab@mcphar.com.ph). McPhar are currently seeking accreditation for ISO-17025 which is specific to analytical laboratories. The address of the McPhar laboratory is:

Office: 3/F P & L Bldg, 116 Legaspi St, Legaspi Village, Makati City (tel: 8158191 to 94).
Assay Laboratory: 1869 P.Domingo St, Makati City, Manila (tel: 8961656, -1681, -7973).
Postal Address: PO BOX 7356, Domestic Airport Post Office, Domestic Rd, Pasay City 1300, Metro-Manila, Philippines – Fax: 8158195.

Subsets of samples from Zone C and Zone A were independently sent to the McPhar Laboratory in Manila for check analyses.

The second check laboratory was ALS Chemex (ABN 84009936029).
Check assays for the Zone C drilling reported in September 2008 were undertaken at the McPhar Laboratory in Manila. Subsequent to the September 2008 NI43-101 technical report on the Zone C inferred resource, the same check samples from Zone C were also sent to ALS Chemex in Perth.

The check samples from Zone A were later sent as a separate batch to ALS Chemex in Brisbane. The samples for fire assay (Au) were then dispatched by ALS Chemex from Brisbane to their Townsville laboratory.

The addresses of these 2 ALS Chemex laboratories are:

ALS Chemex (Perth).
32 Oxleigh Drive
Malaga WA 6090
Australia
Tel: 08-93473222  Fx: 08-93473232

ALS Chemex (Brisbane).
32 Shand St
Stafford QLD 4053
Australia
Tel: 07-32437222  Fx: 07-32437218

15.6 Analytical Methods

Samples received at the Intertek laboratory are checked by laboratory staff against the accompanying Sample Dispatch Sheet. Any discrepancy that is noted in sample numbers is brought to the attention of the company via an emailed Sample Receipt Notification prior to preparing the batch for sample preparation. Samples are submitted to Intertek with Sample Preparation Code PB01. Samples are initially dried (105°C) for as long as it takes to achieve constant weight, and then jaw crushed to minus 5mm. The samples are then riffle split with part stored as a coarse reject. A split of 1-1.5kg was pulverized with 95%
passing 75um. A 250g grab sample is taken from the pulverized pulp and used for the analysis while the remainder is stored as spare pulp.

PB01 - Dry (105°C), Crush (95% <5mm), Riffle Split, Pulverize (95% <75um).

Table 11 summarises the elements that were assayed for each sample, the method of analysis and the detection limit for each method.

Table 11. Method and detection limits for elements analysed in the Zone A drilling program.

<table>
<thead>
<tr>
<th>Element</th>
<th>Method</th>
<th>Code</th>
<th>Det. Limit (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>Fire Assay (30g or 50g)</td>
<td>FA30</td>
<td>0.01 (FA30) and 0.005 (FA50)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>FA50</td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>Atomic Absorption Spectroscopy (AAS)</td>
<td>GA02</td>
<td>2</td>
</tr>
<tr>
<td>Pb</td>
<td>Atomic Absorption Spectroscopy (AAS)</td>
<td>GA02</td>
<td>4</td>
</tr>
<tr>
<td>Zn</td>
<td>Atomic Absorption Spectroscopy (AAS)</td>
<td>GA02</td>
<td>2</td>
</tr>
<tr>
<td>Ag</td>
<td>Atomic Absorption Spectroscopy (AAS)</td>
<td>GA02</td>
<td>1</td>
</tr>
<tr>
<td>As</td>
<td>X-Ray Fluoresence (XRF)</td>
<td>XR01</td>
<td>1</td>
</tr>
<tr>
<td>Sb</td>
<td>X-Ray Fluoresence (XRF)</td>
<td>XR01</td>
<td>1</td>
</tr>
<tr>
<td>Mo</td>
<td>X-Ray Fluoresence (XRF)</td>
<td>XR01</td>
<td>1</td>
</tr>
<tr>
<td>Ba</td>
<td>X-Ray Fluoresence (XRF)</td>
<td>XR01</td>
<td>10</td>
</tr>
<tr>
<td>S</td>
<td>Leco</td>
<td>ST01</td>
<td>0.01 %</td>
</tr>
</tbody>
</table>

The fire assay method was used for all Au assays at Tumpangpitu. Most of the samples from the Zone A 2008 drilling program (88.9%) were assayed by fire assay on a 30 gram aliquot (FA30), however samples from two holes (11.1% of samples; Holes GTD-07-39 and 43) were analysed by fire assay on a 50 gram aliquot (FA50). Recommended ranges for these methods are (FA30: 0.01 – 30 g/t Au; FA50: 0.005 – 50 g/t Au). Assays greater than 50 g/t Au are automatically re-assayed using the FA12 method (Fire Assay followed by gravimetric analysis).

The Fire Assay schemes involve fusing the sample with a litharge based flux and collecting the precious metals in a lead button. After cupellation the resulting prill is dissolved in aqua regia and the gold is determined by AAS for routine samples. Samples exceeding the upper limits of the method are automatically re-assayed utilising a high grade gravimetric method (Intertek 2007).

**FA30:** 30g of sample is analysed. This scheme is suitable for higher concentration samples, those generated in ore body definition drilling and samples with a sulphide and/or copper matrix. Detection Limit (DL) 0.01ppm, Upper Limit (UL) 30ppm (Intertek 2007).

**FA50:** 50g of sample is analysed. This scheme is suitable for exploration samples when the identification of anomalous samples is important. DL 0.005ppm, UL 50ppm. A
recommended option for grassroots exploration is the use of a new fire assay pot for each sample (Intertek 2007).

For Zone A holes drilled by Intrepid-IMN, a total of 3256 samples were assayed by FA30 (GTD 33-34, 36-38, 40-41 and 44-52) while 407 samples were assayed by FA50 (GTD-39 and GTD-43).

The assays of Ag as well as the base metals Cu, Pb and Zn were conducted by atomic absorption spectroscopy (AAS). The laboratory code for this procedure is GA02. Hydrochloric/perchloric digestion (HCL/HClO$_4$) was used for the first pass geochemical analysis of the samples. Silicates are only slightly digested during this procedure.

Table 11 lists the lower detection limits for these elements (Cu - 2 ppm; Pb – 4 ppm; n – 2 ppm; Ag – 1 ppm). The recommended ranges for the GA02 method are Cu, 2-10,000 ppm; Pb, 4-4,000 ppm; Zn, 1-10,000 ppm and Ag, 1-120 ppm.

For samples that exceeded upper limits, the samples were re-assayed by GA30. Triple acid digestion (HCl/HNO$_3$/HClO$_4$) is used for ‘ore-grade’ digestions and is followed by an accurate volumetric finish to enable high concentrations of elements to be analysed. Limitations may still exist with silicates. The lower detection limits for this method are Cu (0.1 ppm), Pb (0.1 ppm), Zn (0.1 ppm) and Ag (5 ppm).

Analyses of As, Sb, Mo and Ba were conducted by X-ray fluorescence on 10g pressed pellets. The laboratory code for this procedure is XR01. The recommended ranges for the XR01 method are As, 1-10,000 ppm; Sb, 1-10,000 ppm; Mo, 1-10,000 ppm and Ba, 10-10,000 ppm. Detection limits are 1 ppm for As, Sb and Mo and 10 ppm for Ba.

Elements that assayed over-range were automatically assayed by the XR02 procedure. The detection limits for the XR02 method for As and Ba are 100 ppm.

Sulphur (S) was analysed using a Leco analyser with detection limits of 0.01%.

15.7 QA/QC Procedures Employed

A total of 3663 drill core intervals and 461 sludge sample intervals were assayed by Intertek from the Zone A drilling conducted by Intrepid-IMN. The principal QA-QC procedures undertaken by Intrepid-IMN and the external laboratories during analysis of these samples comprised:

1) Submission of OREAS Au, Ag & Cu standards by Intrepid-IM with drill core samples. (n=156; 1 in 23.5 samples).

2) Submission of Au, Ag & Cu standards by Intrepid-IMN with sludge samples. (n=11; 1 in 41.9 samples).

3) Submission of 2 types of analytical blanks submitted by Intrepid-IMN with drill core samples (n=60; 1 in 60 samples).
4) Analysis of external standards inserted by Intertek during analysis of drill cores (n=1071; 1 in 3.4 samples).

5) Analysis of external blanks inserted by Intertek during analysis of drill cores (n=534; 1 in 6.8 samples).

6) Analysis of external standards inserted by Intertek during analysis of sludge samples (n=12; 1 in 38.4 unknowns).

7) Analysis of analytical blanks inserted by Intertek during analysis of sludge samples (n=17; 1 in 27 unknowns).

8) Analysis of laboratory replicates.

9) Analysis of laboratory splits.

10) Analysis of 181 Intertek pulps at two check laboratories (1 in 20.2 samples; ~5% of samples).

11) Analysis of 54 field duplicate samples (1 in 67.8 samples; ~1.5% of samples).

Routine quality control procedures used by Intertek for sample preparation include:

1) a 1 in 15 re-split at the sample preparation stage,

2) 1 in 20 samples undergo sieve analysis to monitor grind size and the use of a barren wash is standard in both crushing and pulverising procedures,

3) use of a CCLAS LIMS system which provides built in sample tracking and quality control as well as automatic data capture from the instruments, reducing the risk of data entry errors. This is complimented by a bar-coding system in the Jakarta Laboratory,

4) routine QC generated by the LIMS includes 2nd splits at the sample preparation stage, as well as 2 replicates, 2 reference standards and one blank per batch of 50 samples,

5) laboratory supervisors select additional QC depending on first-pass results.

Internal Standards

Internal standards or certified reference materials (CRM’s) were inserted into the sample strings at a frequency of one standard for every ~20-25 core samples. A total of 156 standards were submitted for the Zone A drilling, resulting in a CRM insertion frequency of about 4.25%. Eight different types of internal standards were routinely submitted by IMN-Intrepid for the Zone A drilling. They were all OREAS standards sourced from Ore Research & Exploration PL, 6-8 Gatwick Road, Bayswater North, Victoria 3153, Australia. Certificates of Analysis for these 8 Certified Reference Materials are attached as Appendix 6.
The standards were purchased as pulps that were pre-sealed in air-tight foil packets labeled with the standard name/number. Prior to insertion of the standards into the IMN-Intrepid sample stream, the label of the CRM was erased from the foil packet using turpentine, and the CRM was then assigned a sample number consistent with the IMN sample string. The assigned sample number was also written on the calico bag in which the foil standard was inserted. These calico bags, as well as the calico bags containing the submitted blanks, were packed with the core samples into rice sacks for transport to the Intertek laboratory. A range of standards was used in order to reflect the range of mineralisation types and grades associated with the Tumpangpitu prospect. Oxide Au standards were only inserted into sequences of core samples from oxidised mineralisation zones whilst sulphide Cu-Au standards were inserted into sequences of samples from hypogene sulphide mineralisation. Three grades of oxide Au standard were used (low, moderate and high) according to the interpreted grade of the mineralisation.

The standards that were used in the Zone A drilling are:

- **OREAS 52 Pb**: 0.307 g/t Au, 3,338 ppm Cu (n=9).
- **OREAS 2Pd**: 0.885 g/t Au (n=57).
- **OREAS 50Pb**: 0.841 g/t Au, 7,440 ppm Cu (n=6).
- **OREAS 6 Pp**: 1.520 g/t Au (n=39).
- **OREAS 7 Pb**: 2.770 g/t Au (n=13).
- **OREAS 61d**: 4.76 g/t Au, 9.27 ppm Ag (n=13).
- **OREAS 131a**: 30.9 ppm Ag, 1.72% Pb, 2.83% Zn (n=9).
- **OREAS 131b**: 33.3 ppm Ag, 1.88% Pb, 3.04% Zn (n=10).
- **OREAS 22P**: Blank

**Blanks**

Blanks were inserted into the sample stream by Intrepid-IMN. Generally between 2 and 5 internal analytical blanks were inserted with the batch of samples from each drill-hole, the number inserted depending on the drill-hole length, with two holes having 7 blanks inserted (GTD-46 and GTD-49). The blanks are not inserted at any pre-defined interval, although on average there is a blank being inserted for every 60 core samples.

Two types of blank samples were used for the Zone A drilling. The initial natural blank (Lampon black sand blank) was phased out part way through the Zone A drilling program and was replaced by a commercial quartz blank from OREAS (Blank 22p).

The Lampon black sand blank was sourced from a beach that extends east of Lampon, a village located on the coastal plain approximately 5 kilometres east of Zone B at Tumpangpitu. The sand is jet-black in color and its mineralogical composition was estimated as:
Pyroxene/hornblende - 90%, olivine - 5%, magnetite - 2%, quartz - 2% & unidentified minerals - 1%.

Whilst the blank typically analyses low in the elements being assayed, there is some possibility that the occasional spikes in Au and Zn (and lesser spikes in As and Mo) in the blanks may be attributed to the complex mineralogy of the sand and its magnetite content rather than to laboratory contamination. It was considered prudent to change to a commercially available blank which was subsequently submitted with samples from the Zone A drilling program in holes GTD-08-44 onwards.

Duplicates

Duplicate core samples (n=54) were collected during part of the Zone A drilling program and submitted to Intertek for assay. The core was re-sampled at a frequency of 1 in every ~68 samples. The re-sampled ‘duplicate’ comprised quarter core, the remaining quarter core was retained as a record of the interval.

15.8 QA/QC Results

A review of QA/QC results for the Zone A diamond drilling program was made by Mr Damien Lulofs, a geochemist from Lulofs Management Services. The executive summary of Mr Lulofs report is inserted below. The full original QA/QC assessment report is included as Appendix 7.

EXECUTIVE SUMMARY (D.Lulofs)

A review of QAQC data for the Tujuh Bukit project (Zones A & C drilling) was conducted during December 2008. A good QAQC program has been implemented for the Tujuh Bukit resource (Zone A & C) and excellent data is available for review.

Samples submitted include:
- **Standards** and **blanks** - commercially purchased from OREAS.
- Field **duplicates** (Two separate quarter core samples as different sample numbers for same analysis at the same laboratory).
- **Check samples** (pulps [same sample number] resubmitted to a second or third lab).
- Laboratory **replicates** (second to fifth split of pulp for same analysis, same laboratory).
  (Internal laboratory standards and blanks have not been assessed here.)

As with previous assessments (May 2008 & August 2008 on Zone C), mean absolute paired difference (MAPD) has been used to estimate variability. MAPD = difference/average and is expressed as a percentage. 15% variability is the suggested tolerance limit.
The internal standards for Au all fall well within accepted thresholds of 15%. Au assays for all laboratories show variance of <7% indicating appropriate analysis methodology and machine calibration (and/or adequate recognition of QAQC samples).

Table 12. Variability of laboratory analyses.

<table>
<thead>
<tr>
<th>Standard name (OREAS)</th>
<th>Intertek (FA30)</th>
<th>Intertek (FA50)</th>
<th>ALS (FA30)</th>
<th>ALS (FA50)</th>
<th>McPhar (FA50)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2Pd</td>
<td>3.9%</td>
<td>3.1%</td>
<td>5.2%</td>
<td>2.9%</td>
<td></td>
</tr>
<tr>
<td>50Pb</td>
<td>n/a</td>
<td>n/a</td>
<td>n/a</td>
<td>n/a</td>
<td>n/a</td>
</tr>
<tr>
<td>52Pb</td>
<td>n/a</td>
<td>6.1%</td>
<td>6.3%</td>
<td>2.1%</td>
<td></td>
</tr>
<tr>
<td>54Pa</td>
<td>n/a</td>
<td>n/a</td>
<td>3.6%</td>
<td>3.2%</td>
<td></td>
</tr>
<tr>
<td>61d</td>
<td>n/a</td>
<td>n/a</td>
<td>6.1%</td>
<td>1.8%</td>
<td></td>
</tr>
<tr>
<td>6Pc</td>
<td>n/a</td>
<td>n/a</td>
<td>4.0%</td>
<td>1.4%</td>
<td></td>
</tr>
<tr>
<td>7Pb</td>
<td>n/a</td>
<td>n/a</td>
<td>3.1%</td>
<td>1.8%</td>
<td></td>
</tr>
<tr>
<td>Standards submitted with Core</td>
<td>Routine Drill Samples (Core)</td>
<td>Check Samples [2nd Lab] (Pulps)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

OREAS standard 6Pc is plotted below as a good example of standards assayed in the check laboratories (ALS & McPhar).

Figure 20. Examples of OREAS standard (6Pc) submitted to Intertek, ALS Chemex and McPhar laboratories.

The internal blank was previously an 'extremely' magnetic black sand which probably consists of 90% black pyroxene, 5% olivine and ~1% silica (from Lampon) which has produced ‘anomalous’ results.

A commercial blank (from OREAS) has now been phased in (Blank 22p). Assays of ‘Blank 22p’ are limited but to date there is no contamination concern. All blanks have returned values <0.02ppm.

The laboratory replicates for Au fall well within accepted thresholds of 15%, indicating good reproducibility at the laboratory level (and hence probably adequate sample preparation). There is no concern here.
Reproducibility is better with FA50 in all laboratories. This is not surprising given the larger sample weight (i.e. 50g). In this dataset, the reproducibility appears to be more erratic near the detection limit for FA30 than the larger sample weight FA50. Variance will generally be higher closer to the detection limit but FA50 gives a better result than FA30. It is the view of the writer that a 50g Fire Assay is preferred over 30g for Au resources due to sample representivity.

There is no marked difference when looking at the variance of laboratory replicates across the two zones (Zone A & C). ALS shows some difference in FA30, but this can be attributed to one sample which assayed near the detection limit (0.011ppm – see graph).

Table 13. Replicate variance (% MAPD) for FA30 & FA50 in Zones A & C.

<table>
<thead>
<tr>
<th></th>
<th>ALL DATA</th>
<th>ZONE A</th>
<th>ZONE C</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>FA30</td>
<td>FA50</td>
<td>FA30</td>
</tr>
<tr>
<td>ALS</td>
<td>8.0%</td>
<td>3.3%</td>
<td>5.1%</td>
</tr>
<tr>
<td>McPhar</td>
<td>6.2%</td>
<td>2.8%</td>
<td>6.2%</td>
</tr>
<tr>
<td>Intertek</td>
<td>6.8%</td>
<td>3.6%</td>
<td>7.2%</td>
</tr>
</tbody>
</table>
Site duplicates were collected and submitted to Intertek for analysis. Duplicates were predominately submitted for 30g Fire Assay (122 samples) but some samples were for 50g Fire Assay (12 samples). Variance for 30g FA and 50g FA was 25% and 23% respectively. Site duplicates indicate whether the sampling is ‘representative’ of the drill interval or the ore block it characterizes. Given the replicate variance is 3 – 8% (i.e. laboratory variance), it can be assumed ‘natural’ variance (‘nugget effect’ and sampling protocol) is <20%. This can be accepted as a fairly good result for gold resources with no concern over current
sample weights/lengths of half core. Site duplicates should continue to be collected as an integral part of the QAQC program.

Prior to August 2008, Intertek assays (from Zone C) were re-submitted as check assays at McPhar. Subsequently, the same batch of check pulp samples was also submitted to ALS Chemex in an attempt to determine the accuracy of Intertek and McPhar analyses. Zone A check samples, submitted after August 2008, were submitted to all three laboratories.

**Check samples - Zone C:**

Assays range up to 25 g/t but to best demonstrate variance, 0 – 1 g/t is plotted here with the red trend line indicating the desired 1:1 linear trend.
ALS assays showed excellent correlation with Intertek assays. It can only be assumed McPhar assays were under reporting for those samples in the Zone C samples.

**Check samples - Zone A:**

Interestingly, this trend is less obvious when reviewing the Zone A (only) data.

![Figure 24. Plots of interlaboratory assays for Au (Zone A).](image)

<table>
<thead>
<tr>
<th>McPhar vs' ALS (FA50)</th>
<th>Intertek vs' ALS (FA50)</th>
<th>McPhar vs' Intertek (FA50)</th>
</tr>
</thead>
<tbody>
<tr>
<td>MAPD</td>
<td>8.7%</td>
<td>7.5%</td>
</tr>
</tbody>
</table>

McPhar assays still show slight underestimation but not as distinct as in Zone C. Regardless, McPhar assays are still suspect and further investigation on McPhar’s Fire Assay methodology is required (especially given analysis of internal standards and replicates are satisfactory).

The check samples provide confidence to Intertek analysis and the use of Intertek as the principal laboratory. Check samples should still continue to be submitted (to ALS) as part of the routine QAQC program.

**16. DATA VERIFICATION**

Spot checks were made of drill hole ledgers containing historic assays against digital data.

The following sample numbers were checked: DC25501, 32, 35, 96, 25870-75, 25637, 38; 25698, 99, 25700, 01, 25743-47, 25759-64, 26050-26075, 25985-88, 26118-124, 25951, 33127-30, 33439-45, 33425-432, 34332-36, 33973-76. Sample numbers 26050-75, 25930-41; DC33088-33260, 33606-33723, had missing Cu, Pb, Zn & Mo and required re-loading of original data. Not all original data was subsequently located.

It was noticed that Au1 and Au2 were averaged in some cases and in others (eg 25957) Au1 was used as the preferred gold value. H&S recommends that Au1 be used unless it is obvious that an analytical mistake was made in which case Au2 (or Au3) should be used.
It was recommended that Intertek be authorized to email laboratory reports directly to H&S. This was instituted and enables spot checking of data in the database with original data received directly from the laboratory.

Fourteen (14) drill core samples were re-sampled by the author and sent for independent check assaying at McPhar Laboratory, Manila, Philippines. An example of these samples is provided in Figure 25. The results, together with original assays, are provided in Table 14. Anonymous standards were also included (results with recommended values in Table 14). The aim was not a rigorous sampling study but rather to provide results from independent sampling and assaying.

![Image of drill core samples](image_url)

**Figure 25: Due Diligence Sample HS02 (GT16, 9.7 & 10.7m, 10328).**

The samples taken were taken from broken sections of core within the stated intervals and as such are not expected to be representative of the original assayed intervals. Given these constraints the results are regarded as excellent. Consultant Dr B. Rohrlach comments that sample HS06 contains a highly ferruginous and crushed zone from 15.9 to 16.0m which may have contributed some variance to the results. The results for the included standards are excellent and 2% high (McPhar/Recommended Value) for OREAS7 and 4% low for OREAS10.
Table 14. Assays of diamond core re-samples.

<table>
<thead>
<tr>
<th>Sample</th>
<th>Hole</th>
<th>Interval Original</th>
<th>Check Assays of Re-samples</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Start End Au Ag</td>
<td>Au (1) Ag (2)</td>
</tr>
<tr>
<td>HS01</td>
<td>GT16</td>
<td>28.6 28.7 0.8 265</td>
<td>1.48 104.2</td>
</tr>
<tr>
<td>HS02</td>
<td>GT16</td>
<td>9.7 10.7 1.85 83</td>
<td>1.72 58.4</td>
</tr>
<tr>
<td>HS03</td>
<td>GT16</td>
<td>81 81.1 0.368 42</td>
<td>0.79 23.0</td>
</tr>
<tr>
<td>HS04</td>
<td>GT16</td>
<td>119 119.1 0.334 6</td>
<td>1.67 28.4</td>
</tr>
<tr>
<td>HS05</td>
<td>GT17</td>
<td>114.4 114.6 0.49 16</td>
<td>0.63 0.63 13.4 13.3</td>
</tr>
<tr>
<td>HS06</td>
<td>GT17</td>
<td>16 16.1 0.02 2</td>
<td>0.34 67.3</td>
</tr>
<tr>
<td>HS07</td>
<td>GT15</td>
<td>16.5 17 3.07 178</td>
<td>9.84 9.79 315.4 310.4</td>
</tr>
<tr>
<td>HS08</td>
<td>GT15</td>
<td>88.9 89 0.859 59</td>
<td>0.96 27.3</td>
</tr>
<tr>
<td>HS09</td>
<td>GT05</td>
<td>133.3 133.4 2.03 37</td>
<td>2.00 16.7</td>
</tr>
<tr>
<td>HS10</td>
<td>GT05</td>
<td>46.5 46.6 0.87 3</td>
<td>0.89 3.2</td>
</tr>
<tr>
<td>HS11</td>
<td>GT05</td>
<td>174.5 174.5 7.02 12</td>
<td>1.42 10.1</td>
</tr>
<tr>
<td>HS12</td>
<td>GT10</td>
<td>7 7.1 0.59 4</td>
<td>0.69 3.4</td>
</tr>
<tr>
<td>HS13</td>
<td>GT10</td>
<td>43 43.1 2.99 6</td>
<td>3.55 45.3</td>
</tr>
<tr>
<td>HS14</td>
<td>GT14</td>
<td>25.3 25.5 1.21 32</td>
<td>4.32 63.6</td>
</tr>
<tr>
<td>Averages</td>
<td></td>
<td></td>
<td>1.61 53.2 2.16 55.7</td>
</tr>
<tr>
<td>HS15</td>
<td>OREAS7</td>
<td>Std RV= 3.00</td>
<td>3.07 IS &lt;0.5 &lt;0.5</td>
</tr>
<tr>
<td>HS16</td>
<td>OREAS10</td>
<td>Std RV= 6.81</td>
<td>6.57 1.4</td>
</tr>
<tr>
<td>Detection</td>
<td></td>
<td></td>
<td>0.03 0.5</td>
</tr>
</tbody>
</table>

McPhar analytical methods
Au - by fire assay on 30g sample.
Ag - by AAS following conc. HCl and HCl/HNO3/HClO4 leach in latter stages on 1g sample
IS = Insufficient Sample

17. ADJACENT PROPERTIES

There are no mineral exploration tenements or mining properties that lie adjacent to this project at the time of the writing of this report.

18. MINERAL PROCESSING AND METALLURGICAL TESTING

In 2000, Placer conducted preliminary bottle roll tests on 27 selected samples of oxide mineralisation from holes GT-10, GT-11, GT-12, and GT-14. The samples were designed to test a range of ore types including Au rich-Ag poor, Ag rich-Au poor, and Au-Ag rich zones. Results indicated an average of 83% Au and 84.5% Ag recovery from higher grade (sub-economic) zones (Campbell, 2000).
Preliminary metallurgical testing has been commissioned by Intrepid Mines on the basis of 4 composite samples from quarter core samples from Zone C. This work was completed in December 2008 and a summary of the results is given below. No additional metallurgical work has been commissioned at this stage using samples from Zone A.

Summary: The following composites representing different locations within the Tujuh Bukit (Tumpangpitu) deposit have been subjected to comminution, gravity concentration and cyanidation testwork.

- **Composite SIOX** – vuggy silica above the base of complete oxidation.
- **Composite SITR** – vuggy silica below the base of complete oxidation and above the base of fracture oxidation.
- **Composite CYOX** – Silica/clay above the base of complete oxidation.
- **Composite CYTR** – Silica/clay below the base of complete oxidation and above the base of fracture oxidation.

The average calculated head grades for the four composites were:

**Table 15. Average calculated head grades of composite metallurgical samples from Zone C.**

<table>
<thead>
<tr>
<th>Composite</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>SIOX</td>
<td>1.01</td>
<td>66</td>
</tr>
<tr>
<td>SITR</td>
<td>1.24</td>
<td>92</td>
</tr>
<tr>
<td>CYOX</td>
<td>0.84</td>
<td>55</td>
</tr>
<tr>
<td>CYTR</td>
<td>1.03</td>
<td>65</td>
</tr>
</tbody>
</table>

All four composites displayed very similar metallurgical characteristics with the exception of composite SIOX, which had a higher ball mill work index.

**Table 16. Ball mill work index values for four composite metallurgical samples from Zone C.**

<table>
<thead>
<tr>
<th>Composite</th>
<th>Ball mill work index kWh/t</th>
</tr>
</thead>
<tbody>
<tr>
<td>SIOX</td>
<td>16.1</td>
</tr>
<tr>
<td>SITR</td>
<td>12.3</td>
</tr>
<tr>
<td>CYOX</td>
<td>12.4</td>
</tr>
<tr>
<td>CYTR</td>
<td>12.2</td>
</tr>
</tbody>
</table>

Gold extractions obtained by gravity concentration were very low, such that gravity concentration would not be required in any future processing plant.
Cyanidation tests under CIL conditions consistently yielded high gold and silver extractions under a range of test conditions. The extractions obtained at a grind size of 80% passing 75µm with a leach time of 24 hours and an initial cyanide concentration of 0.100% were:

Table 17. Percentage gold extraction by gravity concentration.

<table>
<thead>
<tr>
<th>Composite</th>
<th>% Gold extraction by gravity concentration</th>
</tr>
</thead>
<tbody>
<tr>
<td>SIOX</td>
<td>3.2</td>
</tr>
<tr>
<td>SITR</td>
<td>0.6</td>
</tr>
<tr>
<td>CYOX</td>
<td>2.0</td>
</tr>
<tr>
<td>CYTR</td>
<td>1.8</td>
</tr>
</tbody>
</table>

Tests over a range of leach times indicated that a leach time of 20 hours would be sufficient for plant design.

Reducing the initial cyanide concentration from 0.100% to 0.060% had, in virtually all cases, no discernable effect on gold and silver extractions, but resulted in reduced cyanide consumptions and small increases in lime consumptions. The testwork indicated that plant reagent consumptions will be in the following ranges: 0.65 to 0.90kg/t cyanide and 1.0 to 1.2 kg/t lime.

Bottle roll tests on minus 12.5mm material indicated that relatively high gold extractions would be obtained by heap leaching, but silver extractions would be much lower than those obtained by CIL processing.
Table 19. Indicated extractions by heap leaching.

<table>
<thead>
<tr>
<th>Composite</th>
<th>% Gold</th>
<th>% Silver</th>
</tr>
</thead>
<tbody>
<tr>
<td>SIOX</td>
<td>77</td>
<td>28</td>
</tr>
<tr>
<td>SITR</td>
<td>75</td>
<td>30</td>
</tr>
<tr>
<td>CYOX</td>
<td>78</td>
<td>23</td>
</tr>
<tr>
<td>CYTR</td>
<td>75</td>
<td>28</td>
</tr>
</tbody>
</table>

19. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Topography

Topographic surveys dated 12 December 2008 (file topo_121208.csv) were used to define the elevation surface for the reported resource estimates. Subsequently, an updated survey file was received on 17 December 2008 (file Topo_Survey_Point_7B_161208.csv). It was not possible to use this for the reported estimates due to the deadline being the next day (18 December 2008). The new data, however, was used for the resource model that is the basis of preliminary pit optimising studies. Significant (greater than 5m difference) discrepancies between the two survey data sets occur mainly on the peripheries of the main resource area (Figure 26). These discrepancies do not materially affect the reported estimates within the meaning of “Inferred”. The distribution of the data points (Figure 26) highlight the need for a more extensive and uniform survey coverage on the peripheries and within the surveyed area.
Figure 26. Discrepancies between 16 December 2008 and as-used topographic models (12 Dec 08).
(red crosses mark positions of data points)

Data

Zone A holes contain 4494, 4561 and 4448 assays for Au, Ag and Cu, respectively. Some 3750 entries for an oxidation code are present. The lower number reflects the absence of this information for older holes. Table 21 summarises the number of available gold assays sorted by oxidation code.

The depths at which the base of complete oxidation ("BOX") and base of partial oxidation ("BTR") occur are provided in Table 20. The focus of this work is the completely oxidised and partially oxidised zones. Resource estimates are confined to these zones due to the
favourable metallurgical response as indicated by preliminary testing. No resource estimates have been undertaken in the sulphide zone.

<table>
<thead>
<tr>
<th>ID</th>
<th>Depth</th>
<th>Code</th>
<th>ID</th>
<th>Depth</th>
<th>Code</th>
<th>ID</th>
<th>Depth</th>
<th>Code</th>
</tr>
</thead>
<tbody>
<tr>
<td>GT004</td>
<td>240.5</td>
<td>BOX</td>
<td>GTD-08-24</td>
<td>147</td>
<td>BOX</td>
<td>GTD-08-37</td>
<td>138</td>
<td>BOX</td>
</tr>
<tr>
<td>GT004</td>
<td>276.6</td>
<td>BTR</td>
<td>GTD-08-24</td>
<td>185</td>
<td>BTR</td>
<td>GTD-08-37</td>
<td>376</td>
<td>BTR</td>
</tr>
<tr>
<td>GT011</td>
<td>198.7</td>
<td>BOX</td>
<td>GTD-08-25</td>
<td>137.4</td>
<td>BOX</td>
<td>GTD-08-38</td>
<td>308</td>
<td>BOX</td>
</tr>
<tr>
<td>GT011</td>
<td>201.45</td>
<td>BTR</td>
<td>GTD-08-25</td>
<td>167.7</td>
<td>BTR</td>
<td>GTD-08-38</td>
<td>334</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-07-15</td>
<td>24.2</td>
<td>BOX</td>
<td>GTD-08-26</td>
<td>25.4</td>
<td>BOX</td>
<td>GTD-08-39</td>
<td>140</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-15</td>
<td>180.05</td>
<td>BTR</td>
<td>GTD-08-26</td>
<td>48.1</td>
<td>BTR</td>
<td>GTD-08-39</td>
<td>338</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-07-16</td>
<td>128.4</td>
<td>BOX</td>
<td>GTD-08-27</td>
<td>120</td>
<td>BOX</td>
<td>GTD-08-40</td>
<td>75.95</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-16</td>
<td>233.25</td>
<td>BTR</td>
<td>GTD-08-27</td>
<td>193.55</td>
<td>BTR</td>
<td>GTD-08-41</td>
<td>314</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-17</td>
<td>31.2</td>
<td>BOX</td>
<td>GTD-08-28</td>
<td>209.35</td>
<td>BOX</td>
<td>GTD-08-43</td>
<td>312</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-17</td>
<td>219.6</td>
<td>BTR</td>
<td>GTD-08-28</td>
<td>287</td>
<td>BTR</td>
<td>GTD-08-43</td>
<td>436</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-07-18</td>
<td>37.01</td>
<td>BOX</td>
<td>GTD-08-29</td>
<td>13.2</td>
<td>BOX</td>
<td>GTD-08-44</td>
<td>300</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-18</td>
<td>188.5</td>
<td>BTR</td>
<td>GTD-08-29</td>
<td>16.5</td>
<td>BTR</td>
<td>GTD-08-45</td>
<td>186</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-19</td>
<td>213.54</td>
<td>BOX</td>
<td>GTD-08-30</td>
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<td>GTD-08-46</td>
<td>72</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-19</td>
<td>213.55</td>
<td>BTR</td>
<td>GTD-08-30</td>
<td>166.71</td>
<td>BTR</td>
<td>GTD-08-46</td>
<td>266</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-07-20</td>
<td>227.2</td>
<td>BOX</td>
<td>GTD-08-31</td>
<td>111.35</td>
<td>BOX</td>
<td>GTD-08-47</td>
<td>264</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-07-20</td>
<td>312.4</td>
<td>BTR</td>
<td>GTD-08-31</td>
<td>151.65</td>
<td>BTR</td>
<td>GTD-08-48</td>
<td>176</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-08-21</td>
<td>119.4</td>
<td>BOX</td>
<td>GTD-08-32</td>
<td>110.6</td>
<td>BOX</td>
<td>GTD-08-48</td>
<td>204</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-08-21</td>
<td>239.4</td>
<td>BTR</td>
<td>GTD-08-32</td>
<td>120.2</td>
<td>BTR</td>
<td>GTD-08-49</td>
<td>324</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-08-22</td>
<td>136.3</td>
<td>BOX</td>
<td>GTD-08-33</td>
<td>172</td>
<td>BOX</td>
<td>GTD-08-51</td>
<td>116</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-08-22</td>
<td>154.1</td>
<td>BTR</td>
<td>GTD-08-34</td>
<td>84</td>
<td>BOX</td>
<td>GTD-08-52</td>
<td>220</td>
<td>BOX</td>
</tr>
<tr>
<td>GTD-08-23</td>
<td>133</td>
<td>BOX</td>
<td>GTD-08-34</td>
<td>264</td>
<td>BTR</td>
<td>GTD-08-52</td>
<td>268</td>
<td>BTR</td>
</tr>
<tr>
<td>GTD-08-23</td>
<td>167.9</td>
<td>BTR</td>
<td>GTD-08-36</td>
<td>408</td>
<td>BOX</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 20. Summary by drill hole of starting depths of oxidation codes

Spatial Distribution

Figure 27 to Figure 32 illustrate the distribution in plan and cross-section of Au and Ag in Zone A.
Figure 27. Plan View, Zone A (rotated Local Grid), 2m composites, Au.
Figure 28. East-west sectional view, (Local Grid), all 2m composites, Au.

Figure 29. East-west sectional view, 11280N (Local Grid) – 2m composites, Au.
Figure 30. East-west sectional view, 11200N (Local Grid) – 2m composites, Au

Figure 31. East-west sectional view (Local Grid) – all 2m composites, Ag
Oxidation

Summaries of Au, Ag, As and Cu by oxide code\(^1\), lithology, lithology-oxidation and alteration-oxidation are provided in Table 21 to Table 24. The steady decline in gold grade with decreasing oxidation is clearly shown in Table 11 and also depicted in a cumulative frequency diagram in Figure 33. There is a similar, though not as pronounced, effect with Ag. Arsenic (As) is clearly concentrated in the completely oxidised zone. The geochemical variation with oxidation within individual lithologies is shown in Table 23. It is clear that oxidation has acted as a significant control of gold and silver enrichment because individual lithologies with variable oxidation display the overall trend of enriched gold and silver with increasing oxidation. For example, the lithic tuff (Ltu, Table 23) similar to other units, shows enrichment from 0.117 to 0.122 to 0.251 to 0.431 to 0.483 g/t from sulphide to completely oxidised zones.

Silver displays a slightly different behaviour with overall enrichment with increasing oxidation except in the completely oxidised zone. In the Ltu unit, silver increases from 5.7 to 16.8 to 31.1 to 37.9 g/t from sulphide to strongly oxidised zones though decreases to

\(^1\) (5 = complete, 4 = strong, 3 = medium, 2 = weak & 1 = fresh)
11.7 g/t in the completely oxidised zone. This decrease in the oxidised zone for silver is
common to other units and is shown as a set of box-and-whisker plots in
Figure 35. Figure 36 shows the relationship of gold with silver colour coded by domain (ie
oxidation code, 1 = sulphide and 5 = oxidised). Figure 37 shows the distribution of gold
and silver down-hole on local grid section 11280N (compare with Figure 30). Gold shows
a strong tendency to be enriched towards the surface and silver towards the base of
complete oxidation.

Arsenic (As) behaves in a similar fashion to gold (Figure 39 & Figure 40) and is enriched
in the oxidised zones. Copper only reaches levels of interest in the deeper sulphide zone
and shows no obvious relationship with oxidation (Figure 38).

The depths at which the base of complete oxidation (“BOX”) and base of partial oxidation
(“BTR”) occur (Table 20) were used to triangulate surfaces defining BOX and BTR. These
were and then plotted on cross section. To more completely define these surfaces away
from drill hole information, boundaries were digitised and surfaces re-triangulated using
the digitised points as well as the actual logged positions. Figure 41 displays all intervals
with logged oxidation codes. It is evident that although the overall trend is for the degree
of oxidation to decrease with depth there are a number of exceptions such as “islands” of
oxidation within fresh material.

The current oxidation model is, of necessity, simplified and future resource models may be
improved by using an indicator approach whereby blocks are estimated in terms of
proportions of the various degrees of oxidation intensity. The oxidation model consists of
layers with the thickness of the completely oxidised zone averaging 106m and the partially
oxidised zone averaging 104m resulting in an overall total thickness of complete and
partial oxidation averaging approximately 200m.

<table>
<thead>
<tr>
<th>Oxide code</th>
<th>Au</th>
<th>Ag</th>
<th>As</th>
<th>Cu%</th>
<th>Oxide code</th>
<th>N</th>
<th>Length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>0.450</td>
<td>12</td>
<td>997.2</td>
<td>0.026</td>
<td>5</td>
<td>2269</td>
<td>4543.45</td>
</tr>
<tr>
<td>4</td>
<td>0.311</td>
<td>32</td>
<td>588.3</td>
<td>0.043</td>
<td>4</td>
<td>459</td>
<td>919.1</td>
</tr>
<tr>
<td>3</td>
<td>0.138</td>
<td>20</td>
<td>378.4</td>
<td>0.038</td>
<td>3</td>
<td>197</td>
<td>400.5</td>
</tr>
<tr>
<td>2</td>
<td>0.099</td>
<td>8.4</td>
<td>330.3</td>
<td>0.061</td>
<td>2</td>
<td>439</td>
<td>876.1</td>
</tr>
<tr>
<td>1</td>
<td>0.054</td>
<td>1.2</td>
<td>112.6</td>
<td>0.040</td>
<td>1</td>
<td>386</td>
<td>773.5</td>
</tr>
<tr>
<td>na</td>
<td>0.227</td>
<td>2.8</td>
<td>352.8</td>
<td>0.181</td>
<td></td>
<td>830</td>
<td>1879.1</td>
</tr>
<tr>
<td>Totals</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4580</td>
<td>9391.75</td>
</tr>
<tr>
<td>Averages</td>
<td>0.312</td>
<td>11.3</td>
<td>715.8</td>
<td>0.063</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 21. Summary of Au, Ag & Cu by oxide code
(N = number of data; Length = total meterage of samples)
**Table 22. Summary of Au, Ag & Cu by lithology.**

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Au</th>
<th>Ag</th>
<th>As</th>
<th>Cu%</th>
<th>N</th>
<th>Length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>na</td>
<td>0.207</td>
<td>4.5</td>
<td>167.2</td>
<td>0.013</td>
<td>34</td>
<td>67.7</td>
</tr>
<tr>
<td>And</td>
<td>0.032</td>
<td>0.8</td>
<td>28.7</td>
<td>0.012</td>
<td>62</td>
<td>124</td>
</tr>
<tr>
<td>Dac</td>
<td>0.136</td>
<td>9.1</td>
<td>345.6</td>
<td>0.033</td>
<td>26</td>
<td>52</td>
</tr>
<tr>
<td>Gd</td>
<td>0.025</td>
<td>0.5</td>
<td>12.5</td>
<td>0.014</td>
<td>4</td>
<td>8</td>
</tr>
<tr>
<td>Hcy</td>
<td>0.043</td>
<td>1.9</td>
<td>80.5</td>
<td>0.051</td>
<td>71</td>
<td>142.85</td>
</tr>
<tr>
<td>Hcy-si</td>
<td>0.067</td>
<td>3.6</td>
<td>239</td>
<td>0.081</td>
<td>101</td>
<td>201.7</td>
</tr>
<tr>
<td>Hsi</td>
<td>0.186</td>
<td>52.7</td>
<td>314.4</td>
<td>0.035</td>
<td>14</td>
<td>28</td>
</tr>
<tr>
<td>Hsi-al</td>
<td>0.184</td>
<td>24.2</td>
<td>576.6</td>
<td>0.104</td>
<td>9</td>
<td>18</td>
</tr>
<tr>
<td>Hsi-cy</td>
<td>0.146</td>
<td>12.4</td>
<td>438.8</td>
<td>0.150</td>
<td>121</td>
<td>241.1</td>
</tr>
<tr>
<td>Hsi-cy+al</td>
<td>0.172</td>
<td>11.5</td>
<td>403</td>
<td>0.102</td>
<td>34</td>
<td>67.65</td>
</tr>
<tr>
<td>Ibx</td>
<td>0.155</td>
<td>4.4</td>
<td>0.014</td>
<td>12</td>
<td>28.95</td>
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**Lithology** refers to the type of rock or soil, and **Oxide code** refers to a specific oxide composition. **Au**, **Ag**, **As**, **Cu%**, and **N** represent concentration levels and sample counts, respectively, while **Length (m)** indicates the length of the sample or deposit in meters.
**Lithology** | **Oxide code** | **Au** | **Ag** | **As** | **Cu%** | **N** | **Length (m)**
---|---|---|---|---|---|---|---
Hcy-si & 4 & 0.055 & 8.6 & 152.3 & 0.211 & 23 & 46
Hcy-si & 3 & 0.04 & 2.5 & 99.5 & 0.062 & 20 & 40
Hcy-si & 2 & 0.09 & 1.3 & 419.8 & 0.035 & 37 & 73.2
Hcy-si & 1 & 0.064 & 1.4 & 69.7 & 0.038 & 11 & 22.5
Hsi & 5 & 0.203 & 10 & 474.3 & 0.036 & 3 & 6
Hsi & 4 & 0.15 & 53.4 & 399.2 & 0.012 & 5 & 10
Hsi & 3 & 0.07 & 66 & 76 & 0.015 & 1 & 2
Hsi & 2 & 0.236 & 75 & 181.4 & 0.061 & 5 & 10
Hsi-al & 5 & 0.363 & 26 & 985.7 & 0.021 & 3 & 6
Hsi-al & 4 & 0.09 & 127 & 385 & 0.005 & 1 & 2
Hsi-al & 2 & 0.095 & 3 & 402 & 0.198 & 4 & 8
Hsi-al & 1 & 0.1 & 1 & 239 & 0.078 & 1 & 2
Hsi-cy & 5 & 0.246 & 102.3 & 563.4 & 0.036 & 9 & 17.15
Hsi-cy & 4 & 0.125 & 8.8 & 310.7 & 0.116 & 25 & 50.1
Hsi-cy & 3 & 0.095 & 10 & 102.8 & 0.085 & 6 & 12
Hsi-cy & 2 & 0.153 & 5.8 & 668.4 & 0.225 & 41 & 82
Hsi-cy & 1 & 0.138 & 2.4 & 297.2 & 0.126 & 40 & 79.85
Hsi-cy+al & 5 & 0.658 & 23.8 & 1353.8 & 0.022 & 4 & 8
Hsi-cy+al & 4 & 0.17 & 48.8 & 287 & 0.011 & 4 & 8
Hsi-cy+al & 3 & 0.07 & 16.3 & 43.5 & 0.031 & 2 & 4
Hsi-cy+al & 2 & 0.068 & 1.4 & 106.2 & 0.094 & 6 & 11.65
Hsi-cy+al & 1 & 0.109 & 3.3 & 353.5 & 0.151 & 18 & 36
Ibx & 5 & 0.155 & 4.3 & na & 0.014 & 11 & 26.8
Ibx & 4 & 0.15 & 5 & na & 0.014 & 1 & 2.15
Ltu & 5 & 0.483 & 11.7 & 1040.7 & 0.025 & 2025 & 4051.3
Ltu & 4 & 0.431 & 37.9 & 780.3 & 0.030 & 271 & 542
Ltu & 3 & 0.251 & 31.1 & 610.7 & 0.030 & 62 & 130.5
Ltu & 2 & 0.122 & 16.8 & 386.3 & 0.051 & 77 & 156
Ltu & 1 & 0.117 & 5.7 & 143.4 & 0.056 & 16 & 32
Qd & 1 & 0.02 & 0.5 & 8.5 & 0.006 & 2 & 4
Sap & 5 & 0.475 & 0.8 & 427.2 & 0.013 & 12 & 24
Sap & na & 0.07 & 0.5 & 89 & 0.007 & 1 & 2
Soi & 5 & 0.24 & 0.8 & 339 & 0.016 & 2 & 4
Soi & 4 & 0.05 & 0.5 & 42 & 0.009 & 1 & 2
Soi & na & 0.02 & 0.5 & 64 & 0.012 & 2 & 4
TBx & 5 & 1.62 & 12 & 2890 & 0.051 & 1 & 2
TBx & 2 & 0.06 & 38 & 27 & 0.005 & 1 & 2
Tu & 5 & 0.68 & 3.5 & 995 & 0.022 & 2 & 4
Tu & 4 & 0.109 & 19.2 & 481.6 & 0.024 & 7 & 14.35
Tu & 3 & 0.06 & 3.7 & 260.4 & 0.010 & 11 & 22
Tu & 2 & 0.078 & 2.1 & 155.5 & 0.013 & 24 & 47.15
### Lithology

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<th>Cu%</th>
<th>N</th>
<th>Length (m)</th>
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### Averages

|            | 0.331 | 13.3 | 737.1 | 0.034 |

Table 23. Summary of Au, Ag & Cu by lithology and oxide code.

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<th>As</th>
<th>Cu%</th>
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### Table 24. Summary of Au, Ag & Cu by alteration and oxide code.

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<th>Cu%</th>
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<tr>
<td>SCC</td>
<td>2</td>
<td>0.045</td>
<td>0.5</td>
<td>311.4</td>
<td>0.003</td>
<td>17</td>
<td>34</td>
</tr>
</tbody>
</table>

| Totals     |             | 3787| 7586.35 |
| Averages   |             | 0.331| 13.3 | 737.1 | 0.034 |
Figure 33. Cumulative frequency diagram, Au by oxide intensity.
(C = complete, S = strong, M = medium, W = weak & F = fresh)
Figure 34. Cumulative frequency diagram, Ag by oxide intensity. 
(C = complete, S = strong, M = medium, W = weak & F = fresh)
Figure 35. Box and whisker plot, Ag by oxide intensity.
Figure 36. Silver – gold relationship by oxide zone.
(5 = complete, 4 = strong, 3 = medium, 2 = weak & 1 = fresh = domain)

Figure 37. Silver – gold relationship, 11200N (local).
(purple = silver, blue = gold; red surface = base of complete oxidation; grey = pit)
Figure 38. Copper – gold relationship.  
(5 = complete, 4 = strong, 3 = medium, 2 = weak & 1 = fresh = domain)

Figure 39. Arsenic – gold relationship by oxidation intensity.  
(red = complete, yellow = strong, olive = medium, brown = weak & black = fresh)
Figure 40. Arsenic – gold relationship.

Figure 41. East-west sectional view, oxidation intensity—raw intervals².

² 5 = complete, 4 = strong, 3 = medium, 2 = weak & 1 = fresh
Compositing

Two metre length composites were created using Techbase software (see summary statistics in Table 25). These are appropriate to the bulk style of mineralisation. The composites are continuous and do not break at, for example, changes in oxidation. The minimum resulting stored length is 1.2m (1.0m minimum specified). The 2m length was chosen on the basis of a bench height of 5m which on preliminary considerations may be appropriate for a Au-Ag CIL-type open pit mining operation that is a precursor to a larger porphyry copper bulk mineable project. Depending on the ultimate size of the resource, the composite length may be increased to be more appropriate to a larger mining operation with larger bench heights of 10m or more. The bench height chosen for the resource evaluation is 6m.

The oxidation codes were applied by locating their centroids with respect to the two oxide surfaces (base of complete oxidation and base of partial oxidation). An example of the resulting oxidation coding applied to the 2m composites is provided in cross-sectional view in Figure 42 and in Figure 43 for the modelled resource blocks.

![Table 25](image)

<table>
<thead>
<tr>
<th>OX code</th>
<th>Au</th>
<th>Ag</th>
<th>Cu%</th>
<th>As</th>
<th>SG</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mean</td>
<td>2173</td>
<td>2240</td>
<td>2173</td>
<td>2110</td>
<td>745</td>
</tr>
<tr>
<td>Maximum</td>
<td>1</td>
<td>12.10</td>
<td>0.03</td>
<td>976</td>
<td>2.15</td>
</tr>
<tr>
<td>Minimum</td>
<td>1</td>
<td>0.50</td>
<td>0.001</td>
<td>0.50</td>
<td>1.45</td>
</tr>
<tr>
<td>Coef Var</td>
<td>248.6</td>
<td>149.1</td>
<td>204.4</td>
<td>10.0</td>
<td></td>
</tr>
</tbody>
</table>

Table 25. Summary of 2m length composites by oxide code.

(Code 1 = completely oxidised; 2 = partially oxidised; 3=fresh)

Declustered statistics are provided in Table 26. The difference in the means for Au and Ag has resulted from the tendency of clustering of drilling in the higher grade portions of Zone A, particularly in the Au-enriched portions of Zone A.
Table 26. Summary of declustered 2m length composites by oxide code.

<table>
<thead>
<tr>
<th></th>
<th>Completely Oxidised</th>
<th>Partially Oxidised</th>
</tr>
</thead>
<tbody>
<tr>
<td>No. Data:</td>
<td>2173 2173</td>
<td>1315 1315</td>
</tr>
<tr>
<td>mean:</td>
<td>0.36 12.057</td>
<td>0.142 10.544</td>
</tr>
<tr>
<td>Minimum:</td>
<td>0.005 0.5</td>
<td>0.002 0.5</td>
</tr>
<tr>
<td>Maximum:</td>
<td>24.3 733</td>
<td>10.8 601</td>
</tr>
<tr>
<td>CV:</td>
<td>2.679 2.888</td>
<td>2.926 2.712</td>
</tr>
</tbody>
</table>

Figure 42. East-west sectional view, oxidation – 2m composites.
(blue = completely oxidised, green = partially oxidised; fresh not shown)
Variography

Variography was completed using H&S’s GS3 software for Au, Ag and SG for the combined completely and partially oxidised data sets. Figure 44 to Figure 47 provide examples of variograms and models for Au and Ag. Models used are provided in Figure 48 to Figure 5. The range, as defined by the distance at which the sill remains essentially constant, is approximately 80-100m, 30-50m and 6-18m in along-strike (azimuth ~320), down-dip (azimuth ~050) and down-hole directions.
Figure 44. Modelled down-hole variogram, gold.

Figure 45. Modelled down-hole variogram, silver.
Figure 46. Variograms for along strike, down hole and down dip directions, gold.
(-60 is the dominant down hole direction; 135 azimuth is along strike; 40 azimuth is down dip)

Figure 47. Modelled variogram, gold.
(-60 is the dominant down hole direction; 135 azimuth is 315 UTM; 40 azimuth is 050 UTM)
Block Model

A block model was constructed with dimensions of 3,360m x 2,760m x 1,194m per Table 27. The model extents were chosen to cover resources over Zones A, B & C.

Min – Max Easting: 173,000 – 176,360 Column size: 40 Number: 85
Min - Max Northing: 9,045,000 – 9,047,760 Row size: 40 Number: 70
Min – Max RL: +494 – -700 Level size: 6 Number: 200

Table 27. Summary of block model extents.

Ordinary Kriging (OK) using H&S’s GS3 software was used to estimate Au, Ag and density. Search parameters are summarised in Figure 51. Figure 52 depicts the search ellipsoid. Estimation was in three passes though only estimates from Passes 1 & 2 were classified as Inferred. Based on experience with other similar deposits, it is likely that a 160m search for Inferred is reasonable though it is possible, depending upon the results of future drilling, that this may be reduced. It is stressed that all estimates are regarded as Inferred the meaning of which carries the understanding that the majority of the tonnage
should be realised as a resource though a substantial amount may be lost with further sampling.

The search ellipse is divided into eight sectors (octants) from which a minimum of 6 composites coming from at least two octants are required (maximum number of composites is set at 32) for estimating Category 1 resources. Category 2 estimates use the same sample selection criteria as Category 1, however, the search dimensions are expanded by 100% to 160 x 160 x 80 metres. Category 3 resources are the least confident and are estimated by maintaining the expanded search ellipse but the sample criteria are relaxed to be a minimum of six composites coming from one octant. Estimates from this pass have not been reported here. The categorization into 1, 2 & 3 is useful to assist identify areas that require more drilling.

Figure 51. Search parameters.

The search orientation is depicted in Figure 52.
Results

Figure 53 illustrates the position of modelled grades in relation to drill holes. Figure 54 to Figure 56 are examples of cross sections through the modelled gold estimates at 11200N and 11280N (local grid). These can be compared with Figure 29 and Figure 30 which display sections with actual data. Figure 57 and Figure 58 show a comparison of grades for Au and Ag. Figure 59 and Figure show gold and silver data in juxtaposition with the models for 9046600N (UTM grid), respectively. Both figures show the position of the preliminary optimised pit and 60 shows the position of the very low confidence blocks that have not been reported in this report as part of the Inferred Resource. These blocks are shown with black block outlines.

Results on the basis of Au and AuEq cut-off grades are provided in Table 28 to Table 31.
Figure 53. Model in relation to drill hole locations (UTM Grid).
(red outline encloses blocks > 0.2 g/t Au and black > 0.5 g/t)
Figure 54. East-west sectional view, 11200N (Local Grid) – Model, Au.

Figure 55. East-west sectional view, 11200N (Local Grid) – Model, Ag.
Figure 56. East-west sectional view, 11280N (Local Grid) – Au model.
Figure 57. Comparison of composite and model gold grades.
(Domain 1 = model, 2 = 2m composites)

Figure 58. Comparison of composite and model silver grades.
(Domain 1 = model, 2 = 2m composites)
<table>
<thead>
<tr>
<th>Cats 1 &amp; 2</th>
<th>OX code</th>
<th>Au</th>
<th>Ag</th>
<th>SG</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number</td>
<td>1</td>
<td>4277</td>
<td>4277</td>
<td>4277</td>
</tr>
<tr>
<td>Mean</td>
<td>0.35</td>
<td>12.83</td>
<td>2.20</td>
<td></td>
</tr>
<tr>
<td>Maximum</td>
<td>3.08</td>
<td>161.9</td>
<td>2.55</td>
<td></td>
</tr>
<tr>
<td>Minimum</td>
<td>0.01</td>
<td>0.5</td>
<td>1.77</td>
<td></td>
</tr>
<tr>
<td>Coef Var</td>
<td>130</td>
<td>117</td>
<td>6</td>
<td></td>
</tr>
<tr>
<td>Cats 1 &amp; 2</td>
<td>OX code</td>
<td>Au</td>
<td>Ag</td>
<td>SG</td>
</tr>
<tr>
<td>Number</td>
<td>2</td>
<td>3465</td>
<td>3465</td>
<td>3465</td>
</tr>
<tr>
<td>Mean</td>
<td>0.14</td>
<td>11.86</td>
<td>2.36</td>
<td></td>
</tr>
<tr>
<td>Maximum</td>
<td>2.52</td>
<td>125.4</td>
<td>2.6</td>
<td></td>
</tr>
<tr>
<td>Minimum</td>
<td>0.01</td>
<td>0.5</td>
<td>1.89</td>
<td></td>
</tr>
<tr>
<td>Coef Var</td>
<td>106</td>
<td>120</td>
<td>4</td>
<td></td>
</tr>
<tr>
<td>Cats 1 - 3</td>
<td>OX code</td>
<td>Au</td>
<td>Ag</td>
<td>SG</td>
</tr>
<tr>
<td>Number</td>
<td>1</td>
<td>5672</td>
<td>5672</td>
<td>5672</td>
</tr>
<tr>
<td>Mean</td>
<td>0.30</td>
<td>13.81</td>
<td>2.21</td>
<td></td>
</tr>
<tr>
<td>Maximum</td>
<td>3.86</td>
<td>256.9</td>
<td>2.55</td>
<td></td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0.5</td>
<td>1.77</td>
<td></td>
</tr>
<tr>
<td>Coef Var</td>
<td>141</td>
<td>146</td>
<td>6</td>
<td></td>
</tr>
<tr>
<td>Cats 1 - 3</td>
<td>OX code</td>
<td>Au</td>
<td>Ag</td>
<td>SG</td>
</tr>
<tr>
<td>Number</td>
<td>2</td>
<td>4930</td>
<td>4930</td>
<td>4930</td>
</tr>
<tr>
<td>Mean</td>
<td>0.13</td>
<td>10.04</td>
<td>2.36</td>
<td></td>
</tr>
<tr>
<td>Maximum</td>
<td>2.63</td>
<td>125.4</td>
<td>2.63</td>
<td></td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0.5</td>
<td>1.89</td>
<td></td>
</tr>
<tr>
<td>Coef Var</td>
<td>116</td>
<td>130</td>
<td>4</td>
<td></td>
</tr>
</tbody>
</table>

Table 28. Summary of modelled Au, Ag and SG by oxide code.
Figure 59. East-west sectional view, 9046600N (UTM Grid) – Au model.
(colour changes at 0, 0.2, 0.5, 1.0, 2.0 & 5.0 g/t Au – grey, blue, green, orange, red & purple; pit shown as black surface)
Figure 60. East-west sectional view, 9046600N (UTM Grid) – Ag model.
(colour changes at 0, 10, 30, 60, 90 & 120 g/t Ag – grey, blue, green, orange, red & purple;
pit shown as upper black surface; black block outlines mark very low confidence blocks)
Table 29. Summary of Inferred Resource Estimates, by AuEq cut-offs, Zone A.

<table>
<thead>
<tr>
<th>Cut-off (AuEq)</th>
<th>Tonnes</th>
<th>AuEq (g/t)</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Oz AuEq (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.50</td>
<td>43,556,919</td>
<td>1.05</td>
<td>0.62</td>
<td>27.8</td>
<td>1.47</td>
</tr>
<tr>
<td>0.75</td>
<td>24,349,305</td>
<td>1.40</td>
<td>0.86</td>
<td>34.8</td>
<td>1.10</td>
</tr>
<tr>
<td>1.00</td>
<td>16,325,249</td>
<td>1.67</td>
<td>1.08</td>
<td>38.1</td>
<td>0.88</td>
</tr>
</tbody>
</table>

Table 30. Summary of Inferred Resource Estimates, by AuEq cut-offs and Oxidation Zone.

<table>
<thead>
<tr>
<th>Cut-off (AuEq)</th>
<th>Tonnes</th>
<th>AuEq (g/t)</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Oz AuEq (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.50</td>
<td>18,403,992</td>
<td>1.47</td>
<td>1.09</td>
<td>24.5</td>
<td>0.87</td>
</tr>
<tr>
<td>0.75</td>
<td>11,712,594</td>
<td>1.78</td>
<td>1.38</td>
<td>26.0</td>
<td>0.67</td>
</tr>
<tr>
<td>1.00</td>
<td>8,514,152</td>
<td>1.99</td>
<td>1.57</td>
<td>27.2</td>
<td>0.54</td>
</tr>
</tbody>
</table>
Table 31. Summary of Inferred Resource Estimates, by Au cut-offs and Oxidation Zone.

Check estimate

A separate estimate for Areas A and C of the Tujuh Bukit Project in SE Java was completed using Multi Indicator Kriged (“MIK”) whole-panel (“E-type”) estimation of an Au equivalent (“AuEq”) and also using MIK mine-recoverable estimates of Au and Ag in smaller Selective Mining Units (“SMUs”). Panels of 40 x 40 x 6m based on a UTM grid formed the basis of the estimation. Recoverable resources assumed SMUs of 10 x 10 x 6m. Estimation was conducted only on oxide and partially oxidised zone material for Au and Ag.

The results of this estimate are summarized in Table 32 (rows “E” & “L”).

Search radii for informing panels were 110 x 110 x 12m and horizontally oriented. An expansion factor of 0.3 was used making the effective search radius for Inferred Resources 140 x 140 x16m. A minimum of 8 samples from 2 octants was required for Inferred Resources.

For IK 14 variograms in each of 5 directions were constructed for Gold and Silver. Horizontal variograms showed poorly defined horizontal continuity in both Au and Ag and closer spaced drilling will be needed to better understand horizontal continuity. Down hole variograms indicated very good continuity of mineralisation with the 0.8 variance level being more than 12 meters.

It was noted during this work that:
- Closer spaced drilling will be needed to better define controls on horizontal continuity.
- It is probable that horizontal search radii will prove to be too large and mineralisation will have shorter horizontal ranges than currently used.
• It is also probable that a multi metal MIK recoverable resource model will improve estimation results when more data become available.

Other Checks

A check of the effect of cutting was made by completing estimates based on uncut grades (Table 32, J vs K). This resulted in only a 2-3% increase in gold grades and contained gold. The maximum increase was 3.3% in gold grade in the oxide zone. It is evident from an inspection of the occurrence of elevated gold and silver grades that there is clustering of data in the higher grade areas. The effect of de-clustering the data is shown in Table 32 (A & B).

Table 32 also shows the results of the effect of using variable SMU dimensions upon gold grade using a “global change of support”. This technique seeks to predict the distribution of grades, on a global basis, by transforming the original point distribution to a distribution based on blocks with variable dimensions.

The OK model reported in this work (e.g. J & K), of necessity, is based on blocks with the largest dimension of 40 x 40 x 6 metres. A mining operation may be expected to achieve higher grades assuming a smaller mining unit is employed. Figure 61 illustrates the increase in grade that results from using a smaller SMU. Ideally, the MIK-recoverable model (E in Table 32) that used a SMU of 10 x 10 x 6m should approximate results from the 10 x 10 x 3 SMU (C&D). The close agreement of the results suggests that improvements in grade of the order of 20 to 30% will result from a more selective operation than that assumed in the OK model. However, in order to confidently predict outcomes based on smaller SMUs, the drill spacing requires at least an 80 x 80 metres pattern over the whole of the mineralised area with an approximate 40 x 40 pattern required in portions requiring Measured and Indicated status. It is emphasized that the results provided in Table 32 rely on a number of assumptions and the most realistic model from a mining perspective will be an actual MIK-recoverable model based on an increased density of drilling.

<table>
<thead>
<tr>
<th>Details</th>
<th>Support</th>
<th>Cut-off Grade (Au)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>0.5</td>
</tr>
<tr>
<td>A</td>
<td>Data</td>
<td>2m composites</td>
</tr>
<tr>
<td>B</td>
<td>Data_DC</td>
<td>2m composites</td>
</tr>
<tr>
<td>C</td>
<td>GCOS</td>
<td>10 x 10 x 3m</td>
</tr>
<tr>
<td>D</td>
<td>GCOS-DC</td>
<td>10 x 10 x 3m</td>
</tr>
<tr>
<td>E</td>
<td>Model</td>
<td>10 x 10 x 6m (MIK)</td>
</tr>
<tr>
<td>F</td>
<td>GCOS</td>
<td>20 x 20 x 6m</td>
</tr>
<tr>
<td>G</td>
<td>GCOS-DC</td>
<td>20 x 20 x 6m</td>
</tr>
<tr>
<td>H</td>
<td>GCOS</td>
<td>40 x 40 x 6m</td>
</tr>
<tr>
<td>I</td>
<td>GCOS-DC</td>
<td>40 x 40 x 6m</td>
</tr>
<tr>
<td>J</td>
<td>Model</td>
<td>40 x 40 x 6m (OK Cut)</td>
</tr>
<tr>
<td>K</td>
<td>Model</td>
<td>40 x 40 x 6m (OK Uncut)</td>
</tr>
</tbody>
</table>
Table 32. Grades and effect of change of support.
(Data = 2m composites; DC = declustered; GCOS = global change of support using Indirect Lognormal model; OK = Ordinary Kriging; MIK = Multi Indicator Kriging; MIK-panel = panel average, “E-type” estimate; OK Cut model is based on corrected topography and slightly differs from results in Table 31).

Figure 61. Relationship of gold grade to selective mining unit dimensions.

Correlation of Au with Ag

The poor correlation of gold with silver (Figure 36 and Figure 37) and the tendency of Au-rich and Ag-rich zones to be spatially distinct (e.g. Figure 37) suggests that selection of cut-off grades on the basis of a combined Au-Ag equivalent may provide an over-optimistic view of the resource from a mining perspective. This will become clearer with increased drilling.

Proportion of Resource Within Preliminary Optimised Pit

Preliminary pit optimizing was undertaken by a mining consultant. Conclusions discussed here are based on these assumptions: US$900/oz metal price for Au, $11/oz for Ag, pit slope of 45 degrees, CIL recoveries of 90% for Au and 87% for Ag, mining costs of US$2-2.30/t, processing costs of US$11.1/t.
The resources were checked against the resulting pit shell (e.g. Au - Figure 59 and Ag - Figure). Almost the entire resource (97% of contained ounces) above 0.5 g/t Au occurs within the pit. Effectively all the resources above 0.75 and 1.00 g/t Au occur within the pit shell. These results only demonstrate that there is a good basis for a reasonable expectation that the mineralisation discovered qualify as resources. The position of the pit shell will be used to help devise future drilling programs.

It should be stressed that this preliminary study is not a financial study and does not define “Ore Reserves”. It is reported here only to demonstrate the requirements of JORC and NI43-101, that the resource has “reasonable prospects for economic extraction” are satisfied.

The Indonesian Forestry Law restricts non forestry activities within protection forests and prohibits mining using an open pit method in protection forest areas. Intrepid’s Alliance partner, PT IMN, is working with relevant Indonesian authorities to allow for a review of forest land status if the exploration activities support such a decision. The Zone A Resource falls within a protection forest area, and there are risks attendant upon the reclassification process.

20. OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information available at this time. It should be noted that previous exploration companies referred to the project that included the Tumpangpitu prospect as the Bukit Hijau Project. However, this name is considered to be too similar to that of the near-by Batu Hijau copper mine of Newmont and might create some confusion.

Baseline hydrological studies of water quality in local streams, wells and sea water are ongoing on an approximately quarterly basis. Samples are collected from identical locations by Intertek Caleb Brett and PT. IMN local personnel and submitted to Intertek in Jakarta for analysis.

Fieldwork for baseline Flora and Fauna studies have been completed.

No environmental or social baseline studies were compiled by previous workers in the area.

Dr Bruce Rohrlach’s contribution to the geological discussion in this report is acknowledged. His work has been extensively quoted and any original thinking on the genesis of the Tujuh Bukit mineralisation is from Bruce. Mr Damien Lulofs has made a major contribution to the QA/QC of this report. The work of Mr Sam Garrett is also acknowledged in identifying the potential of the property and for organising the geological team that commenced the 2007 drilling. Mr Malcom Norris made a substantial contribution to the document.
21. INTERPRETATION AND CONCLUSIONS

The drilling program met original objectives and resulted in Inferred Resource Estimates that are consistent with the previous drilling targets.

Data density is adequate within the understanding of Inferred Resources. The assay and geological data is considered reliable for the purposes of resource estimation as reported here.

There is some uncertainty regarding topography in parts of Zone A.

More work is required to better model the distribution of variable oxidation.

22. RECOMMENDATIONS

In the qualified person’s opinion, the character of the property is of sufficient merit to justify continued drilling.

The application of “Leachwell”-type CN assays on physical composites should be considered to help define ore-types and help assign metallurgical recoveries to future resource models. This work should be done in conjunction with a consultant metallurgist. Estimated costs are approximately $25/sample. A relationship between CIL-recoveries and Leachwell recoveries should first be established.

A more extensive topographic coverage is necessary to cover the flanks of the deposit in order to constrain resource estimates and pit optimization studies. This is estimated to cost US$25,000.

Core recovery measurements on a sample basis should be recorded. Matrix-matched standard reference materials should be generated from residue samples. “Off-the-shelf” standards are appropriate for the early stages of a project but as the project proceeds it is important that standards match the unknown sample pulps.

More time should be allowed for the interpretation of geology, oxidation and mineralised zones before resource estimation reporting deadlines are set by management.
23. REFERENCES


24. DATE AND SIGNATURES

The effective date of this report is 1st February, 2009.
CERTIFICATE OF QUALIFICATION

I, Phillip Hellman, FAIG, do hereby certify that:

1. I am a Director of:
   Hellman & Schofield Pty Ltd
   Suite 6, 3 Trelawney St,
   EASTWOOD  NSW  2119
   AUSTRALIA

2. I graduated with a BSc(Hons) degree in geology from University of Sydney in 1973. In addition I have obtained a PhD in geochemistry and petrology from Macquarie University in 1979 and a Diploma of Education from Sydney University in 1974.

3. I am a Fellow of the Australian Institute of Geoscientists

4. I have worked as a geologist for over 30 years since my graduation from university.

5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.


7. I visited the Property for three days from 20 to 22 November, 2007, and again for three days in October 2008.

8. I have had an involvement in the Property since June 2006. The nature of this involvement includes resource estimation and general consulting in relation to QA/QC, geological logging and database assembly.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical report misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
Dated 1st February, 2009

__________________________
Signature of Qualified Person

P. Hellman, FAIG PhD

__________________________
Name of Qualified Person
25. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

The Tujuh Bukit Project is not a development property, nor is it a property which is under mineral production. Thus no further information is furnished here.

26. ILLUSTRATIONS

All figures of relevance to this report have been inserted into the relevant sections above.